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Black Butte Copper Project

Feasibility Study (Johnny Lee Deposit) and Mineral Resource Estimate Update (Lowry Deposit) - Technical Report NI 43-101

SANDFIRE RESOURCES BLACK BUTTE COPPER AMERICA INC.

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1. SUMMARY

1.1 Introduction and Scope

Sandfire Resources America Inc. (SRA) is pleased to announce the results of a Feasibility Study (FS) for its 100% owned Black Butte Copper Project (the Project) in Meagher County, Montana.

The Report, which presents the results of the FS, was overseen by Deepak Malhotra of Resource Development Inc. and Pro Solv, LLC, with contributions from other consultants discussed in Section 2.3. This Report has been prepared in accordance with the requirements of National Instrument 43-101 (NI 43-101), "Standards of Disclosure for Mineral Projects", of the Canadian Securities Administrators (CSA) for lodgement on the CSA's "System for Electronic Document Analysis and Retrieval" (SEDAR). The opinions contained herein and effective October 19, 2020, are based on information collected by the various consultants and employees of SRA.

The FS Case builds on, and replaces, the Updated Preliminary Economic Assessment (PEA) published on July 12, 2013, and meets or exceeds all of the standards and obligations required under the Project's stringent 2020 Mine Operating Permit (MOP) issued by the Hard Rock Mining Bureau of the Montana Department of Environmental Quality (MDEQ). The Project considers the design and construction of an integrated processing facility for the treatment of ores from the Johnny Lee deposit. It encompasses geology, resource definition, mine design, metallurgical testing and process design, infrastructure requirements, and associated capital and operating cost estimation, leading to an assessment of the economic potential of the Project.

This Technical Report also includes an updated Mineral Resource for the Lowry deposit which is located approximately 2 km SE of the Johnny Lee deposit. The Lowry deposit Mineral Resources are not part of the Mineral Reserve.

It should be noted that there is a strong potential for the Project to deliver significantly greater tonnages from underground with future additional drilling at both the Johnny Lee and Lowry deposits. With additional drilling, there are potential opportunities at both deposits to convert Inferred Mineral Resources to Measured and Indicated Classifications. The conversion of Inferred Mineral Resources at the Lowry deposit is a future looking statement as the Lowry Mineral Resource will require additional drilling, modelling, economic analysis, and permitting in order to be upgraded to Indicated and Measured Mineral Resource Classification. The Lowry deposit also remains open in several directions. The Black Butte Project mineralized system within the district extends beyond the Johnny Lee and Lowry deposits and contains excellent exploration prospects for additional discoveries that could also significantly extend the life of the Project compared to the FS Case. The previous statement is a forward looking statement (see Section 2.5 for a description of cautionary notes and forward looking information).

1.1.1 Cautionary Statements

The purpose of this Technical Report (Report) is to present the FS findings of the Project. The contents of this Report reflect various technical and economic conditions at the time of writing. Given the nature of the mining business, these conditions can change significantly over relatively short periods of time. Consequently, actual results may be significantly more or less favourable.

Refer to Section 2.5 for more detail on Cautionary Notes and Forward Looking Information.

1.2 Property Description and Ownership

1.2.1 Property Description and Location

The Black Butte Copper Project consists of 3,223.03 hectares (ha) of fee simple lands under mineral lease by Sandfire Resources America (SRA), and 525 unpatented mining claims on US Forest Service lands (USFS), leased by SRA, totalling 4,036.74 ha. The Project is located in south-central Montana in Meagher County, 27 km north of White Sulphur Springs (Figure 1.2). This Technical Report includes the Johnny Lee and Lowry copper (Cu) deposits. The Johnny Lee Cu deposit, which forms the focus of this FS Technical Report, was discovered by a Joint Venture (JV) between Broken Hill Propriety Ltd. and Cominco American Inc. in 1985. The Lowry deposit, a similar style Cu deposit, is located approximately two km to the SE of the Johnny Lee deposit.

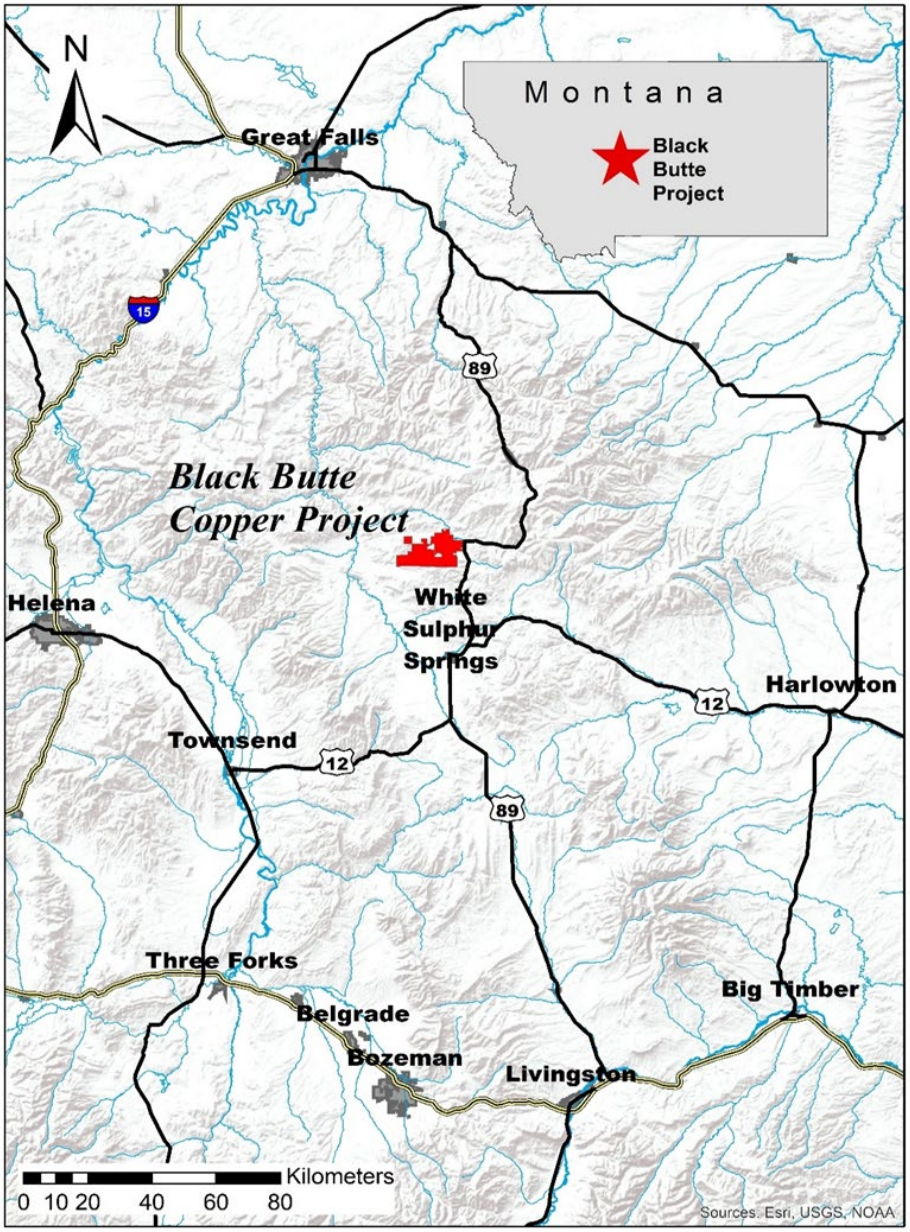

Figure 1.1 Project Location Map

1.2.2 Ownership

SRA has one surface lease and three mineral leases that collectively cover 100% of the Bar Z Tract in the Johnny Lee area. The leases were signed in May 2010 and are valid for a 30-year period, renewable for subsequent periods of 10 years each. Under the Surface Lease Agreement, the landowner has the option to transfer title to the lands necessary for construction and operating a mining and milling operation to SRA and be compensated for the land at the appraised value. Annual payments of $75,000 are to be made by SRA for the duration of the surface lease. Under the Mineral Lease Agreements, a Net Smelter Return royalty (NSR) of 5%, split equally between the three mineral lease holders, is payable on any mineral concentrates produced by SRA. Minimum advance annual royalty payments of $64,600 are payable to each of the three mineral lease holders by SRA. The NSR can be reduced to 2% by payment of $1,666,666 to each of the lease holders. Exercising the NSR reduction option eliminates further advance royalty payments.

There is one five acre parcel, surrounded by the Bar Z Tract, for which SRA has no mineral or surface agreement. This area is called the McGuire parcel. The NE corner of the Johnny Lee Upper Copper Zone (UCZ) projects to surface and outcrops in this area. The portion of the UCZ that occurs within the McGuire parcel is excluded from this FS and Mineral Resources.

The Short Tract Surface Lease and two mineral leases, cover 100% of the Short Tract in the Black Butte Copper Project area, including a portion of the Johnny Lee deposit and the entire Lowry deposit. These agreements are valid for a 30-year period, renewable for subsequent periods of 10 years each. The Short Tract Surface Lease, signed in April 2010, requires annual payments of $4,000 by SRA. All other terms of the surface lease are the same as those described above for the Bar Z Tract. The original mineral lease for the Short Tract was signed in May 2010 but has since been subdivided between the Short Family (15% share of mineral rights) and Holmstrom Sheep Creek Ranch Inc. (85% share of mineral rights). A revised Mineral Lease Agreement signed with the Short family in December 2014 requires an advance royalty payment of $24,120.30 per year plus an additional $10,000 per year in rent. The Mineral Lease Agreement signed with Holmstrom Sheep Creek Ranch Inc., requires an advance royalty payment of $136,681.70 per year. The combined mineral interest has a NSR of 5%, with an option to reduce this to 2% by payment of $5,000,000. Exercising the NSR reduction option eliminates further advance royalty payments.

1.3 History and Exploration

From 1975 to 1984, several companies undertook exploration in the Black Butte Copper Project area, but no significant mineralization was discovered. A JV between Broken Hill Proprietary Ltd. (BHP) and Cominco American Inc. (CAI) discovered the Johnny Lee deposit in 1985. BHP exited the JV in 1990 where after CAI continued to explore, discovered the Lowry zone, but abandoned the Project in 1995.

In 2010 Tintina Resources Inc., (Tintina) acquired the mineral rights formerly owned by CAI and, between 2010 to 2012, completed 168 diamond drillholes, primarily focused on Resource definition and Preliminary Economic Assessment (PEA) work at the Johnny Lee and Lowry deposits. Sandfire Resources NL acquired a majority shareholding in Tintina in 2013 and the company name was changed to SRA.

Subsequent to acquisition by SRA, an additional 87 diamond drillholes have been completed to support Resource definition.

Regional exploration work programs have been performed by the multitude of historic Project owners with work including soil sampling and geophysical surveys with limited regional exploration drilling. Much of the area overlying prospective stratigraphy in the Black Butte Copper Project area has been soil sampled at a line spacing that ranges from 200 to 800 m. Robust Cu in soil anomalies has been identified at the Johnny Lee deposit as well as other Cu prospects in the vicinity.

1.4 Geology and Mineralization

The Black Butte Cu deposits are hosted by Paleoproterozoic sedimentary rocks of the Newland Formation of the Belt Supergroup that were deposited in a portion of the Belt-Purcell basin referred to as the Helena Embayment. The Newland Formation comprises shale, carbonaceous shale, calcareous shale, dolomite, conglomerate, sedimentary breccia and bedded-pyrite massive sulphide (massive sulphide) that was deposited along the fault controlled, northern margin of the Helena Embayment.

The Black Butte Cu deposits are considered hybrid SEDEX (Sedimentary Exhalative Sulphide) – SSC (Sediment-hosted Stratabound Copper) deposits. It is likely that current fault architecture represents reactivated, basin rift and transform fault architecture, the intersections of which formed feeder structures for mineralizing hydrothermal fluids during basin evolution.

Cu mineralization at the Johnny Lee deposit is hosted by two massive sulphide units that occur at different stratigraphic levels within the sedimentary succession. These massive sulphide units are referred to as the Upper Sulphide Zone (USZ) and Lower Sulphide Zone (LSZ). Cu mineralization of economic significance in the USZ and LSZ is developed sub-parallel to bedding in the massive sulphide but is localized and is interpreted to have formed subsequent to the massive sulphide host units.

The zone of >0.25% Cu mineralization in the USZ is referred to as the Johnny Lee Upper Copper Zone (UCZ). The north-eastern part of the UCZ outcrops and is oxidized to a depth of 13 m below surface (most of this oxidized ore is located on the McGuire parcel which is excluded from the Black Butte Copper Project). Within the SRA mineral tenure, the UCZ occurs at depths of 40 to 210 m below surface.

The UCZ is truncated in the north by the Volcano Valley Fault Zone (VVFZ) and is transected and displaced by Fault 1 which divides the UCZ into Eastern and Western blocks. The UCZ Western block has plan dimensions of 1,000 m long by 440 m wide and a thickness that ranges from 4 to 45 m. The UCZ in the Eastern Block has plan dimensions of 950 m long by 140 to 185 m wide and a thickness that ranges from 5 to 37 m. Within the zone of >0.25% Cu mineralization, the UCZ contains one to six, stacked bedding, sub-parallel zones of >1.2% Cu mineralization that range in thickness from 1 to 12 m.

Primary Cu mineralization in the UCZ occurs as copper sulphides (chalcopyrite, tennantite, bornite and cuprian-siegenite) that texturally overprint early framboidal or melnikovite textured pyrite. Copper sulphide grain size and liberation characteristics in the UCZ are a function of both host lithofacies variation and the amount of progressive hydrothermal remobilization that has occurred. The northern part of the UCZ has generally had stronger hydrothermal remobilization and exhibits coarse grained, well-liberated copper sulphides and recrystallized euhedral pyrite. The southern part of the UCZ is generally characterized by fine grained copper sulphides that are interstitial to early pyrite. Heterogeneous hydrothermal activity, and preferential replacement of atoll-textured pyrite, in the southern part of the UCZ has created localized zones of coarse grained, well-liberated copper sulphide.

The intersection zone of Fault 1 and a bedding sub-parallel, brittle-ductile shear zone has resulted in enhanced fracturing and has allowed localized water ingress into the UCZ, below the local level of surficial oxidation. Primary copper sulphides in this zone have been supergene altered to chalcocite, covellite, and secondary bornite. Arsenic (As) in the UCZ occurs in tennantite, marcasite, and pyrite. The concentrations of these minerals in the UCZ is highly variable.

Within the Johnny Lee LSZ is a zone of Cu mineralization that is termed the Johnny Lee Lower Copper Zone (LCZ). The LCZ is situated below the UCZ, at depths of 340 to 520 m below surface. Mineralization occurs in three, >2.0% Cu lenses that occur at the same stratigraphic level, termed the West, Central and East Lenses. These lenses are truncated to the south by the VVFZ. The three LCZ lenses strike EW and dip to the south at 15° to 30°.The LCZ zones range in strike from 350 to 450 m, 45 to 250 m in down-dip extent and 1 to 15 m in thickness.

Cu mineralization in the LCZ is almost entirely hosted by chalcopyrite that is coarse-grained in comparison with the northern part of the UCZ. Trace amounts of tennantite are rarely observed, typically interstitial to early pyrite. As concentrations in the LCZ are considered minor.

At the Lowry deposit, economically significant Cu mineralization of greater than 1.2% Cu is hosted by a succession of sub-parallel, ferruginous sediment with interbedded shale and conglomerate lithofacies. Two distinct zones of > 1.2% Cu mineralization have been termed the Lowry Middle Copper Zone (LMCZ) and the Lowry Lower Copper Zone (LLCZ). Mineralization at the Lowry deposits occurs in both massive sulphide and the interlayered clastic sediment. Copper sulphide mineralization is predominantly chalcopyrite (8.3 – 17.0% in typical mineralized intersections) with minor tennantite (0.01 – 0.23%).

1.5 Mineral Resource Estimate

Summary Mineral Resources for the Johnny Lee deposit and the Lowry Deposit are presented in Table 1.1 and Table 1.2, respectively. The deposits have been classified as per the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) definitions into Measured, Indicated, and Inferred Mineral Resources.

Mineral Resource Category	Quantity (kt)	Cu (%)	Metal (Mlb)
Measured + Indicated Mineral Resources			
UCZ			
Measured	1,386	2.61	59.5
Indicated	8,271	2.32	330.1
Sub-Total M+I	**9,658**	**2.36**	**389.6**
LCZ			
Measured	583	5.68	68.6
Indicated	639	7.88	104.3
Sub-Total M+I	**1,222**	**6.83**	**173.0**
UCZ + LCZ			
Measured	1,969	3.52	128.2
Indicated	8,910	2.72	434.4
Total M+I	**10,880**	**2.86**	**562.5**
Inferred Mineral Resources			
UCZ	2,201	2.25	85.2
LCZ	478	5.92	62.3
Total Inferred	**2,679**	**2.91**	**147.5**

Source: SRK, 2019

Note:

- *The effective date for this Mineral Resource is October 15, 2019. All significant figures are rounded to reflect the relative accuracy of the estimates. Cu assay values were capped where appropriate;*
- *All Mineral Resources are located within land currently under control or lease to Sandfire Resources America Inc., and*
- *Mineral Resources are inclusive of Mineral Reserves and have reasonable prospects for eventual economic extraction. Inferred Mineral Resources have a high degree of uncertainty as to their economic and technical feasibility. It cannot beassumed that all or any part of an Inferred Mineral Resources can be upgraded to Measured or Indicated Mineral Resources;*
- *Mineral Resources are reported using a cut-off grade of 1.00% recoverable Cu based on long-term Cu price assumptions of $3.20/lb, variable recovery assumptions, mining costs, processing costs, G&A costs totalling $71/t;*
- *Metallurgical recovery has been estimated in the UCZ on a block basis with a consistent 94% Cu recovery applied to the LCZ;*
- *There are no known risks to the Johnny Lee Mineral Resources that could materially affect the potential development of the Mineral Resources other than that the Project is subject to legal challenges that are outlined in the Management Discussion and Analyses of the June 2020 Company Quarterly Report (also see Section 14.14 of this Technical Report);*
- *The Mineral Resources for the Johnny Lee Upper Copper Zone (UCZ) and Johnny Lee Lower Copper Zone (LCZ) were reviewed and approved by Erik Ronald, P.Geo (APGO#3050) of SRK Consulting (U.S.), Inc., a Qualified Person as defined by CIM.*

Table 1.1 Black Butte Copper Project Mineral Resource Estimate for the Johnny Lee Deposit as of October 15, 2019 – SRK Consulting (U.S.), Inc.

Category	Quantity (Mt)	Cu (%)	Total Metal (kt)
LMCZ			
Inferred	5.7	2.53	318.7
LLCZ			
Inferred	2.6	2.13	121.3
Combined LMCZ + LLCZ			
Inferred	8.3	2.41	440.0

Source: SRK, 2020

Note:

- *The effective date for this Mineral Resource is October 15, 2020. All significant figures are rounded to reflect the relative accuracy of the estimates. Cu assay values were capped where appropriate;*
- *Mineral Resources have reasonable prospects for eventual economic extraction. Inferred Mineral Resources have a high degree of uncertainty as to their economic and technical feasibility. It cannot be assumed that all or any part of an Inferred Mineral Resources can be upgraded to Measured or Indicated Mineral Resources;*
- *Metallurgical recovery of Cu has been assigned to the Lowry deposit using the mean recovery of 86% Cu;*
- *To demonstrate reasonable prospects for eventual economic extraction of Mineral Resources, a cut-off grade of 1.20% Cu based on metal recoverability assumptions, long-term Cu price assumptions of $3.20/lb, mining costs, processing costs, G&A costs totalling $71/t;*
- *There are potential permitting and other risks to the Lowry deposit that could materially affect the potential development of the Mineral Resources including those outlined in the Management Discussion and Analyses of the June 2020 Company Quarterly Report (see Section 14.14 of this Technical Report);*
- *All Mineral Resources are located within land currently under control or lease to Sandfire Resources America Inc., and*
- *The Mineral Resources for the Lowry Middle Copper Zone (LMCZ) and Lowry Lower Copper Zone (LLCZ) were reviewed and approved by Erik Ronald, P.Geo (APGO#3050) of SRK Consulting (U.S.), Inc., a Qualified Person as defined by CIM.*

Table 1.2 Black Butte Copper Project Mineral Resource Estimate for the Lowry deposit as of October 15, 2020 – SRK Consulting (U.S.), Inc.

1.6 Mineral Reserve Estimate

A NSR was calculated for each block in a block model based on metallurgical recovery, grade, and payability factors. Mine design shapes were created to reach a cut-off value of $70/t which was used for guidance to create detailed designs. All mining blocks then had dilution and recovery applied to them and were tested for economic viability. Summary Mineral Reserve statements for the Black Butte Copper Project are listed in Table 1.3 below.

Class	Diluted Tonnes	Cu Grade	Contained Cu Metal (t)
Proven	1,998,000	3.0%	60,700
Probable	6,804,000	2.4%	165,400
Total	**8,802,000**	**2.6%**	**226,100**

Notes:

1. *The Qualified Person for the Mineral Reserve estimate is Brad Evans MAusIMM CP (Mining).*
2. *Effective date: October 19, 2020. All Mineral Reserves have been estimated in accordance with Canadian Institute of Mining and Metallurgy and Petroleum (CIM) definitions, as required under NI 43-101.*
3. *Mineral Reserves were estimated using a $3.10 /lb Cu price and a NSR cut-off value of $70/t.*
4. *Tonnages are rounded to the nearest 1,000 t, metal grades are rounded to one decimal place. All units are metric.*
5. *Rounding as required by reporting guidelines may result in summation differences.*
6 *Average metallurgical recovery is 84%.*

Table 1.3 Mineral Reserve Estimate for the Johnny Lee deposit as of October 19, 2020 – Mining Plus

1.7 Mining Methods

The Black Butte Copper Project contains two zones, the Johnny Lee Upper Copper Zone (UCZ) and the Johnny Lee Lower Copper Zone (LCZ). Both of these zones are characterized as being high-grade, flat-lying and with relatively narrow widths. All deposits have anomalous Silver (Ag) and Cobalt (Co) mineralization. However Cu is the only economic product considered in the FS.

Geotechnical data was gathered from logging of the diamond drill core performed by SRA geologists as well as part of previous work by MDEng (Kalenchuk et al, 2015). Specific geotechnical holes were drilled along the projected main decline and one of the ventilation raises and logged by Mining Plus. Mining Plus in collaboration with SRA geologists undertook a Quality Assurance and Quality Control (QA/QC) audit of the data gathered that included training SRA geologists. Acoustic Televiewer and oriented core data were used to determine structural information. In addition to the data logging, multiple rock property tests were performed on different rock types including; Uniaxial Compressive Strength (UCS) testing, Triaxial Strength testing, Cerchar Abrasiveness testing and Slake Durability testing (Agapito Associates, 2019; Nazeri, 2019). Generally, the rock mass can be characterized as poor to good though with significant local variability. A prominent feature underground is the Volcano Valley Fault Zone (VVFZ), a very poor-quality rock mass sheared black shale and gouge with blocks of heavily veined dolostone and shale.

Ground support Underground (UG) will consist of a combination of rebar, friction bolts and dynamic bolts for tendon support and welded wire mesh with shotcrete depending on the ground conditions and the expected lifespan of the drift.

The Johnny Lee deposit will be accessed by a single main ramp driven from surface. The ramp dimensions will be 5 m wide by 5 m high and excavated with a flat back to maximize the stability of the flat dipping joint sets that are prevalent throughout the Project. The ramp will be excavated at a maximum gradient of -15% from the surface and pass to the east of the UCZ and then spiral down to the LCZ. Ventilation and secondary egress will be through three main ventilation raises. Figure 1.2 is an isometric view of the UG design looking from the SE.

All material handling will be by trackless UG equipment with 51 tonne (t) haul trucks hauling ore directly from stope areas to either a surface ore pad or the surface crusher.

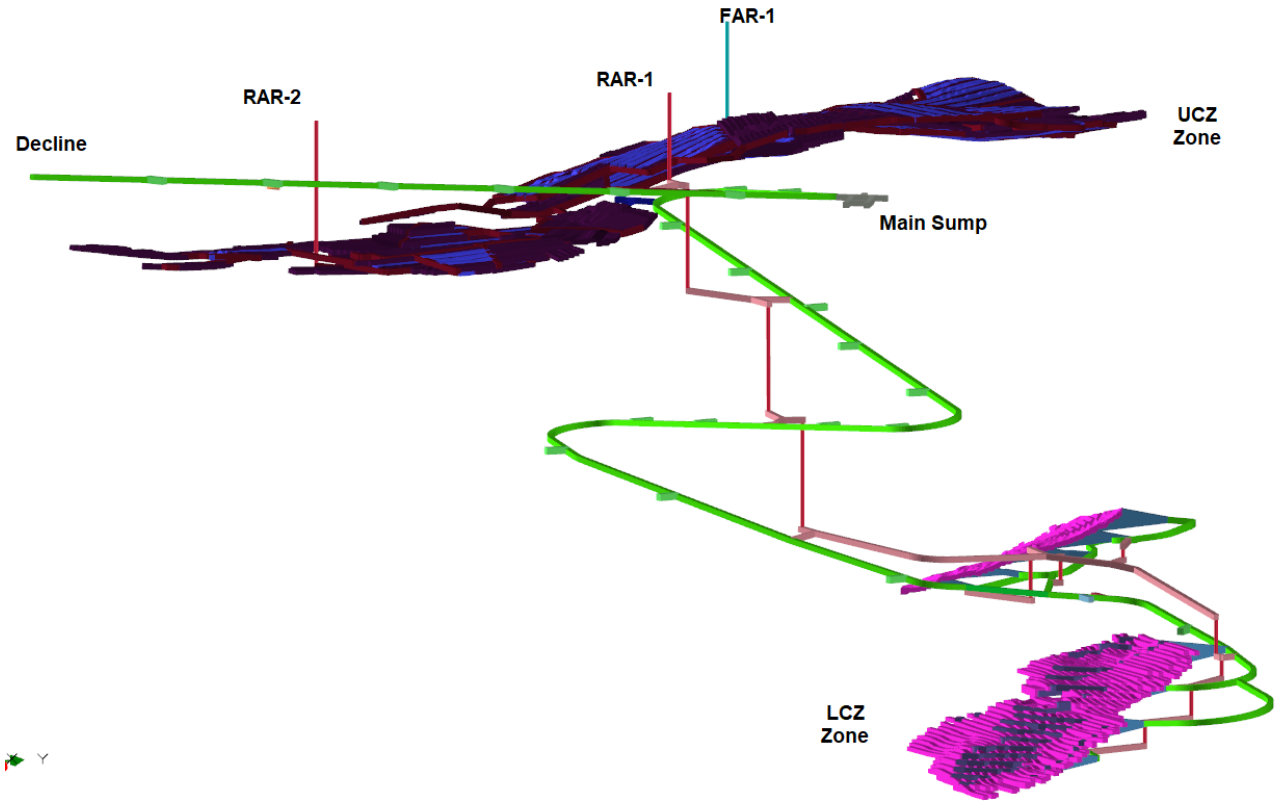

Figure 1.2 Isometric View UG Design from the SE

The mining method will be a combination of Drift and Fill (DF) and Cut and Fill (CF) depending on the height of the orebody. All openings will be completely backfilled with paste backfill to allow for the complete extraction of the orebody. In the UCZ where the orebody is wider, a Primary-Secondary-Tertiary method will be used, with the tertiary stopes being extracted through an unsupported slash retreat. Figure 1.3 shows the mining sequence for the UCZ. The LCZ will be extracted using CF methods with paste fill.

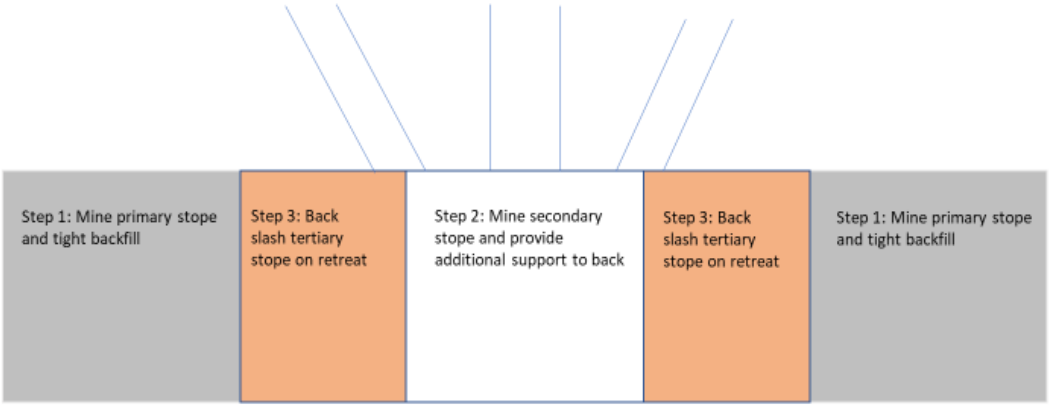

Figure 1.3 Upper Copper Zone Mining Sequence

Table 1.4 below shows the key mine physicals for the eight year Life Of Mine (LOM) for the FS Case.

	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10
Lateral Development (km)	2.3	4.9	17.1	14.6	15.5	15.5	12.5	11.1	12.0	2.2
Vertical Development (m)	80	795	309	0	0	0	0	0	0	0
Waste Tonnes (kt)	230	272	288	9	24	33	5	11	10	3
Ore Tonnes (kt)	0	197	1,203	1,199	1,201	1,198	1,203	1,200	1,181	220
Ore Mined Head Grade (% Cu)	1.2	1.9	2.8	3.3	2.9	2.7	2.3	2.1	2.1	1.8
Paste Backfill (km^3)	0	23	219	335	321	290	297	358	314	81

Table 1.4 Mine Physicals Summary

1.8 Mineral Processing and Metallurgical Test Work

Previous metallurgical test work programs undertaken by SRA indicated that production of a Cu concentrate from the LCZ by froth flotation recovered 93.3% to 96.6% of the Cu resulting in a concentrate grading 27.0% to 30.8% Cu. Tests on UCZ composites during the same test programs showed a wide range of Cu recoveries (61.9% to 91.2%) at concentrate grades of 18.5% to 24.5% Cu. Mineralogical investigation of UCZ metallurgical composites indicated that copper sulphide liberation was the primary metric that defined metallurgical performance.

Systematic mineralogical investigation of UCZ drill intercepts was undertaken to define the vertical and lateral variability in copper sulphide liberation throughout the entire UCZ. This study also allowed the geometry of the supergene alteration zone (at the intersection of Fault 1 and the brittle-ductile shear zone) to be resolved. The supergene altered zone comprises 2.2% of the total volume of the UCZ.

Based on the mineralogy derived geometallurgical model, 19 PQ diameter (85 mm) diamond drillholes were targeted to intersect the complete range of UCZ Cu liberation types. From these drillholes, 21 metallurgical composites were developed, including two composites from the supergene alteration zone.

Comprehensive batch rougher and cleaner flotation tests were completed on all 21 UCZ metallurgical composites to determine the optimum primary grind size, reagent suite, rougher regrind size and flowsheet for UCZ ore. Tests undertaken with site water showed no significant differences to those completed with laboratory tap water. Two rounds of Locked Cycle Tests (LCT's) were conducted, using a representative subset (seven to eight composites) of the UCZ composites using slightly different regrind sizes and different grinding media. Based on the test work the optimized flowsheet for the UCZ was developed:

- Primary Grind to 35 µm P_{80} using stainless-steel grinding media;
- Lime addition to Rougher circuit to maintain pH = 9.5;
- Rougher flotation using Aero 3477, mono-sodium phosphate and dextrin;
- Regrind of Rougher concentrate to 10 µm P_{80};
- Lime addition during Regrind to maintain Cleaner circuit pH = 9.5;
- Additional mono-sodium phosphate and dextrin added during Regrind;
- Three Stage Cleaner circuit with Cleaner Scavenger;
- Additional Aero 3477 added to Cleaner circuit; and
- Polyfroth W31 added to Cleaner circuit.

The LCT's on non-supergene altered composites, using the optimized flowsheet recovered 70.6% to 90.1% of the Cu into a concentrate grading 16.9% to 27.1% Cu. Locked cycle testing of a supergene altered UCZ composite recovered 69.8% of the Cu into a concentrate grading 14.1% Cu. A blend of the six non-supergene altered composites was used to create an UCZ Global Composite. Locked cycle testing of this composite recovered 81.6% Cu into a concentrate grading 24.4% Cu.

Given the amount of variability in non-supergene altered UCZ composites, the relationship between Cu recovery and categorized proportional geometallurgical core logging, comprehensive geochemistry and systematic mineralogy was evaluated in detail. Of these, mineragraphy defined copper sulphide liberation metrics showed the best correlation with recovery. The regression based formula below defines the relationship between variability batch test cleaner Cu recovery (from the 19, non-supergene altered composites) with five mineralogy derived metrics:

Variability Test Cu Cleaner Recovery = 94.144 + (0.10615(A+B)) + (-0.28667*(C+D)) + (-0.26708*E)*

- A =% Chalcopyrite Interlocked with Marcasite/Siegenite;
- B =% Chalcopyrite Interlocked with Gangue;
- C =% Chalcopyrite in Ternary Grains;
- D =% Chalcopyrite in Quaternary Grains;
- E =% Pyrite.

There is a robust linear correlation between the variability test cleaner Cu recoveries and the cleaner recoveries from the six LCT's on non-supergene altered UCZ composites, using the optimized UCZ flowsheet. This linear correlation is defined by:

*Locked Cycle Test Cu Cleaner Recovery = (0.6619 * Variability Test Cu Cleaner Recovery) + 31.231*

The formulae above were used to convert the mineragraphy metrics from 113 non-supergene altered UCZ mineralogy composites spaced throughout the UCZ (both laterally and vertically) into expected Cu recoveries. Inverse Distance Weighted Squared (IDW2) interpolation of these Cu recovery metrics has been used to create a Cu recovery model for the UCZ that has been integrated with the Mineral Resource model. Based on the process outlined above, estimated Cu recoveries for the UCZ range from 68.2% to 87.9%.

The process design concentrate values for treatment of UCZ ore were generated from weighted LCT results. The final concentrate conditions yielded a Cu grade of 23.5% and Cu recovery of 80.8%. These figures have been utilized in the process design criteria and mass balance.

The supergene altered zone has been assigned a Cu recovery estimate of 69.8% based on the LCT of the supergene altered composite.

A batch, single-stage cleaner flotation test on a LCZ composite, using the UCZ flowsheet, recovered 92.3% Cu to a concentrate grading 26.1% Cu. Locked cycle testing was undertaken using a blend of the UCZ Global Composite (76%) and the LCZ composite (24%). Cu in the feed was 93.2% recovered into a concentrate grading 21.5% Cu. The metallurgical balance indicated that there were no negative synergies between blending the two feed sources. Based on previous and recent test work, a global 93% Cu recovery has been assigned to the LCZ.

Analyses of the Cu concentrates from locked cycle testing of UCZ composites has reported potentially deleterious levels of As. There is no correlation between the As concentration of the feed composites and that in the concentrates as only certain As bearing minerals (primarily tennantite) preferentially deport to the concentrate. There is a strong linear correlation between the tennantite percentage of the feed, estimated using systematic mineragraphy, and the As levels in Cu concentrates from LCT's. This correlation is defined by the formula:

Locked Cycle Test Cleaner Concentrate As Grade (ppm) = (8048.4 * Tennantite%) + 3202.6

This formula has been used to convert the tennantite concentrations for the systematic mineralogy composites into expected As concentrations in Cu concentrate. IDW2 interpolation has been used to create an As in concentrate block model which has been integrated with the Cu recovery and Mineral Resource models. Based on the tennantite concentrations, As in UCZ concentrates is expected to range from 3,202 to 14,876 parts per million (ppm).

Based on analyses of the concentrate produced during locked cycle testing of a master LCZ composite a global As in concentrate value (230 ppm) has been assigned to LCZ ore.

1.9 Recovery Methods

Metallurgical test work indicates that the Cu in the UCZ and LCZ can be recovered to a concentrate by crushing, grinding, and froth flotation processes. The UCZ ore requires a fine primary grind (38 µm P_{80}) and a very fine regrind (10 µm P_{80}) of the rougher concentrate to achieve optimized recoveries. The LCZ ore does not require such fine grinds to achieve optimized recoveries. However, LCZ ore will be blended with UCZ ore in small volumes and treated using the process as optimized for UCZ ore. Metallurgical test work has demonstrated that there are no reductions in Cu recovery to concentrate from UCZ or LCZ ore by blending and processing the blend using the flowsheet optimized for UCZ ore.

1.10 Infrastructure

The layout and surface footprint of all aboveground infrastructure for the Black Butte Copper Project has been designed as part of the final approved Mine Operating Permit (MOP) application (Tintina Montana, 2017) and in a subsequent MOP updates and approvals including an October 26, 2018 MOP update letter (Zieg, 2018) submitted to and approved by the Montana Department of Environmental Quality (MDEQ). The aboveground infrastructure in the two aforementioned documents includes: access roads, site roads, mine portal, ventilation raises, processing plant, paste plant, a water treatment plant, two separate reclamation stockpiles, two separate soil stockpiles, a temporary construction stockpile, temporary waste rock storage pad, cemented tailings facility, process water pond, contact water pond, treated water storage pond, non-contact water reservoir, an alluvial underground (sub-surface) infiltration gallery, power lines, pipelines, workshops, store, offices and parking. There are sufficient on-site materials that will be sourced from the facility excavations to both construct the on-site facilities and to later utilize for mine closure and reclamation. Only minimal amounts of imported materials are required for construction of the mine access roads.

1.11 Environmental

The MOP application for the Black Butte Copper Project was first submitted to the MDEQ in December 2015. The application was designed to meet the requirements of the Montana Metal Mines Reclamation Act (MMRA) (Title 82, Chapter 4, Part 3, MCA) and the rules and regulations governing the act.

The final MOP application (revision 3) was submitted to the MDEQ in July 2017 and was declared complete and compliant with the MMRA by the MDEQ in August 2017. This resulted in the MDEQ issuing a draft operating permit in September 2017. Following this determination, the MDEQ retained an independent third-party environmental expert to complete a draft Environmental Impact Statement (EIS) for the Black Butte Copper Project. The draft EIS was released for Public Comment by the MDEQ in March 2019. The Final EIS document was issued by the MDEQ in March 2020 (MDEQ, 2020) and included one Agency Modified Alternative (AMA) that included additional UG backfilling of certain mining voids with low hydraulic conductivity cemented tailings paste in order to potentially achieve higher water quality standards in reclamation.

The MDEQ issued the Record of Decision (ROD) document (MDEQ, 2020a) for the MOP application in April 2020 that requires SRA to implement the AMA. The ROD document separates the Mine Development Construction activities into two Phases (Phase 1 and Phase 2) with five stipulations and allows SRA to pursue an incremental (i.e. phased) bonding approach. SRA is pursuing an incremental Bond 1 and Incremental Bond 2 that correspond to the Phase 1 and Phase 2 Development Construction activities described in the ROD. MDEQ agreed to the incremental bonding approach as defined by the Administrative Rules of Montana (ARM) 17.24.140 and on May 19, 2020 calculated the Incremental Bond 1 amount to be $4,653,348 (MDEQ, 2020b). SRA finalized the Phase 1 Project Bond obligation and the final Hard Rock Mining Operating Permit (MDEQ, 2020c) was subsequently issued to Tintina Montana Inc. on August 14, 2020 that allowed Phase 1 Development Construction activities to proceed.

At the date of publication of this document, the MDEQ is estimating the Incremental Bond 2 amount which is equivalent to the full Project bond amount. The full Project bond amount would be in place in case of a SRA default where the MDEQ would be responsible for reclamation and closure of the Project site using third party contractors.

In addition to the approved MOP, there are 27 other permits or plans that need to be approved by Federal, Montana State, or Meagher County authorities. These permits and plans cover: water quality, water rights, water supply, wetlands and streambed preservation, aquatics monitoring, dam safety, sewerage disposal, air quality, invasive vegetation, tribal communications, cultural resources, community impact, mining infrastructure, mining operations and emergency response. Work has been initiated on all but four of these permits/plans (which are largely administrative). To date, 13 permits/plans have been approved, three applications have been submitted and seven applications are in the process of being compiled.

1.12 Capital and Operating Costs

1.12.1 Capital Cost Estimates

The Project capital cost estimate developed for the FS is based upon an Engineer, Procure and Construction Management (EPCM) approach for the construction and commissioning of the Project facilities. This includes mine plant and infrastructure, the process plant and infrastructure, general mine infrastructure and roads for an eight-year LOM. The Project will require approximately two years of construction and four years of mine reclamation and closure.

A capital cost of $274.7 Million (M), including contingency, has been developed for the Project and includes all costs before the commencement of production. The capital costs have been estimated to a ±15% accuracy.

The capital costs are summarized in the table below.

Area	Capital Cost $M
Mining	$65.1
Site Infrastructure	$91.4
Mineral Processing	$72.7
Project Indirects (EPCM & Owner Costs)	$20.5
Contingencies (mine, process, infrastructure and indirects)	$25.0
Total Project	$274.7

Table 1.5 Total Project Capital Cost Summary

The capital costs have been developed from vendor budgetary supplied quotes and in-house cost estimation for the proposed plant design. The basis for the capital estimate includes:

- Engineering, Procurement and Construction for the surface infrastructure to support the mining operations;
- Engineering, Procurement and Construction for the process plant and surface infrastructure;
- Administration, stores and workshop buildings;
- Construction of a paste plant and cemented tailings storage facility;
- Construction of a water treatment facility and environmental water control pond and reservoirs;
- Construction of a new access road to site;
- Supply and construction of a high voltage overhead power transmission line;
- Construction of diesel fired emergency supply power station;
- Communications facilities to support the operations;
- Owned fixed plant and mobile surface plant;
- Initial fills for the plant; and
- Insurance and capital spares to support the Project.

1.12.2 Mining Operating Costs

Operating costs have been developed using the parameters specified in the process design criteria. The mining operating cost estimate has been developed on the basis of ore to the Run-Of-Mine (ROM) pad at the same rate as the processing plant name plate of 1,204,500 tonnes per annum (tpa). The operating cost estimate is $27.09M per annum or $23.35 /t of ore mined and supplied to the ROM.

1.12.3 Process Plant Operating Costs

Operating costs have been developed using the parameters specified in the process design criteria. Operating costs for the treatment plant have been estimated to an accuracy of ±15%. The costs are presented in United State dollars (USD$) and are based on prices obtained during the second Quarter of 2019 (2Q19) and exclude the VAT cost components.

The processing operating cost estimate has been developed on the basis of a process plant feed tonnage of 1,204,500 tpa. The LOM operating cost estimate is $49.97 M per annum or $41.49/t milled and includes concentrate ground freight (excluding sea freight), Administration costs, and processing costs. The processing cost (excluding concentrate ground freight and sea freight) is $29.43M per annum or $25.52/t milled for mining years 2 through 7 averaged.

1.13 Economic Analysis

The financial analyses were undertaken based on the following parameters:

- Mineral Reserves of 8.8 Mt averaging 2.6% Cu used in the financial model;
- The mining schedule and mining cost estimates were used for the financial model;
- Copper price of $3.20/lb;
- LOM Copper recovery of 83.8% was used in the financial model;
- The economic model used a treatment charge of $90/t and a refining charge of 9.0 cents/lb. throughout LOM;
- An operating cost of $64.80/t ($1.41/lb. copper metal produced) for Mine to Mill was used for the model;
- A capital cost of $274.7 million was used;
- Royalty payments were also incorporated into the model.

The financial results indicated the following:

- The project has a pre-tax NPV of $124.9 M and a post-tax NPV of $77.6 M using a 5% discount rate;
- The project has a pre-tax IRR of $17% and a post-tax IRR of 13%;
- The projected cash flow will allow for the project capital to be paid back in approximately three years;
- Sensitivity analyses indicate that the project NPV is most sensitive to copper price, recovery, and feed grade.

1.14 Project Implementation

The Capital Cost Estimate (CAPEX) was developed on the basis that the Project would be implemented under an EPCM methodology. Under this methodology, SRA will enter into a head contract with a suitably experienced contractor to carry out the following:

- Detailed engineering;
- Procurement, fabrication and delivery to site of all plant, equipment and materials;
- Construction of the facilities;
- Pre, dry and wet commissioning of the facilities, where appropriate;
- Ore commissioning assistance of the processing plant facilities by the contractor, assisted by the owner's operations team.

Equipment and fabricated items will be sourced from local, interstate and international suppliers. Typically greater than 90% percent of the total capital expenditure for equipment and fabricated items will be sourced within the United States.

All work shall be designed and performed in accordance with relevant government, environmental and health and safety regulations. The contractor will commit to a 'no harm, no incident' culture utilizing and employing their own safety management system back-to-back with SRA's safety systems and policies.

The critical path for the Project implementation is the specification, procurement, installation and commissioning of the SAG and Ball mills. The overall schedule for the Project is 92 weeks (contract award to demobilisation), with 62 weeks on site.

Construction labour working on the processing plant and mine infrastructure will peak at approximately 185 personnel including all construction management, supervision and equipment. Personnel levels for the owner's team, mining contract and bulk earthworks construction (roads, cemented tailings facility and ponds) have not been included in these construction numbers.

1.15 Risks and Recommendations

Risk analyses was undertaken for the Project and categorized into low, medium and high risk. Low and medium risk items can be easily mitigated. However, high risk items need dedicated effort to minimize the negative impact on the Project.

The high-risk items identified in the study are litigation suits in the courts, geotechnical and dewatering underground, water supply and management, recovery under performance and no concentrate sales contract.

Several opportunities for improving economics of the Project were also identified. These included additional drilling for increasing resources and converting Inferred resources to Measured and Indicated classifications, geometallurgy testing for improved copper recovery and reduction in operating costs by simplifying transportation of concentrate and optimizing tailings treatment. The potential for recovering cobalt as a by-product can also improve project economics.

1.16 Interpretations and Conclusions

It is the opinion of the QPs that the FS is technically and economically sound under the present conditions. Brief remarks are summarized below for the various disciplines.

1.16.1 Property Description and Ownership

The surface and mineral rights under lease to SRA are of sufficient size to allow development of a mining operation and exploitation of the Johnny Lee deposit. While SRK Consulting has not undertaken a review of the property ownership, an independent review of the surface and mineral leases was undertaken by Crowley Fleck Attorneys of Billings, MT, dated September 25, 2019 (Cook, 2019; 2019a). This review concluded that the mineral and surface leases for the area within the MOP boundary are in good standing.

1.16.2 Geology and Mineralization

The regional and local geology of the Black Butte Copper Project is well understood, and a robust 3-D structural and stratigraphic model of the Johnny Lee deposit area has been constructed. Within the framework of that model, geological models of the UCZ and LCZ mineralization have been developed. The QP of Mineral Resources considers that the constraints used to create the geological models are appropriate and that the geologic models are suitable for Mineral Resource estimation and reporting.

Extensive systematic mineragraphy of drill hole composites from the Johnny Lee deposit has been undertaken to determine the composition, texture grain size and associations of copper sulphide minerals that host the Cu mineralization in the Johnny Lee deposit. The work has been utilized in metallurgical recovery studies.

Grade modelling for the Lowry deposit area provides sufficient detail for the reporting of Inferred Mineral Resources. Each zone of the LMCZ and LLCZ has been well modelled in respect to Cu grade with individual lenses delineated. Initial mineralogical work has been conducted on the Lowry deposit in sufficient detail to provide information on expected recoveries of Cu at the Lowry deposit. It is the opinion of the QP for Mineral Resources that the Lowry model is appropriate for the current level of study and the estimation and reporting of Mineral Resources.

1.16.3 *Status of Exploration, Development and Operations*

A significant amount of Mineral Resource definition diamond drilling has been completed at the Johnny Lee deposit resulting in drill spacing of nominal 50 m by 50 m across the deposit. The limits of the mineralization are well defined and the drill hole spacing within those limits is sufficient to allow classification of the Mineral Resource estimate into Measured, Indicated, and Inferred categories. Additional drilling at tighter spacing of the LCZ and the north-eastern portion of the UCZ could allow upgrading of mineralization in the Inferred category to Indicated and from Indicated to Measured.

Additional infill drilling to convert Inferred Mineral Resources to Measured and Indicated classifications, exploration drilling, and updates to the existing block model in the future could add life to the Project. The QP for Mineral Resources recommends this work to be conducted in the future.

Several exploration targets have also been identified within the current SRA tenement package that have potential to increase the Mineral Resource base with further work.

1.16.4 *Mineral Processing and Metallurgical Testing*

Sufficient metallurgical test work has been completed to demonstrate that Cu recovery of Johnny Lee ore into a Cu concentrate by froth flotation is the preferred method for mineral processing. The LCZ ore is expected to deliver fairly homogenous Cu recoveries averaging 93%. Due to significant variability in copper sulphide liberation characteristics, the UCZ ore is estimated to return variable Cu recoveries ranging from 68.2% to 87.9%. Systematic mineragraphy, correlated with metallurgical test results, has been used to develop a Cu recovery block model for the UCZ that has been integrated with the Mineral Resource estimate to create a recoverable Cu grade field in the block model.

UCZ ore is expected to produce a Cu concentrate containing potentially deleterious levels of As (3,202 to 14,876 ppm). The amount of As that reports to the Cu concentrate is proportional to the amount of tennantite in the feed. Systematic mineragraphy has been correlated with metallurgical test results to create an As in concentrate block model for the UCZ that has been integrated with the Resource block model. The LCZ contains low levels of both As and tennantite. A global As in concentrate grade of 230 ppm has been assigned to the LCZ based on metallurgical test results. A metallurgical Cu recovery of 84% for the Johnny Lee deposit was calculated based on a weighted average of the recovery in the block model of the material scheduled to be mined.

Mineralogical studies of tailings from metallurgical test work has shown fine grained, liberated copper sulphide is reporting to rougher and cleaner tailings. Limited test work to simulate the recovery impact of a Jameson cell was inconclusive. There is potential to increase Cu recovery by capturing liberated fine copper sulphide.

Limited metallurgical test work on blended UCZ and LCZ ore indicated that there are potential positive recovery synergies to be realized from ore blends that require further evaluation.

1.16.5 Mineral Resource Estimate

It is the opinion of SRK that the Mineral Resource estimate has been produced in accordance with Canadian National Instrument 43-101 (NI 43-101) and CIM guidelines.

1.16.6 Mineral Reserve Estimate

It is the opinion of Mining Plus that the Mineral Reserve estimate has been produced at levels appropriate for a FS and in accordance with NI 43-101 reporting.

1.16.7 Mining Methods

Work completed for the FS indicates that DF UG mining is the most appropriate mining method for the Johnny Lee deposit.

1.16.8 Recovery Methods

The FS metallurgical test program has been used to develop a process flowsheet. The flowsheet records a Recovery process that involves: Crushing, Grinding to 38 µm P_{80}, Rougher Flotation Circuit, Regrind of Rougher Concentrate to 10 µm P_{80}, Three stage Cleaner Flotation Circuit and Filtering of the Cleaner concentrate to produce a saleable Cu concentrate.

1.16.9 Project Infrastructure

The location and footprint of all surface infrastructure, including the mine portal, has been determined and included in the 2017 MOP application to the MDEQ. It is the opinion of GR Engineering Services and the QP, Deepak Malhotra, that the Project infrastructure designs have been prepared to a FS level and that there are sufficient on-site materials available to both construct the facilities and at the end of mining and milling operations to utilize for mine closure. Design and costing of all surface and UG infrastructure has been completed to a FS level.

1.16.10 Environmental Studies and Permitting

Extensive environmental monitoring and base-line studies were undertaken as part of the MOP application (Tintina Montana, 2017) and continue to be updated on a systematic basis. The operation plan detailed in the MOP application was designed to have no impact on the Sheep Creek drainage system and to completely rehabilitate the mine area upon closure. The MDEQ retained an independent environmental consulting company to complete a draft EIS in March 2019 and a subsequent Final EIS in March 2020. The Final EIS document (MDEQ, 2020) includes one AMA that identifies additional UG backfilling of certain mining voids with low hydraulic conductivity cemented tailings paste in order to potentially achieve higher

water quality standards in reclamation. The MDEQ issued the ROD document (MDEQ, 2020a) for the MOP application in April 2020 that requires SRA to implement the AMA.

In addition to a positive ROD, there are 27 additional permits required, or plans to be approved, prior to undertaking all proposed mining and development activity.

Intera Incorporated has reviewed the environmental studies, permitting applications, and planned mine closure activities for the Project included in the FS and notes that they meet or exceed all State standards.

1.16.11 Capital and Operating Costs

It is the opinion of the primary QP of the FS Technical Report with help from the QP of Mineral Reserves and Mining Methods and GR Engineering Services that the capital and operating costs have been produced at levels appropriate for a FS and in accordance with NI 43-101 reporting.

An opportunity exists to reduce the paste cement and slag mixture percentage and hence further reduce the operating costs. Paste test work has indicated that a lower percentage of binder from the 2% Portand cement / 2% blast furnace slag can be applied at suitable strengths. This will allow operations to trim the binder once production has started to find the optimum percentage of each.

Simplification of the logistics cycle for concentrate delivery to port is another area that can be further refined. At present a combination of both trucking and rail of the concentrate in containers to the port has been estimated at $105.21/wet concentrate tonne. Further refinement and contracts with rail services could significantly reduce this component of the operating cost.

1.16.12 Economic Analysis

It is the opinion of the primary QP of the FS Technical Report and GR Engineering Services that the Economic Analysis has been produced at levels appropriate for a FS and in accordance with NI 43-101 reporting.

1.16.13 Mine Closure

Mine closure is estimated to take 4 years with tasks following the plans outlined in the MOP Application. Mine closure costs, excluding the AMA tasks and costs, have been estimated by SRK Consulting at $24.3M (McIntosh, 2020). Additional costs for implementing the AMA tasks (currently in progress at the time of this Report date) will need to be added to this total.

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2. INTRODUCTION

2.1 Terms of Reference

This Technical Report (Report) was prepared for the Black Butte Copper Project to the standard of the Canadian National Instrument 43-101 "Standards of Disclosure for Mineral Projects" to summarize the findings of a Feasibility Study (FS or Study) covering the Black Butte Copper Project located in south-central Montana.

The quality of information, conclusions, and estimates contained herein is consistent with the level of effort involved in GRES's services, based on: i) information available at the time of preparation, ii) data supplied by outside sources, and iii) the assumptions, conditions, and qualifications set forth in this Report. This Report is intended for use by SRA subject to the terms and conditions of its contract with GRES and relevant securities legislation. The contract permits SRA to file this Report as a Technical Report with Canadian securities regulatory authorities pursuant to NI 43-101, Standards of Disclosure for Mineral Projects. Except for the purposes legislated under provincial securities law, any other uses of this Report by any third party is at that party's sole risk. The responsibility for this disclosure remains with SRA. The user of this document should ensure that this is the most recent Technical Report for the property as it is not valid if a new Technical Report has been issued.

2.2 Purpose of Report

The purpose of this Report is to present the technical and economic evaluation and viability of the Black Butte Copper Project at a detailed level, taking into account the updated Mineral Resources and Mineral Reserves and the life of mine capital and operating costs.

The work covered in this Report relates to exploration, drilling, Mineral Resource estimation, mining design, Mineral Reserve estimation, metallurgical test work, and process and infrastructure design undertaken in the period between 2011 and October 2020.

2.3 Participants and Inspections of the Property

Personnel	Company	Expertise	Date(s) of Visit	Details of Inspection
Erik Ronald	SRK Consulting (U.S), Inc.	Geology, Exploration, QAQC, and Mineral Resources	November 6, 2018	Site visit by multiple consultants reviewing the Project.
Brad Evans	Mining Plus	Mineral Reserves and Mining Methods	November 6, 2018	Site visit by multiple consultants reviewing the Project.
Deepak Malhotra	Resource Development Inc.	Metallurgy, Infrastructure, and Mineral Economics (Capital Costs and Operating Costs)	November 6, 2018	Site visit by multiple consultants reviewing the Project.
Patrick Williamson	Intera Incorporated	Environmental Studies, Permitting, and Mine Closure	November 6, 2018	Site visit by multiple consultants reviewing the Project.

Table 2.1 Participants and Inspections

2.4 Report Preparation

This Report was prepared by GRES and overseen by the primary author Deepak Malhotra, with further contributions by external consultants and SRA personnel. Specifically, the Report has drawn on the following input resources:

- SRK Consulting – Geology and Mineral Resources;
- Mining Plus – Mining methods and mine plans, geotechnical assessment, mining infrastructure, Mineral Reserves, mining capital and mining operating costs;
- GR Engineering Services Limited (GRES) – recovery methods, process plant design, project and administration infrastructure, capital and operating costs, and project execution;
- Base Metal Laboratories (BML) – 2018-2019 Metallurgical program for the deposit;
- Intera Corporated – Environmental studies, permitting, and mine closure plans;
- Paterson & Cooke USA Ltd. – 2019 Paste fill test work;
- Tierra Group International Ltd. – Reviews of the cemented tailings facility and operating plan, and waste and water management;
- Sundt Construction Inc. – Regional Construction pricing estimates, construction labor rates and site installation rates
- Pi Kappa Enterprises, LLC – Infrastructure and waste and water management.
- Peter Farbish, Mining Consultant, helped design the financial model.

Further contributions were generated by the staff of Sandfire Resources Ltd.

2.5 Cautionary Notes and Forward Looking Information

The contents of this Report reflect various technical and economic conditions at the time of writing. Given the nature of the mining industry, these conditions can change significantly over relatively short periods of time. Consequently, actual results may be significantly more or less favourable.

This Report may include technical information that requires calculations to derive subtotals, totals, and weighted averages. Such calculations inherently involve a degree of rounding and consequently introduce a margin of error. Where these occur, SRA does not consider them to be material.

Certain statements contained in this Report constitute forward looking information, future oriented financial information, or financial outlooks (collectively, "forward looking information"). Forward looking information is considered here to be within the meaning of Canadian securities laws. Forward looking information often relates to statements concerning SRA's future outlook and anticipated events or results and, in some cases, can be identified by terminology such as "may", "will", "could", "should", "expect", "plan", "anticipate", "believe", "intend", "estimate", "projects", "predict", "potential", "continue" or other similar expressions concerning matters that are not historical facts. Statements of historical fact are not considered forward looking information.

2.6 Qualified Persons and Sources of Information and References

The Consultants preparing this Technical Report are specialists in the fields of geology, exploration, Mineral Resource and Mineral Reserve estimation and classification, underground mining, geotechnical, environmental, permitting, metallurgical testing, mineral processing, processing design, capital and operating cost estimation, and mineral economics.

None of the Consultants or any associates employed in the preparation of this Report has any beneficial interest in SRA. The Consultants are not insiders, associates, or affiliates of SRA. The results of this Technical Report are not dependent upon any prior agreements concerning the conclusions to be reached, nor are there any undisclosed understandings concerning any future business dealings between SRA and the Consultants. The Consultants are being paid a fee for their work in accordance with normal professional consulting practice.

The following individuals, by virtue of their education, experience and professional association, are considered Qualified Persons (QP) as defined in the NI 43-101 standard, for this Report, and are members in good standing of appropriate professional institutions. QP certificates of authors are provided in Section 29 of the Report. The QP's are responsible for specific sections as follows:

- Deepak Malhotra, Resource Development, Inc. is the primary author of the Technical Report and is the QP responsible for Mineral Processing and Metallurgical Testing in Section 13, Recovery Methods in Section 17, Project Infrastructure in Section 18, Market Studies and Contracts in Section 19, Capital and Operating Costs in Section 21, Economic Analysis in Section 22, Other Relevant Data and Information in Section 24, and portions of Sections 1, 2, 3, 25 and 26 summarized therefrom, of this Technical Report.
- Erik Ronald, Principal Consultant, SRK Consulting (US), Inc. is the QP responsible for Geology and Mineral Resources Sections 4 through 12 and 14, 23, and portions of Sections 1, 2, 3, 25 and 26 summarized therefrom, of this Technical Report.
- Brad Evans, Mining Plus, is the QP responsible for Mineral Reserves and Mining Methods in Sections 15 and 16, respectively, and portions of Sections 1, 2, 3, 25, and 26 summarized therefrom, of this Technical Report.
- Patrick Williamson, Principal Hydrogeochemist, Intera Incorporated, is the QP responsible for Environmental Studies, Permitting, and Mine Closure in Section 20 and portions of Sections 1, 2, 3, 25, and 26 summarized therefrom, of this Technical Report.

2.7 Units of Measure

The metric system has been used throughout this Report. Tonnes are metric of 1,000 kg, or 2,204.6 lb.

All currency is in US dollars (USD$) unless otherwise stated.

2.8 Effective Dates

The effective date of this Report is October 19, 2020.

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3. RELIANCE ON OTHER EXPERTS

3.1 Exploration and Mining Concession Tenure

The opinion of the Qualified Person (QP) for Mineral Resources contained herein is based on information provided by Sandfire Resources America Inc. (SRA) throughout the course of the investigations and study. Erik Ronald, P.Geo of SRK Consulting (US), Inc. (SRK) acting as the QP for Mineral Resources has relied upon the work of other consultants in the Project area in support of this Technical Report for the areas of exploration and mining concession tenure. Summary information related to the exploration and mining rights is presented in Section 4 of this Technical Report.

The consultants used their experience to determine if the information from previous reports was suitable for inclusion in this Technical Report and adjusted information that required amending. This Report includes technical information which required subsequent calculations to derive subtotals, totals, and weighted averages. Such calculations inherently involve a degree of rounding and consequently introduce a margin of error. Where these occur, the QP for Mineral Resources does not consider them to be material.

The QP of Mineral Resources has entirely relied on the independent land attorneys Crowley Fleck, PLLP of Billings, MT for their opinion on the good standing of SRA's necessary mineral interests, mining concession tenure, and water rights required for the reporting of Mineral Resources (Crowley Fleck, 2020). This reliance applies to the calculation and classification of Mineral Resources included in Section 14: Mineral Resources Estimates. Based upon the reliance on Crowley Fleck PLLP, it is the opinion of the QP of Mineral Resources that the mineral rights under the control of SRA are acceptable for the reporting of Mineral Resources.

3.2 Surface Rights

The QP for Mineral Resources has relied on the independent land attorneys Crowley Fleck, PLLP of Billings, MT for their opinion on the good standing of SRA's necessary surface use agreements, and water rights for the Project. Additionally, all documentation and details of surface rights have been provided to SRK for QP review by SRA. Based on the reliance of other experts, it is the opinion of the QP for Mineral Resources that all required surface rights necessary for the reporting of Mineral Resources is acceptable. The details of SRA's surface rights are presented in Section 4 of this Technical Report.

The QP for Mineral Resources has relied on SRA for the Project property accessibility, climate, local resources, infrastructure, and physiography as described in Section 5. SRA staff and consultants have collected and summarized the data reported in Section 5. This reliance applies to the classification of Mineral Resources included in Section 14: Mineral Resources Estimates.

3.3 Permitting

The QP responsible for permitting, Patrick Williamson of Intera Incorporated, has relied on information presented in Section 20.4 regarding permits or required compliance plans or acts provided by SRA in the form of a summary table (Table 20.10). The QP for Section 20 has not independently verified the status, completeness or accuracy of the permit applications or any issued permits or required compliance plans or acts. Information pertaining to permits was generated by the following experts:

- Ed Surbrugg of Tetra Tech, Inc. helped manage the environmental permitting process for the Final EIS and Record of Decision;
- Greg Bryce of Hydrometrics managed the permitting process for surface and groundwater including water rights;
- Staff of SRA.

All information from other experts used to develop Section 20.4 is referenced in this Technical Report.

3.4 Social and Environmental Impacts

The QP responsible for Section 20 (Environmental Studies, Permitting and Social or Community Impact), Patrick Williamson, has relied in good faith on information provided by other experts retained by SRA regarding potential Social and Environmental impacts. The information, conclusions, opinions, and estimates are based on information prepared for the revised MOP as well as public domain information (e.g. US Geological Survey stream flow data). In preparing Section 20, the QP relied in good faith upon opinions, statements, or reports that were prepared by other experts and provided by SRA concerning environmental, social and community matters. In relying upon the work of others, the QP took reasonable steps to independently verify the potential for environmental liabilities and impacts associated with the Black Butte Copper Project. This involved a review of all the technical reports cited in Sections 20.2, 20.3 and Subsections 20.8.1 - 20.8.3 as sources of information. It is the opinion of the QP that the baseline characterization studies appear to have been performed in accordance with industry best practices and are acceptable.

The authors of reports used to prepare the Social and Environmental Subsections of Section 20 of the FS include:

- Greg Bryce of Hydrometrics for much of the water resource characterization (Section 20.2.1), water balance (Sections 20.2.2 and 20.8), water permits (MPDES) (Section 20.4), water rights permitting (Section 20.8.1), groundwater modelling (Section 20.2.1), and the design/implementation of the hydrogeological drilling and test program (Section 20.2.1);

- Robert Kimball of Wood Environment & Infrastructure Solutions, Inc. for the design of the water treatment plant (Section 20.8.3) and confirmation of the mine closure water treatment and brine disposal plans and costs;
- Jon Hall of AQ2 Pty Ltd for design considerations for the hydrogeologic characterization program (Section 20.2.1);
- Lisa Kirk of Enviromin Inc. completed much of the waste rock geochemical characterization and water quality modelling for operations (Section 20.2.4), and mine closure (Section 20.10.7);
- Numerous consultants from WESTECH Environmental, including (Section 20.2):
  - Ken Scow for vegetation studies;
  - Corey Baker for the soil study;
  - John Beaver, Ken Scow, Lisa Larsen, Dean Culwell, Jessica Allewalt, and Nancy Scow for the wetland delineations;
  - Patrick Farmer, Ken Scow, Jessica Allewalt, John Beaver, Drake Barton, Dean Culwell, and Corey Baker for the Terrestrial Wildlife Resources evaluation;
- David Stagliano of Montana Biological Survey for the aquatic surveys (Section 20.2.2);
- Lynn Peterson, John Mueller, and Kirk Miller from Tetra Tech, Inc. for the Cultural Resource Inventory study (Section 20.2.3);
- Sean Connolly from Big Sky Acoustics, LLC for the noise survey and assessment (Section 20.2.1);
- Steve Heck from Bison Engineering and Brendan Worrall from Knight Piesold for meteorological data (Sections 20.2.1 and 20.8.2);

All information from other experts used to develop Sections 20.2 and 20.3 and Subsections 20.8.1 through 20.8.3 is cited in the body of the Report and referenced at the end the Technical Report.

3.5 Taxation Information

The QP for Mineral Resources has relied on SRA for all taxation related matters.

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4. PROPERTY DESCRIPTION AND LOCATION

4.1 Location

The Black Butte Copper Project is located within Meagher County, Montana, USA, approximately 27 kilometres (km) north of the town of White Sulphur Springs (Figure 4.1). US Highway 89 is located one kilometre east of the Lowry deposit and three kilometres east of the Johnny Lee deposit. (Figure 4.2)

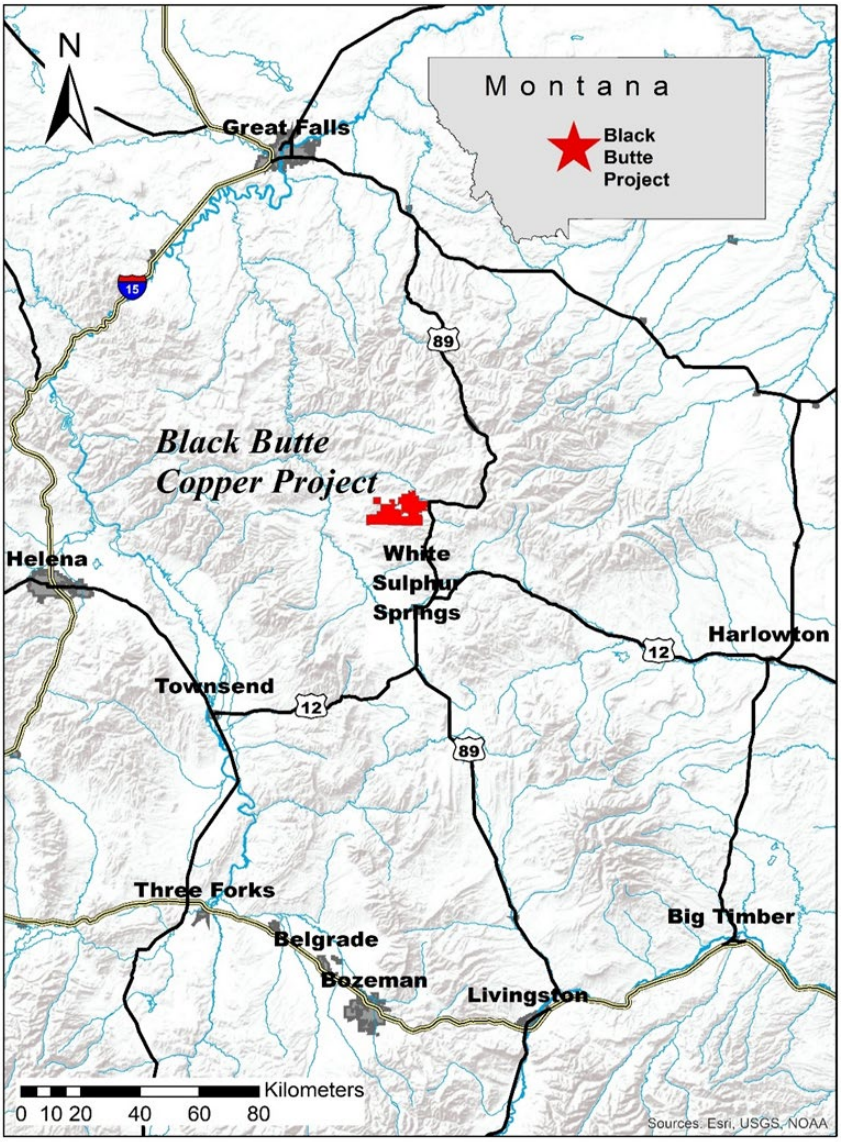

Figure 4.1 Regional map showing the property location in relation to nearby settlements and roads

4.2 Mineral Titles

The Black Butte Copper Project Property consists of approximately 3,223.03 ha (7,964.28 acres) of fee simple lands under mineral lease by SRA through Tintina Montana Inc. (collectively referred to as SRA), and 525 unpatented mining claims on USFS lands covering approximately 4,036.74 ha (9,975 acres) (Table 4.1, Figure 4.2). The Project's land holdings are within Sections 19, 29, 30, 31, and 32 of Township 12 North, Range 7 East; Sections 23, 24, 25, 26, 27, 28, 30, 32, 33, 34, and 35 of Township 12 North Range 6 East; Sections 6, 7 and 13 of Township 11 North and Range 7 East; Sections 1, 2, 3, 4, 5, 6, 7, 8, 9, 10, 11, 12, 13 of Township 11 North and Range 6 East, and Sections 1 and 12 of Township 11 North and Range 5 East.

Tract	Surface Estate	Mineral Estate	Date of Agreement	Acres	Hectares
Bar Z Ranch	Hanson	Hanson, Hanson, Dupea	May 2010	2594.28	1049.87
Short, A & J	Short, A & J	Short, A & J (15%) Davis (85%)	November 2014	2120	857.9
Buckingham	Buckingham	Buckingham, Johnston, Bodell	June 2011	2970	1201.9
Thorson Ranch LLC (Black Butte Portion)	Thorson Ranch LLC	Thorson Ranch LLC	June 2017	280	113.3
US Forest Service Unpatented Mining Claims	US Forest Service	525 Unpatented Mining Claims	---	9,975	4036.7

Table 4.1 Summary of mineral lands held by the Black Butte Copper Project

The MOP application for the Johnny Lee deposit is located on two parcels of fee simple land under mineral lease: The Bar Z Tract and the Short Tract (Figure 4.2, Figure 4.3). The Bar Z Ranch (Hanson Family) owns 100% of the surface rights and three members of the Hanson family own 100% of the mineral interest of the Bar Z Tract. Arthur and Joy Short, a local ranching family, control 100% of the surface rights and 15% of the mineral interests of the Short Tract. Holmstrom's Sheep Creek Ranch LLC (Davis family) controls the remaining 85% of the Short Tract mineral estate.

Steve Buckingham, a local rancher, controls 100% of the surface rights for the Buckingham Tract (Figure 4.2) and shares equal shares of the mineral rights with two siblings. Thorson Ranch LLC owns 100% of the surface estate and 100% of the mineral estate for the portion of the ranch included within the Black Butte Copper Project.

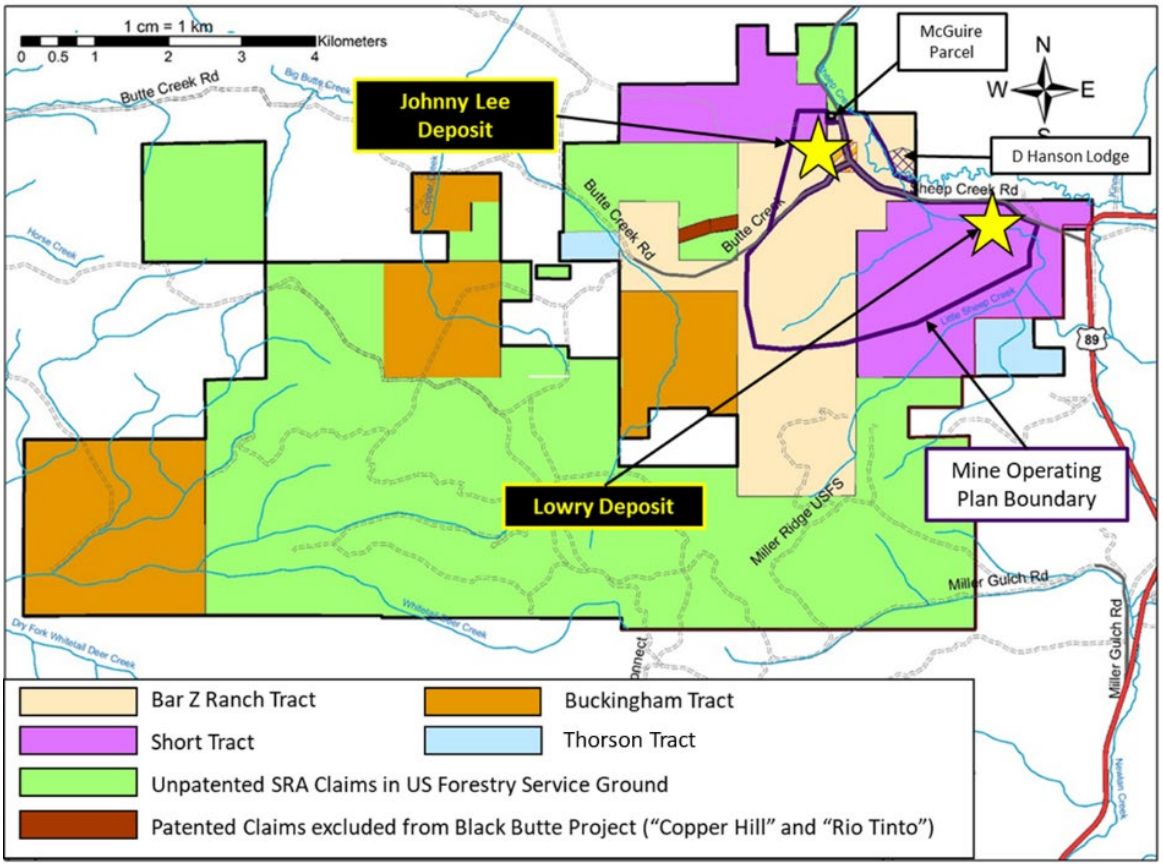

Figure 4.2 Current land tenure of the Black Butte Copper Project showing the boundary of the Mine Operating Plan and the location of the Johnny Lee and Lowry deposits

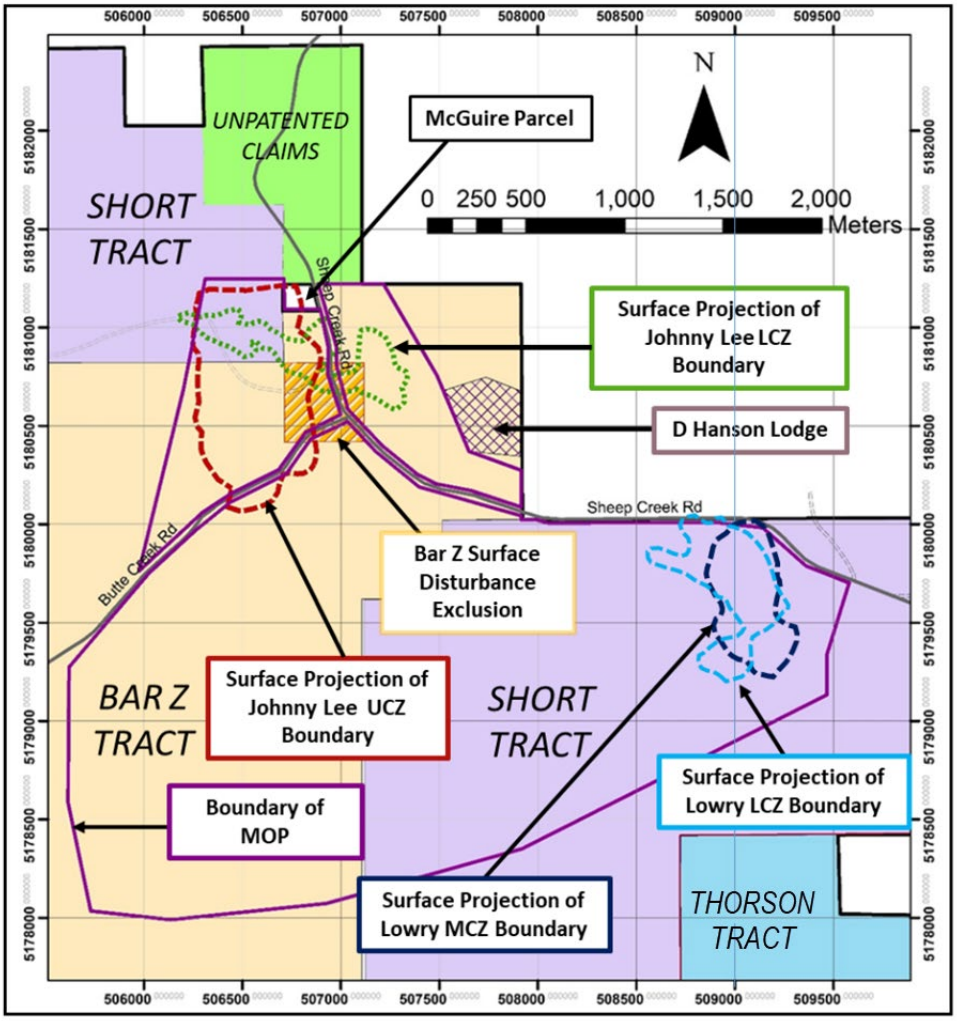

Figure 4.3 ***Map of the study area showing landholdings, the boundary of the MOP and the boundaries of the Johnny Lee and Lowry deposits projected to surface***

The Federal mining claims are distributed over six claim blocks which cover isolated parcels and contiguous blocks of USFS lands in the vicinity of the Black Butte Copper Project. The mining claims give SRA control of the mineral rights covered by those claims. Access for exploration drilling or mining is subject to the rules and processes of the USFS. Appendix A lists the unpatented claims included in the Black Butte Copper Project.

4.2.1 Nature and Extent of Issuer's Interest

The MOP application delineates the outline of the mine facilities planned for the mine operation (Figure 4.3). The plan submitted to the MDEQ envisions mine disturbances on only two of the leased fee tracts, the Bar Z Tract and the Short Tract.

For the Bar Z Tract, SRA has one surface lease and three mineral leases. The surface lease covers 100% of the approximately 1,050 ha (2,594 acres) owned by the Bar Z Ranch. SRA has three separate Mineral Lease Agreements covering that tract with each of the members of the Hanson family who each own portions of the mineral interest. The terms are for 30 years which is renewable for subsequent periods of 10 years each. Included in the Surface Lease are two patented mining claims (Figure 4.2) for the remaining 13 ha (32 acres), the Copper Hill (Mineral Survey #10311) and Rio Tinto (Mineral Survey #10304) patented claims. The mineral rights for these two claims are owned by another party. SRA has no agreement with this party but the Copper Hill and Rio Tinto patented claims are not currently included in the Project.

In June of 2012, the Bar Z Ranch transferred the surface rights of a 31 acre parcel near Sheep Creek to Mr. David Hanson for the purposes of building a lodge (Figure 4.3). This parcel was under the Surface Use Agreement of the Bar Z Ranch at the time of the exchange and an addendum to the Surface Use Agreement was made verifying the Surface Use Agreement is in effect over this parcel.

Upon the death of Rose Holmstrom the original owner of the Short Tract, the surface rights of her leased tract were transferred to Arthur and Joy Short and the mineral rights were split. As to the mineral estate, Arthur and Joy Short were bequeathed a 'life estate' of 15%, with the remaining 85% going to the Holmstrom's Sheep Creek Ranch LLC. At the passing of Arthur and Joy Short, the mineral estate will revert to the complete ownership of Holmstrom's Sheep Creek Ranch LLC, and the surface ownership will be retained by the successors of Arthur and Joy Short. This tract consists of approximately 858 ha (2,120 acres) and is subject to a Surface Use Agreement (dated April 2013) with Arthur and Joy Short. The mineral estate for this tract is covered by the Short Mineral Lease Agreement dated December 2014, and The Holmstrom's Sheep Creek Ranch Mineral Lease Agreement dated November 2017. Both Mineral Lease Agreements are valid for 30 years from the date of the original Rose Holmstrom Lease (May 2010) and are renewable for subsequent periods of 10 years each.

The McGuire parcel (Figure 4.3) is a five acre area, included within the Bar Z Tract, on which SRA has no Mineral or Surface Use Agreement. The NE corner of the UCZ projects to surface and outcrops in this area. The portion of the UCZ that occurs within the McGuire parcel is excluded from this FS.

The Buckingham Tract (Figure 4.2) consists of approximately 1,209 ha (2,970 acres) and is subject to a single Mining Lease Agreement (dated June 2011) with Mr. Steve Buckingham, 100% surface owner and one-third mineral owner, and his two siblings, Kathy Johnston and Marilyn Bodell, each one-third mineral owners. The agreement has a term of 30 years and is renewable in 10-year increments.

Within the Black Butte Copper Project (Figure 4.2) there are two small parcels totalling approximately 113 ha (280 acres) that are part of the Thorson Ranch Mineral Lease Agreement (dated June 2017). These parcels are part of a larger lease agreement that includes parcels not contiguous with the Black Butte Copper Project tenement and that currently have no known mineral resources. The term of the lease is 30 years from the signing date but can be extended for additional periods of five years.

The unpatented lode mining claims (Figure 4.2) are kept active with annual maintenance fees paid to the Bureau of Land Management and Meagher County by September 1st of each year. There are no royalty or lease obligations on these claims.

4.3 Royalties, Agreements and Encumbrances

4.3.1 Bar Z Tract

The Surface Use Agreement required lease payments of $50,000 on signing (May 2, 2010) and on each of the first four anniversary dates. Payments since the fifth anniversary date and each anniversary date since have been $75,000 and will continue for that amount until the end of the lease.

Within this Surface Use Agreement, the landowner has the option to transfer title to those lands “which are necessary for the construction and operation of a mining and milling operation” (Mine Property) to SRA. The Bar Z Ranch would be compensated for the land at the appraised value. At the end of mining and reclamation the Bar Z Ranch may, at their discretion, have the title of the Mine Property returned to them.

SRA has agreed that only underground mining operations will be conducted on the property. SRA also agreed to not disturb any of the pre-existing buildings in the Bar Z Surface Disturbance Exclusion Area (Figure 4.3). These buildings overlie the Johnny Lee UCZ, and so must be considered when designing mine workings.

Each of the three mining leases for the Bar Z Tract required advance minimum royalty payments of $16,150 on signing and on the first and second anniversaries, $32,300 on the third anniversary, $48,450 on the fourth anniversary, and $64,600 on each anniversary thereafter through the term of the lease. The combined mineral interest has a NSR of 5%, with an option to buy this down to a 2% NSR for $5,000,000, thereby reducing each mineral lessors’ royalty to 0.6666% NSR in return for a payment of $1,666,666. Exercising the buy down option eliminates further advance minimum royalty payments.

4.3.2 *Short Tract*

The Short Tract Surface Use Agreement requires a surface lease payment of $4,000 per year and contains the same Mine Property clause as the Bar Z Surface Use Agreement. The landowner may transfer title to those lands deemed Mine Property to SRA. The landowners would be compensated for the land at the appraised value. At the end of mining and reclamation, the owners may have the title of the Mine Property returned to them at their discretion.

SRA has agreed that only underground mining operations will be conducted on the property.

The Short Mineral Lease Agreement (dated December 2014) sets out an advance royalty payment of $24,120.30 per year plus an additional $10,000 per year in surface rent. The Holmstrom's Sheep Creek Ranch Mineral Lease Agreement (dated November 2017) is compensated by an advanced royalty of $136,681.70 per year. The combined mineral interest has a NSR of 5%, with an option to buy this down to a 2% NSR for $5,000,000 which terminates any additional advanced royalty payments.

4.3.3 *Other Tracts*

The following tracts are part of the Black Butte Copper Project property package but are lands for which there is no currently defined mineralization suitable for mining, nor have they been included in any mine plan or permit, but still may have mineral potential. This potential could be realized through future exploration and drilling.

The Buckingham Tract Agreement required advance minimum royalty payments of $5,000 on signing, $15,000 on or before six months after signing, $20,000 on or before the first and second anniversaries, $25,000 on or before the third through fifth anniversaries, $30,000 on or before the sixth through eighth anniversaries, $35,000 on or before the ninth through eleventh anniversaries, $40,000 on or before the twelfth through fourteenth anniversaries, and $50,000 per annum through the remainder of the lease term or until commercial production. At commercial production, the advance royalty payments terminate and a 5% NSR royalty would come into effect, which could be bought down to 2% NSR for a payment of $5,000,000. The agreement specifies that only underground mining will be conducted.

Within the Black Butte Copper Project there are two small parcels totalling approximately 113 ha (280 acres) that are part of the Thorson Ranch Mineral Lease Agreement (dated June 2017). The Mining Lease Agreement prescribed an initial payment of $5,000 on signing, $5,000 on or before the first through the fifth anniversaries, $10,000 on or before the sixth through the tenth anniversaries, $15,000 on or before the eleventh through the fifteenth anniversaries, $20,000 on or before the fifteenth through the twentieth anniversaries, and $30,000 on or before the twentieth anniversary and each anniversary thereafter until the termination of the agreement or the commencement of mining. The agreement includes a 5% NSR, which can be bought down to 2% NSR for a payment of $5,000,000. Advance royalty payments will cease on the

exercise of the royalty buyout. The company has agreed to pay an additional $5,000 per year and $1,000 per drill site for each year that drill exploration is conducted on the property. The agreement specifies that only underground mining will be conducted. The properties subjected to this lease are also covered under a Conservation Easement purchased by a third party. Any mining or exploration conducted on these lands would need to maintain the qualities of the lands under the terms of the Conservation Easement.

4.4 Environmental Liabilities and Permitting

4.4.1 Environmental Liabilities

SRA has conducted exploration under Exploration License #00710 issued by the MDEQ. Regulations include the bonding of exploration disturbances to ensure reclamation is completed. SRA currently has an obligated bond of $137,365 for completion of the reclamation of the 2018/2019 Phase 2 and earlier drill programs. These obligations will be released when the reclamation is completed by SRA and inspected and approved by the MDEQ.

4.4.2 Required Permits and Status

The MOP application for the Black Butte Copper Project was submitted to the MDEQ in December 2015. The application was designed to meet the requirements of the Montana MMRA (Title 82, Chapter 4, Part 3, MCA) and the rules and regulations governing the act. Compliance with regulatory requirements is cross-referenced with components of the MOP application in Table 4.2. Following revisions, the MOP application was declared complete and compliant with the MMRA by the MDEQ in August 2017 and the Draft Mine Operating Permit was issued (MDEQ, 2017). Following this determination, the MDEQ retained an independent third party environmental expert to complete a draft EIS for the Black Butte Copper Project. The Draft EIS was released for Public Comment by the MDEQ in March 2019. The public comment period ended in May 2019 and the Final EIS was published on March 13, 2020.

The MDEQ then issued a ROD on April 9, 2020 (MDEQ, 2020a) that approved the MOP application with one AMA and with five stipulations. Additional details of the AMA may be found in the ROD document (MDEQ, 2020a) and in Section 20 of this Report. The Hard Rock Mining Bureau issued the final Mine Operating Permit on August 14, 2020 (MDEQ, 2020c).

Table 20.10 in Section 20 of this Report presents a list of other permits required, plans that must be submitted, or acts requiring compliance or monitoring in order to comply with a Montana Mine Operating Permit. Additional project bond information and mine closure information is also described in Section 20.10 of this Report.

Section	Rules (ARM)/ACT (MCA) Citation
1.0 Introduction	
1.1 Project Location	ARM 17.24.115(k)
1.2 Brief Project History	MCA 82-4-337(1)(a)
1.3 Land Status	MCA 82-4-335(5)(f) through (h)
1.4 Geology	ARM 17.24.116(3)(i)
2.0 Existing Conditions/Environmental Baseline Studies	
2.1 Climate, Metrological Data & Air Quality	ARM 17.24.116(3)(a)
2.2 Water Resources	
2.3 Wetlands Resources	
2.4 Environmental Geochemistry	
2.5 Soil Resources	ARM 17.24.116(3)(a)
2.6 Terrestrial Wildlife Resources	
2.7 Aquatic Resources	MCA 82-4-335(5)(f) through(h)
2.8 Vegetation Resources	
2.9 Cultural Resources	
2.10 Socio-economic Resources	
2.11 Noise	
2.12 Transportation Resources	
2.13 Land Use	
3.0 Operating Plan	
3.1 Introduction	ARM 17.24.116(3)
3.1.1 Mine Permit Boundary	ARM 17.24.116(3)(d) and (e)
3.1.2 List of Facilities with Surface Disturbance Acres	ARM 17.24.116(3)(d)
3.2 Underground Mine Operations and Mining Methods	ARM 17.24.116(3)(f)
3.2.2 SRA's Underground Mine Plan	
3.2.2.6 Mining Equipment	ARM 17.24.116(3)(j)
3.3 Mineral Production	
3.3.1 Processing Method	ARM 17.24.116(3)(g): ARM 17.24.116(3)(p)
3.3.2 Mining Operations and Schedule	
3.3.3 Mill Support Facilities	
3.4 Mine Site – General Construction	
3.4.1 Overview and Disturbance Acres	
3.4.2 Construction of Facilities	
3.5 Engineering Evaluations	

Section	Rules (ARM)/ACT (MCA) Citation
3.51 Geotechnical Foundation Evaluations	MCA 82-4-301, 82-4-303, 82-4-305, 82-4-335, 82-4-336, 82-4-337, and 82-4-342
3.5.2 Design Standards	MCA 82-4-301, 82-4-303, 82-4-305, 82-4-335, 82-4-336, 82-4-337, 82-4-342 and 82-3-378
3.5.3 Hazard Potential Classifications	
3.5.4 Seismicity	MCA 82-4-301, 82-4-303, 82-4-305, 82-4-335, 82-4-336, 82-4-337, and 82-4-342
3.5.5 Stability Analysis	MCA 82-4-301, 82-4-303, 82-4-305, 82-4-335, 82-4-336, 82-4-337, 82-4-342 and 82-4-378
3.5.6 Longevity of HDPE Geomembranes	
3.5.7 Seepage Analysis	
3.5.8 Tailings Characteristics	ARM 17.24.116(3)(d), MCA 82-4-335(5)(n)
3.5.9 Binder Sources, Cemented Tailings Paste Suitability, and Laboratory Test Results	
3.6 Infrastructure Support and Waste and Water Management Facilities	
3.6.1 Roads	ARM 17.24.116(3)(h) & (r), MCA 82-4-335(5)(i)
3.6.2 Power and Powerlines	
3.6.3 Portal Pad	
3.6.4 Ventilation Raises	
3.6.5 Temporary Waste Rock (WRS) & Operational Storage	ARM 17.24.116(3)(d), MCA 82-4-335(5)(n)
3.6.6 Process Water Pond (PWP)	MCA 82-4-301, 82-4-303, 82-4-305, 82-4-335, 82-4-336, 82-4-337, and 82-4-342
3.6.7 Contact Water Pond (CWP)	
3.6.8 Cemented tailings Facility (CTF)	MCA 82-4-301, 82-4-303, 82-4-305, 82-4-335, 82-4-336, 82-4-337, 82-4-342 and 82-4-378; ARM 17.24 116(3)(g); SB-209: MCA 82-4 335(5)(l)
3.6.9 Non-Contact Water Reservoir (NCWR)	
3.6.10 Stockpiles	
3.6.11 Pipelines	
3.6.12 Equipment & Contract Manpower Required for Support Facility Construction	
3.6.13 Facility Siting Alternative Analysis	
3.7 Water Management	

Section	Rules (ARM)/ACT (MCA) Citation
3.7.1 Introduction	ARM 17.24.116(3)(k); MCA 82-4-336(5)
3.7.2 Water Supply	
3.7.3 Water Balance	ARM 17.24.116(3)(b); ARM 17.24.115 (a-d) and (k)(iv) MCA 82-4-336(2)
3.7.4 Water Treatment	
3.7.5 Treated Water Disposition	
3.7.6 Storm Water	
3.7.7 Erosion Control & Best Management Practices (BMP)	
3.8 Other Operational Management Components	
3.8.1 Total Project Employment with Subcontractors	ARM 17.24.116(3)(q)
3.8.2 Projected Construction & Operational Traffic	
3.8.3 Waters of the US (WOTUS)	
3.8.4 Air Quality & Dust Control	ARM 17.8.308; 17.24.115(1)(h)
3.8.5 Visual Resource Assessment	
3.8.6 Operational Noise	ARM 17.24.116(3)(a): ARM 17.24.116(3)(s)
3.8.7 Fire Protection	ARM 17.24.116(3)(m): 17.24.116(3)(g)
3.8.8 Solid Waste Disposal	ARM 17.24.115(i); ARM 17.24.116(3)(c)
3.8.9 Sewage Treatment	ARM 17.24.116(3)(o)
3.8.10 Hazardous Materials Disposal (Includes Emergency Response Plan)	ERP: ARM 17.24.116(3)(n)
3.8.11 Site Security	
3.8.12 Lighting	
3.8.13 Cultural Resource Protection	ARM 17.24.116(3)(t)
4.0 Modelling Studies	
4.1 Hydrologic Conceptual Model	
4.2 Predictive Water Quality Modelling	
4.3 Post Closure Non-degradation Evaluation	ARM17.30.715
4.4 Closure Compliance with Non-degradation Criteria	ARM17.30.715
5.0 Mitigations	
6.0 Monitoring	
6.2 Ongoing Baseline Monitoring	
6.3 Operational Monitoring	
6.3.1 Water Quality & Quantity Monitoring	ARM 17.24.116(3)(l), MCA 82-4-335(5)(m)

Section	Rules (ARM)/ACT (MCA) Citation
6.3.2 Facility Operational Monitoring	
6.3.3 Facility Geotechnical Monitoring	
6.3.4 Waste Rock Geochemistry Monitoring	
6.3.5 Air Quality Monitoring	ARM 17.8.308; 17.24.115(1)(h)
6.3.6 Wetlands Monitoring	
6.3.7 Aquatic Resource Monitoring	
6.3.8 Noise Monitoring	ARM 17.24.116(3)(s)
6.3.9 Reclamation Monitoring	
6.4 Post Operational Closure Monitoring	
6.4.1 Facility Closure Monitoring	ARM 17.24.115(1)(m)
6.4.2 Water Quality Monitoring	ARM 17.24.115(1)(d),(e),(f),(n);17.24.116(3)(l); ARM 17.24.106
6.4.3 Reporting	
7.0 Reclamation & Closure	ARM 17.24.116(5)
7.2 Disturbed Land Reclamation Compliance	MCA 82-4-336(2), (3), (4), (5), (6), (8), (9)(a), (10), (11)
7.3 Detailed Plan for Permanent Reclamation & Closure	ARM 17.24.150
7.3.1 Post Mining General Construction Measures	
7.3.2 Post Mining Building & Solid Waste Disposal	ARM 17.24.115(1)(i) & (m), MCA 82-4-303(15)(e); ARM 17.50.1405
7.3.3 Site-specific Facility Closure	ARM 17.24.115(1)(m); ARM 17.24.106
7.3.4 Soil Salvage Placement	
7.3.5 Revegetation	ARM 17.24.115(1)(c), (k)(iii) & (l) MCA 82-4-303(15)(c)
7.4 Reclamation Schedule	MCA 82-4-303(15)(i); 82-4-336(3)
7.5 Bond Release	MCA 82-4-338(1),
8.0 References	
9.0 Responses to Comments	

Table 4.2 Black Butte Copper Project Operating Plan cross-referenced with Regulatory Compliance

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

5.1 Location and Access

The Black Butte Copper Project property is located in Meagher County, Montana, 27 km north of the town of White Sulphur Springs (Figure 5.1). Access is via the asphalt-surfaced, US Highway 89 and 4 km of the county maintained, gravel surfaced, Sheep Creek road. US-89 and Sheep Creek road are snow-plowed during the winter, allowing year-round access to the Project location.

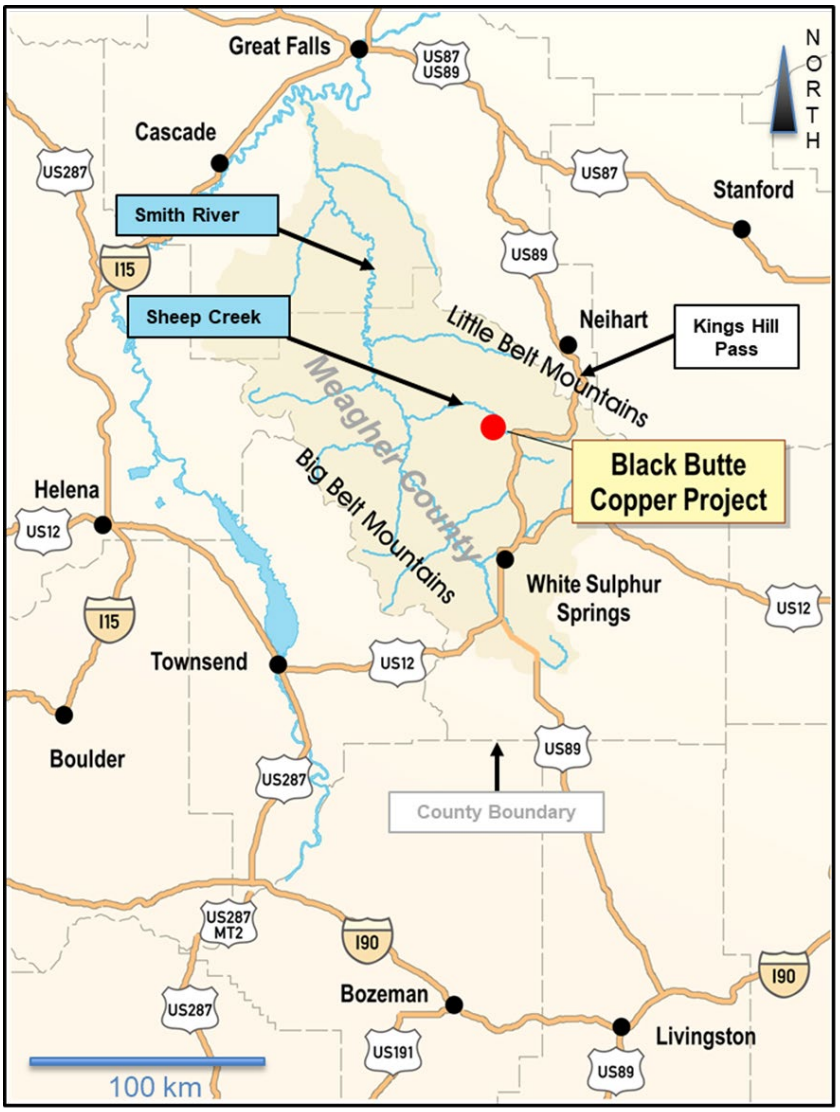

Figure 5.1 Location of the Black Butte Copper Project Showing Significant Cadastral Features

5.2 Population Centres

White Sulphur Springs (Figure 5.1) has a population of approximately 934 people (2017 US Census Bureau estimate) and is the county seat of Meagher County, estimated population 1,866. White Sulphur Springs is 122 km from the state capital Helena (population 31,315), 127 km from Bozeman (population 48,532) and 158 km from Great Falls (population 58,701). All of these cities have international airports. The nearest railheads are Townsend (93 km from site), Belt (100 km from site) and Livingston (149 km from site).

5.3 Physiography, Vegetation and Climate/Topography and Elevation

The Project area is situated on the south flank of the Little Belt Mountains (Figure 5.1) in an area of gently rolling topography. Elevations within the proposed area of operations range from 1,700 to 1,870 metres above sea level (masl). The highest point within the SRA tenement package is Black Butte at 2,071 masl.

Sheep Creek, a tributary of the Smith River, runs through the NE corner of the property (Figure 5.1). The junction between Sheep Creek and the Smith River is approximately 30 km downstream from the Black Butte Copper Project.

Within the Project area, the Sheep Creek valley is a 200 to 800 m wide, flat-lying corridor that is partially utilized for agriculture (irrigated hay meadows). The portions of the valley not being cultivated form willow covered wetlands. The surrounding hilltops, and north facing slopes in particular, are tree-covered, primarily with Douglas fir. Open slopes between the high lying ground and the valley are covered with a mixture of sage-brush and grass.

The Project area occurs in an area with a climate characterized (Köppen-Gieger climate classification) as Cold, Semi-arid or Steppe type (Peel et al, 2007). These climate zones have hot summers and cold winters. Precipitation occurs as a mixture of rain and snowfall during the colder months and rain during the warmer months. Steppe climate zones are subject to significant diurnal temperature variations with the difference between daytime and night-time temperatures often exceeding 20°C.

Long-term weather records for White Sulphur Springs (1,609 masl) record average daily-low and daily–high temperatures of -11 to 0°C for January, -2 to 12°C for April, 9 to 28°C for July and, -1 to 14°C in October. Temperature extremes can reach -44°C in winter and 37°C in summer. Average annual precipitation in White Sulphur Springs is 322 mm of rain and 988 mm of snow for a total of 421 mm precipitation (Western Regional Climate Center, 2019).

In 2012, a meteorological monitoring station was established at the proposed Black Butte Copper Project mine site. Monitoring results from 2013 to 2017 are summarized in Table 5.1.

	2013				2014				2015				2016				2017			
	Temperature (°C)			Precipitation	Temperature (°C)			Precipitation	Temperature (°C)			Precipitation	Temperature (°C)			Precipitation	Temperature (°C)			Precipitation
	Min.	Max.	Ave.	(mm)	Min.	Max.	Ave.	(mm)	Min.	Max.	Ave.	(mm)	Min.	Max.	Ave.	(mm)	Min.	Max.	Ave.	(mm)
Jan	-28.2	6.0	-1.6	16.8	-29.2	6.7	-7.0	40.9	-31.1	9.8	-7.5	35.8	-21.8	7.7	-6.3	33.8	-30.3	5.9	-11.9	9.1
Feb	-23.7	5.0	-6.4	33.8	-39.7	4.9	-12.6	3.6	-30.5	11.7	-3.9	19.8	-23.8	9.1	-3.1	6.4	-32.5	9.5	-5.4	23.4
Mar	-22.0	9.1	-3.4	184.2	-28.3	9.1	-3.3	78.2	-25.5	16.7	0.6	15.2	-16.8	12.4	-1.1	19.8	-11.4	15.9	0.8	19.6
Apr	-20.3	16.3	-0.3	23.1	-14.7	13.7	1.2	46.7	-14.2	19.3	2.0	35.1	-10.7	20.5	3.9	39.4	-13.6	17.0	2.1	45.0
May	-11.0	24.4	7.4	67.6	-7.0	23.0	6.9	21.1	-7.4	21.1	6.6	53.1	-6.5	21.1	6.6	67.6	-6.2	23.5	7.4	41.9
Jun	-3.0	26.5	11.5	F	-0.6	19.7	9.3	144.3	0.1	30.0	14.2	50.0	-2.2	27.0	13.2	33.8	-2.9	26.9	11.6	72.4
Jul	2.8	29.8	16.8	F	1.1	26.4	16.0	30.7	0.7	28.1	14.1	42.9	-1.9	28.5	14.8	44.2	2.2	29.7	18.0	7.6
Aug	F	F	F	F	1.1	28.8	13.9	82.0	-3.6	29.6	14.4	14.7	-2.4	30.1	13.9	29.2	0.1	28.7	14.7	4.3
Sep	-6.1	28.1	11.4	F	-7.8	27.1	9.0	36.6	-4.4	27.1	10.0	42.9	-3.7	26.8	8.3	67.1	-6.5	29.9	8.6	30.0
Oct	-10.9	13.9	0.0	10.2	-9.6	19.8	5.5	18.8	-10.1	23.8	5.4	6.1	-15.8	17.7	4.5	58.7	-11.7	18.9	1.6	6.9
Nov	-22.7	13.5	-2.9	18.5	-28.5	16.5	-5.6	13.5	-26.1	10.8	-5.5	17.8	-20.6	16.3	0.8	4.3	-24.0	13.2	-3.5	28.4
Dec	-35.1	8.5	-9.7	34.0	-36.1	11.6	-5.9	F	-26.0	8.6	-7.2	22.6	-37.4	2.5	-11.7	7.4	-31.3	6.0	-10.3	25.9

Table 5.1 Temperature and Precipitation Observations from the Black Butte Meteorological Station

The area is generally snow-covered from November to March. Surface drilling is typically done between December and March, when the ground is frozen, so as to reduce disturbance in areas of soft ground and avoid surface-use conflicts with agricultural activities. It is envisaged that mine operations will be conducted year-round although heating of both UG and surface operations shall be required during winter.

5.4 Power Supply

The power for the onsite core sheds, water well and meteorological station is currently supplied, from the local power grid, by Fergus Electric Cooperative Inc. (FEC) It is envisaged that an existing 100 kV power line, located at Kings Hill Pass, 25 km along US-89 (Figure 5.1) shall be used to supply electricity for the Project.

5.5 Communications Link

An existing fibre-optic communications cable, operated by Triangle Communications Cooperative, runs alongside the Sheep Creek and Butte Creek county roads through the Project area (Figure 5.2). The cable has sufficient capacity to provide for highspeed internet connection for the proposed mine site as well as the installation of a 4G antenna to provide full cell phone service to the site.

5.6 Water Supply

The site contains significant surface water resources. In addition to Sheep Creek there are two tributaries in the Project area, Little Sheep Creek and Coon Creek.

Mine facilities and operations have been designed to avoid impacts to these surface waters and adjacent wetlands. SRA has conducted and will continue to conduct water and aquatic biosystem monitoring of these surface waters to ensure water quality is maintained.

Groundwater will be encountered during UG mining of the Johnny Lee deposit. It is expected that the water-table shall be intersected by the decline at approximately 320 m from the portal, at a depth of approximately 70 m below surface.

When necessary, groundwater shall be pumped out of the mine to the Contact Water Pond (CWP), located near the portal (Figure 5.2). Runoff from mine surface facilities will also be collected in the CWP. Water required for processing use shall be pumped from the CWP. Any excess water will be treated and infiltrated back into the groundwater system via an Underground Infiltration Gallery (UIG).

SRA requires water rights that will allow for water discharge via the UIG and has negotiated to purchase water rights from upstream agricultural water users. SRA is currently in the process of obtaining a Change-of-Use permit from the Montana Department of Natural Resources and Conservation (MDNRC).

5.7 Surface Ownership

All infrastructure proposed in the Mine Operating Permit (MOP) for the Black Butte Copper Project (Figure 5.2) is wholly located on private ranch lands that are leased by SRA. Section 4 details the surface and mineral rights which are sufficient to conduct mining operations as currently planned.

5.8 Personnel

It is expected that approximately 300 workers will be required during construction and 230 employees once mining activities reach steady state. Meagher County has a small population most of whom are employed in agricultural and service industries. During construction and ramp-up it is envisaged that the majority of skilled personnel will have to be recruited from outside of the County.

Both Montana and the US have a well-developed mining industry from which skilled mining and construction workers can be recruited. Recruitment and training of Meagher County residents will form an important part of the operations strategy.

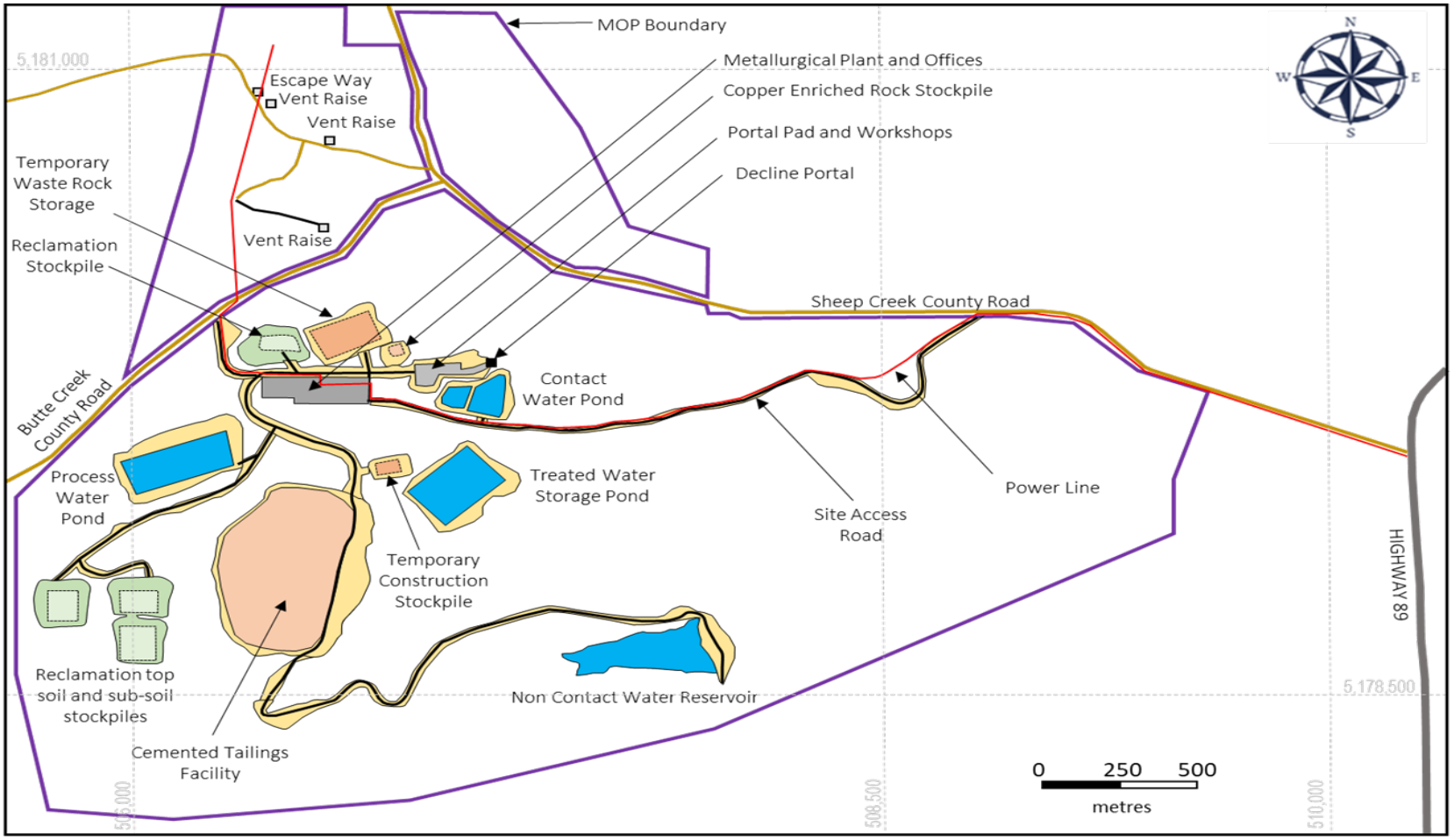

Figure 5.2 Proposed Infrastructure Layout for the Black Butte Copper Project

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6. HISTORY

The first mineral development on the property was by Messrs. Weir and Tyler, local ranchers, who staked claims on copper-stained quartzite at the Virginia Mine prospect, situated approximately 500 m west of the Johnny Lee deposit. By 1894, they had sunk a 20 m shaft and completed a 9 m drift but only encountered minor copper oxide mineralization (Weed, 1899).

A homesteader, John Lee, settled in the area immediately above the current resource in 1906 and in 1910 sank a 15 m deep shaft on the Volcano Valley Fault Zone (VVFZ) (Pers. Comm., Hanson family). He continued to work the property intermittently until 1922. Based on observations in and around the shaft, SRA geologists interpret the workings to have encountered minor copper oxide that had been remobilized during movement along the VVFZ. It is likely that the source for this remobilized Cu was the Johnny Lee deposit. The mineralization that forms the Johnny Lee deposit has been named in recognition of Mr. Lee's efforts.

Until 1973, commercial interest was focused on iron contained by ferruginous gossans in the vicinity of the Black Butte Copper Project property (Goodspeed, 1945; Roby, 1950). These efforts resulted in the staking and patenting of several claims both inside and adjacent to the current Bar Z Ranch Lease (Figure 6.1). The small scale development of the iron ore deposits was accompanied by Cu prospecting but none of the test pits nor exploration adit encountered Cu mineralization of economic significance. No Cu mining has been undertaken within the Project area.

6.1 Prior Ownership and Ownership Changes

The leases that form part of the Black Butte Copper Project property were originally homestead and railroad properties which have been consolidated, over time, into the fee simple tracts now under lease by SRA (Figure 4.2). The remainder of the Project area comprises unpatented claims on USFS land.

Homestake Mining Company Inc. (Homestake) claimed ground within and adjacent to the current Black Butte Copper Project Property in 1973, but by 1975 had abandoned the district.

Cominco American Inc. (CAI) acquired mining claims and leased a number of properties, including the current Bar Z Tract, in 1976.

In 1977, Anaconda Mining Company (Anaconda) acquired land to the west of CAI in the Horse Prairie area. CAI formed a JV with Anaconda in the early 1980s then bought out Anaconda's interest in 1984.

Exxon Minerals leased the tract owned by Rose Holmstrom (now the Short Lease) in 1981. CAI formed a JV with Exxon Minerals between 1984 and 1985 after which CAI purchased Exxon Minerals' interest. In 1985, CAI formed a JV with Utah International Inc. (UII). Broken Hill Propriety Limited (BHP) acquired UII in

1985 and continued the JV with CAI. In 1990, CAI purchased BHP's interest, and assumed control of the entire property. Five years later, CAI dropped all the leases due to falling Cu prices.

In 2010, Tintina Resources Inc. acquired the mineral rights formerly owned by CAI.

Sandfire Resources NL acquired a controlling interest in Tintina Resources Inc. in 2014.

In 2017, Tintina Resources Inc. was renamed Sandfire Resources America Inc. (SRA). In 2019, Sandfire Resources NL was renamed Sandfire Resources Ltd.

6.2 Historical Exploration Chronologically by Company (Prior to SRA)

Homestake conducted geologic mapping, soil sampling, and trenching targeting Pb-Zn mineralization in ferruginous gossans. Homestake completed one drillhole, sited SW of the Johnny Lee deposit, before abandoning the district in 1975.

Anaconda commenced exploration in 1977 and from 1978 to 1979 drilled 12 drillholes that encountered bedded pyritic sulphides but no economic mineralization.

CAI completed 13 drillholes in the Black Butte area from 1978 to 1981. These drillholes intersected bedded pyrite with narrow zones of elevated (<1%) zinc but no commercially significant Cu mineralization.

In 1985, CAI joint-ventured the property with UII, with UII as the operator. As part of the JV agreement, UII was required to complete a number of drillholes recommended by CAI geologists. In the second of these drillholes, SCC-17, UII intercepted both the Upper Zone and Lower Zone of the Johnny Lee deposit. The Upper Zone intersection was 6.09 m at 2.92% Cu from 131.4 m and the Lower Zone intersection was 2.44 m at 2.88% Cu from 355.7 m. This is considered the discovery hole for the Project.

BHP/UII operated the JV until early 1988 drilling a total of 33 drillholes, at which point they had fulfilled their earn-in agreement for a 50% interest. In 1990, CAI purchased BHP's 50% interest with no retained royalties or back-in rights and operational control reverted to CAI.

CAI continued to drill the Johnny Lee deposit, discovered the nearby Lowry deposit, and drilled an additional 28 drillholes before abandoning the Project in 1990.

From 2010 to 2013, SRA completed an additional 168 drillholes in both deposits for Mineral Resource definition, metallurgical test work, and geotechnical studies. This data was used for the PEA released in 2013 (Winckers *et al*, 2013).

From 2014 to present, SRA has focused on gathering environmental, geotechnical, hydrogeological, metallurgical, and geological data to finalize the MOP, provide data for a new Mineral Resource estimate, and complete a FS.

6.3 Historic Mineral Resource and Reserve Estimates

Prior to the establishment of NI 43-101, CAI declared a Mineral Resource estimate of 4.5 Million tonnes (Mt) at 2.5% Cu and 0.1% Co for the Johnny Lee Upper Copper Zone (UCZ) and 1.8 Mt at 6.0% Cu for the Lower Copper Zone (LCZ) (Cominco, 1996). No additional details for this estimate have been located by SRA personnel.

SRA declared separate Mineral Resource estimates for the Black Butte deposit in 2011, 2012, and 2013, all of which were prepared in accordance with NI 43-101. These estimates are documented below.

6.3.1 2010 Johnny Lee Mineral Resource Estimate

In 2010, SRA conducted a six drillhole program to verify drilling, logging, and analyses by previous operators. Resource Modeling Inc. (RMI) used the verification and historic drill data to estimate an Inferred Mineral Resource (Table 6.1) for the Johnny Lee Upper Copper Zone (UCZ) and Johnny Lee Lower Copper Zone (LCZ) (Lechner, 2011).

Area	Resource Classification	Tonnes (000s)	Cu Grade (%)	Co Grade (%)	Ag Grade (g/t)	Cu (Tonnes)	Co (Tonnes)	Ag (000s oz)
Johnny Lee UCZ	Inferred	7,037	2.36	0.12	12.3	166,018	8,618	2,800
Johnny Lee LCZ	Inferred	2,462	4.71	0.06	5.1	116,122	1,315	400

Source: Lechner et al, 2011
1.5% In Situ Cu cut-off grade

Table 6.1 2011 Mineral Resource estimate for the Johnny Lee deposit

6.3.2 2012 Preliminary Economic Assessment Mineral Resource Estimate

During 2011, SRA completed an additional 92 drillholes in the Johnny Lee and Lowry deposits. The updated dataset was used by Tetra Tech Inc. to complete a Mineral Resource estimate, for the Johnny Lee and Lowry deposit, that was used for a PEA (Winckers *et al*, 2012). Table 6.2 summarizes the Mineral Resource estimate.

Area	Resource Classification	Tonnes (000s)	Cu Grade (%)	Co Grade (%)	Ag Grade (g/t)	Cu (Tonnes)	Co (Tonnes)	Ag (000s oz)
Johnny Lee UCZ	Indicated	8,483	2.96	0.12	16.9	241,769	9,979	4,609
	Inferred	1,257	2.64	0.10	16.4	33,113	1,361	663
Johnny Lee LCZ	Inferred	2,462	4.71	0.06	5.1	116,112	1,315	404
Lowry	Inferred	5,139	2.60	0.12	14.6	133,358	6,350	2,412

Source: Winckers et al, 2012
1.6% In Situ Cu cut-off grade for Johnny Lee UCZ and Lowry
1.5% In Situ Cu cut-off grade for Johnny Lee LCZ

Table 6.2 2012 Mineral Resource estimate for Johnny Lee deposit & Lowry deposit

6.3.3 2013 Updated Preliminary Economic Assessment Mineral Resource Estimate

In 2013, Tetra Tech Inc. updated the PEA using a new Mineral Resource estimate that incorporated results from an additional 65 drillholes (Winckers *et al*, 2013). Table 6.3 summarizes this Mineral Resource estimate.

Area	Resource Classification	Tonnes (000s)	Cu Grade (%)	Co Grade (%)	Ag Grade (g/t)	Cu (Tonnes)	Co (Tonnes)	Ag (000s oz)
Johnny Lee UCZ	Measured	2,659	2.99	0.12	16.3	79,380	3,130	1,393
	Indicated	6,250	2.77	0.13	15.5	180,533	8,165	3,249
	Inferred	1,255	2.52	0.10	15.2	31,752	1,270	613
Johnny Lee LCZ	Indicated	2,387	6.40	0.03	4.5	152,863	771	345
	Inferred	205	5.33	0.03	4.1	100,000	45	27
Lowry	Indicated	4,099	2.94	0.10	15.1	120,658	4,082	1,990
	Inferred	801	2.58	0.10	14.1	20,866	907	363

Source: Winckers et al, 2013
1.6% In Situ Cu cut-off grade for Johnny Lee UCZ and Lowry
1.5% In Situ Cu cut-off grade for Johnny Lee LCZ

Table 6.3 2013 Mineral Resource estimate for the Johnny Lee deposit & Lowry deposit

6.3.4 2019 Updated Johnny Lee Mineral Resource Estimate

The 2019 Mineral Resource update for the Johnny Lee deposit (Ronald and Malhotra, 2019) was supervised and reviewed by SRK Consulting (US), Inc. ("SRK"). Table 6.4 summarizes this Mineral Resource estimate.

The 2019 Mineral Resource estimate represents the following notable changes in the defined Mineral Resource from the previously defined Mineral Resource estimate contained within the Updated Technical Report and PEA for the Black Butte Copper Project, Montana (effective date: July 12, 2013) due to the following key factors:

- Additional drilling, analytical data, geological and structural modelling, and interpretation of geological and mineralization for improved confidence;
- Extensive metallurgical and geometallurgical studies have allowed for estimates of Cu recovery that have a higher level of confidence than previously assumed;
- Removal of Co, Ag, and Au by-products as recent metallurgical test work has indicated that froth flotation does not produce a Cu concentrate with Co, Ag, or Au in payable concentrations; and
- Updated cut-off grade used to define the Mineral Resource using recent metallurgical recovery assumptions for Cu and updated price assumptions.

The 2019 Mineral Resource estimate for the Johnny Lee deposit has been used for the FS detailed in this document. The 2019 Mineral Resource estimate for the Johnny Lee deposit is restated herein and an updated Mineral Resource estimate has been defined for the Lowry deposit.

Category	Quantity (Mt)	Cu (%)	Total Metal (kt)
UCZ			
Measured	1.4	2.6	36.2
Indicated	8.3	2.3	191.3
Measured and Indicated	9.7	2.4	227.5
Inferred	2.2	2.2	49.5
LCZ			
Measured	0.6	5.7	32.9
Indicated	0.6	7.9	50.5
Measured and Indicated	1.2	6.8	83.4
Inferred	0.5	6.3	30.3
Combined UCZ + LCZ			
Measured	2.0	3.5	69.1
Indicated	8.9	2.7	241.8
Measured and Indicated	10.9	2.9	310.9
Inferred	2.7	3.0	79.7

Source: SRK, 2019

- *The effective date for this Mineral Resource is October 15, 2019. All significant figures are rounded to reflect the relative accuracy of the estimates. Cu assay values were capped where appropriate;*
- *Mineral Resources are inclusive of Mineral Reserves and have reasonable prospects for eventual economic extraction. Inferred Mineral Resources have a high degree of uncertainty as to their economic and technical feasibility. It cannot be assumed that all or any part of an Inferred Mineral Resources can be upgraded to Measured or Indicated Mineral Resources;*
- *Metallurgical recovery of Cu has been estimated on a block basis in the UCZ, averaging 77.4%, with a consistent 94.0% Cu recovery applied to the LCZ;*
- *To demonstrate reasonable prospects for eventual economic extraction of Mineral Resources, a cut-off grade of 1.00% Cu based on metal recoverability assumptions, long-term Cu price assumptions of $3.20/lb, mining costs, processing costs, G&A costs totalling $71/t;*
- *There are no known risks to the Johnny Lee Mineral Resources that could materially affect the potential development of the Mineral Resources other than that the Project is subject to legal challenges that are outlined in the Management Discussion and Analyses of the June 2020 Company Quarterly Report (also see Section 14.14 of this Technical Report);*
- *All Mineral Resources are located within land currently under control or lease to Sandfire Resources America Inc.*

Table 6.4 Mineral Resource estimate for the Johnny Lee deposit as of October 15, 2019

6.4 Historic Production

There has been no historic production from the Black Butte Copper Project.

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7. GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional Geology

The crystalline basement underlying the Project area consists of Mesoproterozoic crust formed during a period of ocean basin closure, arc magmatism and high-grade metamorphism, termed the Big Sky Orogeny, which occurred during the final assembly of the supercontinent Nuna from 1.86 Giga annum (Ga) to 1.71 Ga (Condit *et al*, 2015; Foster *et al*, 2006; Mueller *et al*, 2016). The Big Sky Orogeny created a zone of crustal weakness coincident with a trend of high-angle faults, and shear zones stretching north-eastward from eastern Idaho to SW Saskatchewan named the Great Falls Tectonic Zone (Figure 7.1).

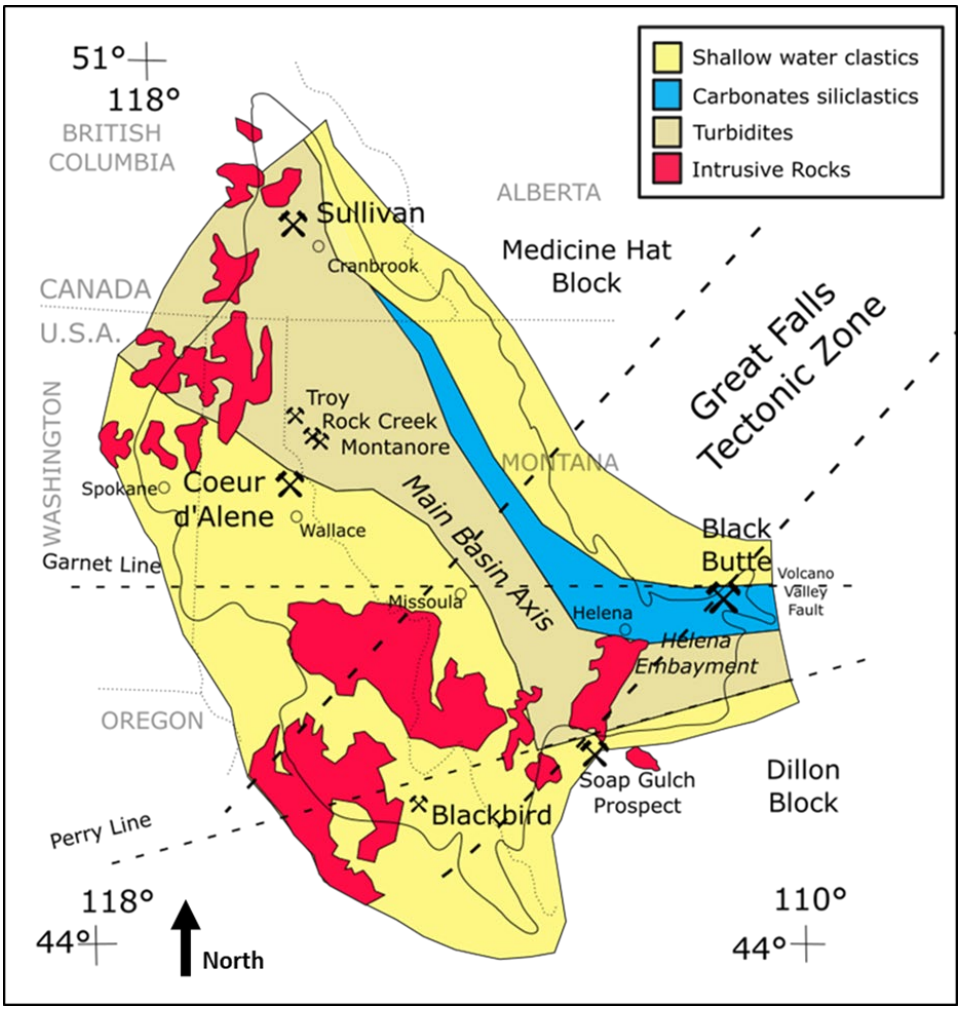


Figure 7.1 The preserved portion of the Belt Basin showing major facies groups of the Belt Supergroup and the location of significant mineral deposits. Modified after Lydon (2007)

Beginning at 1.5 Ga, Nuna began to break-up (Evans & Mitchell, 2011) and an intracontinental basin (the Belt Basin) formed in what is now British Columbia, Alberta, Washington, Idaho and western Montana. This basin rapidly filled with a 15-20 km thick sequence of sediments and minor volcanic rocks collectively termed the Belt Supergroup (Harrison, 1972). Uranium-lead (U-Pb) zircon dating of volcanic rocks in the Belt Supergroup returned ages ranging from 1,468±2 Mega annum (Ma) to 1,390±10 Ma (Evans *et al*, 2000).

Three main lithofacies are recognizable within the Belt Supergroup (Figure 7.1):

- Shallow water clastic sediment;
- A contemporaneous carbonate platform and shelf lithofacies; and
- A deep water turbidite lithofacies.

The Belt Basin was partitioned into a number of sub-basins by differential movement on syn-sedimentary growth faults. One of these sub-basins is the Helena Embayment (Figure 7.1), an EW trending graben bounded by major growth fault zones, principally the Garnet Line on the north, and the Perry Line on the south (Winston, 1986). The Helena Embayment sub-basin contains significantly more carbonate than the main part of the basin and was likely hydrologically restricted from the main basin (Godlewski and Zieg, 1984).

The Belt Basin became increasingly shallow over time and deposition ceased with the onset of the East Kootenay orogeny at approximately 1,350 Ma (Lydon, 2007). Neoproterozoic mafic dikes and sills cut the Belt Supergroup rocks in the Helena Embayment (Reynolds & Brandt, 2005). Cambrian to Devonian sedimentary and volcanic lithofacies unconformably overly the Belt Supergroup of the Helena Embayment.

The Belt Supergroup and overlying Cambrian to Devonian rocks were deformed in the late Cretaceous to early Tertiary by the Laramide Orogeny. This process involved west to east compression and reactivated and inverted many of the Proterozoic faults (Marshak *et al*, 2000).

7.2 Local and Property Geology

Figure 7.2 shows the geology of the Black Butte Copper Project area. The majority of the Project area is underlain by gentle to moderately southward-dipping rocks of the Belt Supergroup. The Belt Supergroup lithofacies are juxtaposed against gently south-dipping Cambrian sedimentary rocks along the VVFZ. To the north of the Project area, Belt Supergroup rocks are also exposed, below the basal Cambrian unconformity (Figure 7.2). Intermediate to mafic, Early Tertiary intrusive dykes and sills occur within the Belt Supergroup. Late Tertiary sedimentary lithofacies unconformably overlie all other units in the area.

7.2.1 *Lithology and Stratigraphy*

Figure 7.3 summarizes the lithostratigraphy within the Project area.

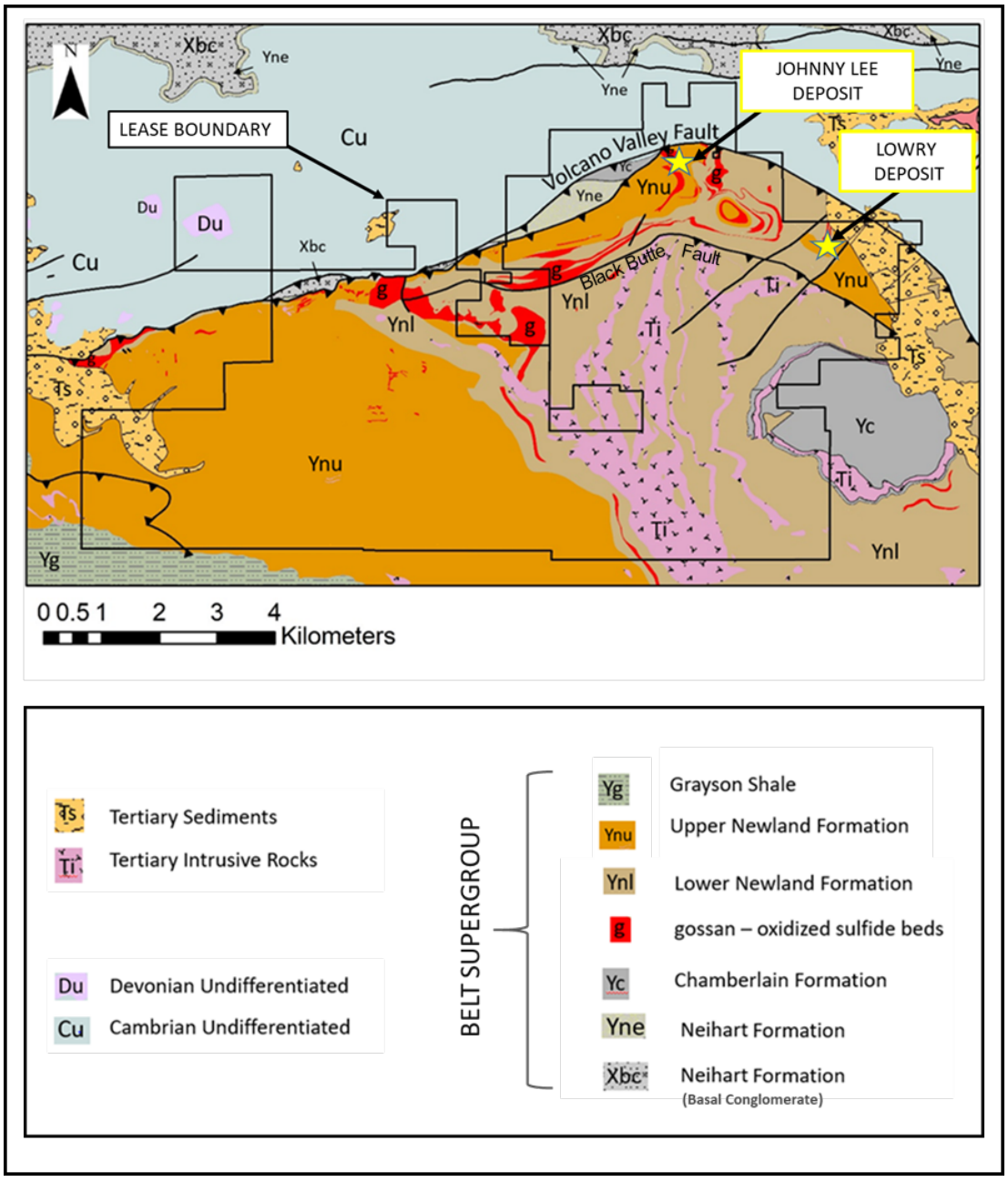


Figure 7.2 Geologic map of the Black Butte Copper Project area showing the Project Boundary and Cu deposits

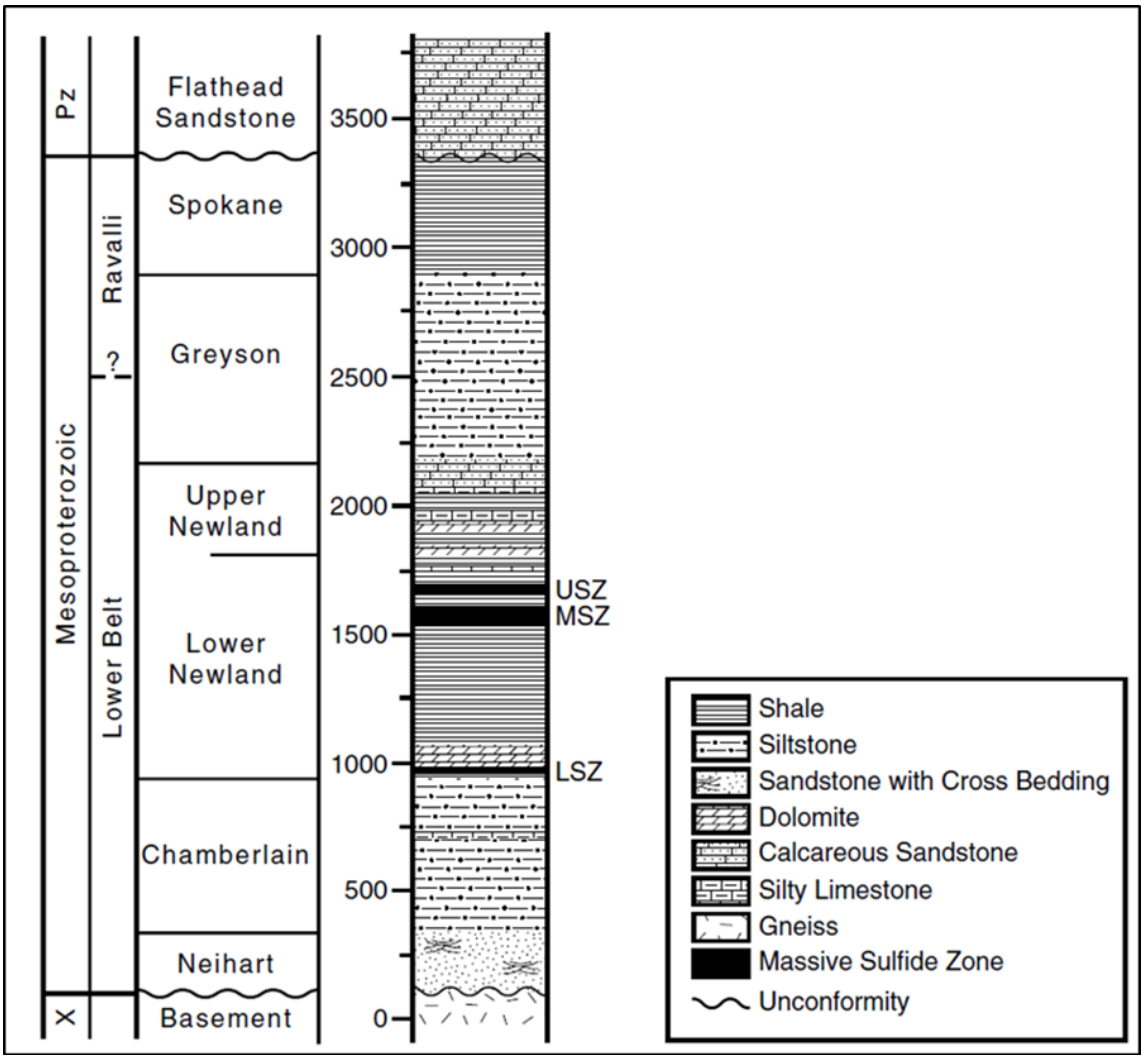


Figure 7.3 Project area stratigraphy with thickness scale in metres (Present et al, 2017)

Mesoproterozoic Basement

Proterozoic basement rocks are not exposed within the Project area but have been intersected in drillholes that penetrate the basal unconformity of the Belt Group. The basement in the Project area consists of coarsely-crystalline amphibolite intruded by meta-granite (granitic gneiss) and meta-diorite (orthogneiss). Granitic gneiss samples collected 20 km north of the Project area have been dated at 1.79 Ga to 1.89 Ga (Mueller *et al*, 2002).

Neihart Formation

The lowest Belt Supergroup unit in the area is the Neihart Formation which unconformably overlies basement rocks (Figure 7.3). Within the lease boundaries, the Neihart Formation is exposed as fault bounded blocks within the VVFZ (Figure 7.2) and has been intersected in drillholes to the north of the VVFZ. The Neihart Formation comprises a locally developed basal conglomerate overlain by a well-sorted, fine-grained quartz-rich sandstone. Due to its well-cemented quartz-rich nature, this unit is informally referred to as the Neihart Quartzite.

Chamberlain Formation

The Neihart Formation grades-up into Chamberlain Formation comprising interlayered shale and sandstone with the first shale unit marking the base of the Chamberlain Formation. The Chamberlain Formation is exposed in a fault-bounded block within the VVFZ and in the axial region of a breached, periclinal anticline along the eastern margin of the Project area (Figure 7.2).

The upper part of the Chamberlain Formation has been intersected in drillholes north of the VVFZ, where it comprises a carbonaceous shale with sandstone lenses and layers. The top of the Chamberlain Formation is defined by the last sandstone unit before Newland Formation shale lithofacies.

Newland Formation

The Chamberlain Formation grades upward into the Newland Formation (Figure 7.3) which consists of shale, carbonaceous shale, calcareous shale, dolomite, conglomerate and massive sulphides. The Newland Formation can be subdivided into two parts: a lower shale and conglomerate-dominated portion, and an upper carbonate-rich portion (Figure 7.3 and Figure 7.4).

Lower Newland

The Lower Newland is approximately 800 m thick in the property area, though it thickens considerably to the south. This unit consists of dolomite, dolomitic shale, carbonaceous shale, shale, and conglomerate. Numerous layers of massive sulphide, comprising bedded pyrite accumulations, are found throughout the unit. These massive sulphide units range from centimetre scale layers to units that exceed 40 m in thickness. Shale and conglomerate lenses often occur within the massive sulphide units. All economically significant Cu mineralization discovered in the Project area to date occurs within massive sulphide units of the Lower Newland. The Johnny Lee Upper Zone deposit is hosted by the Upper Sulphide Zone (USZ), the Johnny Lee Lower Zone deposit is hosted by the Lower Sulphide Zone (LSZ). Mineralization at the Lowry deposit is hosted by the Middle Sulphide Zone (MSZ) and LSZ (Figure 7.4).

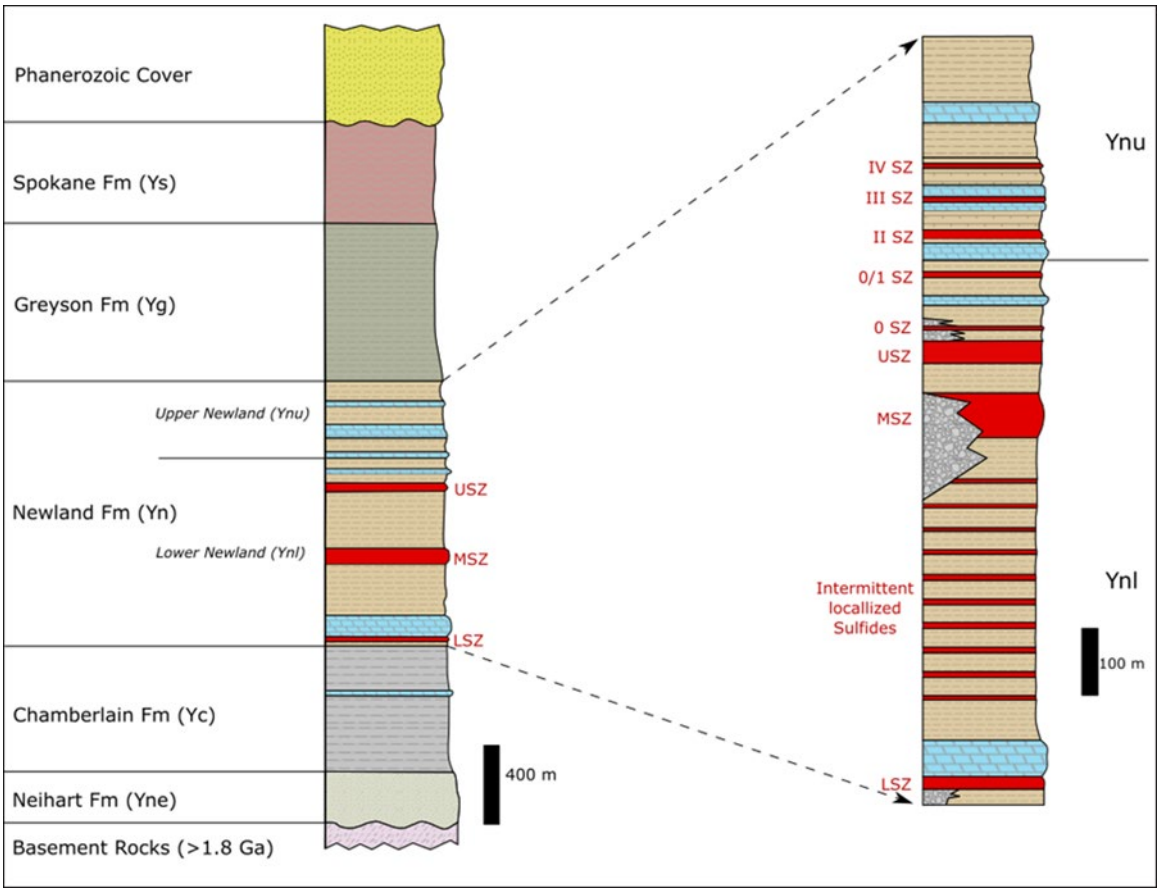


Figure 7.4 Detailed stratigraphy of the Newland Formation within the Project area

Upper Newland

The Upper Newland is approximately 350 m thick and comprises dolomite, dolomitic shale, carbonaceous shale, and shale. Significant conglomerate horizons are absent from the Upper Newland and the proportion of carbonate rocks (dolomite and dolomitic shale) is higher than that of the Lower Newland. The Upper Newland contains massive sulphide units but no economically significant Cu mineralization has been identified to date.

Greyson Formation

The Greyson Formation is exposed in the SW corner of the Project area. It comprises non-calcareous shale with occasional sandstone beds and minor local carbonate. There are no known bedded sulphides in this unit.

Flathead Formation

Much of the area north of the VVFZ is covered by an extensive sheet of Middle Cambrian Flathead Formation that rests unconformably on Belt Supergroup rocks. The Flathead Formation is a moderately to poorly sorted, fine to coarse-grained sandstone that contains abundant hematite. In the area north of the VVFZ, drillholes have intersected Flathead Formation unconformably overlying both Neihart and Chamberlain Formations.

Igneous Rocks

The Project area contains numerous Tertiary, intermediate to mafic dykes and sills that have intruded the Belt Supergroup. The dykes exhibit a preferential N/NE to S/SW strike. These dykes have been identified as near vertical in the Lowry deposit though due to their orientation, limited intercepts, and variable nature, true thicknesses of these dykes are not known. It is the opinion of SRA that the total volume of post-mineralization intrusive mafic dykes represents a minor volume with low risk for grade dilution in the Black Butte Copper Project.

7.2.2 Structure

A three dimensional (3-D) lithostratigraphic model of the Johnny Lee area, constructed in 2019 by SRA using Seequent Leapfrog Geo™ software, allows significant geological units and geological structures to be visualized (Figure 7.5 and Figure 7.6). All proposed mine development and infrastructure occurs within the boundaries of the 3-D lithostratigraphic model.

Buttress Fault

The oldest structural element that has been recognized in the Johnny Lee area is the Buttress Fault (Figure 7.6) which is an EW striking, 70° south dipping brittle fault characterized by a 3 to 5 m wide zone of brecciation and gouge. The Buttress Fault is not exposed at surface as it is truncated by the VVFZ in the central portion of the lithostratigraphic model (Figure 7.5) and sub-outcrops below the Flathead Formation in the eastern and western portions of the model (where the arcuate nature of the VVFZ allows the Buttress Fault to project to the base of the Flathead Formation).

No linear fabric elements, nor kinematic indicators have been recorded from drill core intersections of the Buttress Fault, thus actual movement sense on the fault cannot be determined. Juxtaposition of the Newland and Chamberlain Formations against the Neihart Formation and Proterozoic basement across the Buttress Fault indicate a minimum of 300 m of apparent normal displacement on the Buttress Fault.

The Buttress Fault is interpreted by Godlewski and Zieg (1984) to be a syn-sedimentary, basin-margin fault that developed during formation of the Belt Supergroup depository. The Buttress Fault truncates the LSZ (Figure 7.7), and adjacent Newland Formation lithofacies, that are the host for the LCZ deposit.

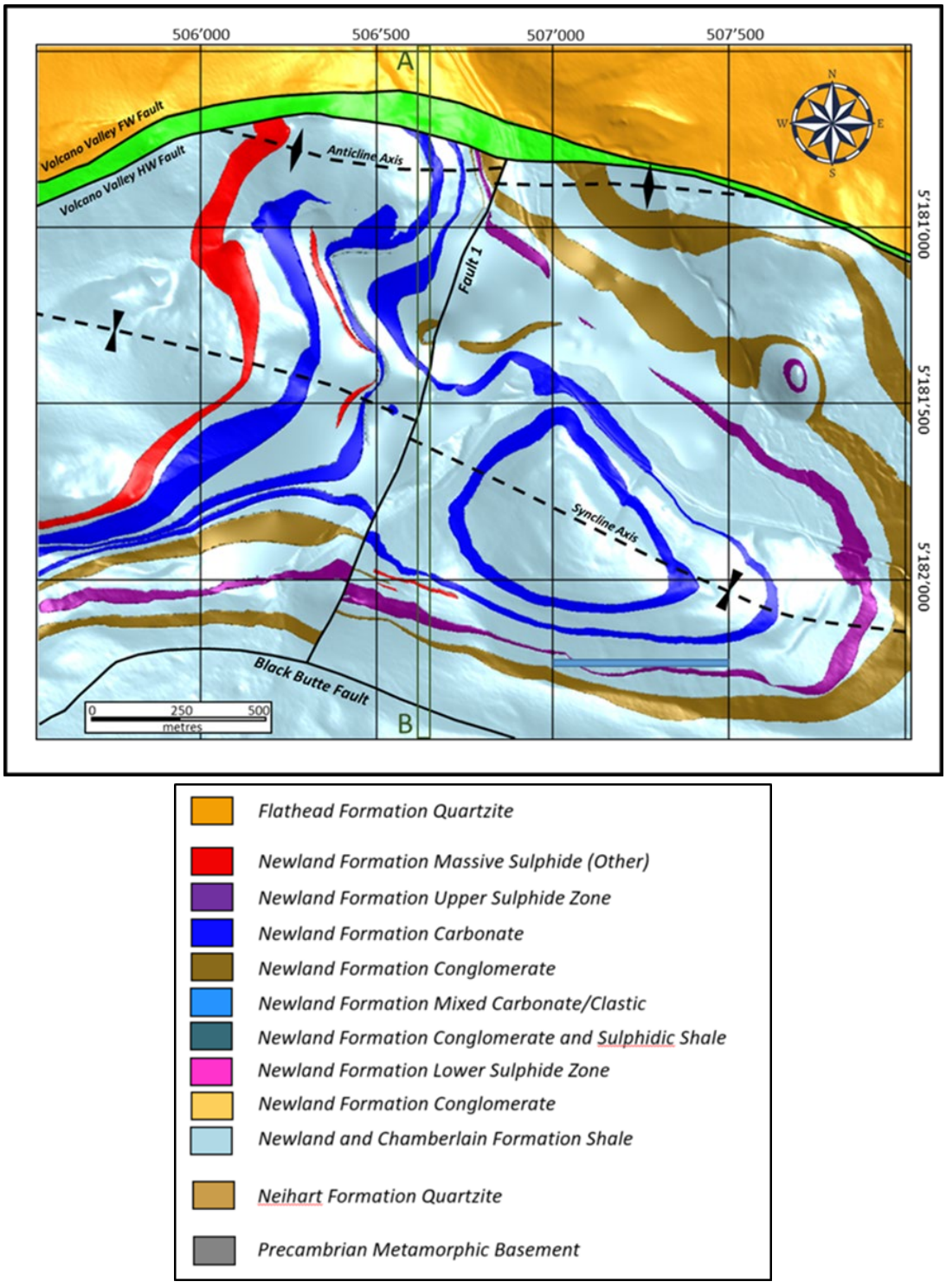


Figure 7.5 Plan view of the 3-D Lithostratigraphic model of the Johnny Lee deposit area showing major structures and the location of cross-section line A-B (Figure 7.6)

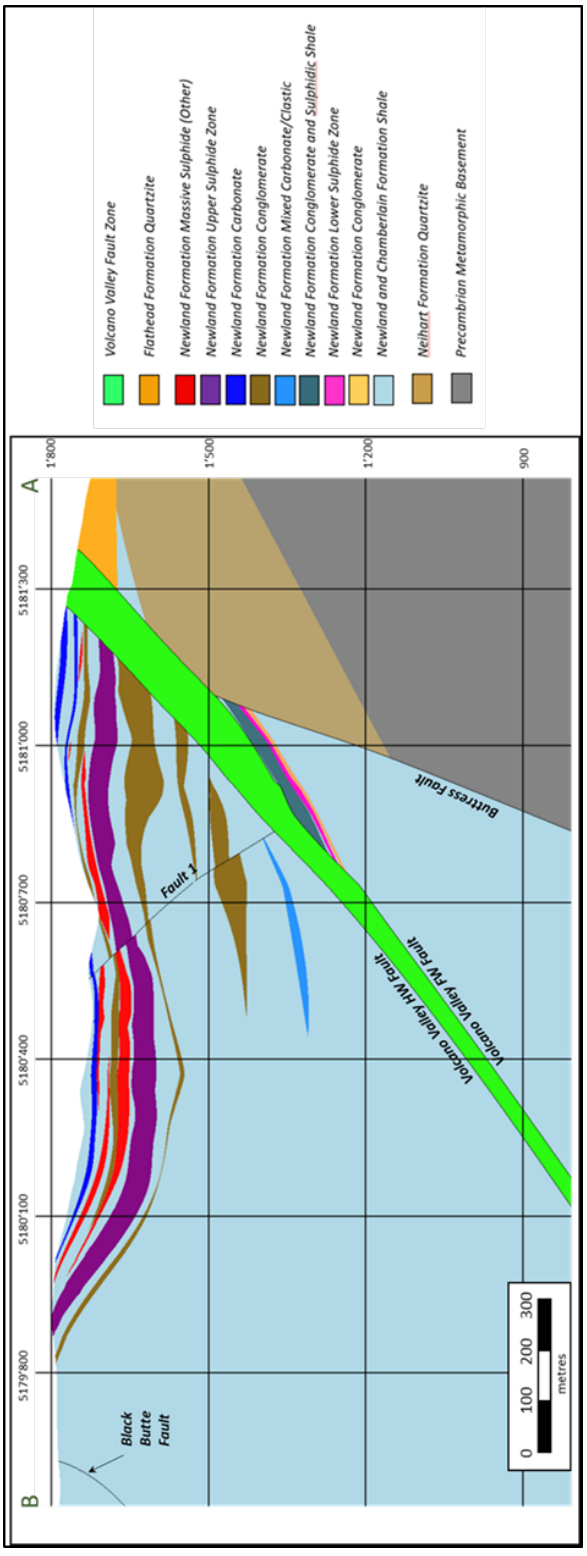


Figure 7.6 Cross-section from A to B, looking west, of the lithostratigraphic model shown in Figure 7.5

Folding

The Newland Formation strata in the Hangingwall (HW) of the VVFZ have been gently folded to form a syncline-anticline pair with upright, W/NW to E/SE striking axial planes (Figure 7.5 and Figure 7.6). In the Johnny Lee deposit area, these folds have a variable plunge that range from 2° to 15° to the W/NW. Within the greater Black Butte Copper Project area (Figure 7.2), folds are periclinal, plunging to both the W/NW and E/SE.

Folding was accompanied by the development of bedding parallel faults and brittle-ductile shear zones, consistent with folding by flexural-slip (Lebrun, 2018). Folding was accompanied by the development of fold-related longitudinal and diagonal joint sets (Lebrun, 2018).

Layer-parallel movement during folding by flexural slip has not caused significant displacement of mineralization but, outside of the fault zones that have been identified, poor geotechnical ground conditions can often be related to zones of closely spaced, shallow dipping flexural-slip fractures or steeply dipping fold-related joints. In zones where both steeply dipping and shallow dipping fold related fracture zones are developed, with a high fracture density, poor ground conditions may have developed.

Volcano Valley Fault Zone

Within the Project area (Figure 7.2 and Figure 7.5), the VVFZ is an arcuate, W/NW to E/SE striking, 50 to 170 m wide, southward dipping zone of brittle-ductile to brittle deformation. The edges of the VVFZ are marked by two significant faults termed the Volcano Valley HW and Footwall (FW) Faults. These faults are characterised by zones of clay-gouge and/or fault breccia that range in width from 2 to 10 m.

Between the two bounding faults, the VVFZ comprises a tectonic melangé of Neihart Formation, Newland Formation, and Chamberlain Formation lithofacies fault blocks bounded by brittle-ductile shear zones, brittle faults or brittle-ductile shear zones that have been reactivated as brittle faults. The brittle-ductile shear zones are characterized by zones of foliation, quartz-carbonate veining and mineral elongation lineations defined by rod-shaped aggregates of quartz and micacous minerals. The brittle faults comprise zones of clay gouge (typically in shales) or fault breccia (in more competent lithotypes) and often have slickenlines on bounding surfaces. Where both brittle-ductile and brittle structural elements are developed at the same location, the brittle elements overprint the brittle-ductile fabric elements.

Both mineral elongation lineations and slickenlines within the VVFZ dip at moderate angles (30° to 50°) to the west suggesting they formed during the same progressive deformation event. Kinematic indicators such as composite planar fabrics, slickenside steps, asymmetrical clasts and asymmetrical pressure shadows indicate oblique reverse-sinistral movement sense during deformation along the VVFZ (Lebrun, 2018). Accurate displacement on the VVFZ within the Project area has not been determined but based on stratigraphic juxtaposition of the upper Newland Formation and the Flathead Formation to the east of the Project area, a minimum of 2,500 m apparent displacement has occurred.

The Volcano Valley FW Fault truncates the UCZ deposit. The Volcano Valley HW Fault truncates the LCZ deposit (Figure 7.6).

Black Butte Fault

The Black Butte Fault has been mapped south of the Johnny Lee deposit (Figure 7.2 and Figure 7.5) and has been intersected in one drillhole, displaying an 8 m downhole width of brecciation and fault gouge. It is a curvi-planar fault striking and dipping sub-parallel to the VVFZ. It is interpreted to have developed during the same deformation event that formed the VVFZ and have similar kinematics. The Black Butte Fault does not impact any of the currently proposed mining and development activity.

Fault 1

Fault 1 is a N/NE striking, 70° to 75° west dipping fault that transects and displaces the UCZ. The fault zone comprises a 0.1 to 0.6 m wide zone of clay-gouge and breccia surrounded by a 0.7 to 3.0 m wide zone of fracturing. No kinematic indicators have been observed and, as core orientation typically fails within the fracture zone, no lineations have been measured within Fault 1.

As there is a high drill density in the vicinity of Fault 1, accurate lithological and mineralization wireframes have been constructed and have been used to determine movement vectors on the fault plane. The dip-slip component of movement was 20 m of reverse-sense displacement, the strike slip component was 120 m of dextral displacement. Fault 1 is thus a reverse-dextral fault with approximately 122 m displacement. Fault 1 is truncated by the VVFZ in the north and is interpreted to be truncated by the Black Butte Fault in the south.

With the exception of the Buttress Fault, all structural elements described above are interpreted to have formed by progressive deformation during the Cretaceous Laramide Orogeny. NE to SW orientated compression produced E/NE - W/NW trending folds. Further shortening led to the development of oblique reverse-sinistral faults, such as the Volcano Valley and Black Butte Faults. Although the oblique reverse-dextral Fault 1 is truncated by the VVFZ, and the Black Butte Fault, the similarity in fabric elements between Fault 1 and the VVFZ suggests that Fault 1 is a second-order fault that developed as a conjugate to the VVFZ during NE-SW directed compression.

7.3 Deposit Geology and Mineralization

Potentially economic Cu mineralization has been identified in the Newland Formation at the UCZ and the LCZ. The mineralization in both of these deposits is largely hosted by massive sulphide units referred to as the USZ and LSZ, respectively. Although not included in the MOP application, Cu mineralization also occurs at Lowry situated 3 km SE of the Johnny Lee deposit (Figure 7.2). Mineralization at Lowry is hosted by the LSZ and the MSZ. In order to discriminate between Johnny Lee and Lowry, the MSZ and LSZ at Lowry are herein referred to as Lowry Middle Sulphide Zone (LMSZ) and Lowry Lower Sulphide Zone (LLSZ). Mineralization at Lowry that occurs in these zones is referred to as the Lowry Middle Copper Zone (LMCZ) and Lowry Lower Copper Zone (LLCZ).

Cu mineralization in the USZ, MSZ, and LSZ is localized (i.e., does not occur throughout the entire stratigraphic unit), nor does it occur in the numerous other massive sulphide zones in the Newland Formation (Figure 7.4).

Whilst Cu mineralization in the southern part of the UCZ is mostly restricted to massive sulphide lithofacies, it is common for economically-significant Cu mineralization to occur in shale and conglomerate lithofacies, interlayered with the massive sulphide in the northern part of the UCZ. Cu mineralization also occurs in shale and conglomerate lithofacies adjacent to the LSZ and in shale, carbonate, and conglomerate lithofacies adjacent to the LMSZ.

McGoldrick and Zieg (2004) interpreted the bedded pyrite massive sulphides to have formed at syn-sedimentary hydrothermal vent sites during deposition of the host clastic sediments. Sulphides are deformed by soft-sediment folding and sulphide accumulations include evidence of vent biota. Barite is commonly observed within both massive sulphide units and sediment horizons and is interpreted to have formed during hydrothermal vent activity.

The massive sulphide units in the Newland Formation range from several metres of centimetre-scale beds interbedded with shale or massive lenses of >50% pyrite up to 40 m thick. The pyrite that forms these accumulations shows significant microtextural variation but can be broadly classified into three categories (McArthur, 2017a, 2017b, 2018, 2019):

- Framboidal: Spherical, biogenic forms sometimes coalescing into bands;
- Melnikovite: Porous, amorphous or sub-microcrystalline, often seen as colloform growths; and
- Crystalline: Euhedral or annealed showing planar crystal faces.

The framboidal and melnikovite forms are considered early (i.e., primitive). Subsequent hydrothermal activity resulted in increasing recrystallization until the presence of planar crystal faces leads the pyrite forms to be classified as crystalline (McArthur, 2018).

Non-supergene altered Cu mineralization occurs as chalcopyrite, tennantite, and bornite. Textural variations of these copper sulphide minerals reflect a complex paragenesis. Whilst minor amounts of copper sulphides occur in association with primitive pyrite (such as in colloform melnikovite bands), the vast majority of copper sulphides texturally overprint the early pyrite forms, typically replacing the gangue minerals and interstices surrounding the pyrite (McArthur, 2018). These observations are interpreted to indicate that whilst some copper sulphide mineralization accompanied formation of the bedded pyritic units, the majority was introduced during progressive hydrothermal activity that post-dated formation of the massive sulphide units.

In all three deposits (UCZ, LCZ, and Lowry), veinlets and disseminations of copper sulphides occur within shale, conglomerate, and carbonate lithofacies adjacent to bedded pyrite, as well as in the bedded pyrite itself. These Cu mineralizing events have been interpreted as either an extended progressive hydrothermal event(s), or copper sulphide remobilization during a later tectono-thermal event.

7.3.1 Johnny Lee Lower Copper Zone

The LCZ occurs at depths of 340 to 520 m below surface, strikes approximately EW and dips at 15° to 30° to the south. Mineralization in the LCZ is primarily hosted by the LSZ located in the FW of the VVFZ and HW of the Buttress Fault (Figure 7.7 and Figure 7.8). The LSZ is overlain by a unit comprising interlayered shale and conglomerate and is underlain by a conglomerate unit. The LCZ deposit comprises three lenses of mineralization termed the East, Central, and West Lenses (Figure 7.7). These lenses are defined by the outer limit of >2.0% Cu mineralization which extend outside of the LSZ into the HW intercalated conglomerate and shale unit (Figure 7.8 and Figure 7.9). Minor Cu mineralization also occurs in the conglomerate below the LSZ but does not exceed 2.0% Cu. Table 7.1 shows the approximate dimensions of the three mineralization lenses that collectively form the LCZ deposit.

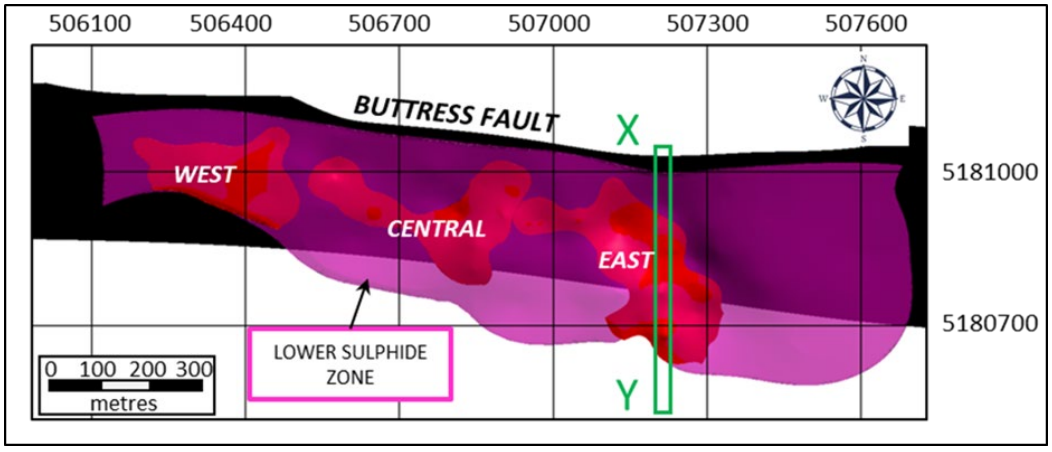


Figure 7.7 Plan view of the LSZ showing West, Central and East Lenses of the LCZ. The VVFZ has been removed

LCZ Segment	Strike Length (m)	Down Dip Extent (m)	Thickness (m)
East	450	45 to 250	1 to 15
Central	360	35 to 270	1 to 8
West	350	45 to 200	1 to 6

Table 7.1 Summary of LCZ dimensions

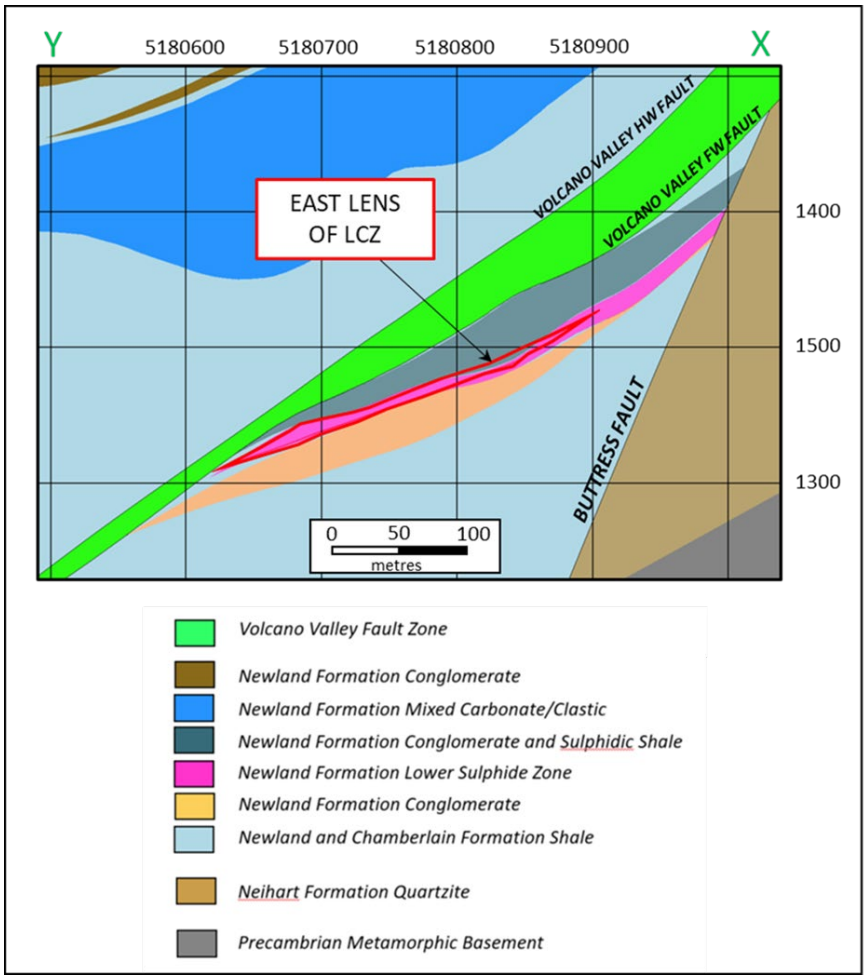


Figure 7.8 Cross-section from X to Y (Figure 7.7) of the East Lens of the LCZ, looking west

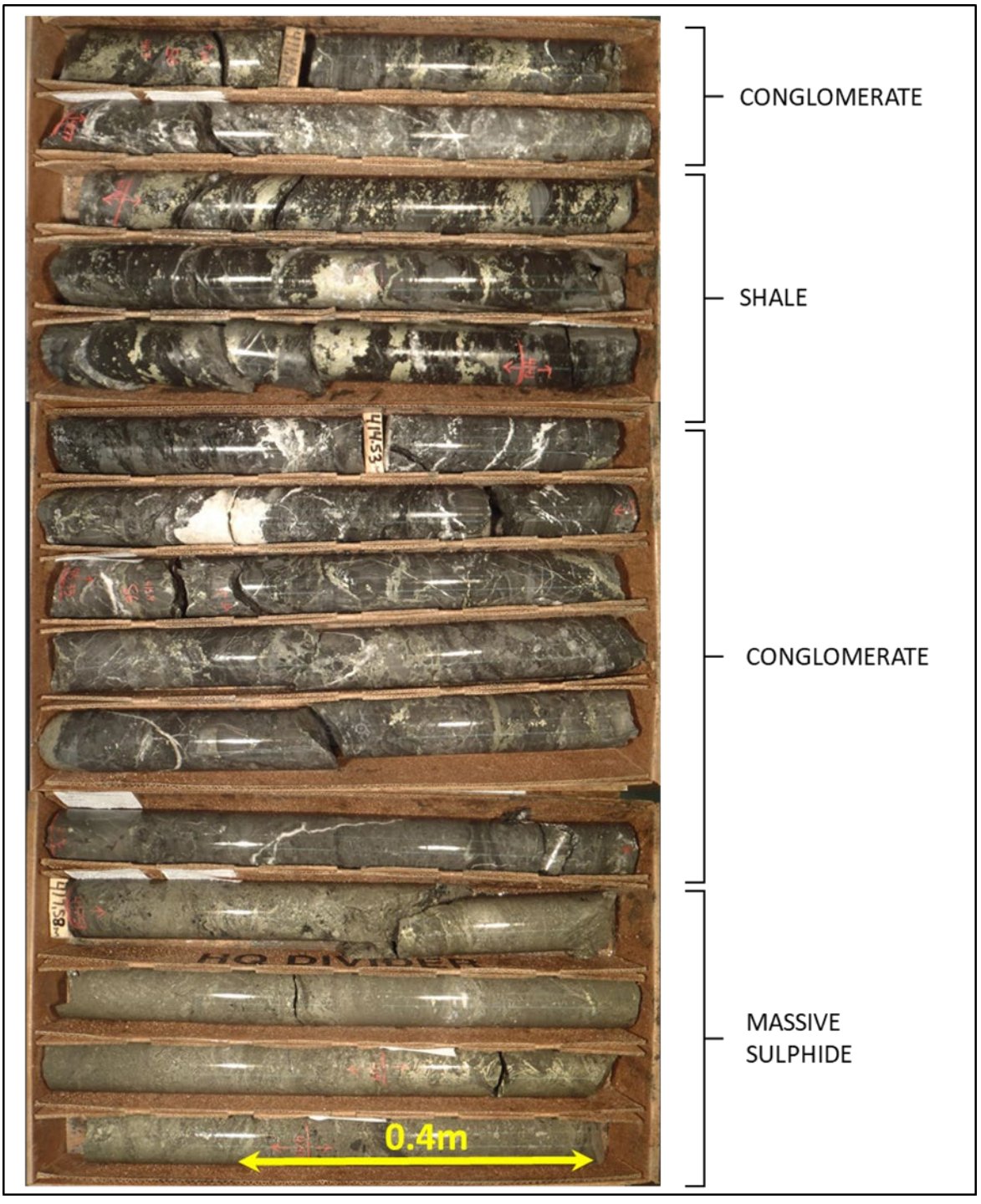


Figure 7.9 *Core photographs of drillhole SC11-011 from 411.2 m (top) to 419.7 m (base) showing chalcopyrite hosted Cu mineralization extending from massive sulphide of the USZ into interlayered shale and conglomerate*

Pyrite in the LCZ is predominantly crystalline, although minor relict, fine-grained, partially recrystallized framboidal or melnikovite pyrite occurs. Cu mineralization in the LCZ is almost entirely hosted by chalcopyrite that is coarse grained in comparison with that of the UCZ (McArthur, 2017b). Trace amounts of tennantite are rarely observed, typically interstitial to primitive pyrite.

Both siliceous and dolomitic alteration occur in association with the LCZ. Silica alteration is most intense in the conglomerate unit immediately below the LSZ where it presents as very fine-grained replacement of the host lithotype. Variable amounts of non-pervasive dolomitic alteration occur through the LSZ and into the overlying interlayered conglomerate and shale unit. The dolomitic alteration is characterized by zones of dolomite crystals up to 5 cm (Figure 7.10).

Figure 7.10 Coarse dolomite alteration in the LCZ in SC19-251 at 533.6 m depth

The previous Mineral Resource estimate for the LCZ (Winckers et al, 2013) included estimates for Ag, Au, and Co. Insufficient work has been done to determine the residency of these elements in the LCZ, although Steadman and Large (2015) noted that high Au (>30 g/t) and Co (>10,000 ppm) concentrations occur within discrete bands in colloform pyrite. Metallurgical tests conducted during this study were focused on Cu recovery with resultant Au, Ag and Co within flotation concentrates below current economic thresholds for the Project.

7.3.2 *Johnny Lee Upper Copper Zone*

The north-eastern corner of the UCZ is exposed on surface (Figure 7.11) and the top of the mineralized zone extends to a depth of 210 m below surface. The high-grade portion of the UCZ (>1.2% Cu) is entirely encapsulated by the USZ although >0.25% Cu Halo mineralization extends into the shale that is located in the HW and FW of the USZ (Figure 7.11). The UCZ is gently folded by a W/NW plunging syncline-anticline pair such that strike is variable and dip ranges between 0° and 20°.

With the exception of its extreme north-eastern corner, the UCZ is situated below the level of surficial oxidation. Acidic groundwater, preferentially focused along a layer parallel, brittle-ductile shear zone, resulted in localized supergene alteration of copper sulphides below the base of oxidation. The volume of supergene altered copper sulphide minerals along the shear zone is generally low, except at the junction of the shear zone with Fault 1.

The UCZ is truncated in the north by the Volcano Valley HW Fault and is transected by Fault 1 which offsets the UCZ with 122 m of oblique reverse-dextral displacement. Fault 1 has been used to subdivide the UCZ into Eastern and Western blocks (Figure 7.12). The UCZ in the Western block has plan-view dimensions of 1,000 m (NE-SW) by 200 to 440 m (NW-SE), in the Eastern Block 950 m (NE-SW) by 140 to 285 m (NW-SE). True width of the UCZ in the Western block ranges from 4 to 45 m and the Eastern block from 5 to 37 m.

Discontinuous lenses of conglomerate occur interlayered with pyritic massive sulphide in the USZ. Shale and carbonaceous shale layers, ranging in thickness from 1 mm to 2.5 m, are ubiquitous throughout the USZ. The proportion of clastic sediment layers is highly variable but shows an overall decrease from north to south.

The bedded pyrite in the UCZ exhibits a significantly larger degree of textural variation when compared with that of the LCZ; whereas the LCZ is dominated by crystalline pyrite, with only minor relict fragments of primitive textures (e.g., melnikovite or framboidal pyrite). UCZ samples show a complete continuum ranging from 26.6% to 96.9% volume primitive pyrite (McArthur, 2018). Although a significant amount of local variation occurs, there is a general trend of increasing pyrite crystallinity from the southern half of the orebody (average 72.1% primitive pyrite) to the northern part of the orebody (average 62.7% primitive pyrite) (McArthur, 2018).

Bedded pyrite in the USZ commonly contains layers of primitive pyrite (variably recrystallized colloform melnikovite or framboidal pyrite) ovoids (Figure 7.13) that when viewed in section present as small pyrite halos surrounding cores of later barite ± quartz infill (Figure 7.14). These intervals are termed Atoll-Textured Units (ATU's).

In contrast to the LCZ, where the Cu is hosted almost entirely by chalcopyrite (with rare occurrences of trace tennantite), the non-supergene altered copper sulphide mineralogy of the UCZ is both mineralogically and texturally diverse. The dominant copper sulphide mineral is chalcopyrite, comprising up to 15.8% (McArthur, 2018). Tennantite is present in concentrations of up to 1.88%, primary bornite up to 0.50%, and cuprian siegenite up to 0.85% (McArthur, 2018). Samples from the southern half of the UCZ orebody have lower chalcopyrite concentrations (average 3.60%) when compared with the northern half of the orebody (average 6.33%), higher tennantite concentrations (average 0.49% in the south and 0.32% in the north), higher bornite concentrations (average 0.12% in the south and trace amounts in the north) and lower cuprian siegenite (average 0.05% in the south and average 0.13% in the north) (McArthur, 2018).

Figure 7.15 shows all mineralogical composites in which supergene alteration of the copper sulphides exceeding 0.1% was observed. In these composites, primary copper sulphides are partially altered to one or more of: secondary bornite, chalcocite, and covellite (McArthur, 2018). In cross-section (Figure 7.16), it is apparent that the supergene alteration is focused along a layer-parallel, brittle-ductile shear zone identified by Lebrun (2018). Fracturing that formed during development of the shear zone has led to water infiltration and supergene alteration. The amount of supergene alteration of the copper sulphides is <5% except where Fault 1 intersects and offsets the shear zone (Figure 7.17). In this area, amplified fracturing in the intersection zone has resulted in enhanced water infiltration resulting in between 17% to 90% supergene alteration of primary copper sulphides.

Table 7.2 shows the Cu metal deportment for the northern and southern parts of the UCZ as determined using 86 mineralogy composites from ore intersections, with average mineral Cu content determined by electron microprobe and high-resolution Scanning Electron Microscope (SEM) microanalyses (McArthur, 2018). The majority of the Cu in both the northern and southern halves of the orebody is hosted by chalcopyrite but in the southern part of the orebody, a much larger contribution is made by tennantite and bornite (both primary and secondary). An average of 3.4% of the Cu occurs in pyrite grains. The contribution to total Cu metal from chalcocite and covellite is higher in the southern half of the orebody reflecting more intense supergene alteration at the intersection of the shear zone with Fault 1. Although only a small proportion of the contained Cu, cuprian siegenite and marcasite contribute higher amounts of Cu in the northern half of the orebody.

Cu Metal Distribution %									
	Chalcopyrite	Tennantite	Pyrite	Bornite	Chalcocite	Covellite	Cuprian Siegenite	Colusite	Marcasite
North	90.1	5.4	2.6	0.0	0.3	0.3	1.0	0.0	0.3
South	77.0	12.0	3.9	3.6	2.1	1.0	0.2	0.1	0.1
Total	81.8	9.6	3.4	2.3	1.5	0.8	0.5	0.1	0.1

Table 7.2 Cu metal distribution for the UCZ based on mineralogical study

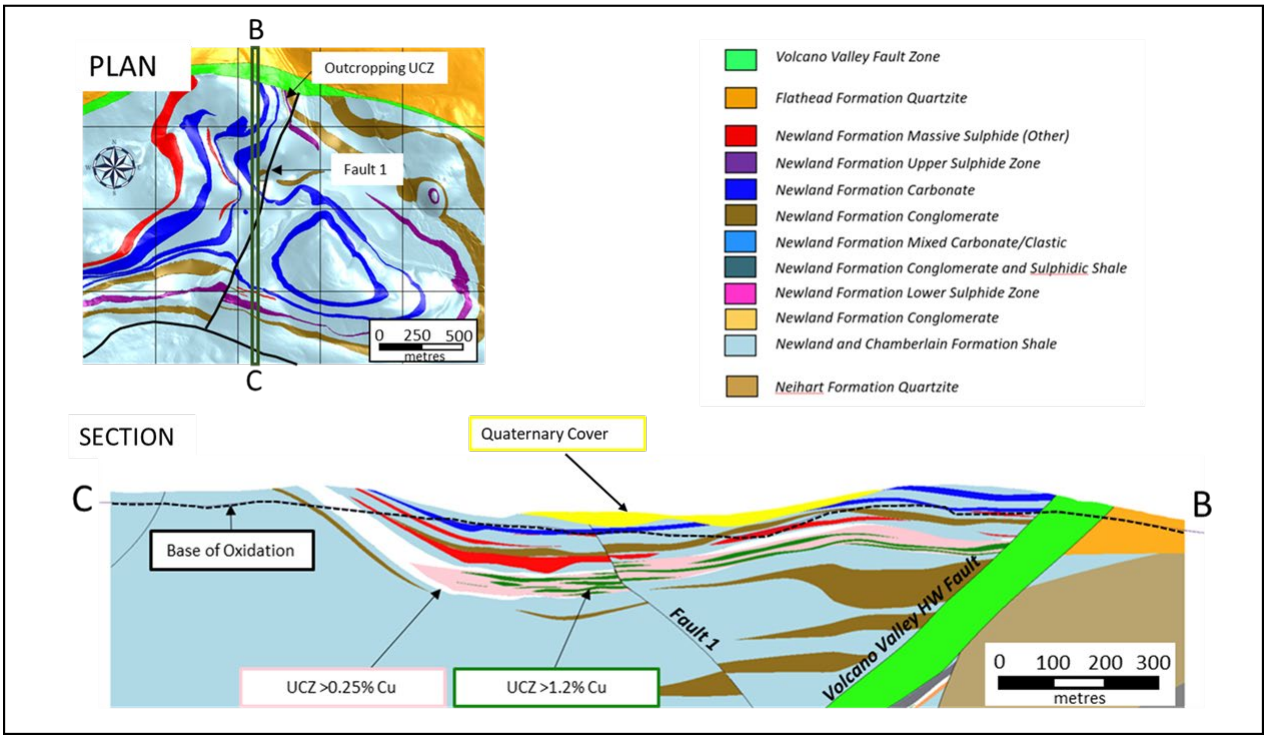


Figure 7.11 Plan view and cross-section of the lithostratigraphic model in the UCZ area. USZ (purple in plan-view) has been made transparent in section such that the UCZ can be observed

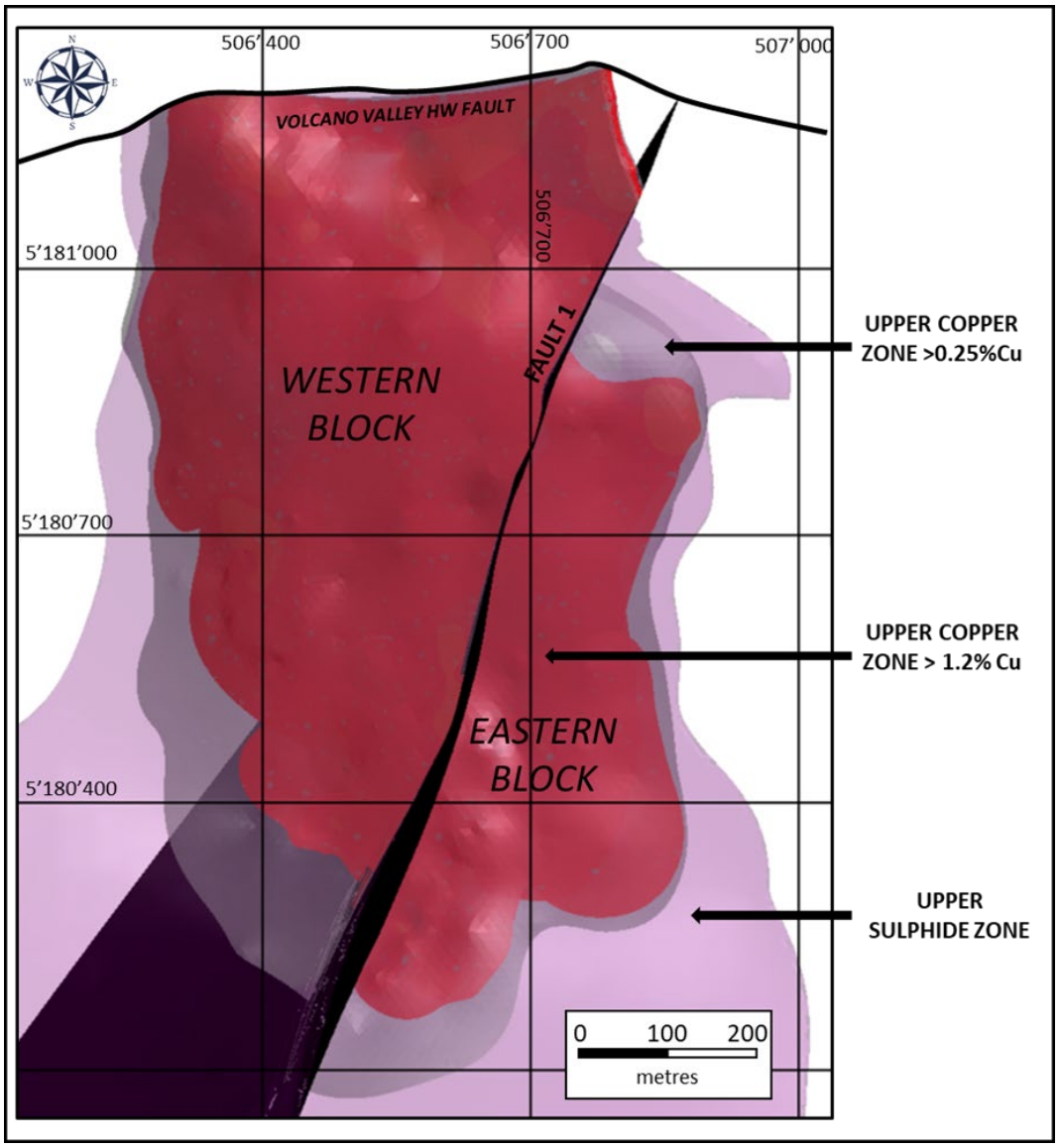


Figure 7.12 Plan view of the UCZ and part of the USZ

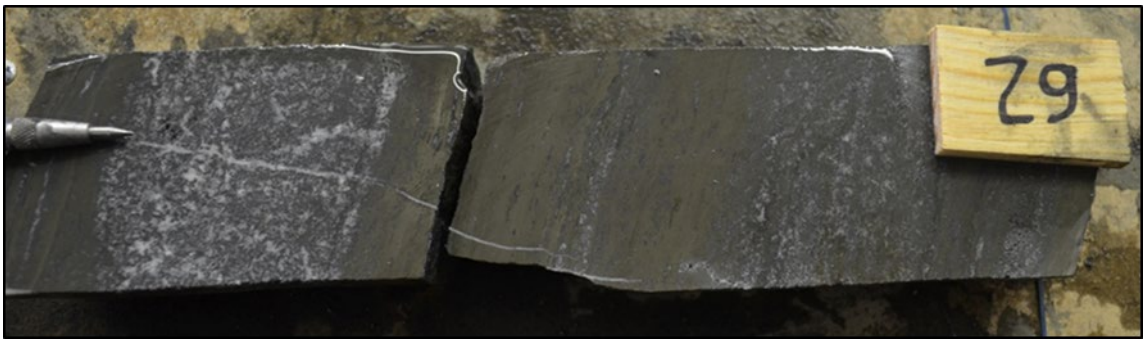


Figure 7.13 Photograph of USZ bedded pyrite massive sulphide containing light colored ATU's

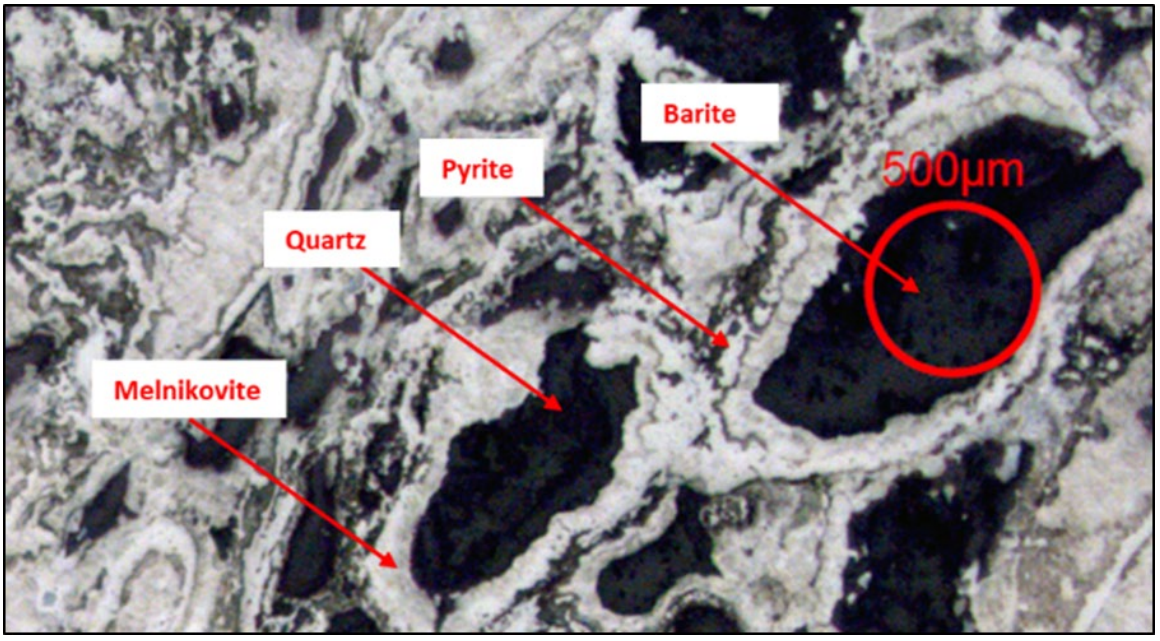


Figure 7.14 Photomicrograph of sample BBPET50 (drillhole SC12-121 at depth 118.2m) showing Atoll texture

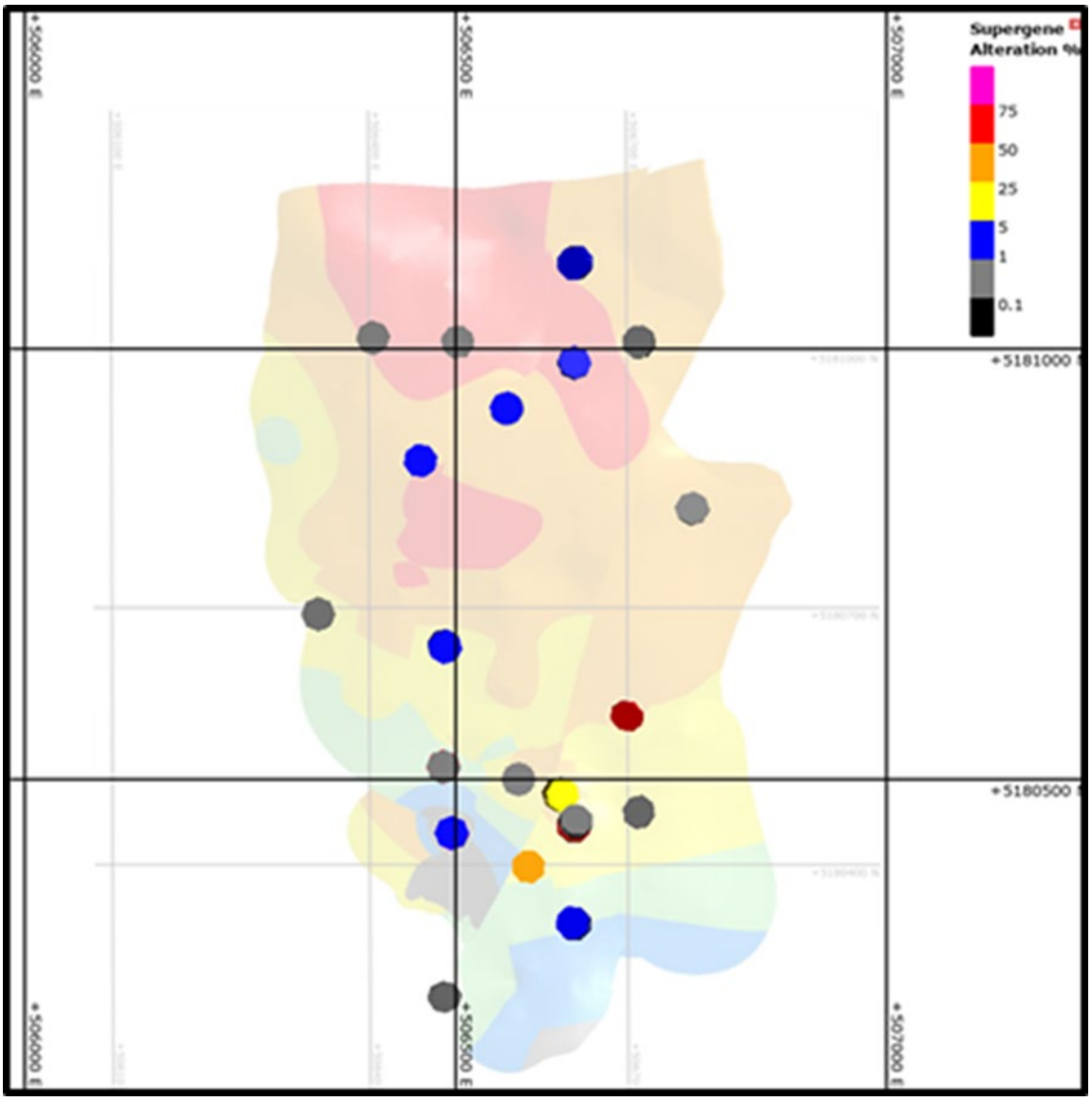


Figure 7.15 Plan view of the UCZ showing mineralogy composites that exhibit supergene alteration, colored by percentage of supergene alteration. UCZ contoured using a Radial Base Function recovery model

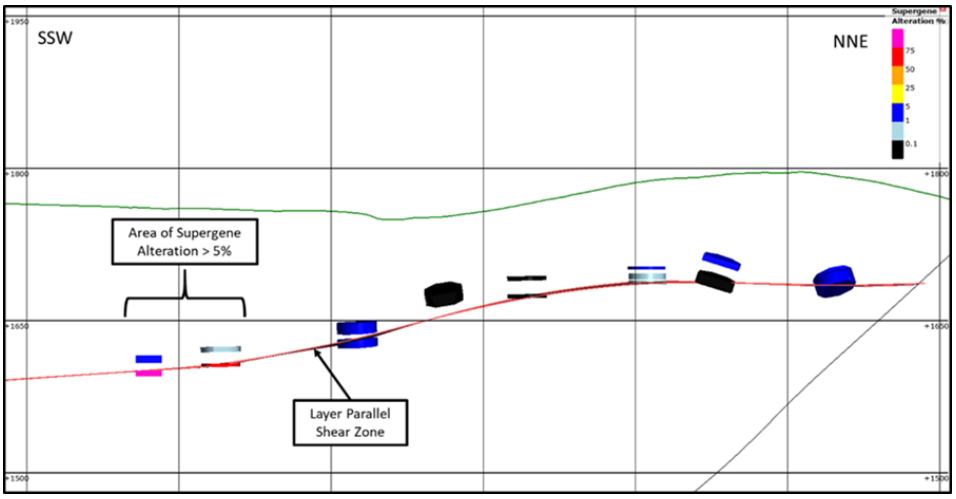


Figure 7.16 *N/NE – S/SW cross-section of the UCZ showing supergene alteration of copper sulphides developed along the brittle-ductile shear zone*

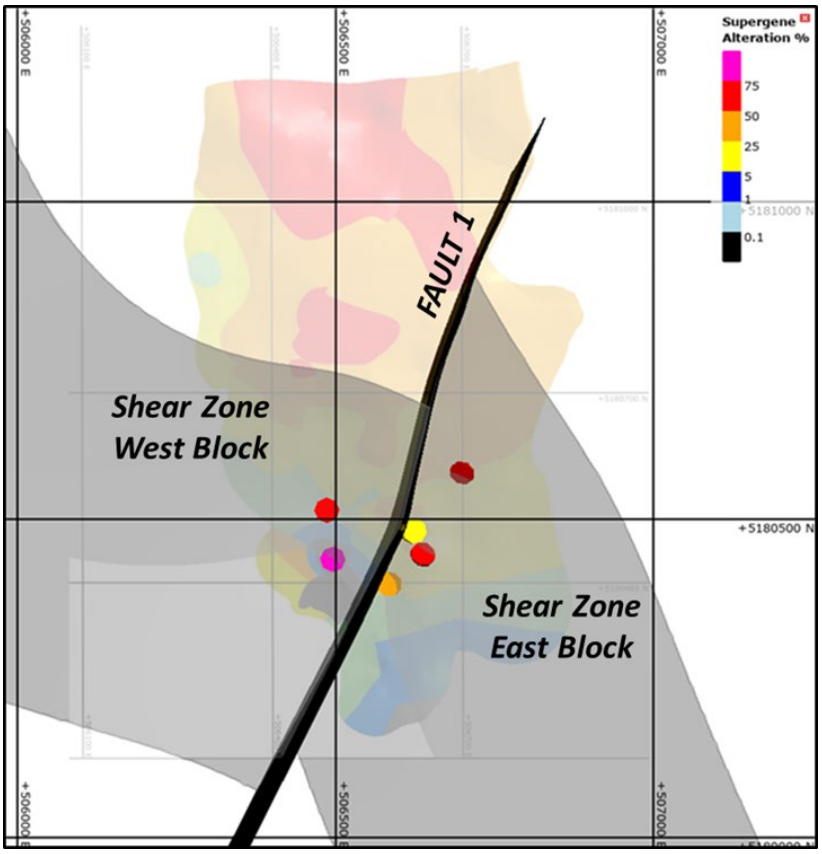


Figure 7.17 *Plan view of UCZ showing mineralogical composites with >5% supergene alteration of copper sulphides*

The mineralogical differences, albeit with significant local variability, between the northern and southern parts of the UCZ are also accompanied by differences observable in core samples. Representative observations regarding pyrite texture (proportions of primitive versus crystalline) and the amount of fine-grained copper sulphide (which includes most tennantite and an appreciable amount of the chalcopyrite) are physically not possible, even when using a pocket microscope (this was attempted during geometallurgical logging of all UCZ drillholes from the 2010 to 2018 Phase 1 Drill Programs but proved unsuccessful). The metrics that can be consistently recorded during logging are the proportions of; bedded pyrite, sediment, ATU's, and coarse (i.e. visually observable) chalcopyrite.

Coarse chalcopyrite in the southern half of the UCZ typically occurs as irregular veinlets that cross-cut layering in the bedded pyrite that have infiltrated and caused partial or complete replacement of bedding layers within the USZ (Figure 7.18). Whilst this does occur along bedded pyrite layers it is much more prevalent along ATU's (Figure 7.19).

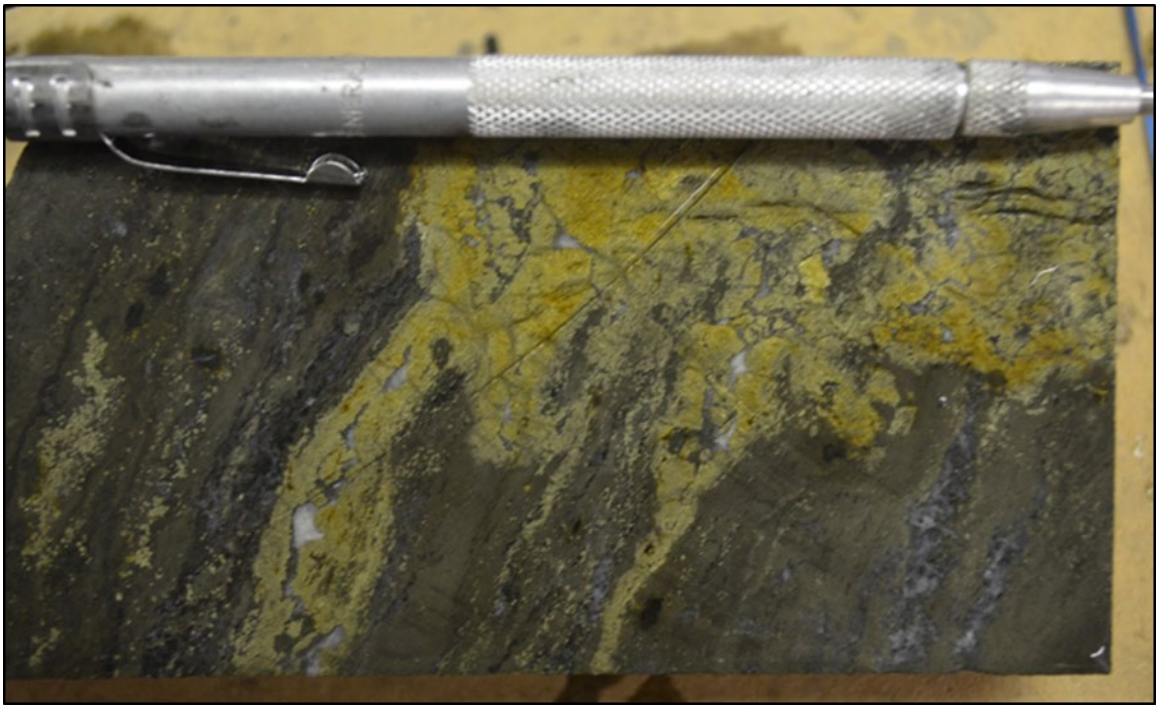


Figure 7.18 Coarse chalcopyrite veinlet cross-cutting layered bedded pyrite with narrow ATU layers. Note the replacement along certain layers.

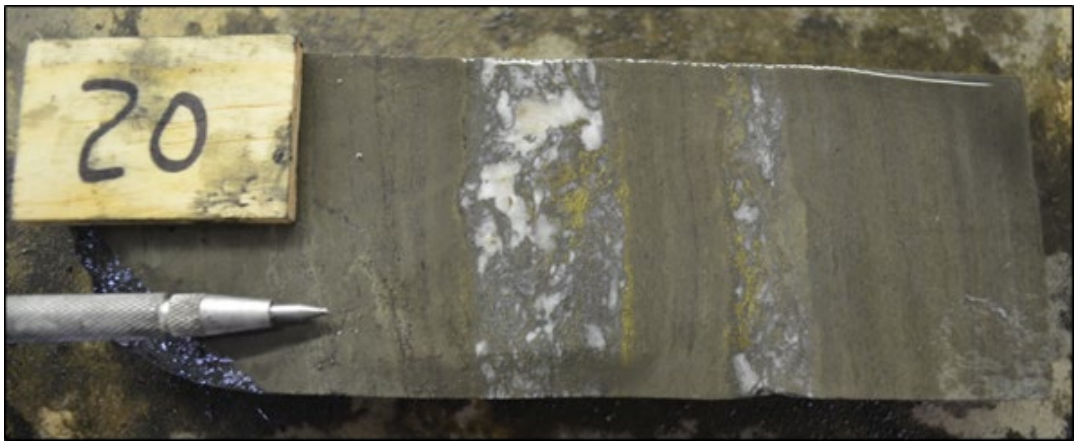


Figure 7.19 Preferential replacement of ATU layers within bedded pyrite of the UCZ

Attempts to correlate the proportion of coarse chalcopyrite with Cu grade in the southern part of the UCZ were unsuccessful. Typically, the calculated Cu grade (from visual estimates of coarse chalcopyrite percentage) was significantly lower than the analyzed grade. In certain drillhole intersections (typically bedded pyrite with no ATU's), grades in excess of 2.0% Cu occurred with no visible coarse chalcopyrite. Coarse chalcopyrite was rarely observed in sediment horizons in the southern part of the UCZ.

Similar observations to those in the southern part of the UCZ were also observed in the northern part of the UCZ but there are a number of additional macro-scale features. The mineralization in the northern part of the UCZ exhibits extensive brittle fracture, quartz ± carbonate veining, and hydraulic brecciation. Whilst some of the coarse chalcopyrite is cross-cut by the fracturing and veining (Figure 7.20), a significant amount appears to have either been introduced, or remobilized, during the fracturing, veining, and hydraulic brecciation (Figure 7.21 to Figure 7.24).

Figure 7.20 Coarse chalcopyrite, cross-cut by quartz-carbonate veins. Core diameter = 63.5 mm

Figure 7.21 Coarse chalcopyrite that is synchronous with hydraulic brecciation, fracturing and quartz-carbonate veins. Core diameter = 63.5 mm

Figure 7.22 Interbedded pyrite and siltstone from the northern part of the UCZ. Coarse chalcopyrite and crystalline pyrite have replaced tabular barite crystals. Coarse chalcopyrite also forms veinlets cross-cutting bedding. Core diameter = 63.5 mm

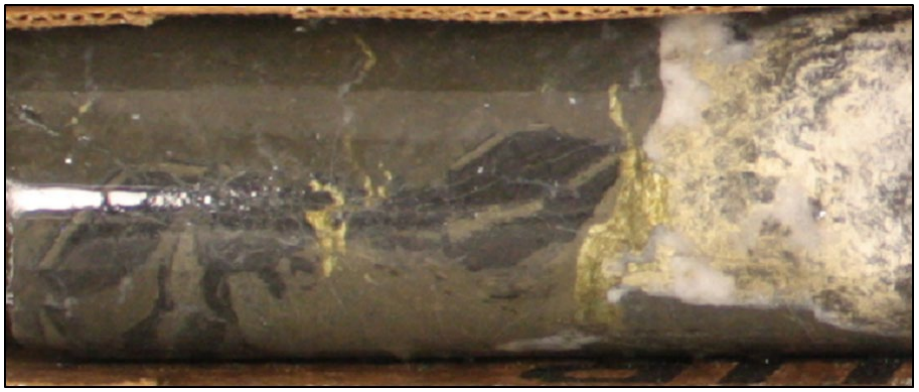


Figure 7.23 Coarse chalcopyrite both synchronous and post-dating quartz-carbonate veining in the northern part of the UCZ. Core diameter = 63.5 mm

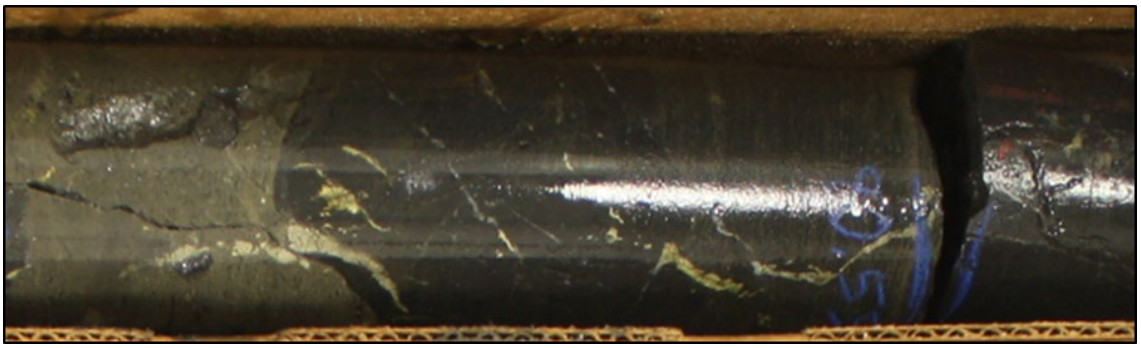


Figure 7.24 Quartz + carbonate + coarse chalcopyrite veinlets cross-cutting siltstone and bedded pyrite in the northern part of the UCZ. Core diameter = 63.5 mm

Although some localized quartz-carbonate veining and hydraulic brecciation occur in the southern part of the UCZ, it is more consistently developed in the northern part of the UCZ (Figure 7.25). The mineralogical differences observed between the northern and southern parts of the UCZ are thus mirrored by core observations that reflect post-lithification hydrothermal activity that occurred primarily in the northern part of the UCZ.

As only the proportions and grain size of coarse copper sulphide were estimated from core, the metrics for all other copper sulphides, and fine-grained chalcopyrite have been constrained using mineralogical investigation (McArthur, 2018). Table 7.3 summarizes the average grain-size of all minerals for the northern and southern parts of the UCZ. In terms of average grain-size of copper sulphide minerals (e.g., chalcopyrite, tennantite, cuprian siegenite, bornite, chalcocite and covellite), those in the southern half of the UCZ are smaller than those of the northern half.

	Gangue	Pyrite	Chalcopyrite	Cuprian Siegenite Marcasite	Tennantite	Bornite
	(µm)					
North	43	33	29	25	19	15
South	38	38	22	13	15	11

Table 7.3 Average grain size of all minerals in the northern and southern parts of the UCZ based on systematic mineralogy by McArthur (2018)

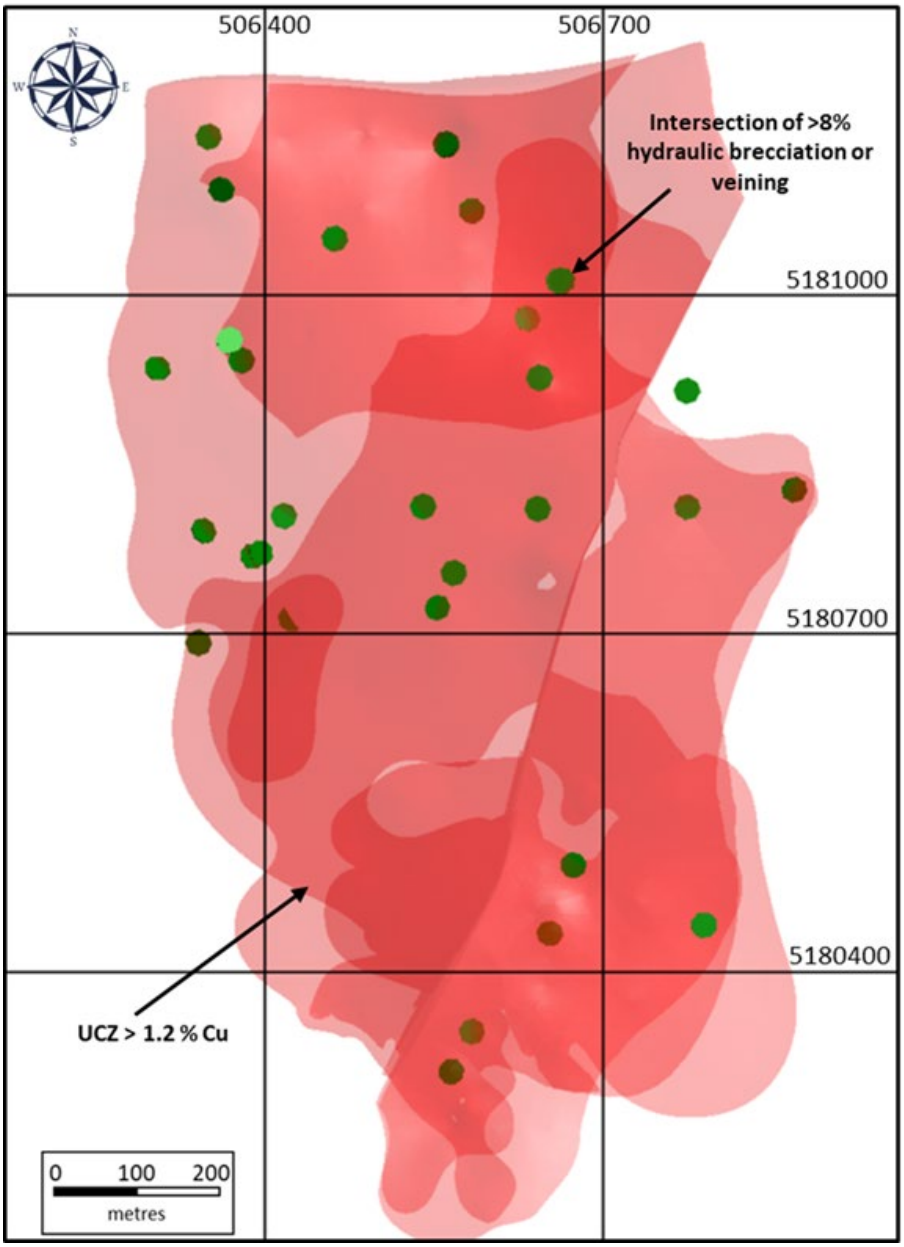
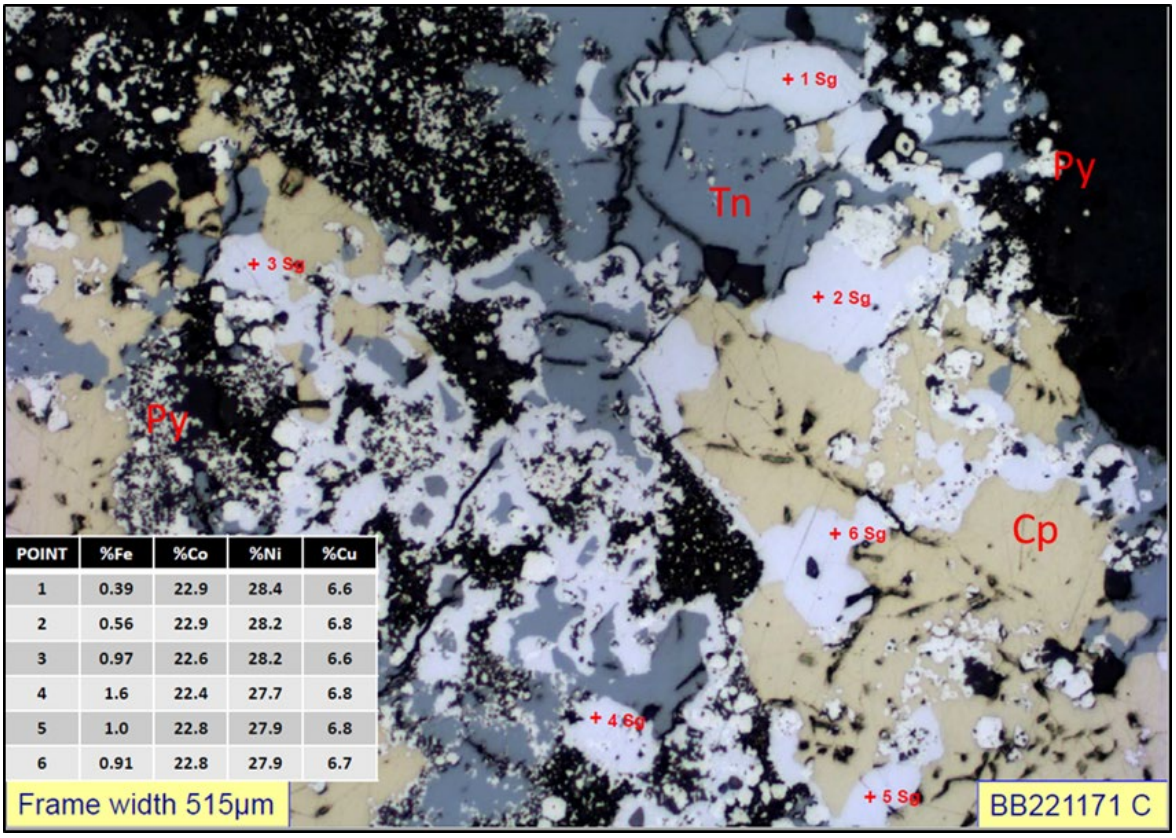


Figure 7.25 Plan view of UCZ showing drillhole intersections that have >8% quartz-carbonate veining and hydraulic brecciation

While the average grain size of the copper sulphides differs in the northern and southern parts of the UCZ, there is a significant amount of local textural variation. Whilst much of the southern part exhibits very fine-grained copper sulphides, there are other parts that have coarse-grained copper sulphides. The reverse relationship is observed in the northern part of the UCZ (Hilliard, 2019; Ronald and Malhotra, 2019).

The 2013 Mineral Resource estimate for the UCZ included estimates for contained Co, Ag, and Au (Winckers et al, 2013). Estimated Au grades for the UCZ were very low (7 to 9 parts per billion [ppb]) and have no economic significance. Co grades reported were 0.10% to 0.13% and Ag grades were 15.2 to 16.3 g/t (Winckers et al, 2013), both of which have potential economic significance.

SEM and Electron Microprobe analyses of UCZ composite samples (McArthur, 2018) has indicated that Co is resident in pyrite (average 0.11% Co), tennantite (average 0.28% Co), and cuprian siegenite (average 23.1% Co). The average weight percent of these minerals in the UCZ is pyrite 50.13%; tennantite 0.45%, and cuprian siegenite 0.07% (McArthur, 2018). Approximately 76% of the Co in the UCZ occurs in pyrite, 23% in cuprian siegenite, and 2% in tennantite. Although cuprian siegenite is not observed in every composite; where it has been observed, it is closely associated with chalcopyrite and/or tennantite (Figure 7.26) and shows similar textural relationships to pyrite and gangue minerals.



POINT	%Fe	%Co	%Ni	%Cu
1	0.39	22.9	28.4	6.6
2	0.56	22.9	28.2	6.8
3	0.97	22.6	28.2	6.6
4	1.6	22.4	27.7	6.8
5	1.0	22.8	27.9	6.8
6	0.91	22.8	27.9	6.7

Figure 7.26 ***Reflected light photomicrograph of an UCZ ore sample showing microprobe analyses points for cuprian siegenite (sg) and analytical results. Tn = tennantite, Cp = chalcopyrite, Py = pyrite. (McArthur, 2018)***

Marcasite micro-analyses returned average Ag concentrations of 200 ppm and rare sphalerite grains had average Ag concentrations of 100 ppm (McArthur, 2018). Laser Ablation, Inductively Coupled Plasma Mass-Spectrometry (LAICP-MS) microanalyses by Steadman and Large (2015) reported increased Ag concentrations (100 to 1,000 ppm) from crystalline pyrite and the rims of primitive pyrite whereas lower values of 10 to 100 ppm reported from pyrite cores. No Ag analyses above detection limit were reported from tennantite. Therefore, it is concluded that the Ag in the UCZ is hosted by late crystalline pyrite and marcasite.

Electron microprobe analyses of UCZ composites showed that As occurs in commonly observed pyrite, marcasite, and tennantite, as well as in trace minerals colusite and enargite (McArthur, 2018).

As concentrations in pyrite vary between the three main textural types. Melnikovite contains on average 0.23% As, whereas framboidal and crystalline forms contain, on average, 0.04% and 0.03% As respectively (McArthur, 2018). Pyrite content in the northern part of the UCZ (31.1%) is lower than that of the southern part (42.4%) (McArthur, 2018). The average weight percent of crystalline pyrite in the northern and southern parts of the UCZ is similar at 11.6% versus 11.8%, whereas those for melnikovite (north = 1.6%, south = 2.2%) and framboidal (north = 17.9%, south = 28.5%) differ significantly (McArthur, 2018).

The As concentration of marcasite in the UCZ averages 0.06% As and there is a higher proportion of marcasite in the northern part of the UCZ (1.98%) than that of the southern part (0.39%).

Tennantite in the UCZ is an As-rich form (18.91% As) with mean composition, determined by McArthur (2018) of:

$$(Cu_{10.34}Fe_{1.22}Co_{0.07}Zn_{0.05})_{11.68}(As_{3.64}Sb_{0.05}Bi_{0.07})_{3.76}S_{13}$$

The northern part of the UCZ contains an average of 0.32% tennantite whereas the southern part contains an average of 0.49% (McArthur, 2018).

Integrating the observations above allows the average As deportment for the southern and northern parts of the UCZ to be estimated (Table 7.4). In terms of average As deportment, the majority occurs within tennantite. The difference between the southern and northern parts of the UCZ largely reflects the average lower proportion of tennantite observed in the north.

		Northern UCZ		Southern UCZ	
	As Microanalysis (%)	Mineral %	Grade Contribution %As	Mineral %	Grade Contribution %As
Melnikovite Pyrite	0.23	1.6	0.004	2.2	0.005
Framboidal Pyrite	0.04	17.9	0.007	28.5	0.011
Crystalline Pyrite	0.03	11.6	0.004	11.8	0.004
Marcasite	0.06	2.0	0.001	0.4	<0.001
Tennantite	18.91	0.3	0.061	0.5	0.086
		TOTAL	**0.077**	**TOTAL**	**0.106**

Table 7.4 Average As content of main As containing mineral species and proportions of those mineral species in the northern and southern portions of the UCZ

7.3.3 Lowry Middle Copper Zone

Drillhole intersection spacing in the LMCZ ranges from 40 – 100 m. The LMCZ is hosted by a succession of ferruginous sediment (massive sulphide and ferruginous shales) with interbedded conglomerate, carbonaceous shale and shale lithofacies. The LMCZ occurs in the HW of the VVFZ (Figure 7.27).

Three zones of >1.2% Cu mineralization occur in the LMCZ, termed LMCZ Vein 1 – LMCZ Vein 3 (Figure 7.27). These mineralized zones dip to the south at 25 - 30°. The >1.2% Cu zones are surrounded by a zone of >0.25% Halo mineralization. The northern tip of the LMCZ occurs at 245 m below surface and the southern tip occurs at 755 m below surface. The LMCZ is situated below the depth of surficial oxidation.

The LMCZ has plan view dimensions of 830 m (NS) by 280 m (EW). The maximum true thickness of the LMCZ is 45 m and it progressively reduces in width and pinches out to the north, south, east, and west.

Mineralization in the LMCZ occurs in both massive sulphide and the interlayered clastic sediment. Pyrite + Marcasite concentrations of typical mineralized intersections range from 9.1 – 45.2% and gangue mineral contact ranges from 38.7 – 76.0% (McArthur, 2019). Copper sulphide mineralization is predominantly chalcopyrite (8.3 – 17.0% in typical mineralized intersections) with minor tennantite (0.01 – 0.23%) observed in 30% of mineralogical samples studied to date (McArthur, 2019). The amount of primitive pyrite in the LMCZ ranges from 10.4 – 60.0% (average 49.1%) which is lower than that of the northern part of the UCZ (average 62.7%) indicating that the amount of recrystallization in the LMCZ is higher than that of the northern part of the UCZ. From a mineralogical and copper sulphide liberation perspective, the LMCZ may be considered similar to those portions of the northern part of the Johnny Lee UCZ that have the largest amount of recrystallised coarse chalcopyrite and low tennantite.

7.3.4 *Lowry Lower Copper Zone*

Many of the drillholes that intersected the LMCZ were stopped-short of the LLCZ, consequently drillhole spacing in the LLCZ is larger than that of the LMCZ, ranging from 60 – 200 m. The LLCZ occurs in the FW of the VVFZ (Figure 7.27) and both the host and mineralization is truncated in the SE by the Volcano Valley FW Fault.

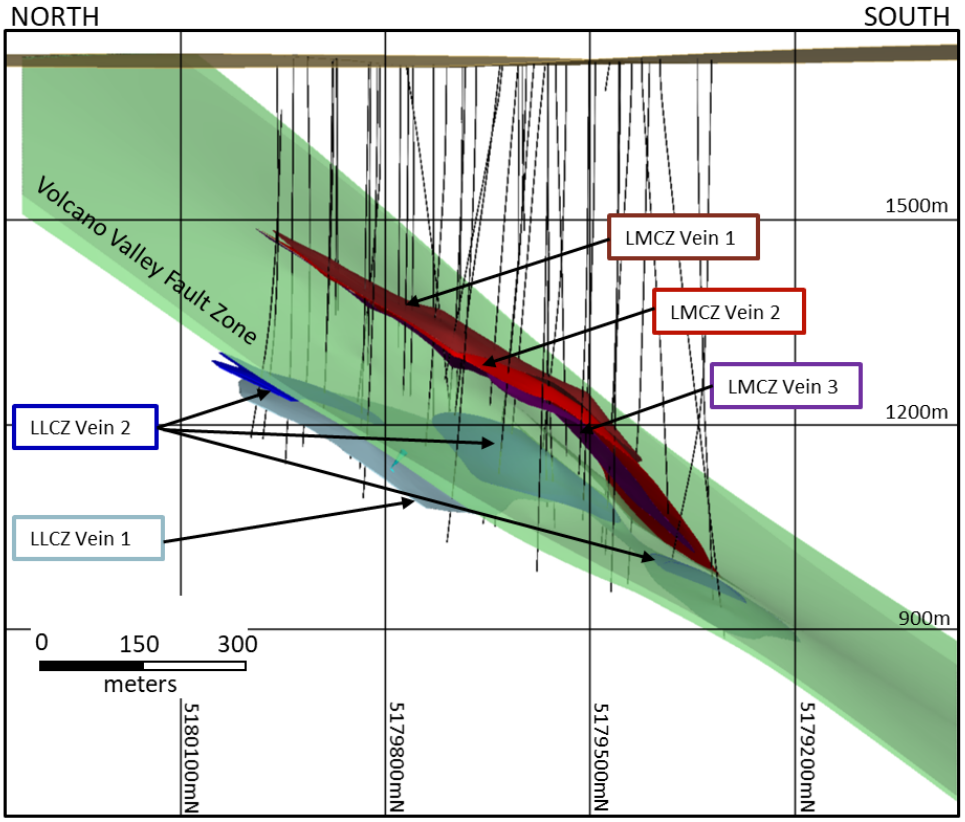


Figure 7.27 ***View of Lowry, looking east, showing the LMCZ and the LLCZ. The VVFZ has been rendered transparent such that the LLCZ can be observed***

The LLCZ is hosted by a succession of ferruginous sediment (massive sulphide and ferruginous shales) with interbedded shale and conglomerate lithofacies. Two bedding-sub-parallel zones of >1.2% Cu mineralization occur within the LLCZ: LLCZ Vein 1 and LLCZ Vein 2 (Figure 7.27). Although some halo style mineralization is developed in the LLCZ, the drilling to date has shown this to be localized. No halo mineralization has been modelled for the LLCZ. The mineralized zones dip to the SW at 25 – 40°.

LLCZ Vein 1 has plan view dimensions of 890 m (NS) and 80 – 630 m (EW). LLCZ Vein 1 ranges in thickness from 1 - 9 m.

LLCZ Vein 2 is discontinuous and forms three lenses. The dimensions of these lenses are as follows:

Lens A: Plan View Dimensions: 330 m (NW-SE) X 130 – 180 m (NE-SW)

True Width: 1 – 5 m

Lens B: Plan View Dimensions 280 m (NW-SE) X 100 – 250 m (NE-SW)

True Width: 1 – 10 m

Lens C: Plan View Dimensions 150 m (NW-SE) X 100 m (NE-SW)

True Width: 1 – 6 m

No mineralogical study has been completed for the LLCZ, visually, the mineralization in the LLCZ presents very similar to that of the LMCZ.

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8. DEPOSIT TYPES

8.1 Mineral Deposit

The Black Butte Copper Project deposits exhibit attributes of both Sedimentary Exhalative sulphide deposits (SEDEX deposits, e.g. Emsbo *et al*, 2016) and Sediment-hosted Stratabound Copper (SSC) deposits (SSC deposits, e.g., Hayes *et al*, 2015).

The deposits feature large pyrite-rich sulphide lenses that occur within marine sediments deposited in a continental rift, a host lithofacies, and paleo-tectonic setting consistent with that of SEDEX deposits. Whereas SEDEX deposits are commonly Pb- and Zn- rich and form on or near the seafloor (Leach *et al*, 2005), the deposit is enriched in Cu-Co-Ag and lacks significant Pb-Zn mineralization. Textural evidence indicates that some copper sulphides at Johnny Lee and Lowry formed synchronous with primitive, early pyrite but that the majority of Cu-Co-Ag sulphide mineralization occurred by replacement of early pyrite and that mineralization/remobilization continued post-burial and lithification.

The Johnny Lee and Lowry deposits share some features with a sub-class of SSC deposits termed reduced-facies SSC deposits: Cu-Co-(Ag) mineralization hosted by reduced, organic- and pyrite-bearing shale, silt and carbonaceous dolomitic siltstone (Hayes *et al*, 2015). SSC deposits are epigenetic, and mineralization is typically found as pore fillings or replacement of existing minerals. Mineralization in typical SSC deposits generally shows a zonation from relatively Cu-rich at the base (native copper, chalcocite, digenite) to more iron-rich at the top (i.e. chalcopyrite). No zonation is evident in the deposits but the association of copper sulphide mineralization with post-lithification veins and hydraulic brecciation supports a partially epigenetic origin.

The deposits are considered hybrid SSC-SEDEX deposits.

8.2 Geological Model

The Black Butte deposits have been interpreted to have formed in two stages: an early Fe- and Ba-rich hydrothermal event, and a later Si- and Cu-rich event (Graham *et al*, 2012; White *et al*, 2014). The early hydrothermal system vented onto the seafloor where it precipitated extensive sheets of fine-grained pyrite with minor copper sulphide and associated barite. The second pulse of hydrothermal activity locally replaced the earlier pyrite with crystalline pyrite, introduced considerable copper sulphide and caused silica ± dolomite alteration of the host rock.

The Helena Embayment (Figure 8.1) was asymmetrical, with deep water and a steep drop-off in the south and more subdued terrain in the north (Schmidt and Garihan, 1986; Zieg, 1986). Heat from dykes and sills emplaced along the southern edge of the embayment during basin extension, are interpreted to have created metalliferous basinal brines (Schmidt and O'Neil, 1982). The asymmetry of the basin allowed these brines to flow through more permeable lithofacies to the northern edge of the depository (Figure 8.1) where they are interpreted to have flowed upwards along the basin-marginal faults and vented to the sea floor, forming finely laminated sulphide mounds (Schmidt and O'Neil, 1982). It is these sulphide mounds that are host to the copper mineralization in the deposits.

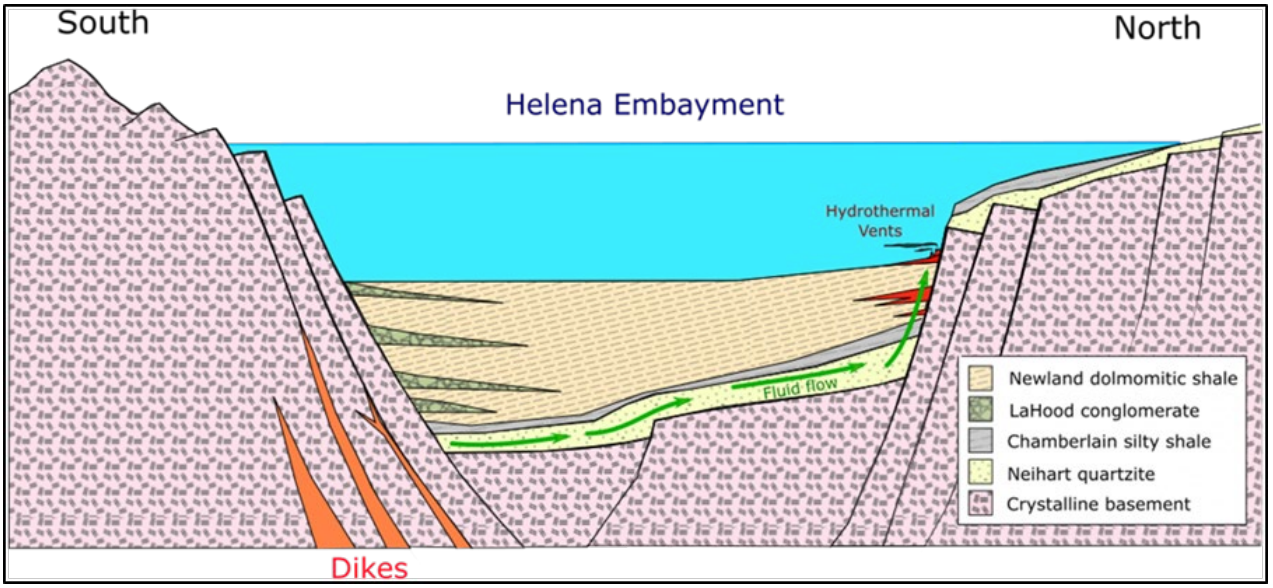

Figure 8.1 Schematic cross-section through the Helena Embayment showing interpreted basinal brine fluid flow and the formation of sulphide mounds on the northern margin of the depository

Subsequent to formation and burial of the massive sulphide units, and continuing after lithification, late stage Cu-rich hydrothermal fluids were generated and ascended along fluid pathways along the basin margin (Figure 8.2). The origin of this later generation of mineralizing hydrothermal fluids is uncertain. White *et al* (2014) proposed that reduction of hematite in the Neihart Formation to magnetite would have generated an oxidizing fluid and liberated Cu from both the Neihart Formation and adjacent sediments. Continued intrusive activity associated with crustal extension prolonged the heat source for basinal fluid flow.

Whereas the formation of the pyrite massive sulphide layers was a regional event throughout the Helena Embayment (where it occurred along basin margin faults), the later main Cu mineralizing event was focused in specific areas. As the Cu mineralization occurred in buried and potentially lithified sediment, it is reasonable to consider the extant structural framework as a control on permeability and hydrothermal fluid flow.

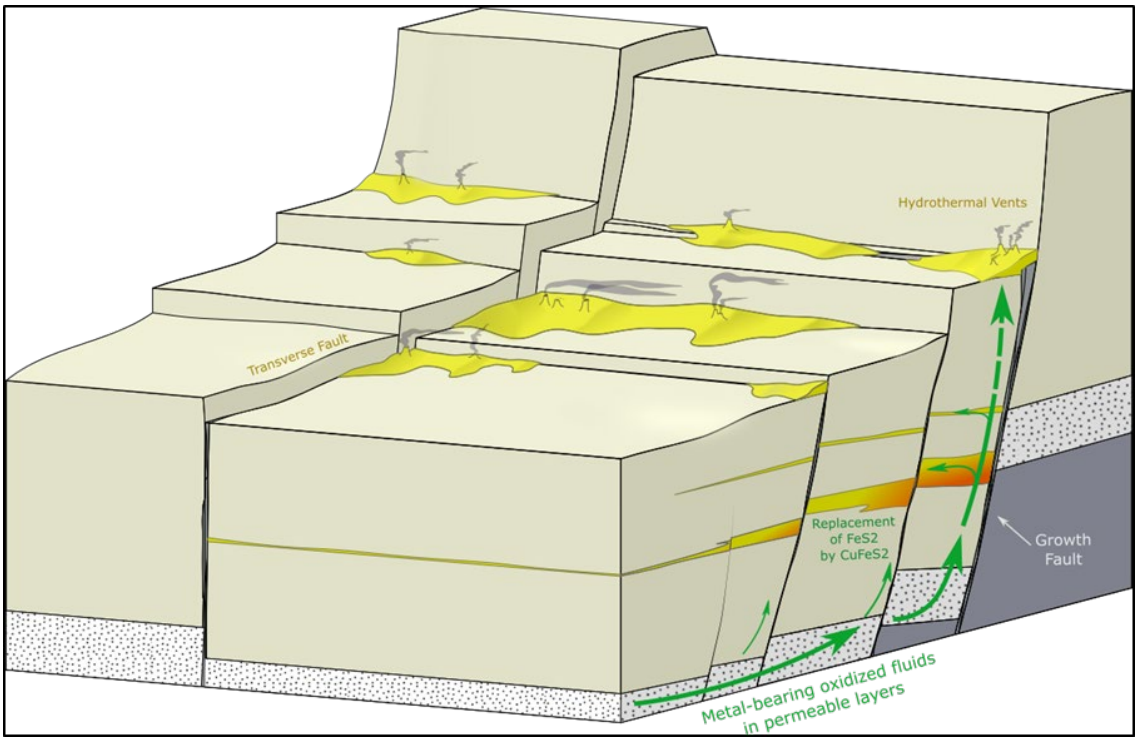

Figure 8.2 Schematic diagram showing fluid flow patterns and replacement of Stage 1 pyrite by Stage 2 Cu mineralizing fluids

The UCZ is transected by Fault 1, along which the last movement was post-mineralization. The current trend of the intersection of Fault 1 with the UCZ is coincident with the most robust Cu mineralization observed within the deposit (Figure 8.3). These observations suggest that Fault 1 is actually a long-lived feature that influenced Cu mineralization during the second stage of hydrothermal activity at Johnny Lee.

It is proposed that Fault 1 was a transform fault (Figure 8.2) developed orthogonal to the basin-margin Buttress and Volcano Valley Faults during formation of the Helena Embayment. It is likely that Fault 1 and the basin margin faults acted as fluid conduits for the initial stage fluids that formed the bedded pyrite-massive sulphide. However, unlike the stage two event, hydrothermal fluid flow was not constrained to the intersection zone. Subsequent to burial and lithification of the bedded massive sulphide, stage 2 Cu mineralizing hydrothermal fluids were focused along permeable zones formed by the intersections between Fault 1, the VVFZ and the Buttress Fault. Importantly, it was this later event that formed the bulk of the Cu mineralization in the Johnny Lee deposit.

Fault 1 and the VVFZ were both reactivated, post-mineralization, during compressional Laramide orogenesis. It is uncertain whether the Buttress Fault was reactivated, or retained the original, extensional growth fault geometry.

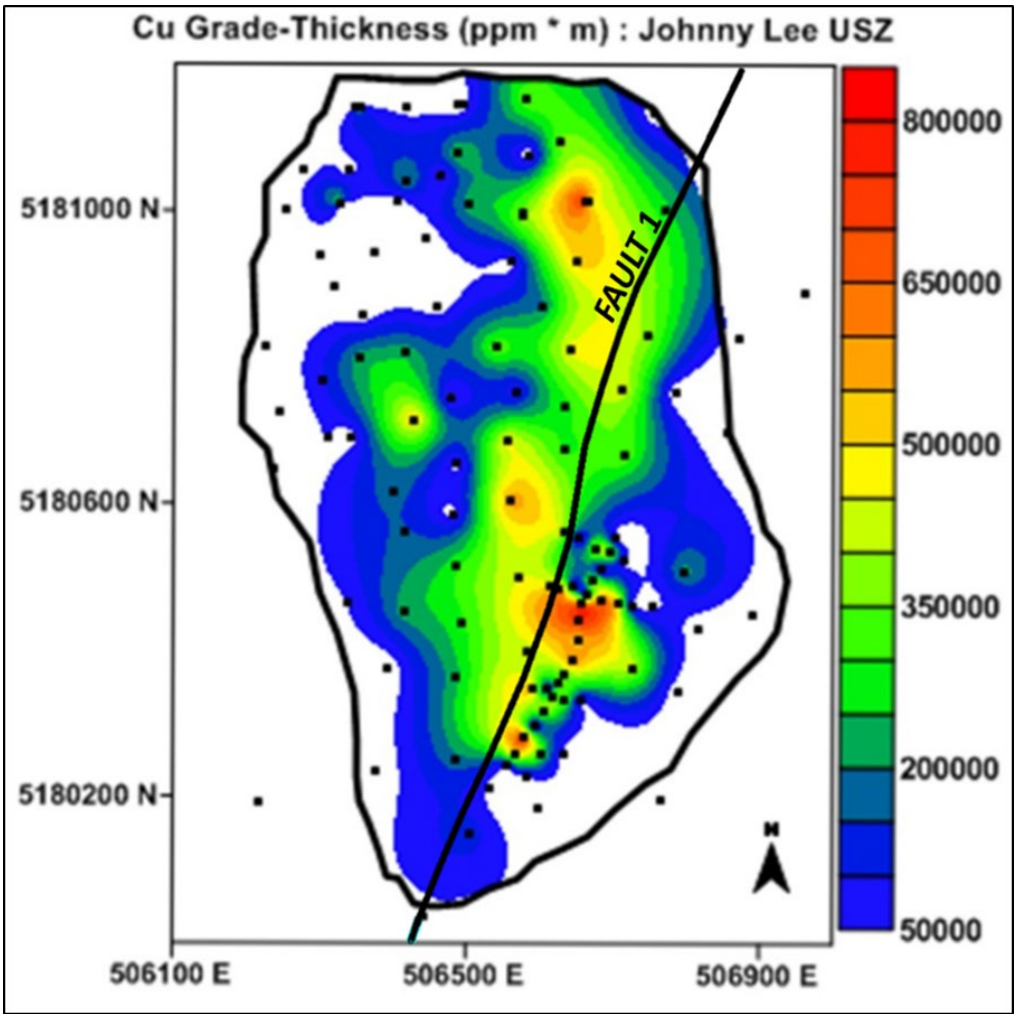

Figure 8.3 *Plan view of a Cu grade X Thickness contour map of the UCZ showing the surface trace of Fault 1*

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9. EXPLORATION

9.1 Relevant Exploration Work

CAI, UII, and BHP conducted a variety of geological, geochemical, and geophysical exploration programs between 1976 and 1993 in the Black Butte Copper Project area. This historic work included surface mapping, surface soil and rock sampling, various geophysical surveys, and core drilling. Geologic maps, down-hole geochemical data, surface geochemical data, drill logs, down-hole surveys, and various compilation maps from these programs were recovered from the University of Montana Belt Research Center, to which CAI donated core and geologic information at the end of their tenure at Black Butte. No geophysical data is presently available. Within the area described as "resource" by CAI, the drillhole spacing was approximately 150 m, and 19 holes penetrated the UCZ and sub zones, and 12 holes penetrated the LCZ. Only three holes encountered both zones. 64 holes in total were drilled on the present leases and an additional four holes were drilled on unpatented claims currently owned by SRA.

In 2011, SRA began a detailed compilation of available geologic mapping data. Historic maps included work that was completed by CAI and BHP. SRA staff compiled a number of previous geologic maps of the district, focusing on areas located adjacent to known Cu occurrences at Black Butte, as well as areas that had historic drilling.

During 2011, SRA crews collected 744 soil samples over previously un-sampled and geologically prospective areas. Field crews collected samples with a hand auger and attempted to collect from the "B" soil horizon. If the B horizon was not present, samples were collected from the "C" horizon. The survey consisted of collecting soil samples from 19 NS oriented lines spaced 300 m apart with samples collected at 60 m intervals along each line. In areas considered more prospective, the line spacing was reduced from 300 m to 150 m, with samples collected every 30 m along the line. Three EW soil lines spaced 500 m apart with samples collected at a spacing of 60 m, covered ground with NS striking stratigraphy. The area sampled ranged from UTM 501650 East to 506480 East and 5176000 North to 5180000 North. Field crews located all sample sites with a handheld Global Positioning System (GPS) unit.

ALS in Reno, Nevada, screened soil samples to 180 µm (ALS PREP 41), digested them in aqua regia, and analyzed them for 51 elements with Inductively Coupled Plasma - Mass Spectrometry (ICP-MS) (ALS code ME-MS41L) and for Au with Inductively Coupled Plasma - Atomic Emission Spectroscopy (ICP-AES) (ALS code Au-ICP21). Results showed anomalous samples consistent with the strike extent of known mineralized areas. Other modestly anomalous results require further field review.

In March 2012, SRA contracted Aeroquest to conduct an airborne electromagnetic survey over the district. Results show that the highly conductive properties of the UCZ and many other conductors are generally consistent with known trends of sulphide mineralization. Total Reduction To Pole (RTP) magnetic data shows the transition from a highly magnetic area to the north that correlates with thin or absent sequences of Belt rocks, to a weaker magnetic area to the south that correlates with thick sequences of Belt rocks. Some strong NE and NW trending linear magnetic features correlate with Eocene intrusives on the map. Other linear magnetic features correlate with the surface trace of gossan exposures.

During 2012, SRA contracted Minex Exploration of Sandpoint, Idaho, to conduct a detailed ground based magnetic survey over the area of the Johnny Lee and Lowry deposits. This greatly enhanced the accuracy of ground locations of features identified from the airborne magnetics survey results, especially locations of Eocene intrusive dikes and sills.

The exploration potential of the Black Butte area is considered highly prospective for SEDEX type deposits. A domain of prospective Newland Formation stratigraphy encompasses a zone over 24 km in length. While the larger airborne electromagnetic survey covers a sizeable portion of this trend, it does not cover all of it. Within the survey area, the prospective Newland stratigraphy, only a few areas have been tested with sufficient data density to identify potentially mineralized areas similar in size to the Johnny Lee or Lowry deposits.

Exploration by SRA, subsequent to 2013, has been limited to a 2016 soil sampling and mapping campaign and continued evaluation and permitting work focused on the Johnny Lee deposit. During 2016, SRA conducted localized 1:3,000 scale geologic mapping to better locate the surface expression of the USZ. The 2016 mapping and soil sampling campaign was focused on the lower Newland Formation stratigraphy (Figure 9.1).

9.2 Sampling Methods and Sample Quantity

The soil sampling in 2016 consisted of 413 samples across the Newland-Chamberlain Formation contact on the east-central portion of the property (Figure 9.1). The aim of the program was to test the lower portion of Newland Formation stratigraphy for a geochemical signature of a potential extension to the LCZ mineralization. Predetermined sample lines were located with a handheld GPS, samplers used a compass to maintain azimuth and a handheld GPS to monitor distance. Samples were collected every 30 m along the lines. A small shovel was used to dig below the A soil horizon, and the sampler collected 50 to 100 g and manually removed coarse fragments.

Sample analysis was performed by ALS in Reno, NV and Vancouver, BC. The samples were screened at the laboratory to minus 180 microns, then digested in aqua-regia and analyzed for 35 elements using ICP-MS.

9.3 Significant Results and Interpretation

Figure 9.2 shows the Cu results from all soil sampling campaigns over the property with associated named prospects. The 2016 soil sampling program failed to indicate any significant anomaly. Historic soil sampling outlined significant Cu anomalism at the Johnny Lee deposit and the Copper Creek prospect. Minor surface Cu anomalism occurs at the Lowry deposit, Sawmill Hill prospect, and Butte Creek prospect. The QP for Mineral Resources has not reviewed the data associated with any prospect beyond the Johnny Lee and Lowry deposits, as these exploration prospects are not the focus of this FS Technical Report. Additional information on Cu prospects in the Black Butte Copper Project area are described in more detail in Section 23.

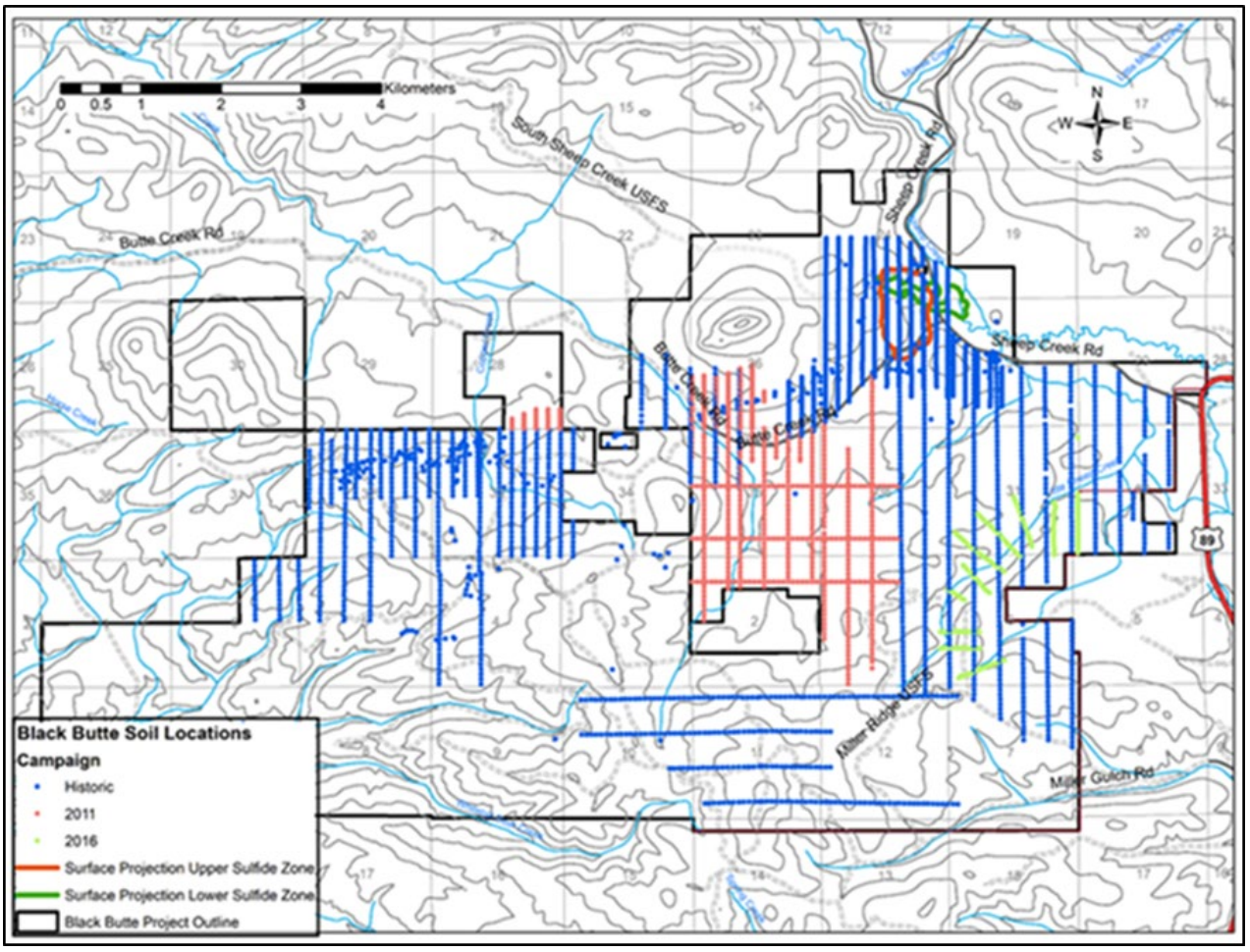

Figure 9.1 Plan showing soil sampling campaigns on the property

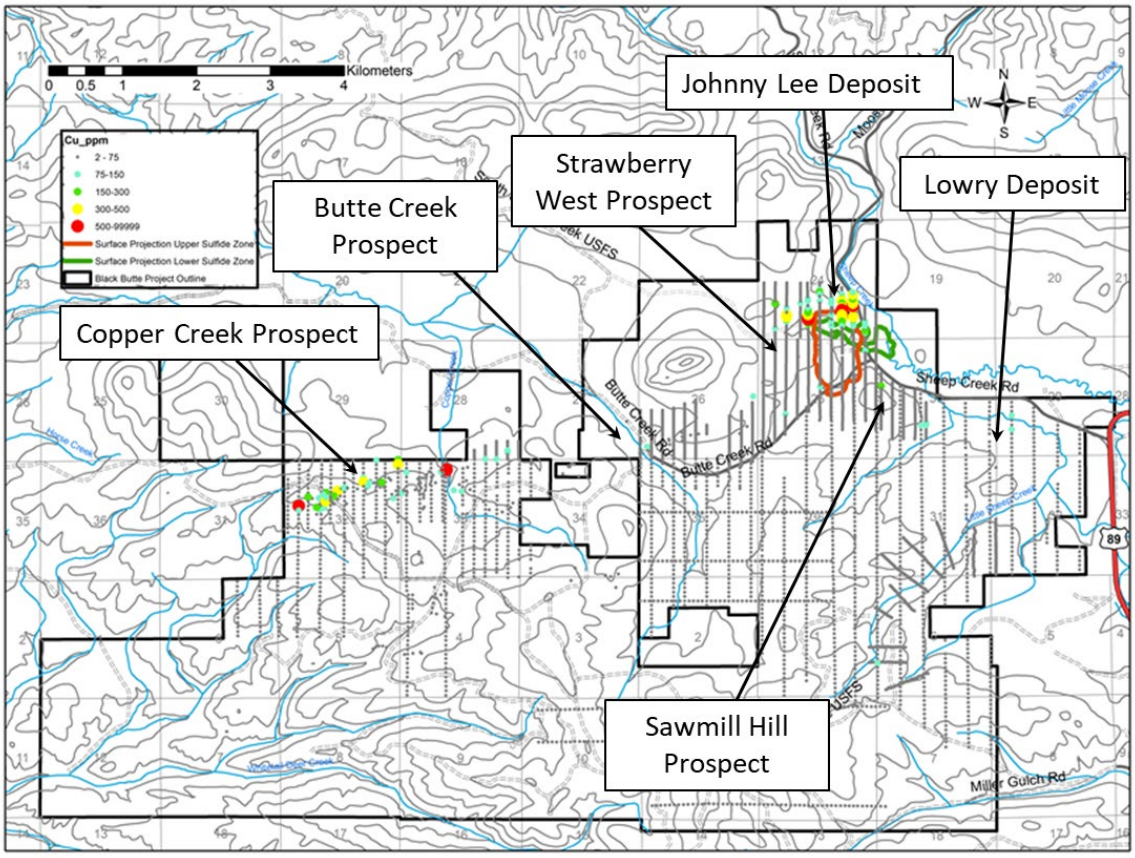

Figure 9.2 *Plan showing soil sampling Cu results, deposits and exploration targets*

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10. DRILLING

10.1 Type and Extent

There are a total of 342 drillholes on the current tenement package that have been completed in numerous campaigns by seven companies (Figure 10.1, Table 10.1). The majority of these drillholes are focused on the Johnny Lee and Lowry deposits.

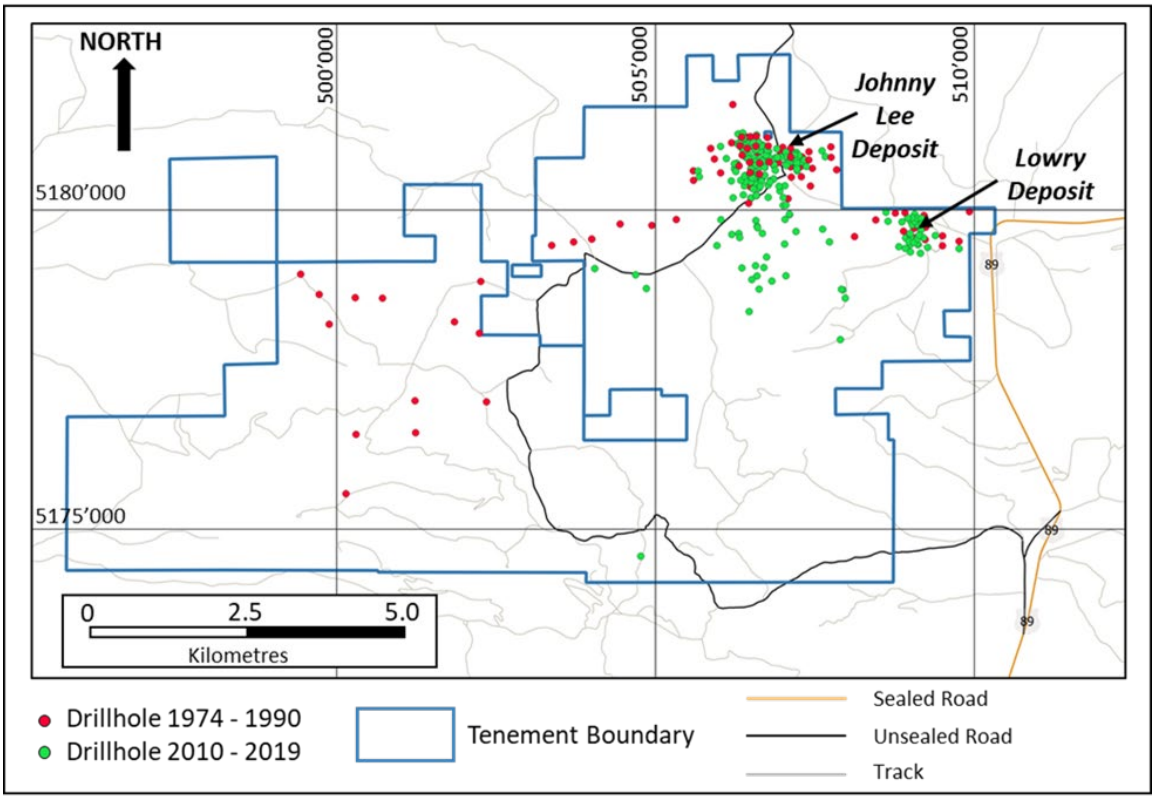

Figure 10.1 Map of the Project area showing tenement boundaries and collar positions of drillholes completed from 1974 – 1990 (Pre-SRA) and 2010 to 2019 (SRA). Coordinate system is NAD83 UTM Zone 12 North

Homestake Mining Company completed one exploration diamond drillhole in 1974 and abandoned the Project shortly thereafter. Cominco American Inc. (CAI) began work in the area in 1975 and drilled 13 diamond drillholes. Anaconda Mining Company acquired tenements west of CAI in 1977 and drilled one Reverse Circulation (RC) drillhole. Exxon Minerals drilled one diamond drillhole in 1984. CAI entered into a JV on the Anaconda tenements in the early 1980's and drilled two diamond drillholes. None of these early drill programs intersected mineralization of economic significance and the data has not been used for the determination of Mineral Resources of this Project.

Company	Years	No. of Holes	Core Metres Drilled	Core Diameter	RC Metres Drilled	Drill Purpose
Homestake Mining	1974	1	148	Unknown	--	Exploration
Cominco American Inc.	1978 - 1992	43	21,952	Unknown	--	Exploration/Resource/Metallurgy
Anaconda Mining	1978	1	--	--	374	Exploration
EXXON	1984	1	220	Unknown	--	Exploration
Utah International Inc.	1980 - 1984	8	5,498	Unknown	--	Exploration
BHP	1985 - 1990	33	12,764	Unknown	116	Exploration/Resource
Tintina Montana Inc.	2010 - 2012	168	58,677	HQ	--	Resource/Metallurgy/Geotechnical
Sandfire Resources America	2014	11	3,245	HQ	--	Resource/Metallurgy
Sandfire Resources America	2015	26	999	HQ	--	Geotechnical
Sandfire Resources America	2018 Phase 1	24	4,924	PQ/HQ	--	Metallurgy/Resource/Geotechnical/ Sterilization
Sandfire Resources America	2018/19 Phase 2	26	6,115	HQ	--	Resource/Metallurgy/Geotechnical
TOTAL		**342**	**114,544**		**490**	

Table 10.1 Black Butte Copper Project drilling history

CAI entered into a JV with Utah International Inc., (UII) a BHP subsidiary, who completed eight diamond drillholes between 1980 and 1984. The first ore intercept of the UCZ and LCZ was drilled by BHP in 1985. BHP completed 32 diamond drillholes and one RC drillhole from 1985 to 1990. No mineralization was intersected in the single RC drillhole and it has not been utilized for the determination of Mineral Resources of this Project.

In 1990, CAI purchased BHP's share in the Project, continued to drill out the Johnny Lee deposit and discovered the Lowry deposit. From 1990 to 1992, CAI drilled an additional 28 diamond drillholes in the Project area: 14 in the Lowry deposit, eight in the Johnny Lee deposit, and the remainder regional exploration drillholes. CAI used this drilling to estimate a Resource and to undertake preliminary metallurgical testing. The resources estimated by CAI were performed prior to the establishment and are not in accordance with NI 43-101. CAI abandoned the Project in 1995.

SRA leased the mineral rights containing the Johnny Lee and Lowry deposits in 2010. From 2010 to 2012, SRA completed 168 diamond drillholes. These drillholes were utilized for Mineral Resource definition at the Johnny Lee and Lowry deposits, metallurgical test work, and geotechnical studies. Six pre-2010 drillholes were twinned by SRA to determine the validity of the data for a NI 43-101 Mineral Resource estimate (Winckers *et al*, 2013). All historic drilling was evaluated by Winckers *et al* (2013) as part of a PEA completed in 2013. The PEA included a Mineral Resource declaration for the UCZ, the LCZ, and the Lowry deposit..

SRA acquired the Project in 2014 and, in that year, drilled 11 diamond drillholes into the UCZ for Mineral Resource infill, geotechnical evaluation, and metallurgical test work. 26 shallow diamond drillholes were completed to evaluate geotechnical conditions below proposed infrastructure in 2015. The metallurgical test work on drillholes from the 2014 program highlighted the variability in the UCZ recoveries that had previously not been identified. A geometallurgical study was initiated in 2017 that allowed the spatial distribution of UCZ mineralization with different geometallurgical characteristics to be constrained.

During the 2018 Phase 1 drill program, 19 diamond drillholes were completed into the UCZ to acquire samples for ore characterization for additional metallurgical test work. These drillholes were planned between existing intercepts such that the information could also be used for Mineral Resource infill purposes and to further constrain the geometallurgical model. During the same program, two drillholes were completed in the LCZ, targeting a gap that had been identified in the Mineral Resource definition drilling. Two sterilization drillholes were completed in areas of proposed infrastructure development and one drillhole was completed at a proposed water injection site.

Utilizing the results from the 2018 Phase 1 drill program, SRA developed a conceptual mine plan that was used as a guide for the additional Project definition drilling. The 2018/19 Phase 2 drill program was designed to provide data to underpin further definition of the resource and comprised the following elements:

- Mineral Resource infill / geometallurgical definition drilling in critical parts of the UCZ and LCZ;
- Acquisition of geotechnical information for infrastructure, UG development and stope design;
- Hydrogeological testing of the proposed decline; and
- Collection of additional metallurgical sample material from the LCZ.

A total of 26 diamond drillholes were completed in the 2018/19 Phase 2 drill program for a total of 6,115 m.

The updated Mineral Resource estimate is focused on the UCZ, LCZ (Figure 10.2) and Lowry deposit (Figure 10.3). All drillholes in these areas are diamond drillholes that were either pre-collared using a tri-cone bit or cored from surface.

47 drillholes (19 for the UCZ, 15 for the LCZ and 16 for Lowry) from pre-SRA drilling are used to support the Mineral Resource estimate presented in this Technical Report. In 2010, SRA obtained a copy of the electronic drillhole database, including the data from these drillholes, from the Belt Research Center, University of Montana. CAI donated this data to the Belt Research Center after they terminated their interest in the Project. Although efforts were made to acquire supporting documentation from Teck (who had acquired CAI subsequent to CAI abandoning the Project) these efforts were unsuccessful. Lacking this data, it is not possible to describe in detail the drilling, logging, and analytical protocols used by historic operators prior to 2010.

Given the absence of assay certificates, downhole survey records, and QA/QC results for the historic drill data, SRA twinned six of the historic drillholes in 2010. Winckers *et al* (2013) compared the historic versus new drillhole data from each of the drillhole pairs and concluded that the historic drill data was acceptable for use in a Mineral Resource estimate.

Since 2013, SRA has drilled 87 diamond core holes over four drilling programs. The sections below outline the drill procedures utilized in those campaigns and highlights procedural changes that were implemented as the Project advanced.

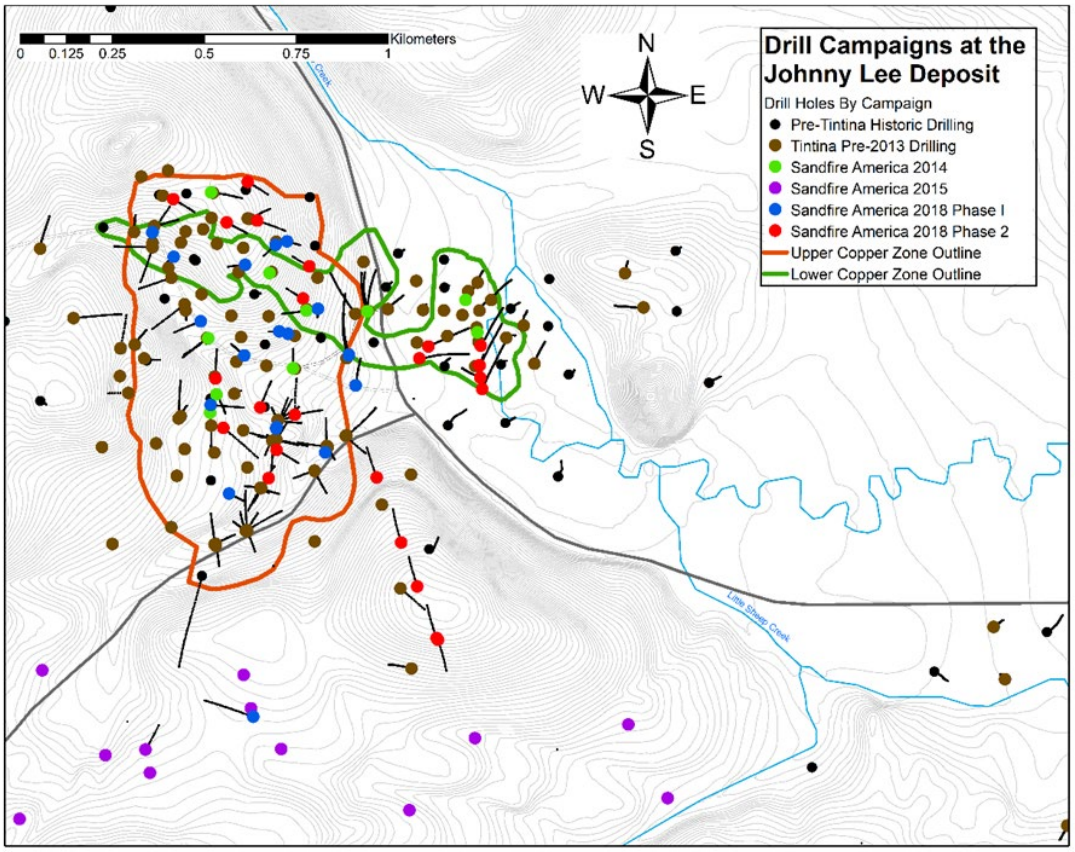

Figure 10.2 Topographic map of the Johnny Lee area showing drillhole collars from all drill programs

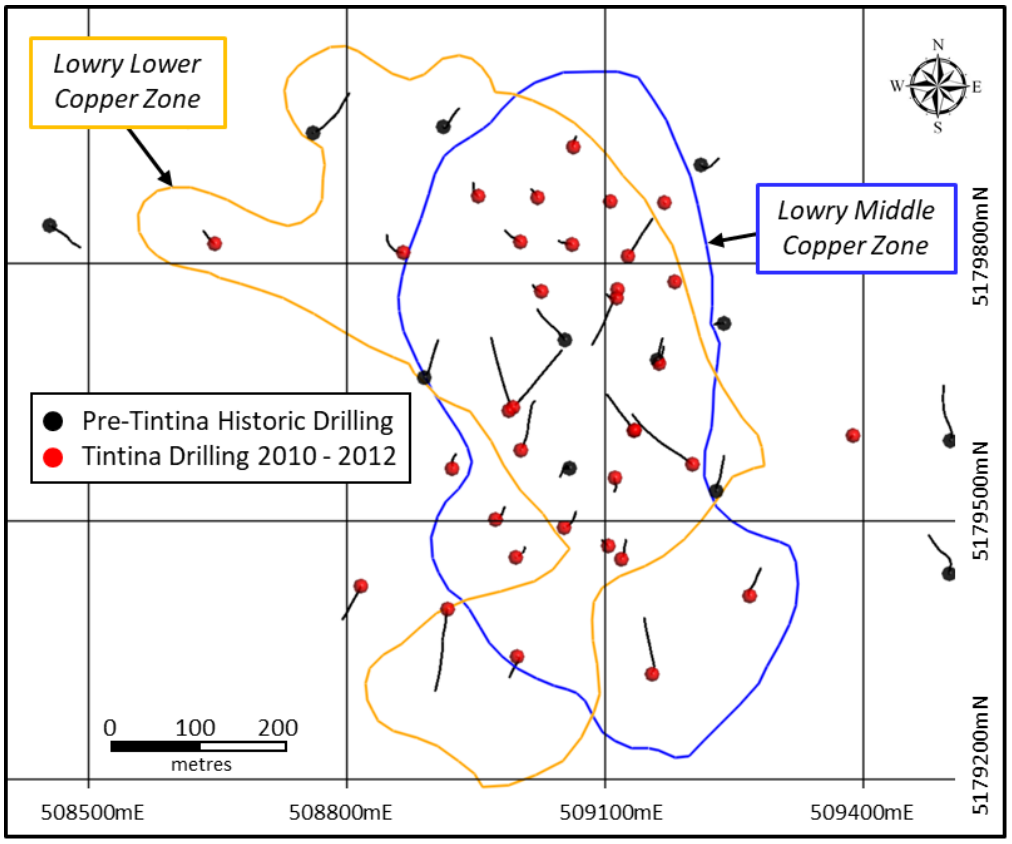

Figure 10.3 Map of the Lowry deposit showing drillholes and the surface projection of the LMCZ and LLCZ

10.2 Procedures

10.2.1 Grid

All exploration and resource infill drilling and installed wells and piezometers have been surveyed at the Black Butte Copper Project using the North American Datum of 1983 (NAD83) / Universal Transverse Mercator (UTM) Zone 12 North coordinate system. Project control was established using several static observation sessions on control points surrounding the Project. The static observations were processed through the Online Positioning User Service (OPUS) of the National Geodetic Survey to derive the coordinates and then the Project was adjusted and scaled to ground.

10.2.2 Collar Survey

Drillhole collars were pegged using either a handheld GPS or a Real Time Kinetic (RTK) GPS instrument. Prior to the 2018/19 Phase 2 drill program, drill rig alignment was completed using a sighting compass and inclinometer. For the 2018/19 Phase 2 program, a Reflex TN14 Gyrocompass™, north-seeking gyroscopic alignment tool was used to ensure accurate azimuth and dip alignment.

Upon completion of drillholes, the collars were accurately surveyed by a registered surveyor from WWC Engineering of Helena, Montana using an RTK GPS instrument (Trimble R8 GNSS™) with horizontal and vertical tolerance set to 0.05 ft (approximately 15 mm). WWC Engineering located and surveyed all historic drillhole collars used for the determination of Mineral Resources.

10.2.3 Downhole Survey

For the 2014 and 2015 SRA drilling programs, a Reflex EZ-Trac™ electromagnetic survey instrument was used to record downhole survey data. Survey shots were taken at 30 m intervals downhole. During the 2018 Phase 1 and 2018/19 Phase 2 drill programs, a Reflex EZ-Trac™ survey instrument was used to record downhole survey data during drilling. A Reflex EZ-Gyro™ (north seeking gyroscope) was used to survey each hole at 3 m to 6 m intervals, upon drillhole completion. Acceptable correlations between EZ-Trac and EZ-Gyro instruments and low magnetic susceptibility readings indicate that magnetic interference of electromagnetic survey instruments was not occurring.

10.2.4 Core Orientation

Core orientation was undertaken on all inclined holes completed from 2014 to present. For inclined drillholes, a Reflex ACT™ II or III core orientation tool was utilized by the drillers to record core orientation. Bottom of hole orientation marks were marked on the core by the drill crew. Core orientation quality was generally good although orientation often failed in highly fractured ground such as that within the VVFZ and adjacent to Fault 1 in the Johnny Lee deposit.

10.2.5 Core Handling

Upon completion of each drill run, the core was extracted from the core barrel, cleaned, and placed in core racks by the drill crew. Core blocks showing hole-depth (in ft.) were placed at the end of each drill run. Core was then placed in waxed cardboard core boxes. Where it was necessary to break core to fit into the core boxes, the artificial break was marked on the core so it was disregarded during Rock Quality Designation (RQD) or geotechnical logging.

Core boxes were picked up twice daily from the drill rig site by SRA technicians and delivered to the logging facility in White Sulphur Springs. Drilling depths were marked at 1 m intervals with core recovery and RQD recorded by geologists or geotechnical assistants. Where lost core intervals greater than 0.1 m were encountered (and not recovered in subsequent drill runs), the lost core intervals were marked and integrated with the depth markup.

Where core orientation was undertaken, the core for successive runs was laid out and the quality of core orientation was determined. If orientation marks from adjacent runs were within 10°, a solid bottom-of-hole line was drawn. If orientation marks were >10° and <20° a dashed line was drawn. If the disconnect between orientation marks was >20° no orientation line was drawn.

Subsequent to logging and sampling, each core box was photographed using a digital camera mounted on a frame for consistency of distance and image size. Digital photograph files were named using the drillhole ID and depth and stored on the SRA server.

Specific Gravity (SG) determinations were then made of whole-core intervals using an Archimedean (wet/dry) scale. Further description of SG testing methodology is described in Section 12.

Intervals selected for sampling were then halved using a core saw. The sample cut was made approximately 5° clockwise (looking downhole) from the orientation line. The half-core that did not contain the orientation line was sampled. Where a field duplicate had been requested by the logging geologist, the remaining half-core was quartered and the quarter without the orientation line was sampled. If core orientation was not performed, or had failed, the core was cut perpendicular to bedding and the same half-core was consistently sampled. Samples were placed in calico bags and stored in a secure facility until dispatch to the laboratory. Remaining core is stored either at the logging facility in White Sulphur Springs or in the secure core sheds at the Project site.

An exception to this workflow is the 2018 Phase 1 UCZ PQ core that was utilized for metallurgical testing. The PQ core was processed in the manner described above but instead of cutting and sampling the core, whole-core of each resource sample interval was dispatched to Base Metallurgical Laboratories in Kamloops, British Columbia, Canada. For each sample interval, the entire PQ core interval was fine crushed. A 1,000 g split of the fine crush was pulverized and sent to ALS Vancouver for analyses.

10.2.6 Geological Logging

Prior to the 2018 Phase 1 drill program, all logging was conducted using graphic logging sheets. The information recorded included: depth, color, lithology, mineralogy, oxidation, grain size, texture, sedimentary structures, alteration, mineralization, and structure. Where core orientation had been successfully completed, alpha and beta angles of planar features (bedding, veins, foliation, faults) were recorded using a Kenometer core protractor.

In the 2018/19 Phase 2 drill program, digital logging software (OCRIS™) was utilized to record all logging information using a portable computer. The digital logging was accompanied by summary graphic logging such that sedimentary facies could be identified in a similar manner to the 2014 through 2018 Phase 1 drill programs. All geological logging data was validated prior to use in geological and resource modelling as described in Section 12.

10.2.7 Geotechnical Logging

In 2014, SRA drilled a 10 drillhole, HQ diameter program (2,775 m) designed to collect geotechnical data and acquire core samples for metallurgical test work from the UCZ and LCZ. The geotechnical and structural data was used to inform a geotechnical evaluation of the UCZ and LCZ by Mine Design Engineering. A geotechnical engineer from Mine Design Engineering trained SRA geologists in geotechnical core logging and monitored data quality during the logging process. The logging protocols utilized in this program and geotechnical lab test results are described in detail in Kalenchuk *et al* (2015).

All drillholes completed during the 2018/19 Phase 2 drill program were geotechnically logged. A geotechnical engineer from Mining Plus trained the SRA geologists in the appropriate logging process and monitored data acquisition to ensure it was fit for purpose. One HQ3 geotechnical drillhole was completed at a proposed vent raise location in the UCZ. As the drillhole was vertical, a Reflex Verti-Ori™ core orientation system was utilized to orient the core. The hole was geotechnically logged by Mining Plus geotechnical engineers and strength testing and other geotechnical determinations to support the FS were conducted by Agapito Associates (2019) and Advanced Terra Testing Inc. (Nazeri, 2019).

10.2.8 Core Recovery

Core recovery in the UCZ is considered acceptable with average recoveries greater than 90%. Within the Eastern Block of the UCZ (including both >1.2% Cu mineralization and >0.25% Cu Halo material) mean core recovery from all drill programs is 93.0% (median = 97.1%). Within the Western Block of the UCZ, mean core recovery is 91.5% (median = 96.4%). Very low core recoveries (<60%) in the UCZ generally occur in isolated individual core runs and thus do not have a significant impact on overall grade estimation or Mineral Resources.

Four drillholes that intersected Fault 1 at low angles (Figure 10.4) exhibited multiple adjacent intervals of <60% recovery in individual drillholes. Even in the drillhole with the lowest recovery (SC18-244), visual inspection of core recovered from each core run suggested that the core loss was systemic, rather than isolated, allowing for meaningful sampling and analyses.

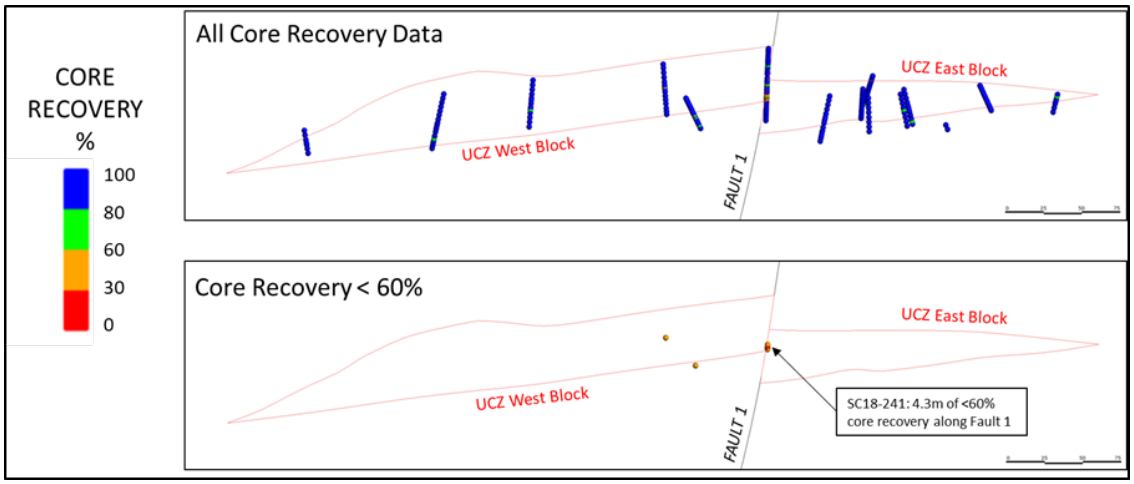

Figure 10.4 Cross-section, looking north, showing raw, un-composited core recovery data in the Eastern and Western Blocks of the UCZ

Although poor core recoveries were encountered in the VVFZ, the recoveries within the LCZ are considered acceptable. Core recoveries in the East, Central, and West zones have mean values of 98.2%, 99.0% and 98.1% respectively. Corresponding median recovery values for the three zones were 99.3%, 100% and 98.0%.

Core recoveries from the Lowry deposit are considered acceptable with average recoveries for the different mineralization zones ranging from 86.0 – 93.2%.

Core recoveries in the LLCZ Vein 1 from all drill programs range between 60.1 – 100.0% and average 87.0% (median = 88.5%). Those for LLCZ Vein 2 range between 70.5% and 100.0% and average 89.3% (median = 91.8%).

Core recoveries from LMCZ Vein 1 range from 6.5% - 100% and average 89.7% (median 95.3%). Five core runs out of a total of 78 returned recoveries of <50%. These core runs were all from different drillholes and represent isolated zones of poor recovery rather than wide zones of below average recovery. Core recoveries from LMCZ Vein 2 range from 32.6% – 100% and average 86.0% (median 92.3%). Five core runs out of a total of 94 returned recoveries <50%, three of these core runs were from consecutive runs in drillhole SC11-079, the remainder were isolated core runs. Core recoveries from LMCZ Vein 3 range from 72.1% – 100.0% and average 93.2% (median = 97.0%). Core recoveries for the entire LMCZ (encapsulated within the LMCZ Halo boundaries) range from 6.5% – 100.0% and average 89.9% (median = 95.7%).

10.3 Interpretation and Relevant Results

The UCZ (Figure 10.4) is a flat lying to shallow dipping zone of Cu mineralization that ranges in thickness from 3 m to 42 m. Gentle folding about a W/NW trending syncline-anticline pair results in a variable dips that range from 0° to 20°. The northern edge of the UCZ is truncated by the HW fault of the VVFZ. The UCZ is transected and displaced by the N/NE striking, steeply W/NW dipping Fault 1.

The LCZ (Figure 10.5 and Figure 10.6) strikes E/NE and dips to the S/SE at between 5° and 37°. The variable dip reflects gentle folding of the deposit. The LCZ is truncated in the south by the FW fault of the VVFZ and to the north by the Buttress Fault.

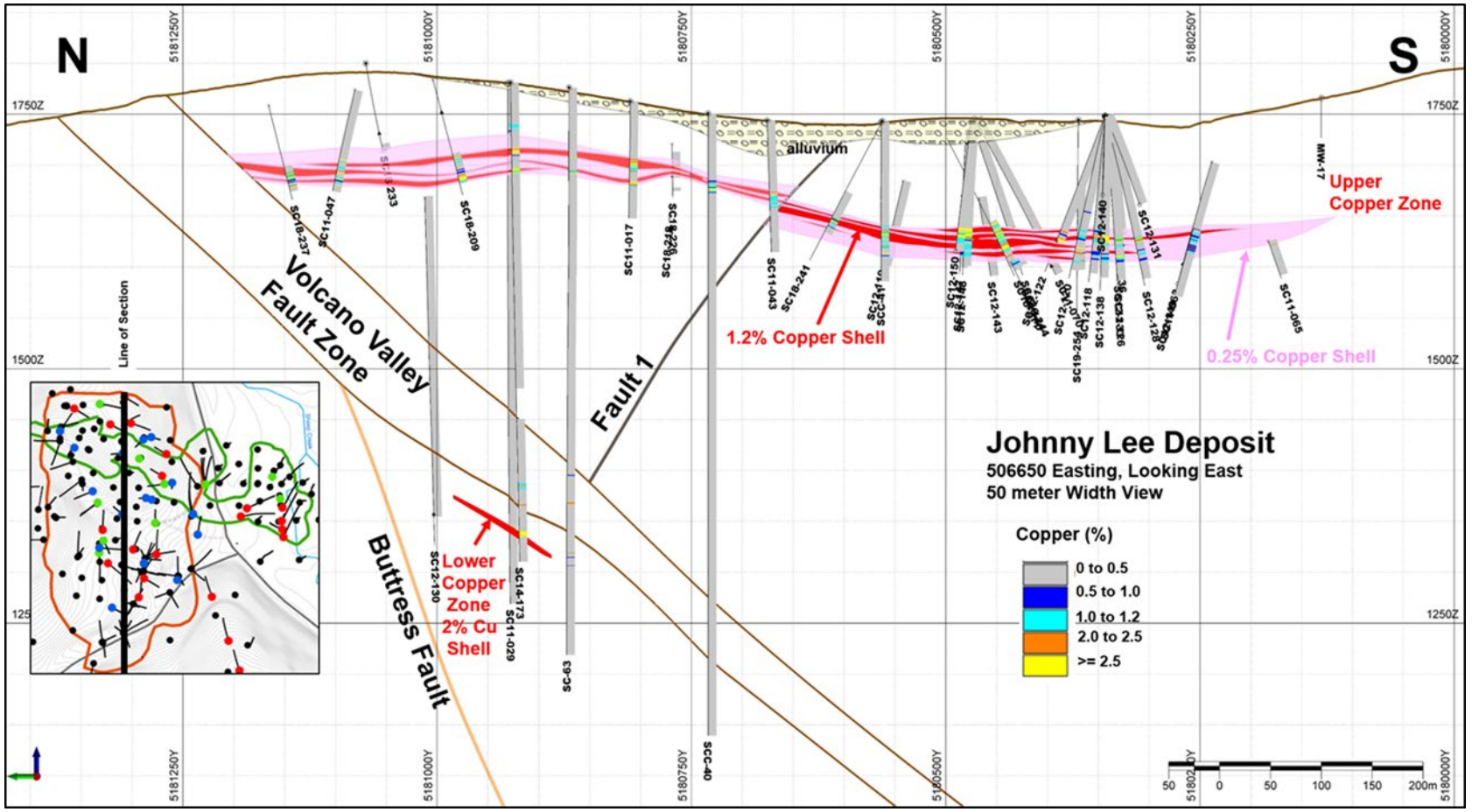

Figure 10.5 Cross-section of the Johnny Lee deposit. See inset for line of section. Legend for inset as per Figure 10.2

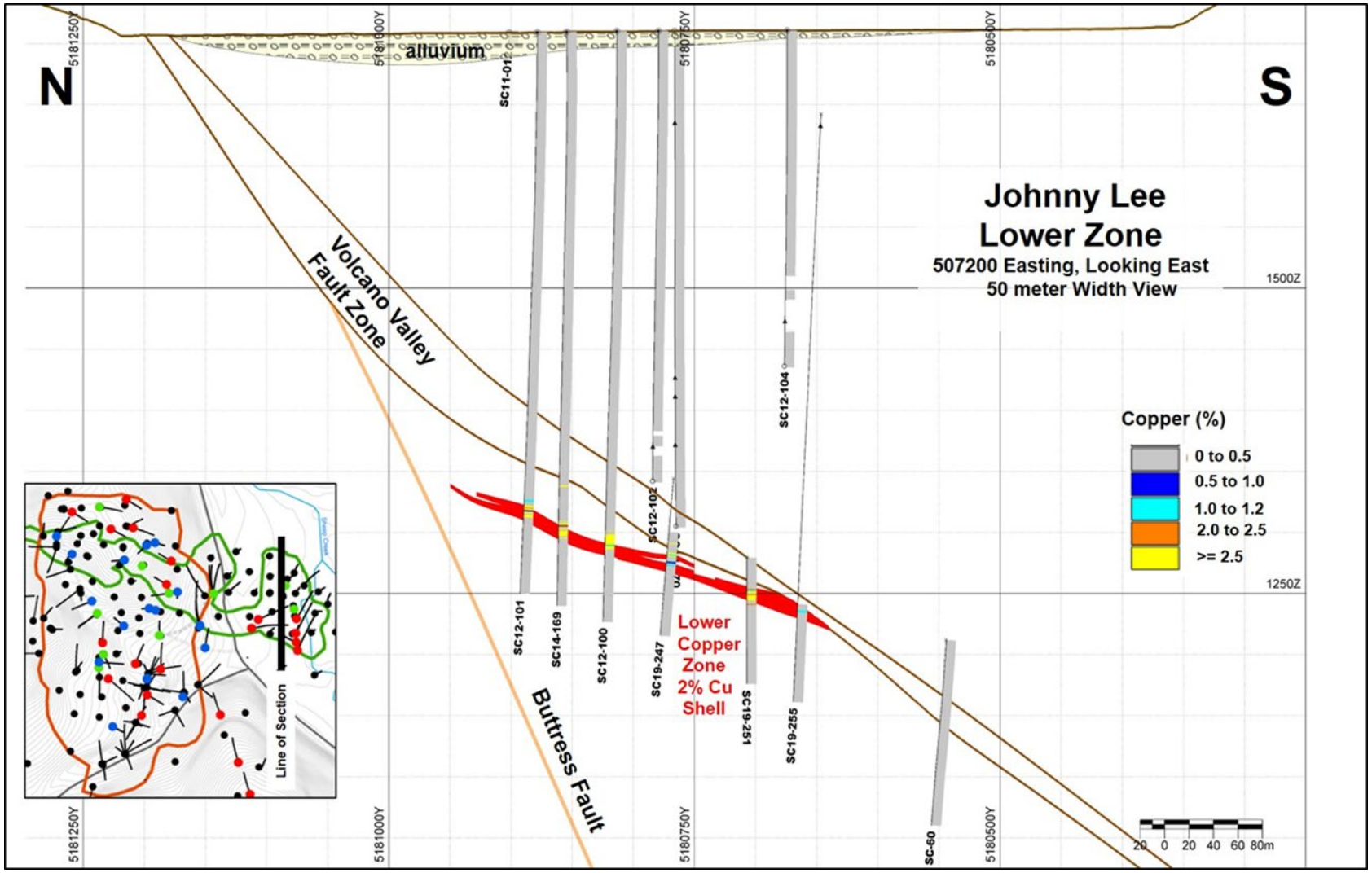

Figure 10.6 Cross-section of the LCZ. See inset for line of section. Legend for inset as per Figure 10.2

The LMCZ is a shallow south dipping zone of mineralization that occurs in the HW of the VVFZ (Figure 10.7 and Figure 10.8).

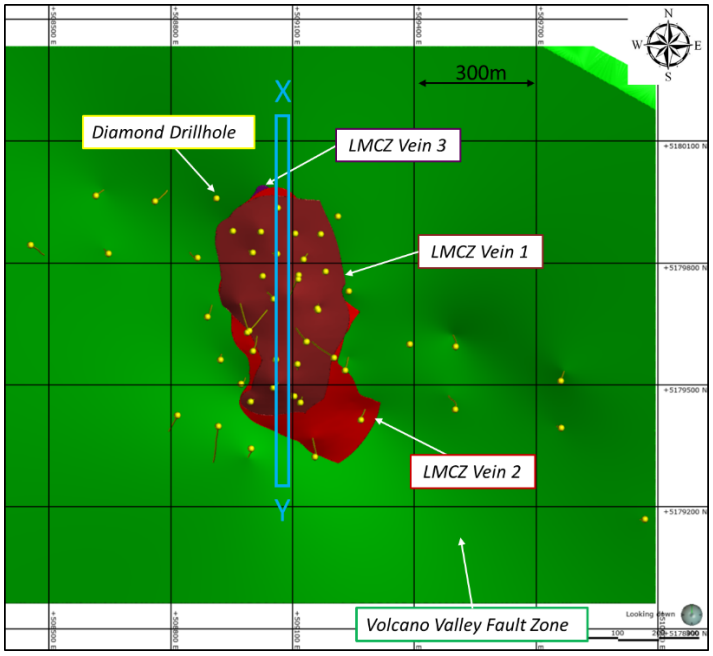

Figure 10.7 Plan view of LMCZ and the VVFZ

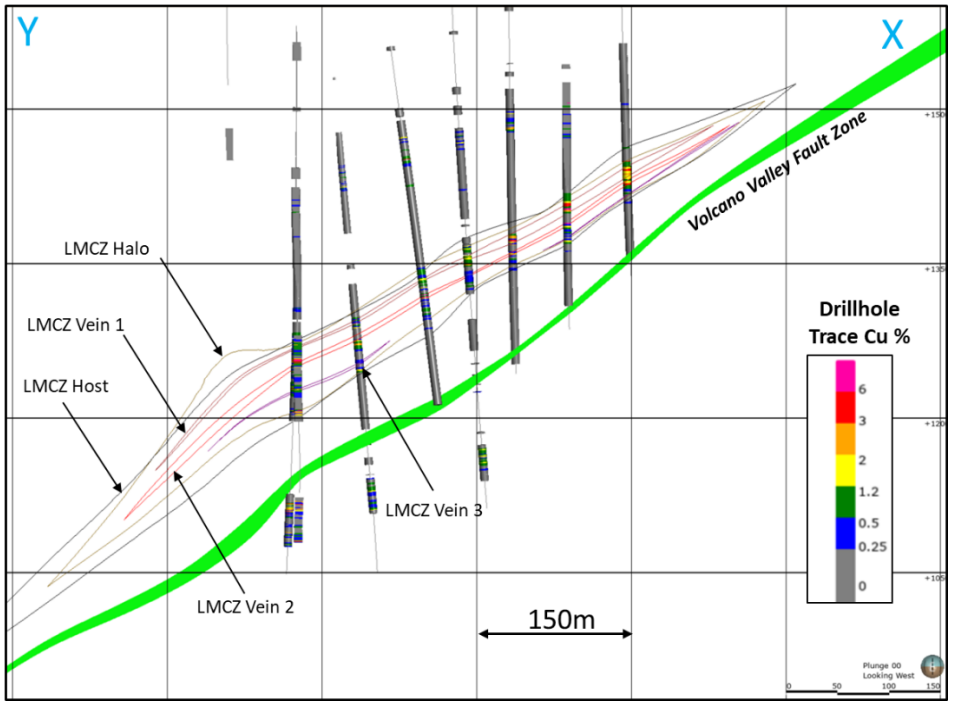

Figure 10.8 Cross-section of the LMCZ along line X-Y looking west

The LLCZ is a shallow S/SE dipping zone of mineralization that occurs in the FW of the VVFZ (Figures 10.9 and 10.10).

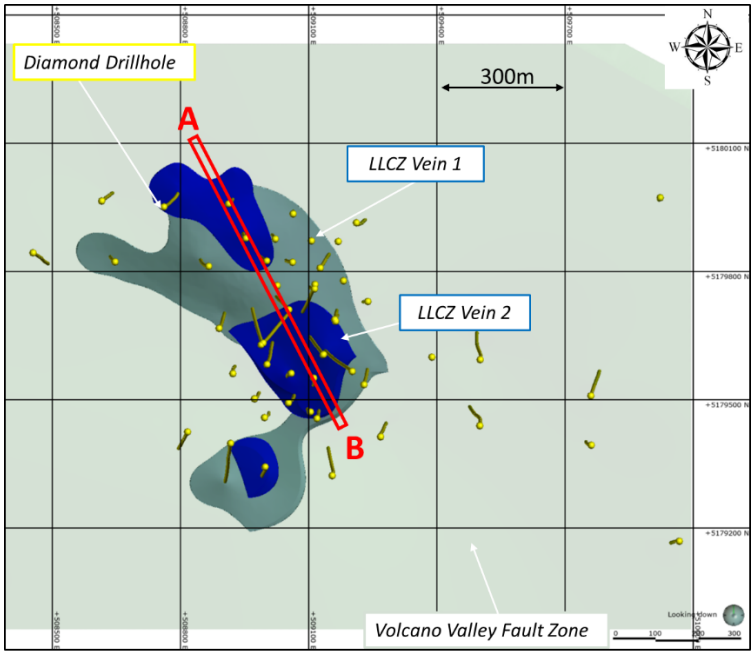

Figure 10.9 Plan View of the LLCZ and the VVFZ (rendered transparent)

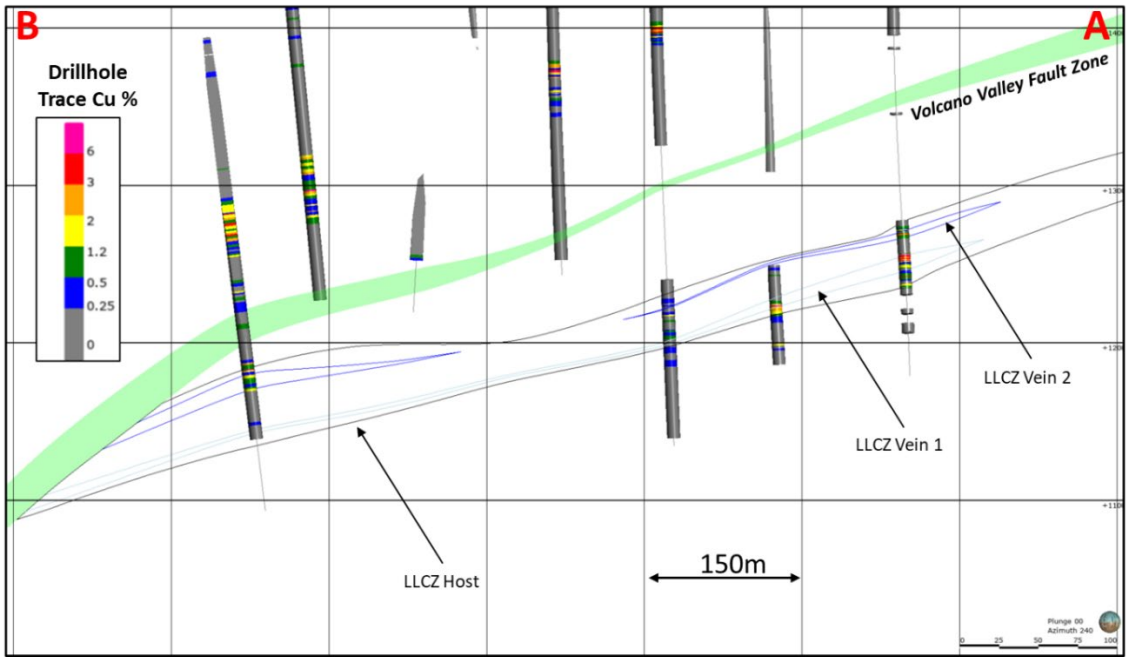

Figure 10.10 Cross-section of the LLCZ along line A-B looking SW

10.4 QP Opinion on Accuracy

Erik Ronald, P.Geo (APGO# 3050) of SRK, acting as QP of Mineral Resources has reviewed the drilling data, procedures, and methodology including a site visit to an active diamond core drilling pad. It is the opinion of the QP that all drilling data included in this Mineral Resource has been performed to good industry standard and the resultant data is appropriate for use in the Mineral Resource estimation.

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11. SAMPLE PREPARATION, ANALYSIS AND SECURITY

11.1 Sample Preparation for Analysis

All Sandfire Resources America, Inc. samples were sent for sample preparation to ALS Global, 4977 Energy Way, Reno, Nevada (ALS Reno). ALS Reno is ISO/IEC 17025 accredited for sample preparation. The 2018 Phase 1 drilling of the UCZ using PQ core was an exception, with samples prepared by Base Metallurgical Laboratories of 970 McMaster Way, Kamloops, British Columbia, Canada (BML Kamloops). Metallurgical laboratories such as BML Kamloops are not certified to ISO 17025 standards. BML Kamloops follows industry accepted practices and all outputs are signed off by a Professional Engineer.

Upon receipt at ALS Reno, samples were sorted, checked against the submission sheet, weighed, barcoded, and logged into the ALS sample management system.

The samples were dried for a minimum of 8 hours at 100°C. The samples were then coarse-crushed to 70% minus 6 mm using a swing jaw-crusher. Every 30th sample was passed through a dry-sieve to ensure that required crush specifications were obtained.

The coarse-crushed material was then fine-crushed to 70% minus 2 mm using a Boyd jaw-crusher and a 1,000 g analytical sample was split off using a Boyd rotary splitter. The fine-crushed material from every 20th sample was passed through a dry-sieve for quality control of fine-crushing.

The 1,000 g analytical sample was then pulverized to 85% minus 75 µm using an Essa LM2 vibratory pulverizing mill. A split of the pulverized material from every 20th sample was wet-sieved to ensure that at least 85% of the pulverized material was <75 µm.

The 1,000 g pulverized sample (pulp) was tipped out of the grinding bowl onto a mat and an approximately 130 g subsample collected, for fire assay, by scooping an x-pattern through the pulp pile (similar to cone and quartering). A 25 to 50 g subsample was collected in the same way for acid digest ICP-AES. The remaining pulp material (pulp residue) was bagged and stored. Envelopes containing the acid digest ICP-AES subsample were shipped to ALS Vancouver.

Whole core intervals from the 2018 Phase 1 UCZ PQ drilling were sent to BML Kamloops. The PQ samples were dried, coarse-crushed to 70% minus 20 mm using a Savona Equipment Gladiator jaw-crusher and then fine-crushed to 70% <2 mm using a Denver Cone crusher. The sample was then split using a Gilson Riffler SP1. A 1,000 g split of the fine-crushed material was pulverized to 90% <75 µm using a TM Engineering ring and puck mill.

The 1,000 g pulverized splits of the 2018 Phase 1 PQ core were sent from BML Kamloops to ALS Global, 2155 Dollarton Highway, North Vancouver, British Columbia (ALS Vancouver) which is ISO 17025 accredited for sample preparation. Prior to undertaking any analytical work, ALS Vancouver completed wet-sieve testing of a proportion of the pulverized splits to check that at least 85% of the pulverized material was passing 75 µm.

At both ALS Reno and BML Kamloops a 250 g fine-crush split was also produced and stored for mineralogical investigation.

At ALS Reno, a duplicate 1,000 g fine-crush split was created for selected samples (Laboratory Duplicate) and pulverized to 85% minus 75 µm. ALS Reno was also instructed to retain all analytical sample pulp residues such that a certain proportion could be reanalyzed at a different laboratory (Umpire Samples).

11.2 Security Measures

For all SRA drilling, drill core was collected from the drill rigs daily by SRA staff and delivered directly to a secure core logging facility, attached to the SRA office in White Sulphur Springs, MT. After logging, the drill core was stored in a secure warehouse/core cutting facility, also attached to the SRA office, until it was cut and sampled. Access to the logging facility and warehouse/core cutting facility was restricted to SRA geological staff.

Once drill core samples were cut, they were placed in labelled calico bags. Multiple calico bags were placed in polypropylene sacks and sealed with cable ties. Polypropylene sacks were placed on wooden pallets and secured using plastic wrap. Samples in preparation were kept in the secure warehouse. Once a pallet of samples was ready for dispatch it was moved to the secure core logging facility.

All samples were shipped to ALS Reno or BML Kamloops by FedEx Corporation (FedEx). FedEx collected the samples from the SRA secure logging facility at which point they assumed responsibility for the chain of custody until delivery to the laboratory.

Upon delivery to the laboratory the samples were unpacked and checked by laboratory staff. Both ALS Reno and BML Kamloops have industry standard sample security protocols at all sample preparation and analytical facilities.

11.3 Sample Analysis

A split of the analytical pulp was sent to ALS Vancouver. ALS Vancouver is ISO/IEC 17025 accredited for Au assay by lead collection fire assay, four acid sample digestion and multi-element analysis using an ICP-AES for low grade to high grade ores.

A four acid digestion was performed on a 0.4 g aliquot and the analysis of the digest was performed using an ICP-AES, calibrated for intermediate level analyses (low to medium grade ore). Table 11.1 lists analytes and detection ranges for the intermediate level ICP-AES analyses.

Element	Lower Limit of Detection	Upper Limit of Detection	Element	Lower Limit of Detection	Upper Limit of Detection
Ag	1 ppm	200 ppm	Mo	10 ppm	5%
Al	0.05%	30%	Na	0.05%	30%
As	50 ppm	10%	Ni	10 ppm	10%
Ba	50 ppm	5%	P	50 ppm	10%
Bi	20 ppm	5%	Pb	20 ppm	10%
Ca	0.05%	50%	S	0.05%	10%
Cd	10 ppm	1%	Sb	50 ppm	5%
Co	10 ppm	5%	Sc	10 ppm	5%
Cr	10 ppm	10%	Sr	10 ppm	10%
Cu	10 ppm	10%	Th	50 ppm	5%
Fe	0.05%	50%	Ti	0.05%	30%
Ga	50 ppm	5%	Tl	50 ppm	5%
K	0.1%	30%	U	50 ppm	5%
La	50 ppm	5%	V	10 ppm	10%
Mg	0.05%	50%	W	50 ppm	5%
Mn	10 ppm	5%	Zn	20 ppm	10%

Table 11.1 Analytes and detection ranges for intermediate level ICP-AES analyses. ALS Code ME-ICP61a

When the upper limits of detection for the intermediate level ICP-AES were exceeded for Cu, Ag, Pb or Zn, the digests from the overlimit samples were reanalysed for the overlimit elements in an ore grade level ICP-AES circuit. Table 11.2 lists detection ranges for these elements in the high level ICP-AES.

Element	Lower Limit of Detection	Upper Limit of Detection	Element	Lower Limit of Detection	Upper Limit of Detection
Ag	1 ppm	1,500 ppm	Pb	0.001%	20%
Cu	0.001%	50%	Zn	0.000%	30%

Table 11.2 Analytes and detection ranges for ore-grade ICP-AES analyses. ALS Code OG62

Au analyses of the analytical sample were either conducted at ALS Reno or ALS Vancouver using lead collection fire assay on a 30 g charge followed by acid digestion and Atomic Absorption Spectroscopic (AA) analysis. This technique allowed for Au analyses up to 10 ppm with a lower detection limit of 5 ppb. On the occasions that Au concentrations exceeded 10 ppm repeat analyses were performed by fire assay with a gravimetric finish.

All umpire sample analyses were completed at American Assay Laboratories (AAL), 1506 Glendale Ave, Sparks, Nevada which is ISO/IEC 17025 accredited. Au analyses of umpire samples were performed using a 30 g charge, lead collection fire assay, acid digest and an ICP-AES finish. Detection range for Au analyses using this technique at AAL is 0.003 to 10 ppm. AAL use a five acid digest of a 0.5 g aliquot to produce a digest for 35 element ICP-AES analysis. Table 11.3 outlines analytes and detection ranges for this technique.

Element	Lower Limit of Detection (ppm)	Upper Limit of Detection	Element	Lower Limit of Detection (ppm)	Upper Limit of Detection
Ag	0.5	100 ppm	Mo	1	1%
Al	100	20%	Na	100	10%
As	2	1%	Ni	1	1%
Ba	5	0.5%	P	10	1%
Be	0.1	0.1%	Pb	3	1%
Bi	5	1%	S	100	10%
Ca	100	35%	Sb	2	1%
Cd	0.5	0.1%	Sc	1	1%
Co	1	1%	Sr	1	1%
Cr	1	1%	Th	20	1%
Cu	1	1%	Ti	10	10%
Fe	100	50%	Tl	10	0.1%
Ga	10	1%	U	10	1%
K	100	10%	V	1	1%
La	10	1%	W	2	1%
Mg	100	40%	Y	1	0.5%
			Zn	2	1%

Table 11.3 Analytes and detection ranges for umpire sample ICP-AES analyses. AAL Code: ICP-5AO35

Upper limits of detection for the ICP-AES were sometimes exceeded for Cu and Zn. Where this occurred the digests from the overlimit umpire samples were reanalyzed for the overlimit elements in an ore grade level ICP-AES circuit. Table 11.4 lists detection ranges for these elements in the high level ICP-AES.

Element	Lower Limit of Detection	Upper Limit of Detection	Element	Lower Limit of Detection	Upper Limit of Detection
Cu	0.001%	50%	Zn	0.001%	30%

Table 11.4 Analytes and detection ranges for overlimit ICP-AES analyses at AAL

Where concentration of Ag in umpire samples exceeded the upper limit of detection in the ICP-AES these samples were reanalyzed for Ag using fire assay with a gravimetric finish (lower limit of detection = 7 ppm, no upper limit of detection).

11.3.1 Specific Gravity Analyses

SG data is collected from diamond drill core and analysed at SRA facilities located in White Sulphur Springs, MT. The Archimedean or immersion method is used to determine SG based on the weight of a sample and the volume of water displaced.

Prior to the 2018 Phase 1 drilling program, individual pieces of representative drill core were selected by SRA geologists and tested for SG. In the SRA database, these SG's are considered POINT data as they represent a specific depth based on the drill sample. Commencing with the 2018 Phase 1 drilling, the sample selection process was modified so that the SG interval corresponded with the sampling interval continuously across mineralized intervals. This data is referred to in the database as INTERVAL SG data.

After SG testing is completed, the data is entered into the SRA database for determination of SG and quality control checking.

Figure 11.1 Specific Gravity Testing Apparatus at the SRA Logging Facility, White Sulphur Springs, MT

11.4 Quality Assurance/Quality Control Procedures

QA/QC utilized at the Black Butte Copper Project have evolved subsequent to acquisition of the Project by SRA in 2014. Changes have been made to the QA/QC protocols at the onset of each of the drill programs that inform the Mineral Resource estimate:

- 2014 Drill Program;
- 2018 Phase 1 Drill Program; and
- 2018/19 Phase 2 Drill Program.

The descriptions and documentation of QA/QC procedures that follow are subdivided by drill program.

11.4.1 Standards

2014 Drill Program

Three Certified Reference Materials (CRM's) were purchased from WCM Minerals, 7729 Patterson Ave, British Columbia (Table 11.5) and inserted into the sample sequence at a minimum ratio of one CRM every 20 samples. The actual CRM insertion rate during the drill program was 7%.

WCM Minerals CRM ID	Cu (%)		Ag (g/t)	
	Expected Value	Standard Deviation	Expected Value	Standard Deviation
Cu-145	3.10	0.090	93.0	3.366
Ni-116	0.78	0.002	N/A	N/A
Pb-129	0.28	0.012	23.0	1.696

Table 11.5 Certified values of the WCM Minerals CRM's used during the 2014 drill campaign

Figures 11.2 to Figure 11.4 show the CRM performance for Cu during the 2014 drill program. With the exception of Ni-116, all CRM's returned analyses within acceptable limits (one analysis of Cu-145 assayed slightly outside two standard deviations). Ni-116 showed a negative bias with three analyses plotting outside of the three standard deviation envelope. Ni-116 is a CRM derived from pyrrhotite – pentlandite, nickel sulphide ore. It was concluded that the negative bias for Ni-116 is related to matrix and analytical technique rather than laboratory inaccuracy.

Figure 11.5 and Figure 11.6 show the performance of the two CRM's that were certified for Ag. One analysis exceeding the two standard deviation limit was returned for Cu-145. All other CRM Ag performance was acceptable although a positive bias was noted for Cu-145.

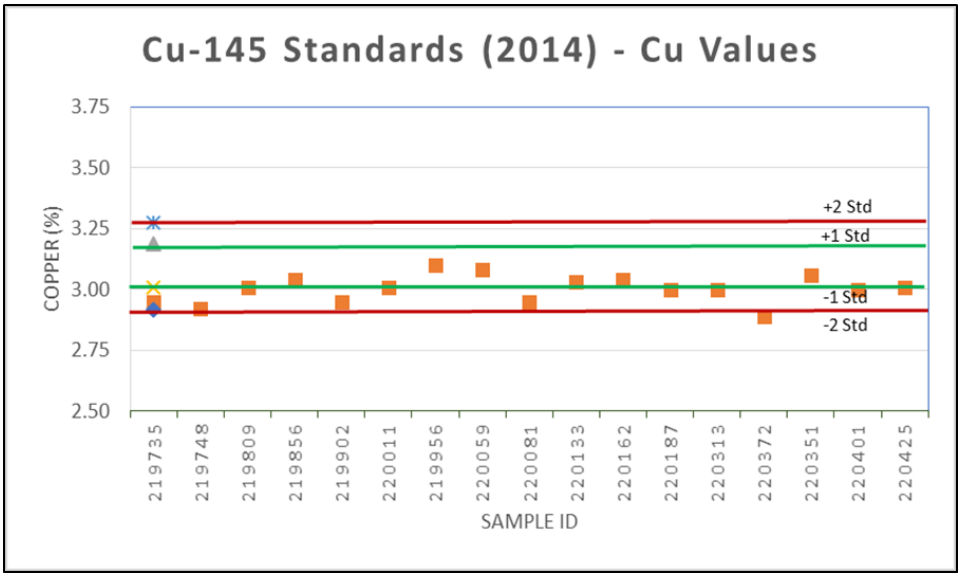

Figure 11.2 Cu-145 Cu values from the 2014 drill program

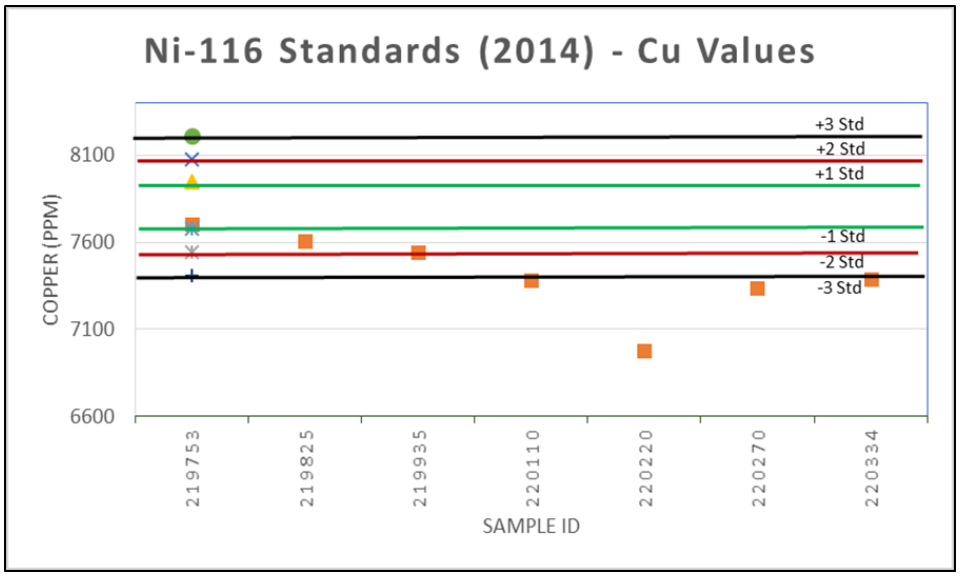

Figure 11.3 Ni-116 Cu values from the 2014 drill program

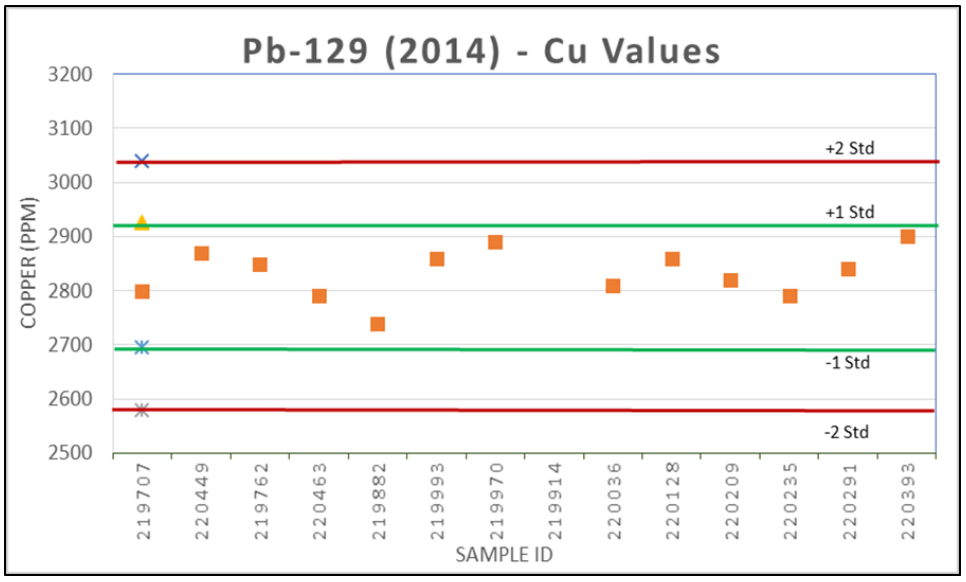

Figure 11.4 Pb-129 Cu values from the 2014 drill program

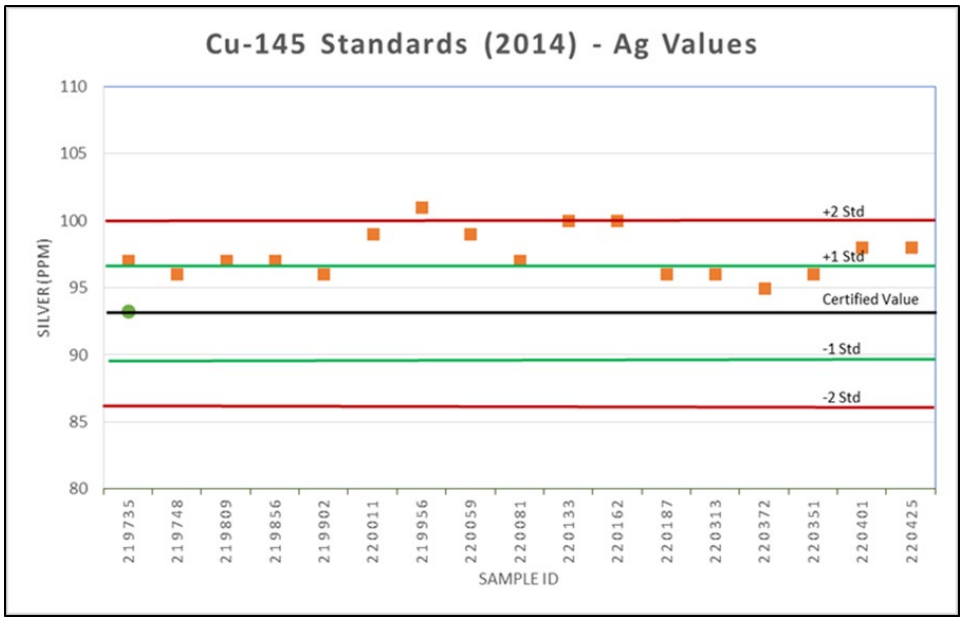

Figure 11.5 Cu-145 Ag values from the 2014 drill program

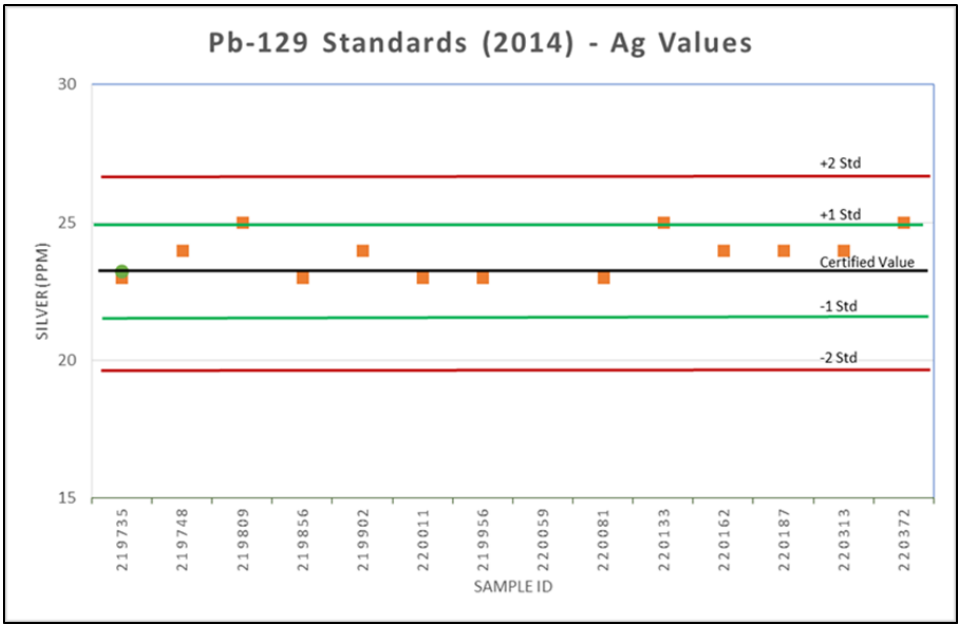

Figure 11.6 Pb-145 Ag values from the 2014 drill program

2018 Phase 1 Drill Program

Four CRM's were purchased from ORE Research and Exploration Pty Ltd, 37A Hosie Street, Bayswater North, Victoria, Australia. Table 11.6 lists the certified Cu and Ag values for these CRM's.

Ore Research CRM ID	Cu (%)		Ag (g/t)	
	Expected Value	Standard Deviation	Expected Value	Standard Deviation
OREAS-624	3.101	0.079	45.30	1.26
OREAS-522	0.916	0.026	1.31	0.11
OREAS-134b	0.135	0.005	209.00	9.00
OREAS-136	0.031	0.001	151.00	5.00

Table 11.6 Certified values for Cu and Ag for ORE Research CRM's used during the 2018 Phase 1 drill program.

CRM's were inserted into the sample sequence at a minimum rate of two CRM's for every 20 samples. For the 2018 Phase 1 drill program actual CRM insertion rate was 14%. Table 11.6 to Figure 11.10 show CRM Cu performance during the 2018 drill program. One CRM (OREAS-522) exceeded the two standard deviation tolerance on one occasion, prompting reanalysis of the portion of the batch affected. The reanalysed samples passed all QA/QC.

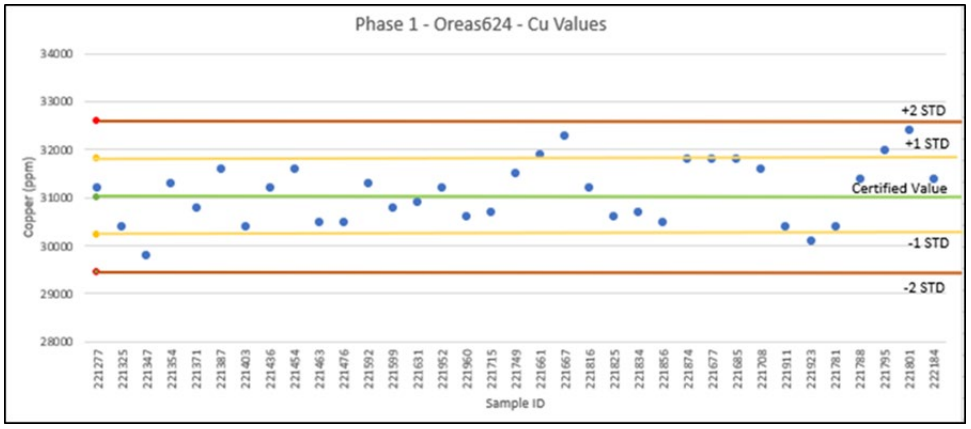

Figure 11.7 OREAS-624 Cu values from the 2018 Phase 1 drill program

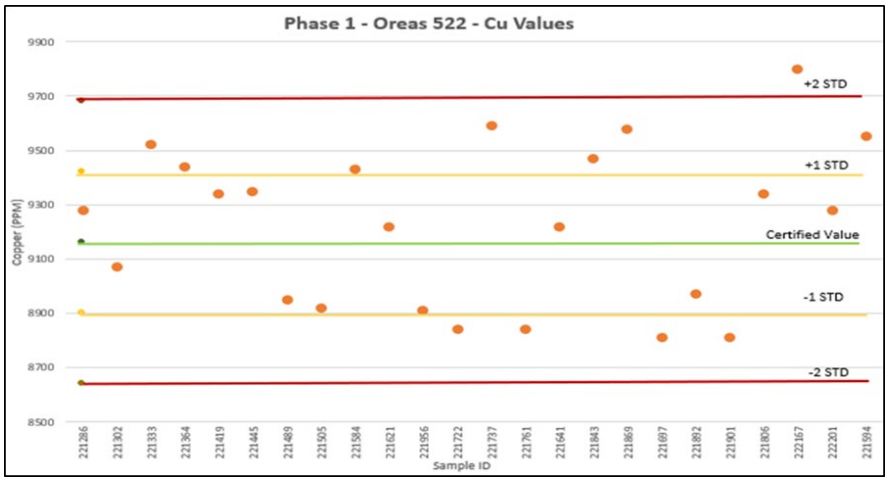

Figure 11.8 OREAS-522 Cu values from the 2018 Phase 1 drill program

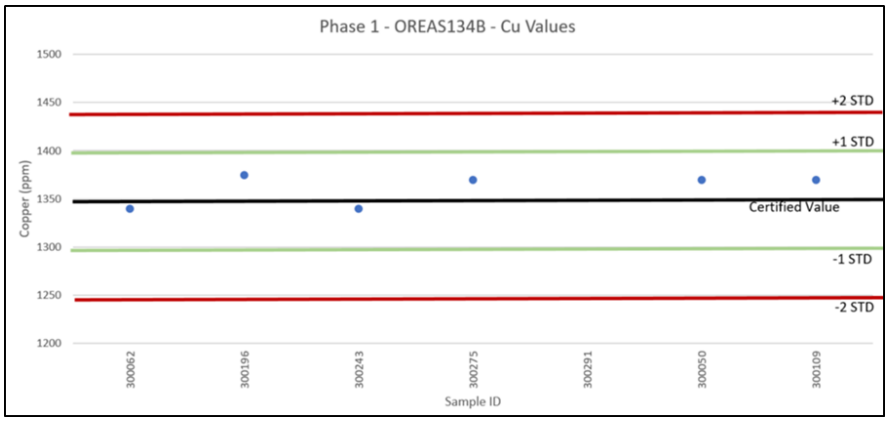

Figure 11.9 OREAS-134B Cu values from the 2018 Phase 1 drill program

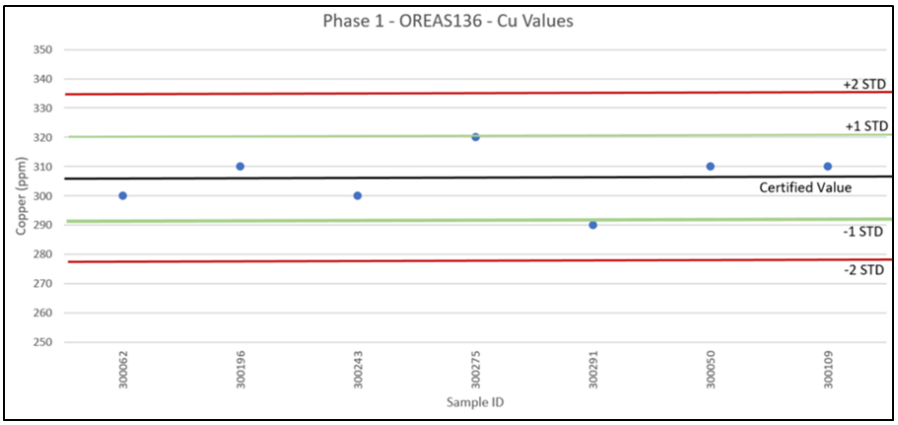

Figure 11.10 OREAS-136 Cu values from the 2018 Phase 1 drill program

Figure 11.11 to Figure 11.14 show CRM Ag performance during the 2018 Phase 1 drill campaign. Ag analyses were generally within or close to two standard deviations of the certified value. One OREAS-134B CRM returned an analysis that significantly exceeded the two standard deviation threshold.

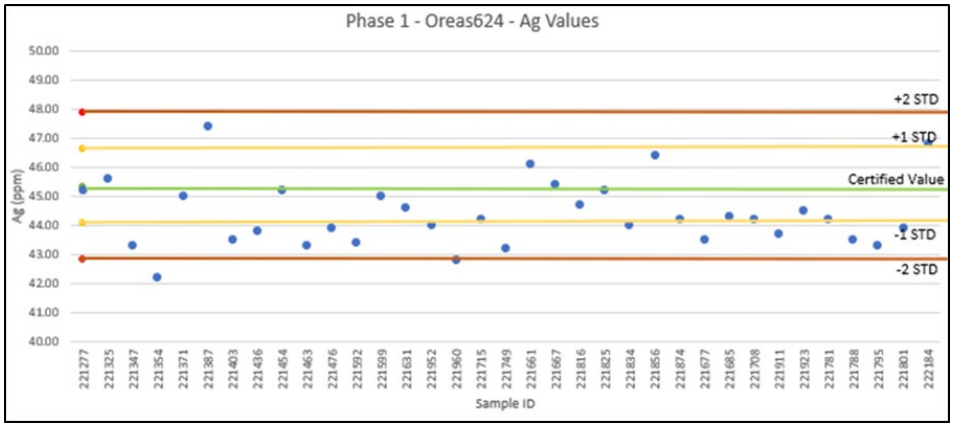

Figure 11.11 OREAS-624 Ag values from the 2018 Phase 1 drill program

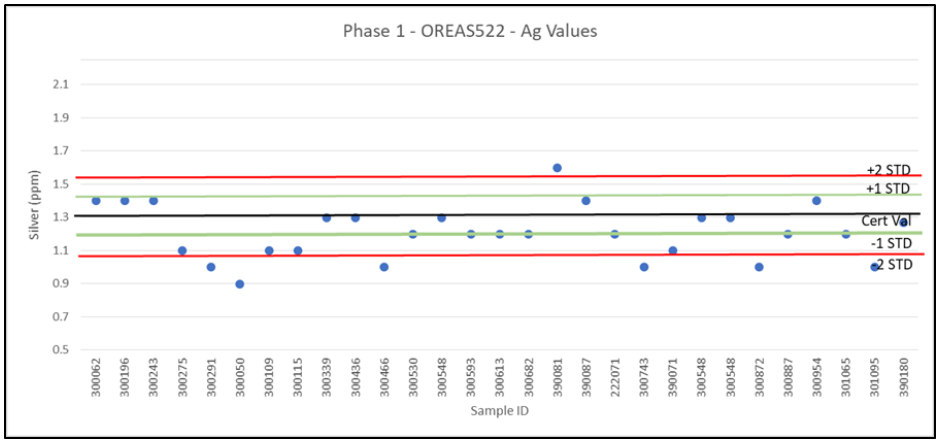

Figure 11.12 OREAS-522 Ag values from the 2019 Phase 1 drill program

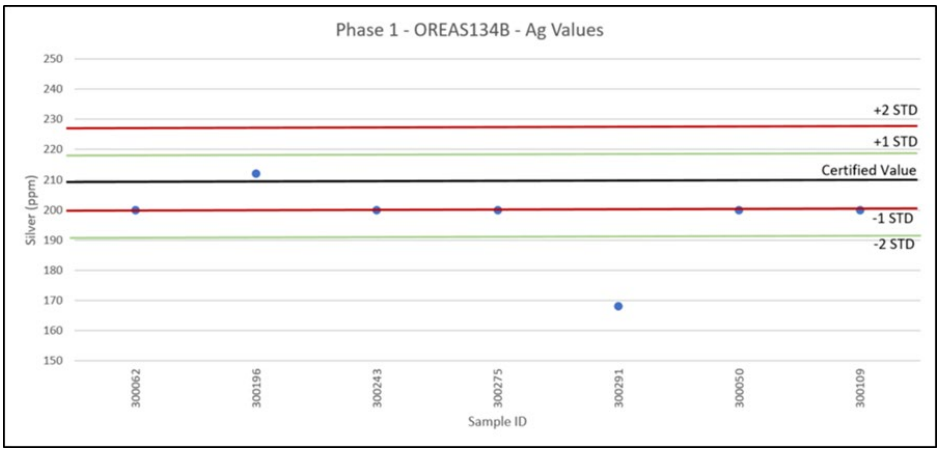

Figure 11.13 OREAS-134B Ag values from the 2018 Phase 1 drill program

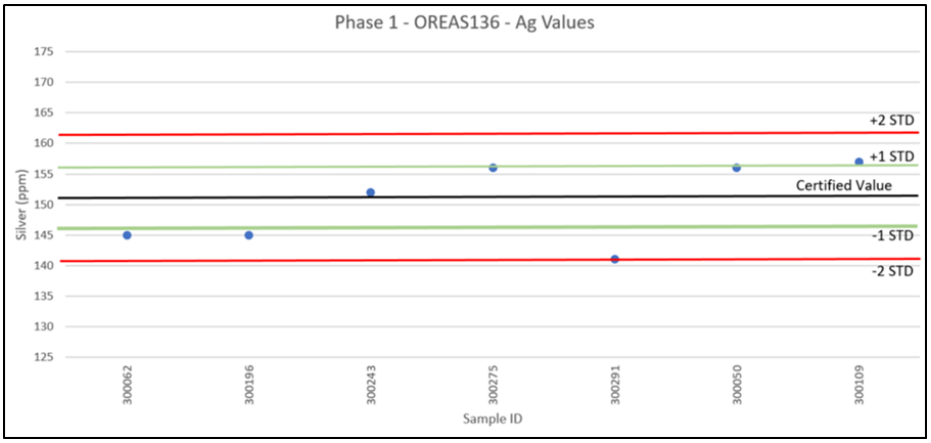

Figure 11.14 OREAS-136 Ag values from the 2018 Phase 1 drill program

2018/19 Phase 2 Drill Program

The same CRM's utilized for the 2018 Phase 1 drill program were also used during the 2018/19 Phase 2 program although an additional CRM (OREAS-935) was added to the suite given the high Cu grades expected from the LCZ (Table 11.7).

CRM's were inserted into the sample sequence at a minimum insertion ratio of one CRM for every 20 samples. Actual CRM insertion rate for the 2018/19 Phase 2 drill program was 7%.

Figure 11.15 to Figure 11.19 show CRM Cu performance for the 2018/19 Phase 2 drill program. Three OREAS-624 CRM analyses exceeded two standard deviations triggering reanalysis of the sample sequences affected. All CRM's in the reanalysed sequences passed QA/QC.

Ore Research CRM ID	Copper (%)		Silver (g/t)	
	Expected Value	Standard Deviation	Expected Value	Standard Deviation
OREAS-935	12.550	0.388	43.87	3.84
OREAS-624	3.101	0.079	45.30	1.26
OREAS-522	0.916	0.026	1.31	0.11
OREAS-134b	0.135	0.005	209.00	9.00
OREAS-136	0.031	0.001	151.00	5.00

Table 11.7 Certified values for Cu and Ag for ORE Research CRM's used during the 2018/19 Phase 2 drill program

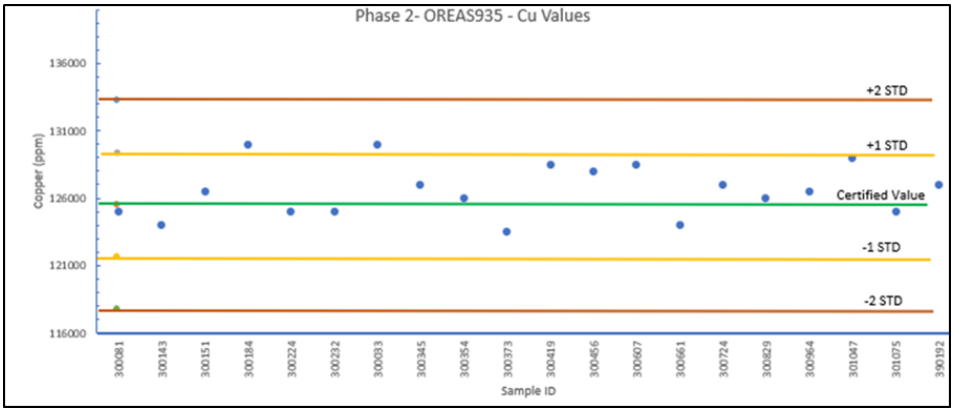

Figure 11.15 OREAS-935 Cu values from the 2018/19 Phase 2 drill program

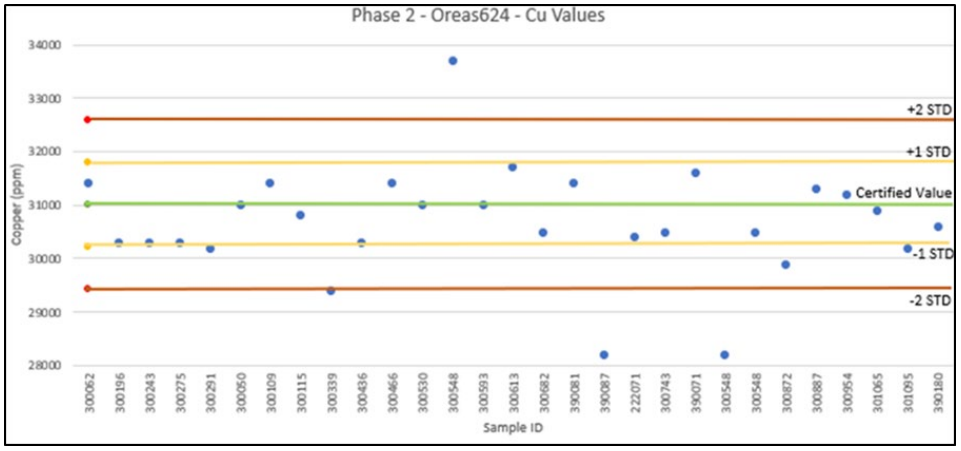

Figure 11.16 OREAS-624 Cu values from the 2018/19 Phase 2 drill program

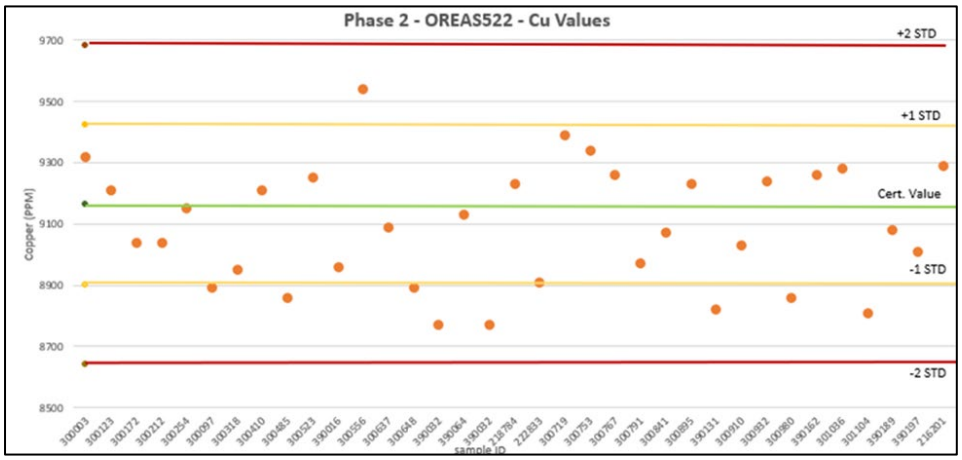

Figure 11.17 OREAS-522 Cu values from the 2018/19 Phase 2 drill program

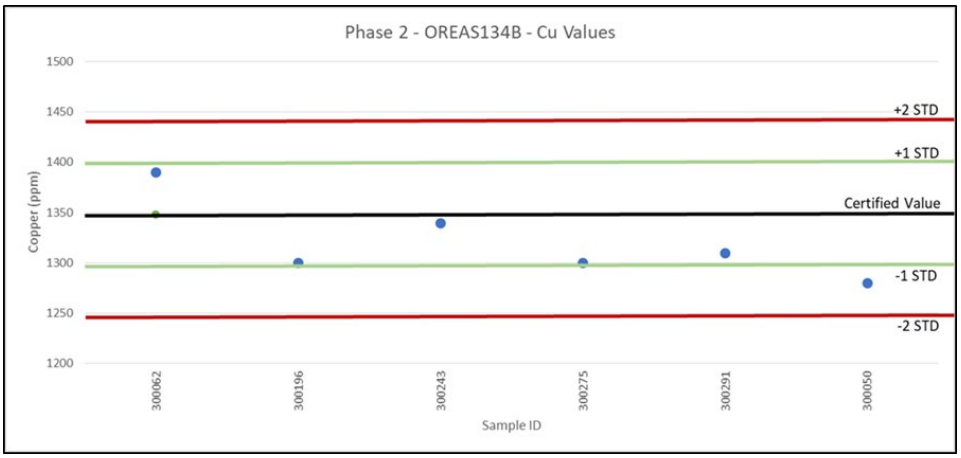

Figure 11.18 OREAS-134B Cu values from the 2018/19 Phase 2 drill program

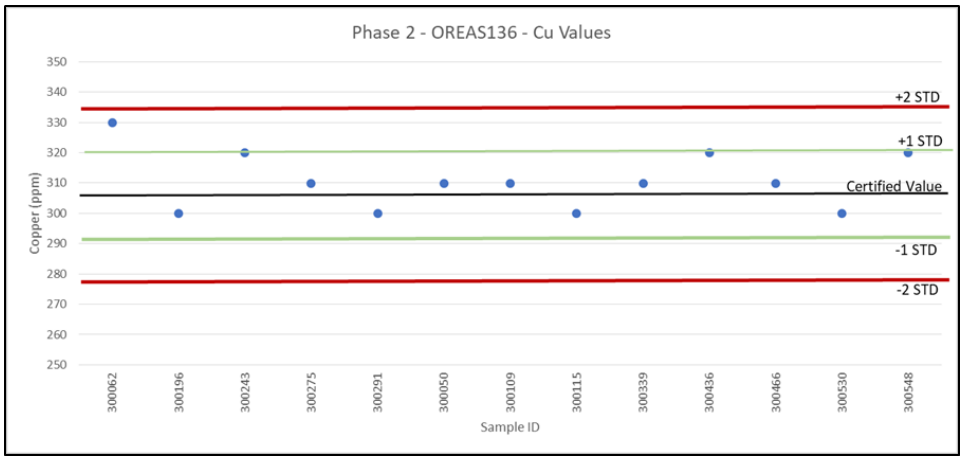

Figure 11.19 OREAS-136 Cu values from the 2018/19 Phase 2 drill program

Figure 11.20 to Figure 11.24 show CRM Ag performance for the 2018/19 Phase 2 drill program. Analyses outside of the two standard deviation thresholds were returned for eight CRM's (OREAS-935 OREAS-624, OREAS-522).

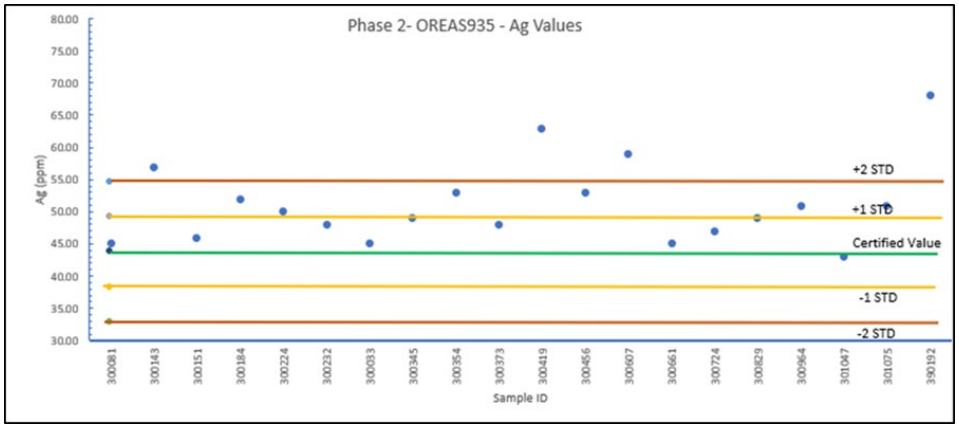

Figure 11.20 OREAS-935 Ag values from the 2018/19 Phase 2 drill program

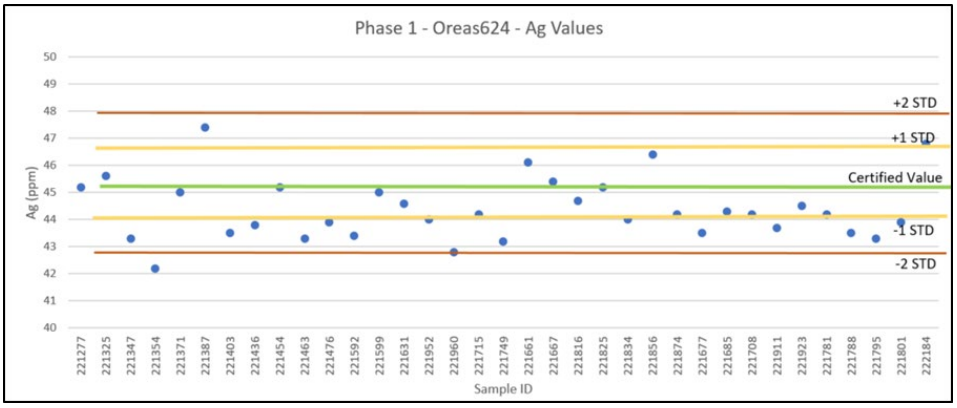

Figure 11.21 OREAS-624 Ag values from the 2018/19 Phase 2 drill program

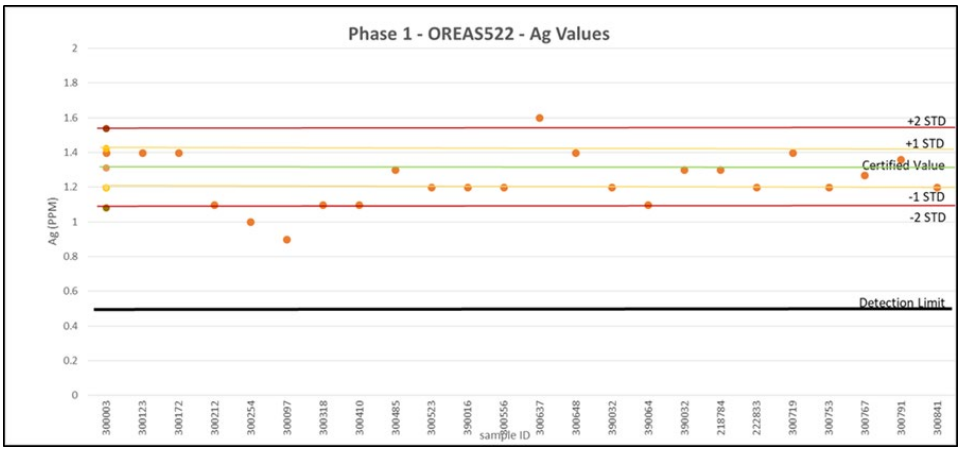

Figure 11.22 OREAS-522 Ag values from the 2018/19 Phase 2 drill program

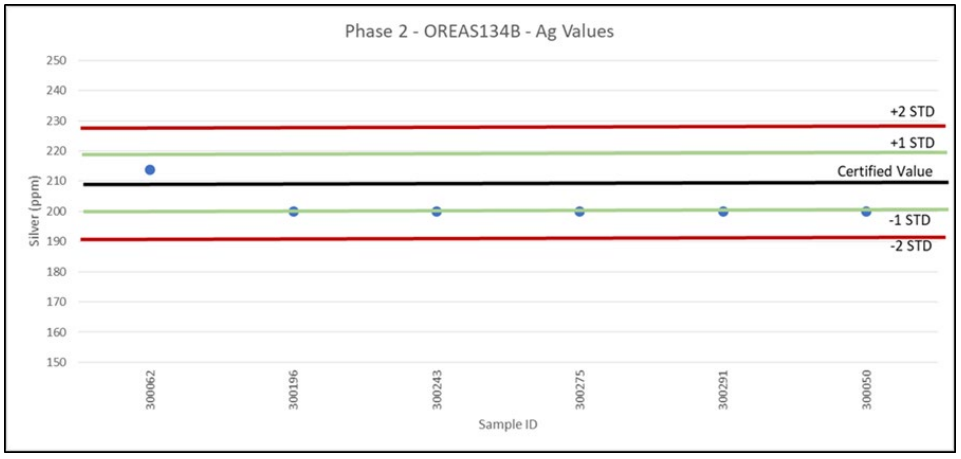

Figure 11.23 OREAS-134B Ag values from the 2018/19 Phase 2 drill program

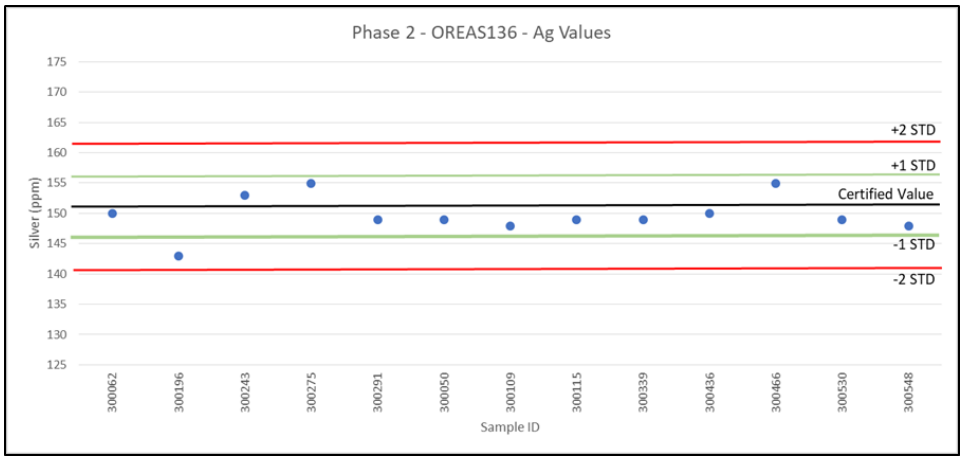

Figure 11.24 OREAS-136 Ag values from the 2018/19 Phase 2 drill program

11.4.2 Blanks

2014 Drill Program

Landscaping marble chips that had been used as coarse blanks in the SRA 2012 drill program were also used in the SRA 2014 drill program. Coarse blanks were inserted at a minimum ratio of one blank every 20 samples. Actual blank insertion rate for the 2014 drill program was 6%.

As indicated on Figure 11.25, these blanks returned poor performance during 2014, often exceeding Cu detection limits by >15 times detection. Similar poor coarse blank performance was reported by Winckers *et al* (2013) from the same landscaping marble chip used as coarse blanks in the 2012 drill program. It is uncertain whether low level Cu analyses returned from the marble are related to contamination in the laboratory crushing and pulverizing circuit or whether some of the marble contained low levels of Cu. The use of the landscape marble was discontinued after 2014.

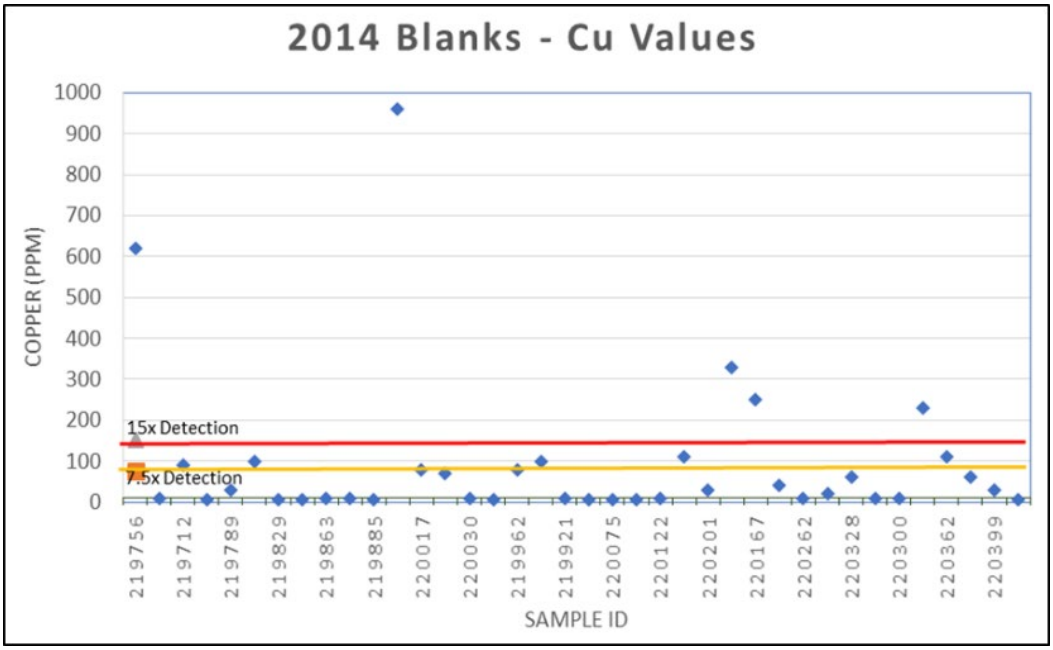

Figure 11.25 Cu analyses of landscape marble chips used as coarse blanks during the 2014 drill program

2018 Phase 1 Drill Program

With the exception of two LCZ drillholes, all 2018 Phase 1 drill program samples were whole PQ core, designated for metallurgical testing, that were prepared at BML Kamloops. No coarse blank material was inserted into the sample sequence during crushing and milling. Fine-crushed and pulverized 1,000 g splits were dispatched to ALS Vancouver after BML had inserted CRM's and pulp blanks into the sample sequence, at positions specified by SRA geologists.

The pulp blank utilized by SRA was OREAS-22e purchased from ORE Research and Exploration Pty Ltd. OREAS-22e has a certified value of 7.97 ppm for Cu (standard deviation = 0.746 ppm) and <0.05 ppm for Ag (standard deviation = 0.0495 ppm). All OREAS 22e Cu values plotted below the 15 times detection threshold and the majority below the 7.5 times detection threshold (Figure 11.26).

Ag analyses for OREAS-22e during the 2018 Phase 1 drill program were all below the detection limit for the analytical technique (Figure 11.27).

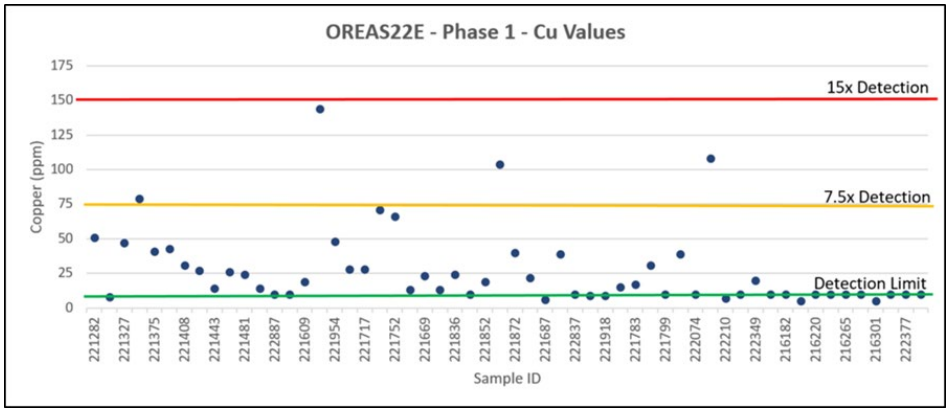

Figure 11.26 Cu analyses of pulp blank OREAS-22e during the 2018 Phase 1 drill program

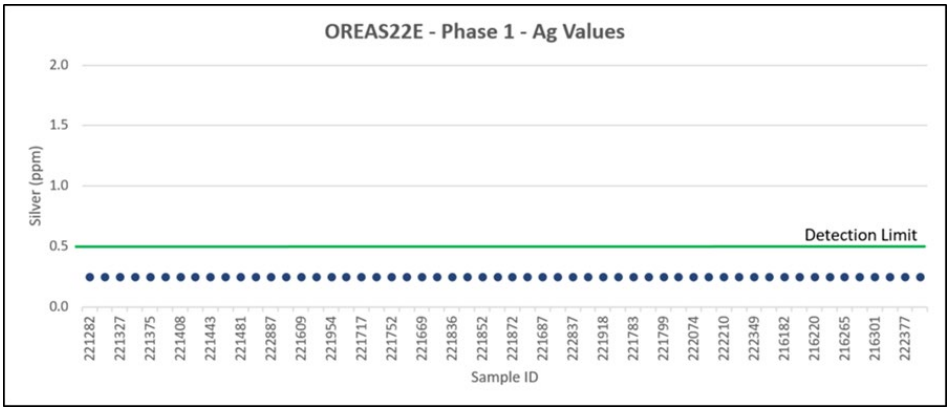

Figure 11.27 Ag analyses of pulp blank OREAS-22e during the 2018 Phase 1 drill program

2018/19 Phase 2 Drill Program

During the 2018/19 Phase 2 Drill program, both pulp blanks (OREAS-22e) and coarse blanks were inserted in the sample sequence by SRA geologists. Coarse blanks (coarse silica blank material) were obtained in 0.5 kg packets from Analytical Solutions Ltd. of 878213 5th, Line East, Mulmur, Ontario, Canada. The recommended upper Cu threshold for the coarse blanks is 25 ppm and for Ag is 1 ppm. Blanks were inserted at a minimum ratio of one blank every 20 samples and alternated between a coarse blank and a pulp blank. Actual insertion rate of blanks for the 2018/19 drill program was 7%.

All pulp blank Cu analyses plotted below the 7.5 times detection threshold (Figure 11.28). All pulp blank Ag analyses reported at, or below, the detection limit (Figure 11.29).

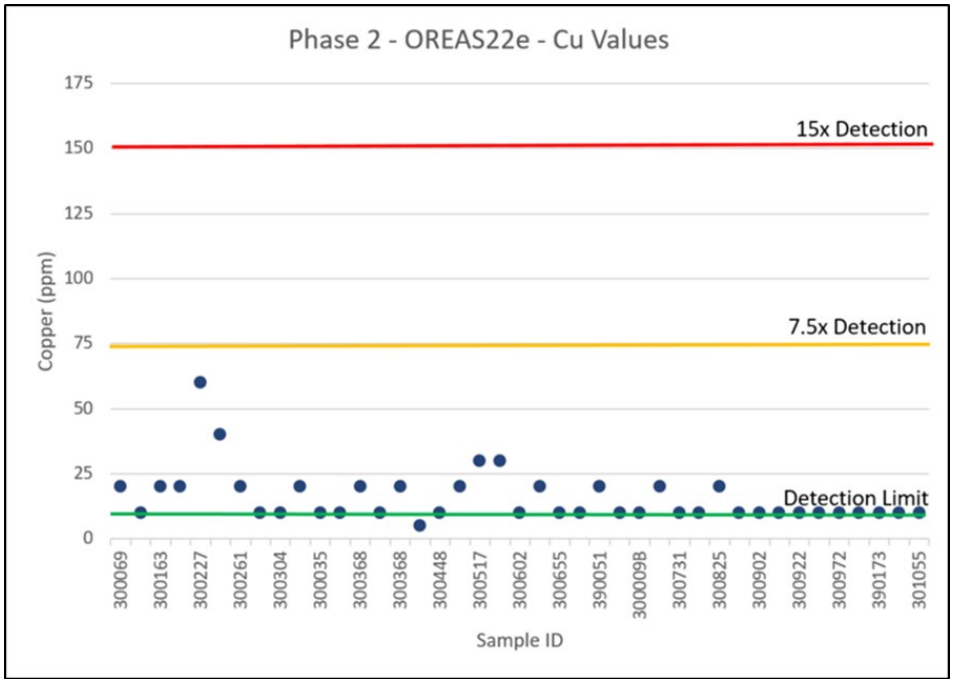

Figure 11.28 Cu results for pulp blank OREAS-22e during the 2018/19 Phase 2 drill program

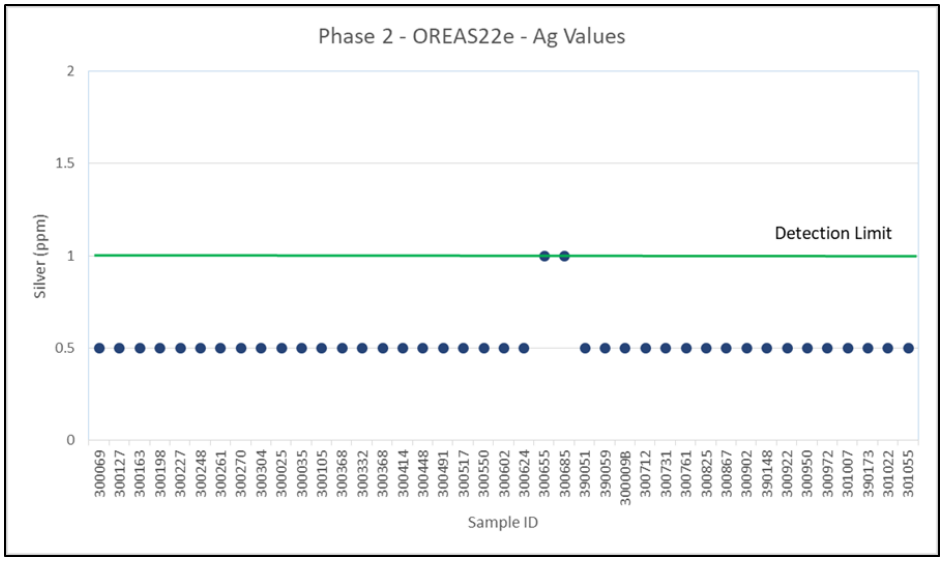

Figure 11.29 Ag results for pulp blank OREAS-22e during the 2018/19 Phase 2 drill program

Figure 11.30 shows coarse blank Cu performance for the coarse silica blank. Analyses reported were generally below 7.5 times detection limit with two analyses exceeding this limit and two analyses exceeding the 15 times detection limit. The coarse blank analyses exceeding the 15 times detection limit were both from the same batch and indicated minor contamination during sample preparation. The two other coarse blanks from this batch both returned values <7.5 times detection. Ag analyses for coarse silica blank all plotted at or below the detection limit (Figure 11.31).

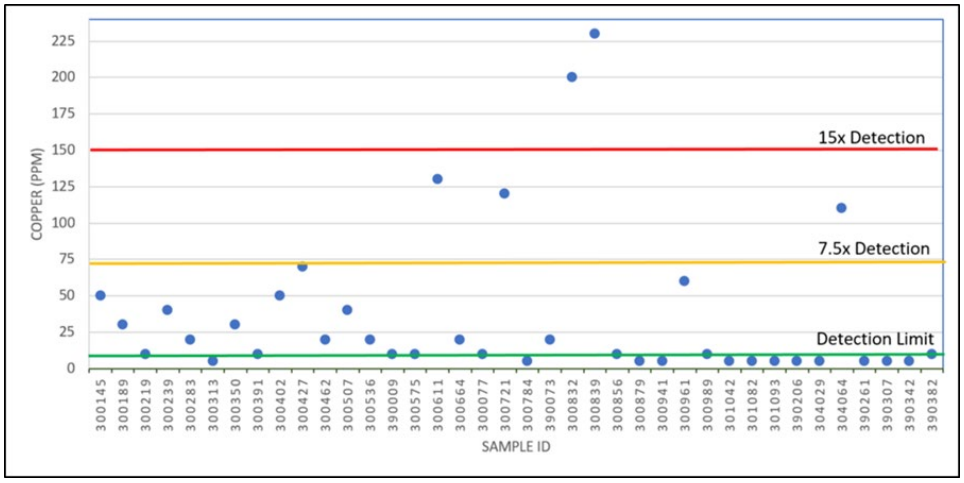

Figure 11.30 Cu results for Analytical Solutions coarse silica blank for the 2018/19 Phase 2 drill program

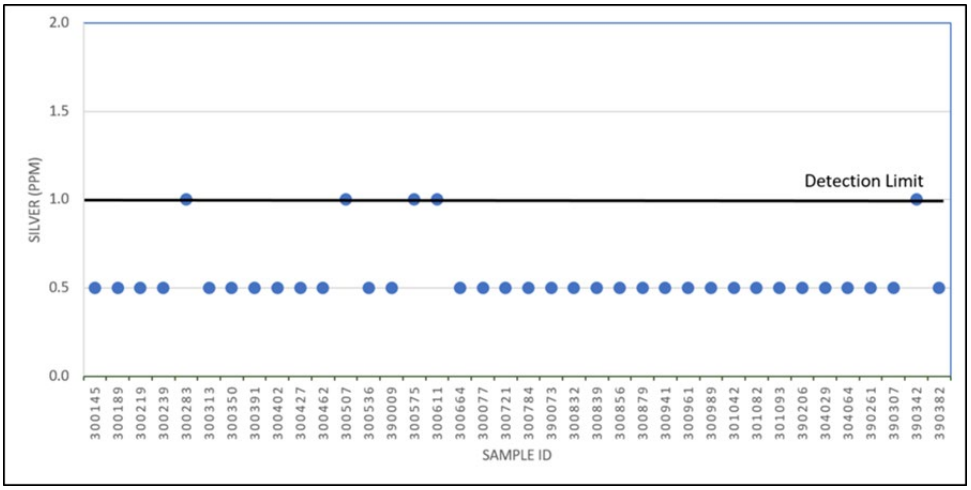

Figure 11.31 Ag results for Analytical Solutions coarse silica blank for the 2018/19 Phase 2 drill program

11.4.3 Duplicates

2014 Drill Program

A quarter-core field duplicate was taken at a minimum rate of one sample per 20 samples and inserted into the sample sequence immediately after the original half-core sample. The actual field duplicate insertion rate for 2014 was 9%. The comparison of Cu values (Figure 11.32) shows an acceptable correlation. Ag analyses of field duplicates show more variation when compared with the original analyses, especially at lower values, but with no clear bias (Figure 11.33).

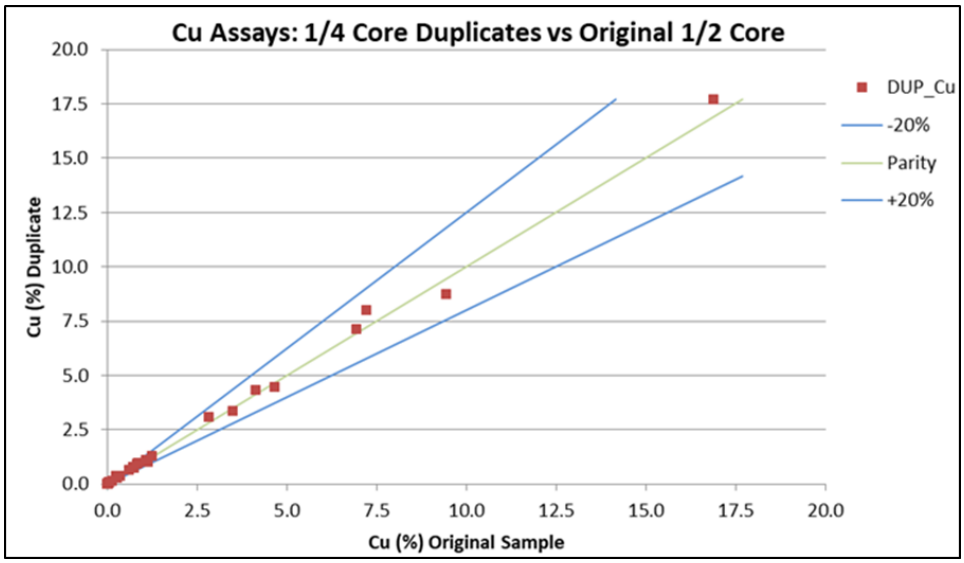

Figure 11.32 Comparison of Cu analyses from original sample and field duplicates for the 2014 drill program

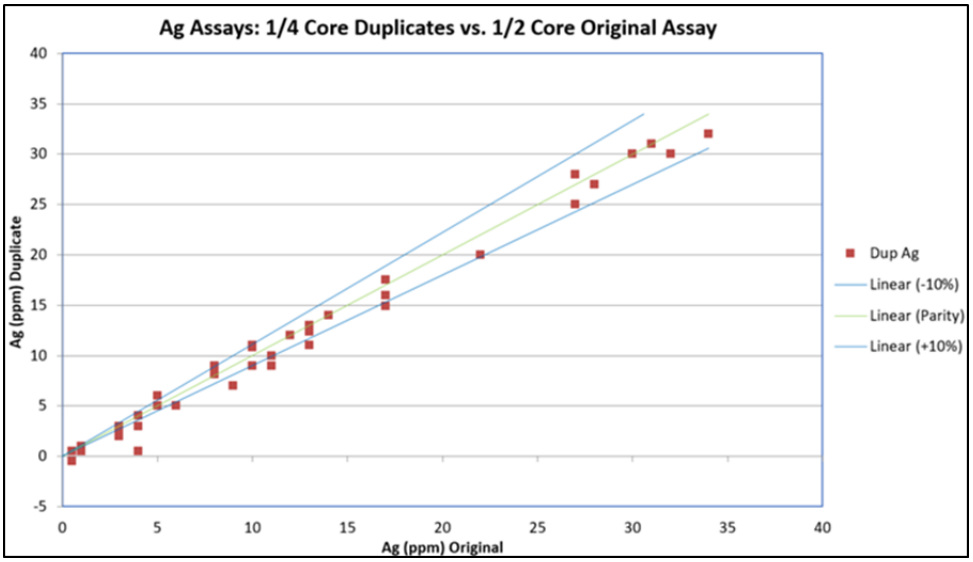

Figure 11.33 Comparison of Ag analyses from original sample and field duplicates for the 2014 drill program

2018 Phase 1 Drill Program

As whole PQ core from the UCZ drillholes was sent to BML Kamloops, no field duplicates were included for these drillholes. Quarter-core field duplicates were included in the samples from the LCZ that were dispatched directly to ALS Reno. Field duplicates were inserted at a minimum ratio of one field duplicate per 20 samples. Given the small number of samples sent to the laboratory, only six field duplicates were inserted. For the 2018 Phase 1 drill program the actual insertion rate was 3%.

Figure 11.34 and Figure 11.35 show comparison of original versus field duplicate analyses for Cu and Ag. The sample set is too small to be statistically meaningful but the correlation between originals and field duplicates is considered acceptable.

Upon completion of all analyses, ALS Vancouver and ALS Reno returned the pulp residues to SRA. 39 pulps were selected, representing a range of Cu grades, for umpire duplicate analyses at AAL Sparks. The pulps were renumbered by SRA geologists and submitted to AAL for 35 element analyses (including base metals) by five acid digest ICP-AES (AAL Code ICP-5A035) and Au and Ag by fire assay with ICP-AES finish (AAL Code FA-Pb30). A comparison of the umpire Cu analyses with original assays is presented in Figure 11.36 and Figure 11.37. These diagrams indicate a reasonable correlation but a positive bias towards the ALS Vancouver results. The four CRM's inserted in the umpire laboratory sample sequence all returned values within one standard deviation of the certified values. It is interpreted that the bias reflects the influence of a different digest rather than laboratory error.

A comparison of the umpire Ag analyses with original assays, shown in Figure 11.38 and Figure 11.39, shows acceptable correlation with no bias.

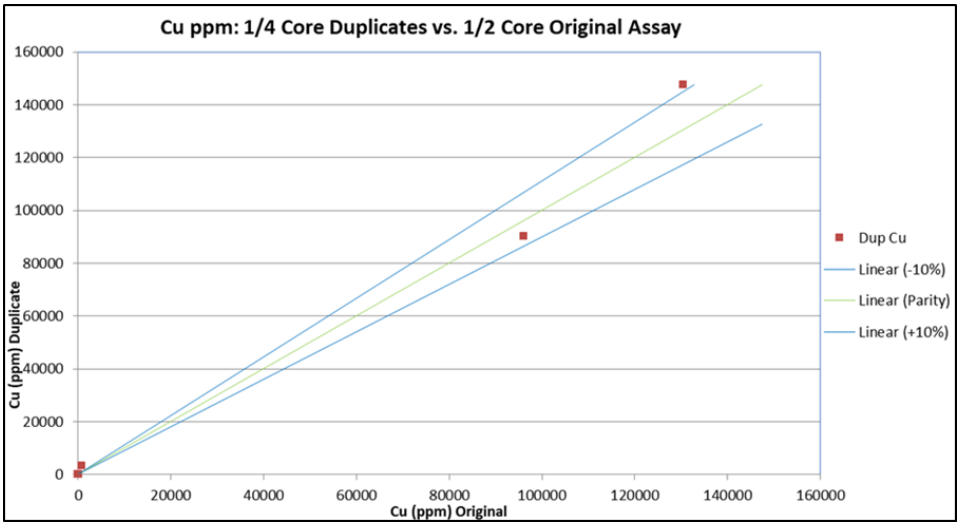

Figure 11.34 Comparison of Cu analyses from original sample and field duplicates for the 2018 Phase 1 drill program

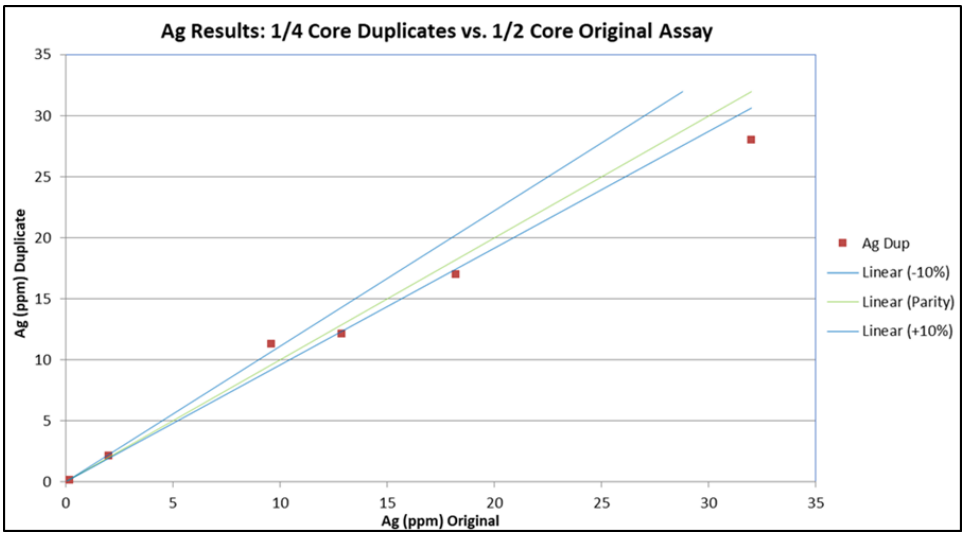

Figure 11.35 Comparison of Ag analyses from original sample and field duplicates for the 2018 Phase 1 drill program

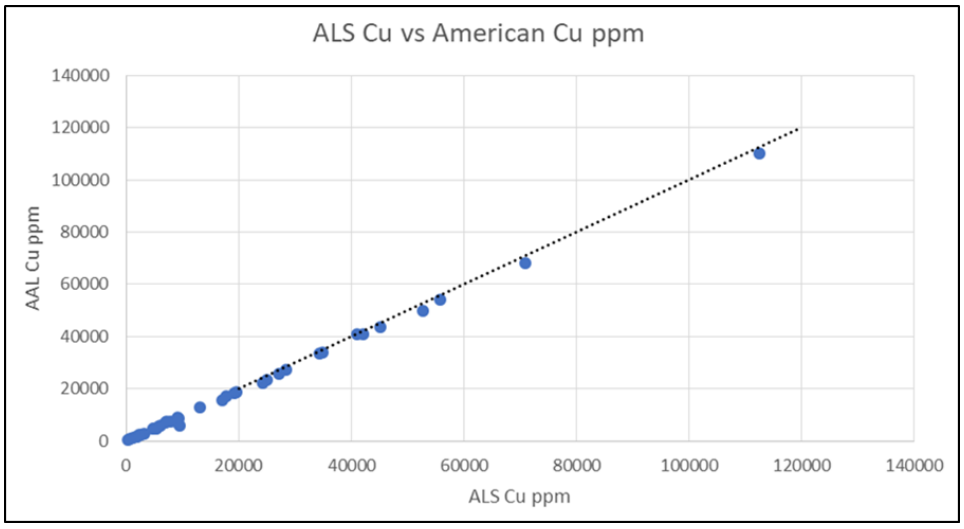

Figure 11.36 Comparison between AAL umpire Cu analyses and ALS original Cu assays for the 2018 Phase 1 drill program

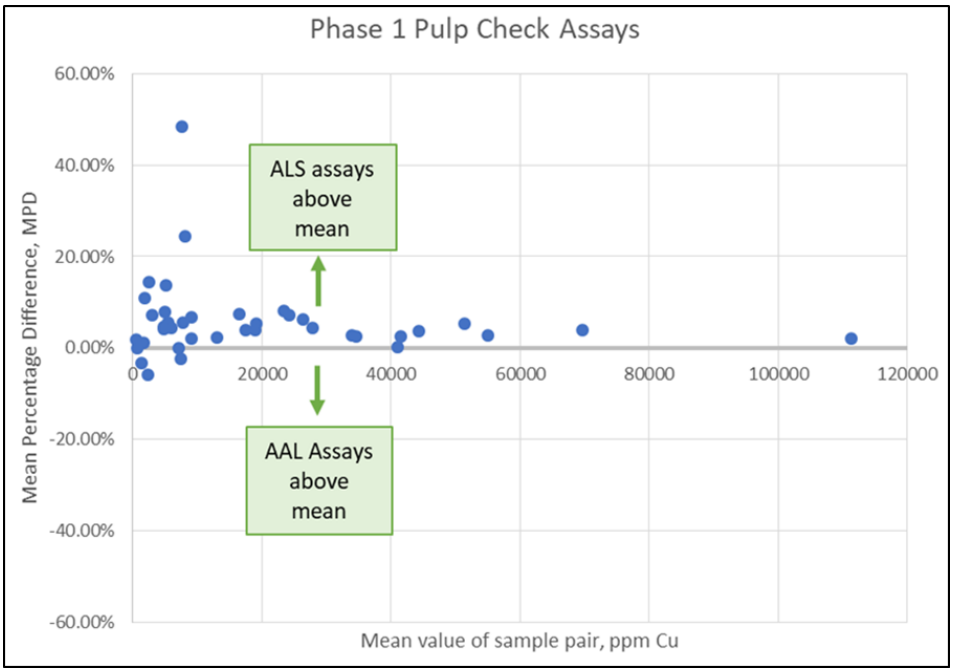

Figure 11.37 Relative error plot of AAL umpire Cu analyses and ALS original Cu analyses for the 2018 Phase 1 drill program, demonstrating positive bias in the ALS data

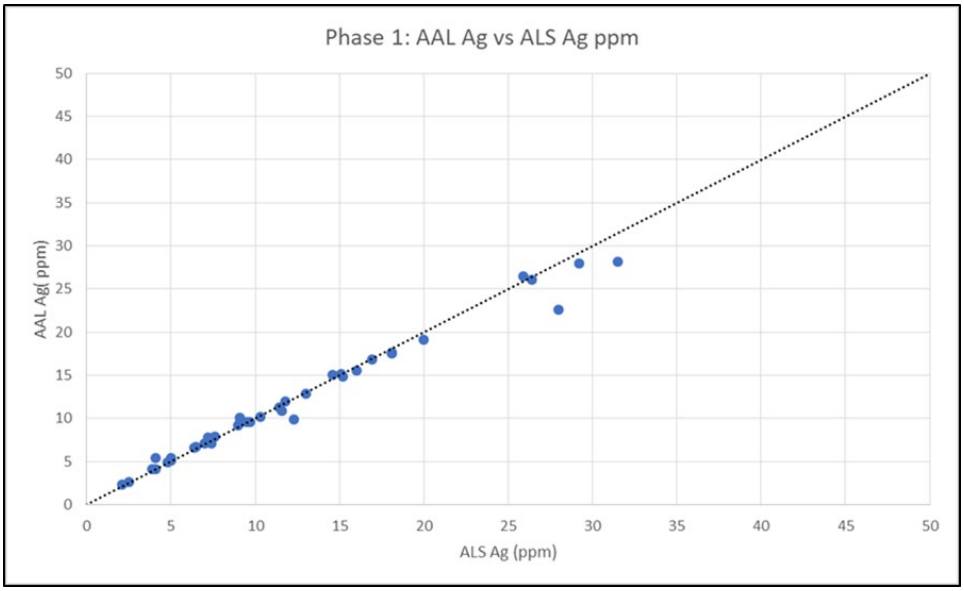

Figure 11.38 Comparison between AAL umpire Ag analyses and ALS original Ag analyses for the 2018 Phase 1 drill program

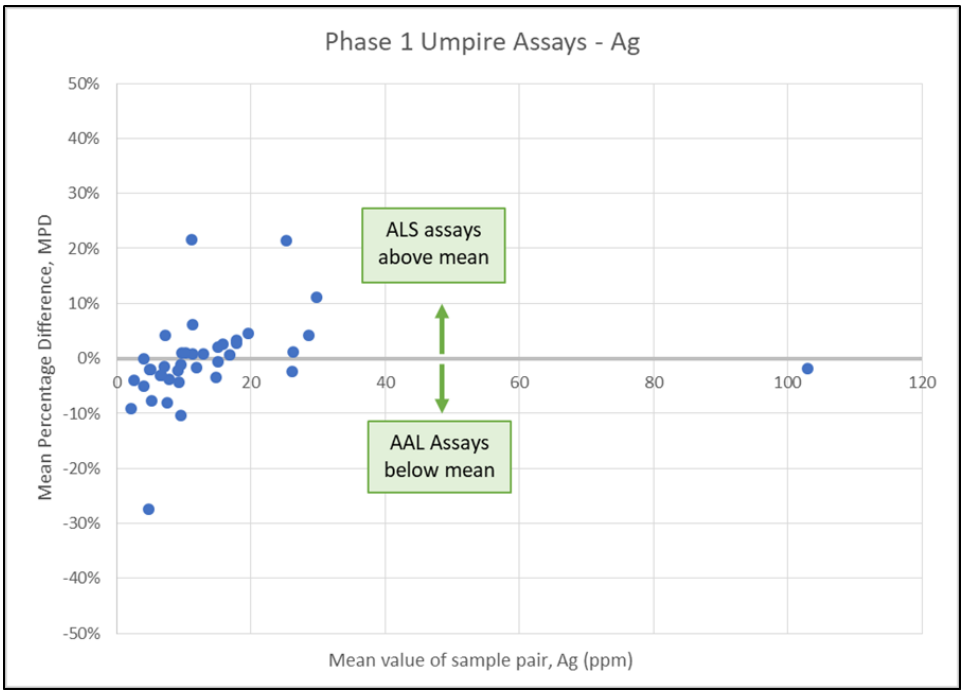

Figure 11.39 Relative error plot of AAL umpire Ag analyses and ALS original Ag analyses for the 2018 Phase 1 drill program

2018/19 Phase 2 Drill Program

No quarter-core field duplicates were inserted in the 2018/19 Phase 2 drill program.

SRA geologists selected 211 samples for laboratory duplicate analysis. This represented 14% of the original samples analyzed during the 2018/19 Phase 1 program. ALS Reno were instructed to construct duplicate pulps from a separate 1,000 g split of fine-crushed material. These duplicate pulps were transported to AAL Sparks. AAL renumbered these samples, inserted CRM's and pulp blanks as supplied by SRA geologists and resubmitted the samples to ALS Reno for analysis.

Figure 11.40 shows the laboratory duplicate Cu analyses compared with that of the original analyses. With the exception of one outlier, duplicate analyses compare favourably with original analyses with no material bias.

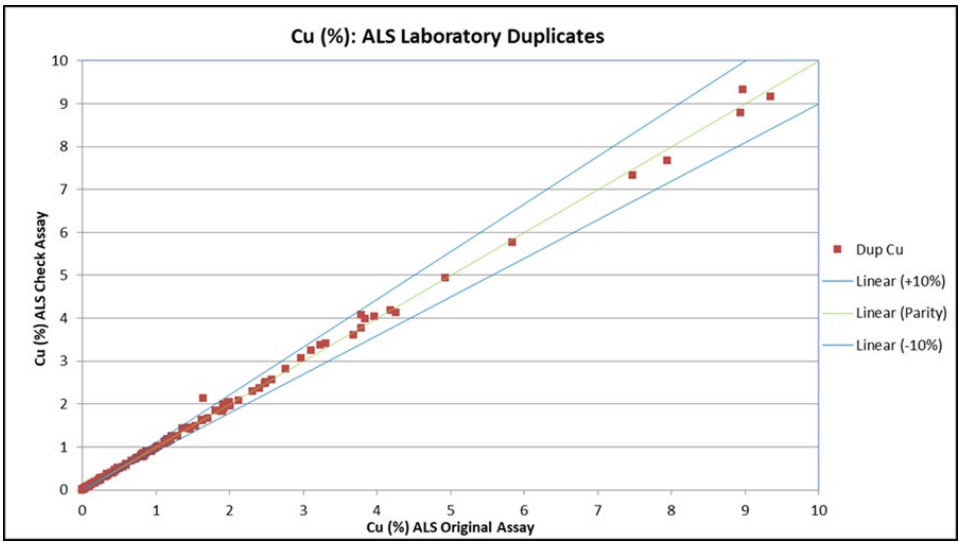

Figure 11.40 Scatter plot of laboratory duplicate Cu analyses and original Cu analyses for the 2018/19 Phase 2 drill program.

Figure 11.41 shows laboratory duplicate Ag analyses compared with original analyses. There is minor variation at low concentrations (which is considered analytical noise) but overall correlations are acceptable.

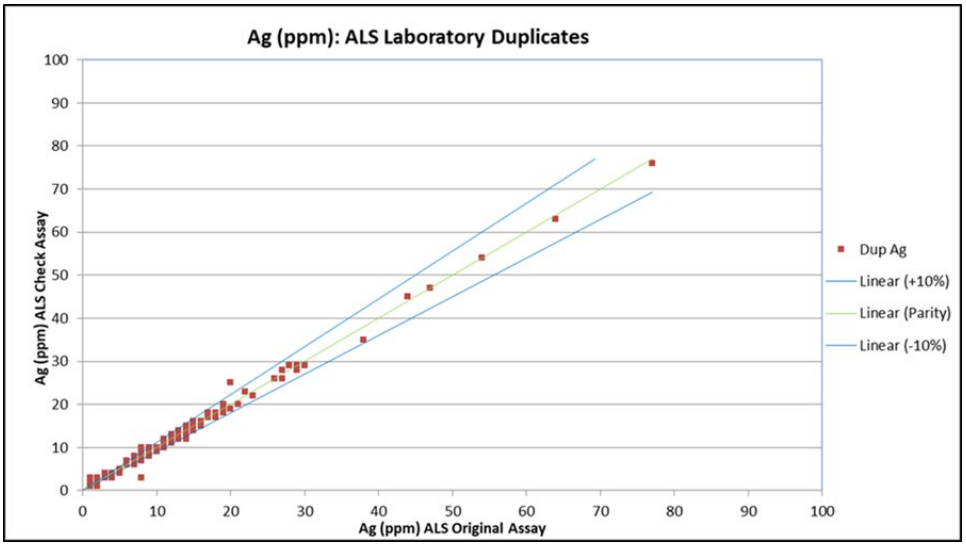

Figure 11.41 Scatter plot of laboratory duplicate Ag analyses and original Ag analyses for the 2018/19 Phase 2 drill program

SRA geologists selected 192 pulp residues from ALS (13% of the original assays) for umpire analysis at AAL using five acid digestion ICP-AES (AAL Code: ICP-5AO35) and Au by fire assay with ICP-AES finish (AAL Code: FA-Pb30). This represents 13% of the total number of original analyses.

A comparison of the Cu results is presented in Figure 11.42 and Figure 11.43. An acceptable correlation with a slight positive bias to the ALS results is evident.

A comparison of the Ag results is presented in Figure 11.44 and Figure 11.45. Whilst the correlation is relatively good there is a positive bias in the ALS results.

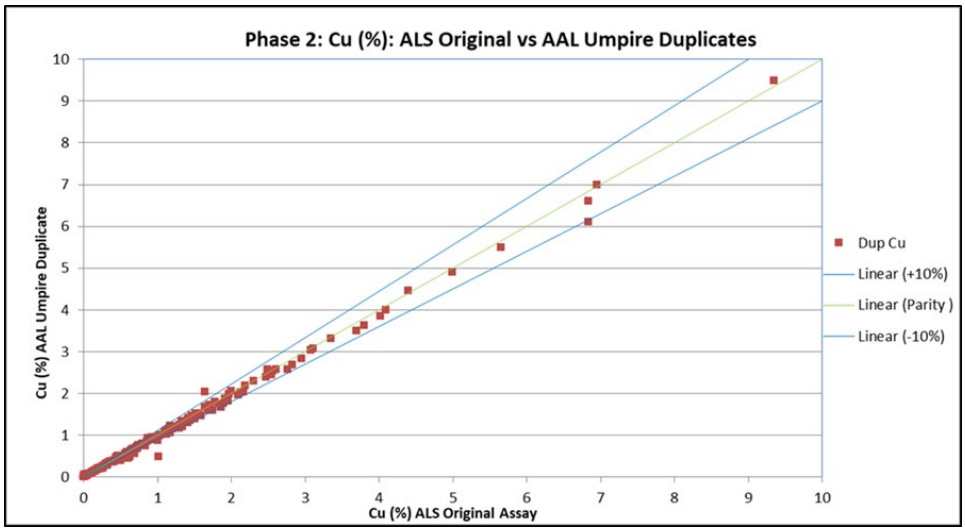

Figure 11.42 Scatter plot of umpire duplicate Cu analyses and original Cu analyses for the 2018/19 Phase 2 drill program

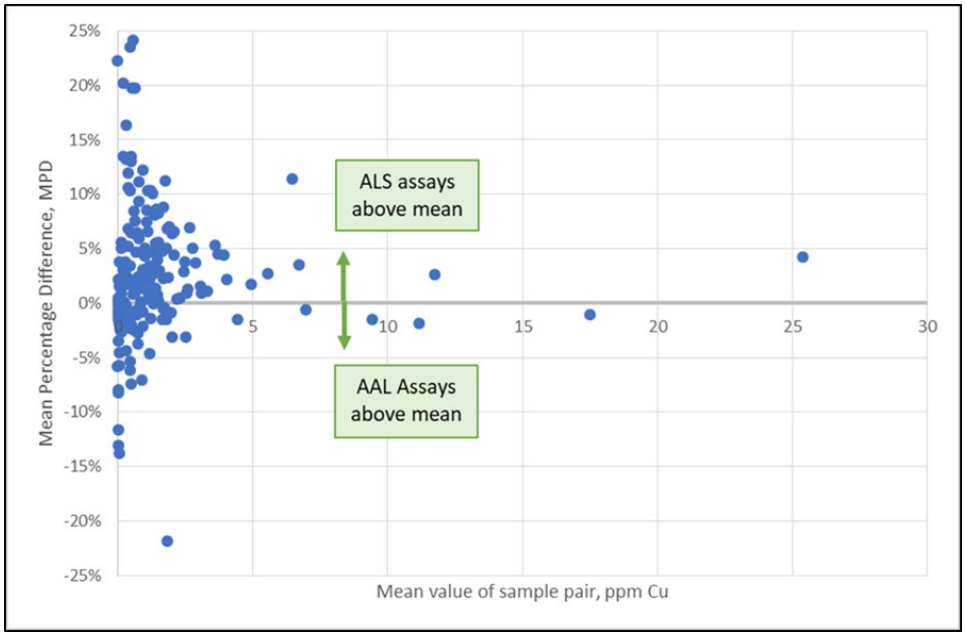

Figure 11.43 Relative error plot of umpire duplicate Cu analyses and original Cu analyses for the 2018/19 Phase 2 drill program

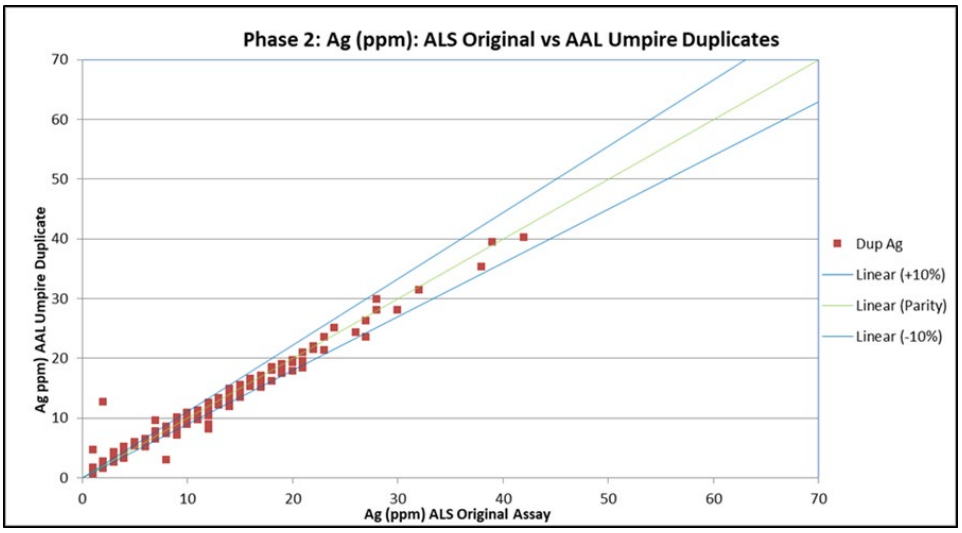

Figure 11.44 Scatter plot of umpire duplicate Ag analyses and original Ag analyses for the 2018/19 Phase 2 drill program

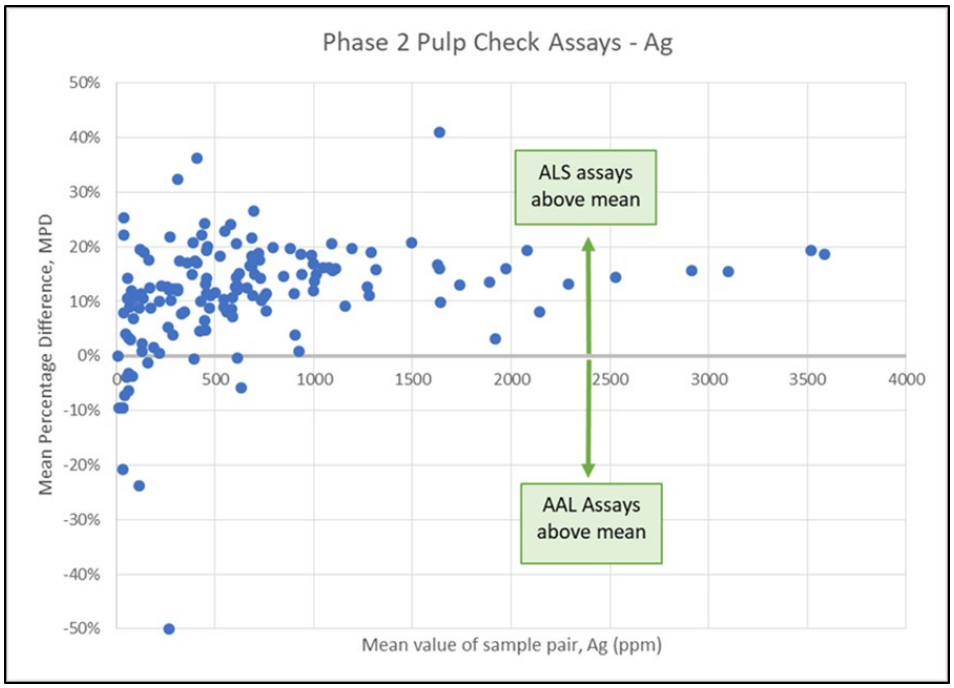

Figure 11.45 Relative error plot of umpire duplicate Ag analyses and original Ag analyses for the 2018/19 Phase 2 drill program

11.4.4 *Results and Actions*

QA/QC evaluation and actions during the 2014 to 2018/19 Phase 2 drill programs were largely conducted using Cu results from CRM's. Analyses of a CRM yielding a Cu analyses greater than two standard deviations from the certified value triggered a CRM failure.

The only CRM failures during the 2014 drill program were on CRM Ni-116 that is interpreted to have a matrix that is unsuitable for use for the analytical protocol utilized for the SRA samples. All other CRM's performed adequately.

During the 2018 Phase 1 and 2018/19 Phase 2 drill programs there were four CRM Cu failures on OREAS standards. All of these CRM's triggered reanalyses of the affected sample series with the insertion of different CRM's and pulp blanks. The reanalysed sample series passed all QA/QC criteria and were entered into the database with a Priority assigned as 0. The original analyses, that had failed QA/QC, were assigned a Priority of 1. Only analytical data with a Priority = 0 was exported for use in Mineral Resource estimation protocols.

Cu analyses of coarse blanks, constructed from landscaping marble, that were inserted in the sample series from the 2014 drill program often returned analyses that would have triggered failure (>15 times detection limit). Given the large number of coarse blanks that returned Cu values significantly above the detection limit it was concluded that the landscaping marble contained variable amounts of Cu. The use of the landscaping marble was discontinued after the 2014 drill program.

Pulp blanks were utilized during the 2018 Phase 1 drill program with all blanks performing acceptably.

Pulp blanks and commercially available coarse blanks were used during the 2018/19 Phase 2 drill program. All pulp blanks performed acceptably. Two coarse blanks exhibited minor levels of contamination (<240 ppm Cu) in the sample preparation circuit. Reanalyses of the sample series affected by these coarse blanks was not undertaken.

11.5 Opinion on Adequacy

Erik Ronald, P.Geo (APGO#3050) of SRK, acting as Qualified Person (QP) for the Geology and Mineral Resources, has reviewed the summary QA/QC program, results, and interpretation of data. It is the opinion of the QP that the quality assurance program is fit-for-purpose and suitable given the type of deposit and nature of mineralization. Quality control samples, results, and interpretation are performed to industry standard and are acceptable for the reporting of Mineral Resources. The limited QC sample failures and interpreted biases are considered immaterial.

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12. DATA VERIFICATION

12.1 Procedures

During November 2018, Erik Ronald, P.Geo (APGO# 3050) of SRK, acting as QP for Mineral Resources conducted a site visit and tour of the Project site, core shed, and facilities located in White Sulphur Springs, MT. Mr. Ronald reviewed site geology, core logging, density measurements, active diamond core drilling, core storage and handling, and discussed general items associated with the Project and future mining operations. The site visit and tour were hosted by SRA and included a team of multi-discipline consultants associated with the Project. Mr. Ronald and the site visit team were given unfettered access to all aspects of the Project and staff, with all questions satisfactorily addressed by SRA.

It is Mr. Ronald's opinion that SRA manages the geology and Mineral Resources of the Black Butte Copper Project in an organized and disciplined manner than adheres to accepted industry standards.

12.1.1 Data Management

SRA employs a Structured Query Language (SQL) database as the central data storage system using DataShed™ v4.6.3 as the front-end. User access to the database is restricted and regulated by specific user permissions. Existing protocols maximize data functionality and quality, while minimizing the likelihood of error introduction at primary data collection points and subsequent database upload, storage, and retrieval points.

Prior to 2018, the Project drill database (which includes data from the broader geological district) was managed in a SQL database that was maintained by an off-site independent consultant (Mr. J. Cote, 50914 North 292nd Ave, Phoenix, AZ). When additional data was loaded, Mr. Cote provided an updated database to SRA in Microsoft Access™ format. During 2018, this legacy database was imported into an SQL database that is hosted on the Sandfire Resources Ltd server in Perth, Australia. The Australian server performs an automatic daily export of data, in a Microsoft Access™ format, suitable for importing into Micromine™, to the SRA server in White Sulphur Springs, MT. Subsequent to 2018, the structure of the SQL database has been modified and augmented as requested by the SRA geological team.

Expedio OCRIS™ Mobile logging software was utilized by SRA geologists for the 2018 and later drilling programs to capture data including collar, survey, lithology, mineralization, structure, sample information, specific gravity, recovery, geotechnical and other meta-data. The OCRIS™ logging templates have been designed to match the tables of the SQL database and have library drop-down boxes where only pre-defined codes can be recorded that allows interactive, on-the-fly data validation. Once a hole has been validated, an .oxo file is exported and sent to Database Administrators (DBA) to load into the database.

Assay laboratory files are electronically supplied to the DBA and SRA geologists in .sif and text file format. The assay data is loaded into the database by the DBA. SRA geologists assess the QA/QC of the assay batch and decide whether it passes or fails.

The SQL server database is configured for optimal validation through constraints, library tables, triggers, and stored procedures. Data that fails these rules during import is rejected or quarantined until reviewed by a geologist.

12.1.2 Pre-SRA Johnny Lee Drill Data Verification

In 2010, SRA twinned five historic drillholes to confirm the geology, mineralization grade and thicknesses recorded in the CAI database that was obtained from the Belt Research Center at the University of Montana. The results of this drilling, as well as additional comparison between historic and SRA drillholes, allowed Winckers *et al* (2013) to conclude that the historic data, included in the CAI database, was suitable for NI 43-101 Mineral Resource estimation purposes.

As part of the 2013 PEA, Winckers *et al* (2013) checked assay certificates against database entries for 935 analyses of SRA drillhole samples used for the Mineral Resource estimate. No errors were found in the checked subset.

Winckers *et al* (2013) performed spot checks of driller's downhole surveys against the records in the database for the SRA drilling and found no errors. The QP for Mineral Resources reviewed the work performed by Winckers et al (2013) and determined the historic data verification is acceptable for the determination of Mineral Resources.

12.1.3 Pre-SRA Lowry Drill Data Verification

In 2010, SRA twinned one historic drillhole to confirm the geology, mineralization grade and thickness. The drill data from the historic drillhole (SC-80) and the twin hole (SC10-006) were reviewed based on historic paper logs, digital database, with re-logging performed in 2020 by SRA personnel. Though minor inconsistencies were present between the different generations of logging, the overall primary lithostratigraphy was sufficiently consistent to provide confidence in general logging integrity and alignment between various campaigns. Validation and re-logging data was reviewed by the QP of Mineral Resources and determined to be acceptable for use in modelling, estimation, and reporting of Mineral Resources.

The logging codes used in the CAI database (utilized for the Lowry deposit Mineral Resource estimate presented in this document) have been converted to those utilized by SRA for the Johnny Lee deposit for consistency across the property. As the data has been modified subsequent to the review by Winckers *et al* (2013), a number of additional validation steps were performed.

The data from one randomly selected drillhole from each drill program (SCC-39, SC-86, SC10-006, SC11-082 and SC12-155) representing 10% of all drillholes used to inform the Lowry deposit Mineral Resource was validated. Although some inconsistencies were noted in the historic (pre-SRA) Cu analytical data that had been captured, these inconsistencies are not considered material. All other validation checks performed on analytical data showed a high level of accuracy with no errors identified.

The original paper sampling sheets, or sample intervals on paper logs, from the drillholes were captured in a digital format and compared with the sample intervals in the database. The total number of sample intervals validated was 947 which represents 11.5% of the total sample intervals in the Lowry deposit. A single, 0.05 m data entry error was identified in one drillhole: SC10-006. This sample was from an unmineralized interval that does not inform the Mineral Resource estimate. The sample interval data is considered suitable for Mineral Resource estimation purposes.

Assay certificates are unavailable for the pre-SRA SC- and SCC- prefix drillholes that comprise 31.4% of the drillholes used to inform the Lowry deposit Mineral Resource. Historic paper copies of the analytical data, compiled by CAI, for drillholes SCC-39 and SC-86 were located. These records show that initial low-level (<2% Cu upper limit of detection) analyses by acid digest and AA Spectroscopy were conducted and analyses >0.2% were reanalysed in a high-grade AA circuit. For SCC-39 and SC-86 the analytical data from the low-level Cu analyses was digitally captured for all <2% Cu analyses and the high-grade analyses for any analyses that exceeded the 2% detection limit for the low-level Cu circuit. The Cu analytical data was then compared with that in the SRA database (which had been derived from the original CAI database).

It was noted during the comparison of the analytical data that database Cu values in the >0.2 – 2.0% Cu range had not been consistently sourced from the low-level or high-level AA circuit analytical data. For SC-86, 16 analyses (from a total of 186) had been sourced from the high-level circuit data. This results in an average Cu value for the 16 samples that is 0.03% higher than that which would be derived if the low-level Cu analyses were used consistently. For SCC-39 14 analyses (from a total of 188) were sourced from the high-level Cu circuit data. This results in an average Cu value for the 14 samples that is 0.05% higher than that which would have been derived if the low-level analyses were used consistently. Apart from inconsistency of source data no other issues or errors were identified in the analytical data from the pre-SRA drilling. All sample numbers from the paper records matched those in the database.

Analytical certificates from ALS (Reno or Vancouver) are available for all SRA drilling. Cu analyses were digitally captured from the certificates for drillholes SC10-006, SC11-082 and SC12-155 and compared against the records in the digital database. No errors or inconsistencies were observed. All sample numbers from the analytical certificates matched those in the database.

Geological logging information, from the five drillholes selected for validation, was digitally captured from paper graphic logs. Rather than capture the logging codes utilized at the time of logging (which have changed during different drill programs), the percentages of different lithotypes in each interval were recorded. This data was then compared against the current SRA database in which logging codes have been standardized.

A total of 271 logging intervals were reviewed from the five drillholes. A total of 21 inconsistencies were noted between the original paper logs and the database. Of these inconsistencies, five were in or adjacent to mineralized intervals. In two of these examples, zones of strongly fractured massive or semi-massive sulphide had been recorded in the database as high-strain zones rather than the pre-deformation lithotype. In three examples, disconnects were related to different logging resolution (minimum width of logged lithotype). As geological modelling has been completed using both core photographs and the logging records in the database the impact of these minor inconsistencies is not considered material.

12.1.4 2014 SRA Johnny Lee Drill Data Verification

In 2014, SRA completed 11 drillholes in the Johnny Lee deposit (SC14-169 to SC14-179) that are used in the Mineral Resource estimate presented in this document.

Validation of these holes was conducted in two parts: 1) SRA geologists validated the physical data utilizing an audit checklist. The audit checklist included verification of Collar Survey, Downhole Survey, Sample Locations, Lithology, Structure, and Specific Gravity. 2) SRA geologists audited the assay information to ensure source data matched with respect to sample number, sample interval, and assay certificates.

A full review of QA/QC data from 2010 to 2015 drilling was undertaken in 2018 by Sandfire Resources Ltd personnel to ensure the validity of assay data. No material issues were identified during this review.

12.1.5 2018-2019 SRA Johnny Lee Drilling Data Verification

Beginning with the 2018 Phase 1 drill program, SRA geologists validated all data within the database using a two stage process. Physical validation was completed upon finalization of drilling. Final validation was completed upon receipt of analytical results.

The first stage of the validation process involved validation of the recorded physicals using the checklist shown in Figure 12.1. The validation included checking database entries for Collar Data, Downhole Survey, Recovery, RQD, Lithology, Structure, Sample Mark-up and Specific Gravity data against the source documentation or digital files. If required, corrections were sent to the DBA for entry into the database. As a final check, the data was then reviewed in 3-D visualization software (Micromine™) to assure that it is spatially consistent with surrounding drillholes in terms of geology and mineralization positions. Once all these checks were performed, the drillhole was assigned a PV (Partial Validated) code in the Validation field of the Collar table in the database.

The second stage of the validation process was performed once assay results were received from the laboratory. During this stage, analytical data and laboratory reported sample masses were compared against sampling data and core photographs to ensure the Cu grades and sample masses were visually consistent with what had been reported from the laboratory. QA/QC of the assay data, including review of certified reference material, blanks, and duplicates was then performed. If assay data was considered acceptable, it was assigned a V (Validated) code in the Validation field of the database.

All pre-2018 data has been assigned an H (Historic) code in the Validation field of the database.

12.1.6 SRA Verification of Upper Copper Zone Specific Gravity Data

The previous Mineral Resource model for the Johnny Lee UCZ was developed for the PEA (Winckers *et al*, 2013). The PEA Mineral Resource model used the average of 181 SG measurements to assign a fixed SG of 3.99 to the rocks within the two UCZ mineralization wireframes (UCZ Wireframes 31 and 32). The average of 357 density measurements were used to assign an SG of 3.60 to laminated sulphide zones outside of UCZ Wireframes 31 and 32 but within the USZ. All SG measurements for the PEA Resource Model were taken using Archimedean / immersion method measurements on individual pieces of core (not entire core runs or resource sample intervals). These SG measurements are referred to here as POINT SG Data. From the 2018 Phase 1 drill program onwards, all SG measurements were taken using entire resource sample intervals (if sampled), on a continuous basis through the mineralized zone, or on entire core runs (if not sampled). This data is referred to here as INTERVAL SG Data.

The POINT and INTERVAL SG data in the database were reviewed and, during error checking, it was noted that there were a large number of overlapping intervals for POINT SG Data (approximately 160) and a smaller number for INTERVAL SG data (approximately 40). The INTERVAL overlaps were duplicates that were not correctly coded as such within the Duplicate Column in the database. This error was corrected. The historic POINT SG data is stored in the database as intervals (corresponding to the Resource Sample intervals they occur within) and it is uncertain whether the overlapping POINT samples are duplicates or measurements taken on a different piece of core within one Resource sample interval.

Comparison of the SG's of overlapping samples indicated relatively small differences between the measured SG's so the second data record from all the overlapping intervals was removed from the SG dataset.

The POINT and INTERVAL SG data were imported into Leapfrog Geo™ as separate interval tables. A spatial filter was applied to extract the SG data that occurred within the >1.2% Cu UCZ Eastern Block and UCZ Western Block wireframes (Eastern and Western Blocks are the two portions of the UCZ separated by Fault 1). This was done separately for both the POINT and INTERVAL data.

Descriptive statistics were calculated for the POINT and INTERVAL data for both the UCZ Eastern Block and Western Block wireframes (Table 12.1 and Table 12.2). These statistics show that the Eastern Block data for the POINT data has slightly higher values than that of the INTERVAL data. The POINT data for the Western Block shows noticeably higher values relative to the INTERVAL data.

Drill Hole Physical Data Validation		
Note that all these checks are done against data that has been exported from the Database		
Drillhole number:	SC18-	
COLLAR:		
Drillhole ID Correct and not a duplicate of an existing drillhole		
Drillhole collar Survey completed using differential GPS (DGPS) by qualified surveyor		
Collar co-ordinates in correct projection, datum, grid system		
Co-ordinates in DB match those of Surveyor DGPS report		
DGPS survey closely matches Planned Collar location and that which was recorded during Collar Pegging		
SURVEY:		
Downhole survey Azi provided in correct projection, datum, grid system		
North seeking gyroscopic downhole survey (GYRO) completed of entire drillhole (including 0m survey at collar)		
GYRO 0m survey closely matches Planned collar Azi/Dip and the Setup Azi/Dip that has been verified by Driller and Rig Geologist		
GYRO survey closely matches Single Shots (SS) taken using Electronic Survey camera (Reflex EZ-Trak or similar) during drilling. Review magnetic susceptibility readings from EZ-Trak if large difference between GYRO and SS		
GYRO has been recently calibrated and tested in test jig/test hole (recommend weekly testing)		
GYRO survey does not exceed end of hole depth in drill report		
RECOVERY:		
Geologist to check that all zones of core loss have been recorded appropriately		
RQD:		
RQD measurements of all recovered core have been completed		
Spot checking comparison RQD measurements against Core photos should be completed by Geologist		
Ensure all RQD calculations range between 0 and 100		
LOGGING:		
Ensure logging does not cross-cut zones of core loss		
Check Lithology, Sulphide and Structure logging against core photos (focus on ore zones in particular)		
SAMPLE MARK-UP:		
Ensure that samples do not cross zones of lost core		
Ensure all samples respect minimum and maximum sample widths		
Ensure that samples do not cross-cut significant lithological or ore boundaries		
Ensure that all intervals with logged Cu sulphides (with an appropriate halo buffer) have been sampled		
Ensure no duplicate sample IDs		
SPATIAL VALIDATION:		
Import the drillhole data into a 3D visualisation package (Micromine or similar)		
Does the drillhole correspond approximately with the planned collar, azi, dip and target?		
Does the drillhole trace follow a relatively smooth trend with no abrupt "jags"?		
Does the logged Lithology make sense when compared to the surrounding drillholes?		
Do the logged Copper sulphides make sense when compared with surrounding drillholes?		
Do any logged structures and bedding correspond with that of surrounding drillholes?		

Figure 12.1 Drill hole physical data validation form

Data Type	Count	Mean	Std. Dev	Coeff. Var.	Variance	Min	Lower Quartile	Median	Upper Quartile	Max
INTERVAL	209	3.96	0.34	0.09	0.12	2.96	3.79	4.03	4.21	4.60
POINT	169	4.05	0.33	0.08	0.11	2.90	3.97	4.24	4.27	4.69

Table 12.1 Statistics for UCZ East Block >1.2% Cu wireframes, SG data

Data Type	Count	Mean	Std. Dev	Coeff. Var.	Variance	Min	Lower Quartile	Median	Upper Quartile	Max
INTERVAL	160	3.58	0.33	0.09	0.11	2.52	3.36	3.56	3.82	4.31
POINT	104	3.82	0.33	0.09	0.11	3.01	3.57	3.81	4.10	4.34

Table 12.2 Statistics for UCZ West Block > 1.2% Cu wireframes, SG data

In order to evaluate the SG data spatially, the POINT and INTERVAL data were plotted separately (Figure 12.2). The POINT data was collected from a large number of drillholes over several drilling programs prior to 2018 and thus has a comprehensive spatial coverage of the entire UCZ. The INTERVAL data was collected only during the 2018 drilling, which was focused on certain parts of the UCZ, so while locally the data may be of better quality, the spatial coverage is less robust than that of the POINT data.

A significant proportion of the Western Block INTERVAL data occurs within the northern portion of the Western Block (characterized by lower SG's than the southern portion). Consequently, the statistically higher SG values from the POINT data in the Western Block are consistent with the spatial distribution of data from the two datasets. The Eastern Block INTERVAL data provides more spatially comprehensive coverage of the Eastern Block (with corresponding greater statistical similarity between the POINT and INTERVAL data).

Spatial comparison between SG values from POINT and INTERVAL data from similar areas within the UCZ from both Eastern and Western Blocks show similar values. It is concluded that the SG data for the POINT and INTERVAL data, whilst not directly comparable in sampling density and methodology, are best utilized together to better constrain the highly variable nature of the UCZ density for Mineral Resource estimation purposes.

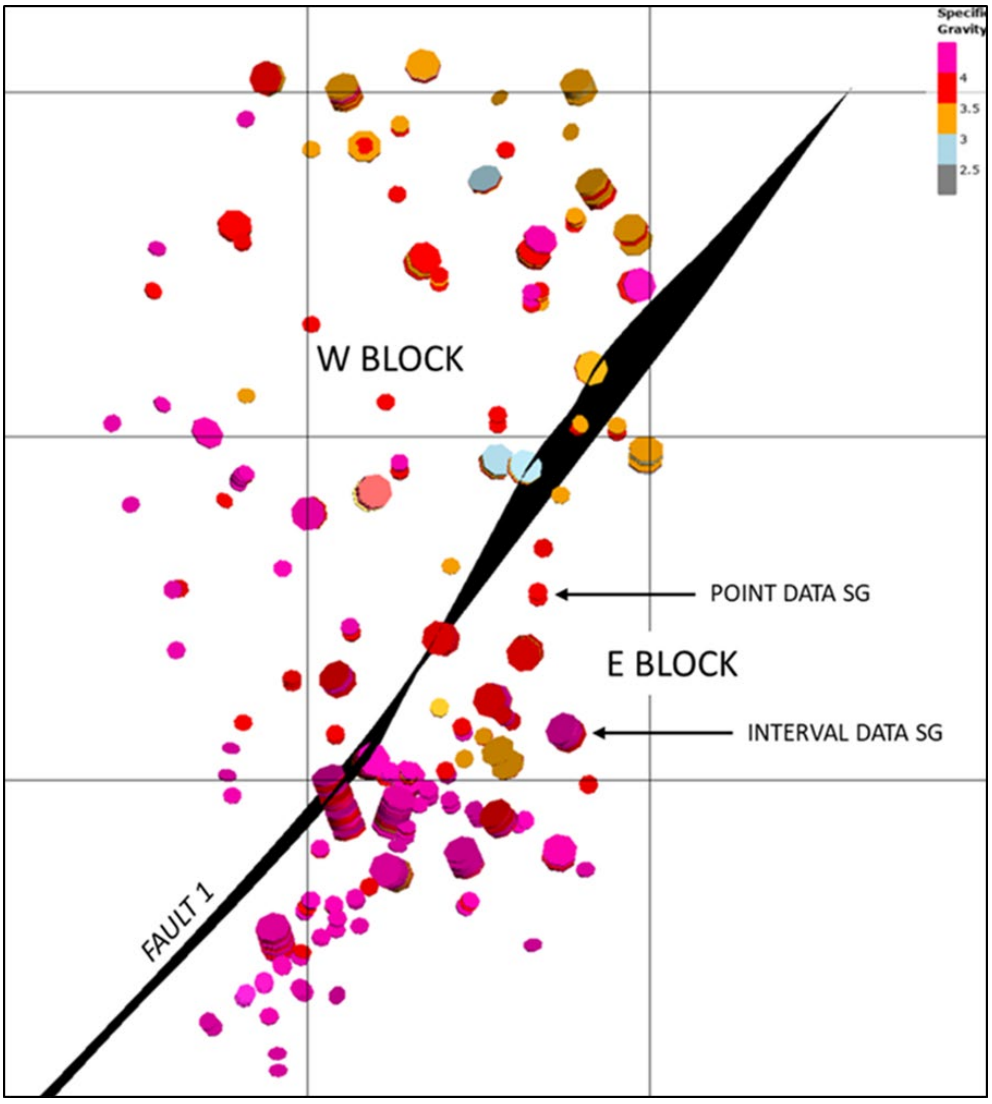

Figure 12.2 Plan view of the UCZ showing SG data for POINT and INTERVAL samples

12.1.7 Data verification by QP

SRA provided SRK with a drilling database of exploration drilling focused on the Johnny Lee and Lowry deposits. SRK was provided with Collar, Survey, Geology, Assay, Recovery and RQD, Density, and Structure tables as comma-separated value (.csv) files. Data associated with adjacent properties or other prospects were not provided nor reviewed by SRK.

SRK performed a series of visual, statistical, and software validation checks on the drilling database using Geovariances Isatis™, Seequent Leapfrog Geo™, Maptek Vulcan™, Phinar X10-Geo™ and Microsoft Excel™ software packages. Visual and statistical inspection of drilling data resulted in no material errors or biases observed in the provided database. Automated software data checks from Vulcan™ and Leapfrog Geo™ were used as an additional level of validation resulting in no material errors.

As part of the QP site visit, Mr. Ronald observed drilling activities, sampling, logging, cutting, and core photography activities. Each step in the observed process followed established procedures and was considered good industry practice in data collection and management.

12.2 Limitations

SRK notes no data limitations.

12.3 Opinion on Adequacy

It is the opinion of the QP, that the data verification checks performed internally by SRA staff in combination with external and independent checks by SRK, have resulted in sufficient validation of the fundamental drilling database at the Black Butte Copper Project, and that the data used in the modelling of the geology and calculation of the Mineral Resources is satisfactory.

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13. MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Introduction

Test work has been carried out since 2011 for the proposed Black Butte Copper Project involving the processing of ore mined from underground in a conventional copper sulphide flotation plant. Composites of drill core were included in the test programs. The programs completed were:

- 2012 PEA study comprised of five composites and a master Over-All composite (OA) of Upper Copper Zone (UCZ) and a master overall composite of the Lower Copper Zone (LCZ) from the Johnny Lee deposit. Tests included optical and QEMSCAN mineralogical analysis, comminution tests, batch flotation tests and a single LCT. The purpose of the program was to establish the basic flowsheet and examine the effects of different conditions and parameters. Figure 13.1 displays the associated drillholes used in the inspectorate program with the following symbol. ● *(red circle – black dot);*
- 2016 PFS test program comprised of 18 variability samples from the UCZ and eight variability samples from the LCZ. From the variability samples, four development composites were generated for the UCZ (Dev-Comp 1 to 4) and one development composite for the LCZ (Dev-Comp 5) covering zones from the North to the South of the Johnny Lee deposit. The purpose of the program was to further develop and optimize the flowsheet development from the previous program against the mineralogical variations across the Johnny Lee deposit. Figure 13.1 displays the associated drillholes used in the SGS program with the following symbol. ◉ *(white circle – black dot);*
- 2018 FS test program focused on optimizing the previous flowsheet development and reagent schemes with an additional 21 variability composites. These were aligned with the previous development composites (Dev Comp 1 to 4) with additional samples between the tested holes. The test work specifically targeted the liberation characteristics of the deposit with the emphasis on improvements to the flotation grade and recovery relationship from the previous programs. An extensive LCT program provided the optimal flowsheet design for the expected variability in flotation performance based on the mineralogical variability from the North to South zones within the deposit. Figure 13.2 displays the associated drillholes/composites used in the BML program. (BBFT-01 to BBFT-21).

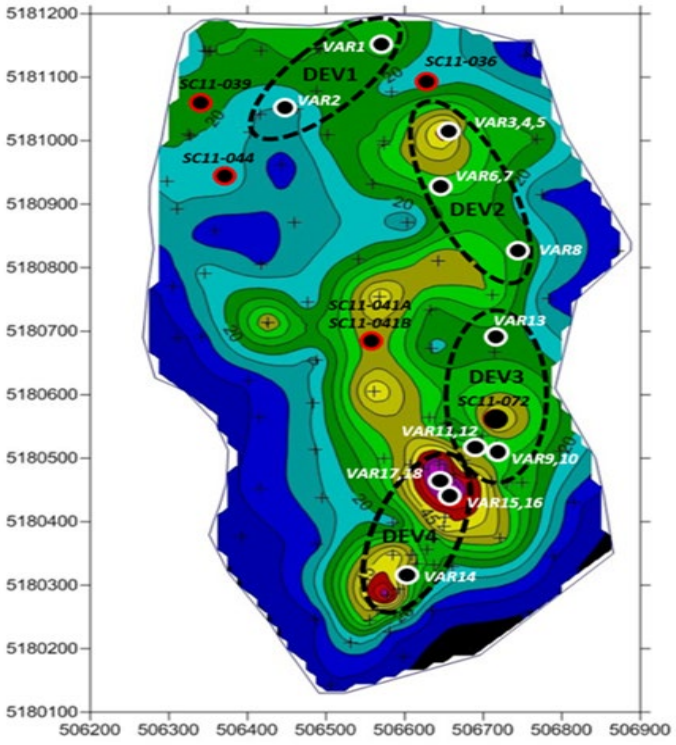


Figure 13.1 PEA And PFS Sample Location Map

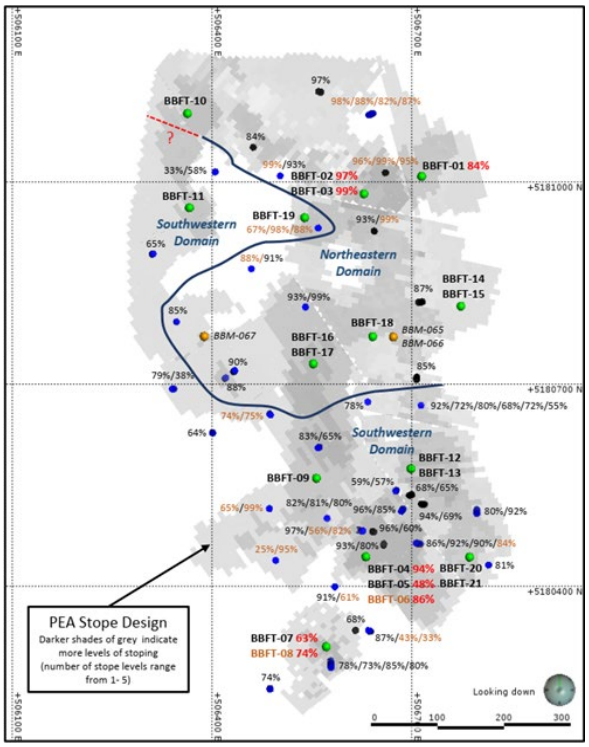


Figure 13.2 FS Sample Location Map

13.2 Historical Metallurgical Test Work Programs

13.2.1 2012 Preliminary Economic Assessment Test Work

Inspectorate Exploration and Mining Services Ltd, Metallurgical Division conducted a testing program for the 2012 PEA study (Shi and Redfearn, 2012 and 2012a). The PEA tested five composites from the UCZ, one overall composite from the UCZ (UCZ-OA) and one overall composite from the LCZ (LCA-OA). The test work included comminution testing, mineralogical studies, and LCT's.

Comminution tests reported a Bond Rod Mill Work Index (BRWi) of 15 kWh/t using an 1,190 µm closing screen and a Bond Ball Mill Work index (BBWi) of 14.8 kWh/t using a 53 µm closing screen on samples from the UCZ. LCZ tests reported BRWi of 12.6 kWh/t using a 1,190 µm closing screen and BBWi of 11.8 kWh/t using a coarser 75 µm closing screen. The abrasion index was only determined for the UCZ and returned an Ai figure of 0.6885 g.

No SAG Mill Comminution (SMC) testing was conducted during the 2012 PEA test work program.

Following initial batch rougher and cleaner optimization for the UCZ, the flowsheet selected utilized a primary grind of 38 µm K_{80} and a rougher concentrate regrind of 8 µm K_{80}. Lime was added to suppress pyrite. SIPX and Cytec 3418A were used as collectors and MIBC as the frother.

Subsequently, a single LCT cycle test was conducted on the UCZ-OA composite using the developed flowsheet from the initial test runs. Results demonstrated Cu recoveries of 82.2% at an average Cu concentrate grade of 21.7%. The flowsheet for the PEA LCT is shown in Figure 13.3 with overall results reported in Table 13.1.

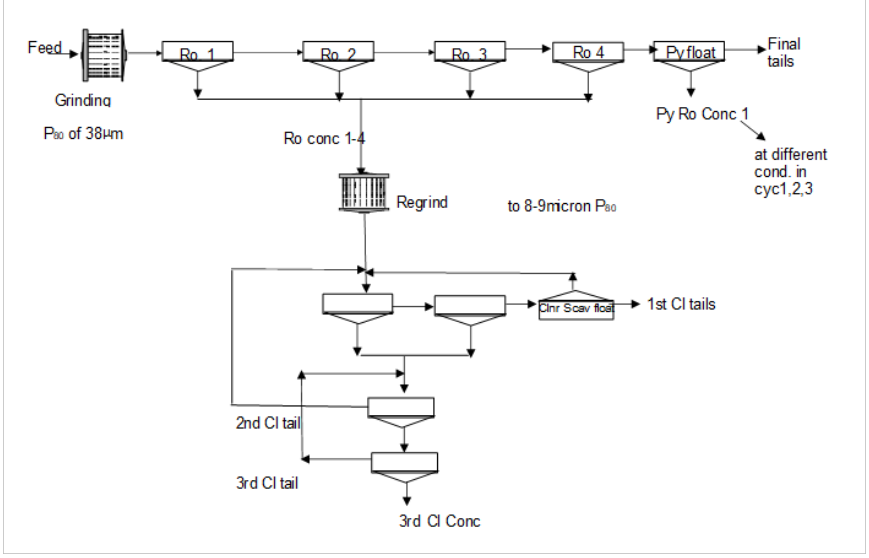


Figure 13.3 LCT PEA Flowsheet

Product	Weight	Assay			Distribution		
		Ag	Cu	Co	Ag	Cu	Co
	(%)	(g/t)	(%)	(%)	(%)	(%)	(%)
Third Cleaner Concentrate	6.9	16.8	**25.80**	0.14	7.7	**79.2**	6.4
Second Cleaner Concentrate	8.5	15.4	21.70	0.15	9.0	82.2	8.2
First Cleaner Scavenger	10.4	23.2	1.10	0.24	16.2	5.0	16.7
Rougher Concentrate	18.9	19.9	10.30	0.20	25.2	87.2	24.8
Total Final Tails	81.1	13.8	0.35	0.14	74.8	12.8	75.2
Calc Head	100.0	14.9	2.24	0.15	100.0	100.0	100.0

Table 13.1 LCT's (Cycle 4 to 6) Projected Metallurgy

Batch open cleaner variability tests were conducted on four of the five drill core used to construct the UCZ-OA composite. Cu recoveries from these variability tests ranged from 71.6% to 79.5%, with concentrate grades ranging from 12.9% to 19.7% Cu. Although the batch variability tests are not directly comparable to the LCT, they demonstrated the amount of variability in flotation response from samples sourced in the northern part of the deposit to those in the central part (SC11-041 generated the 12.9% Cu grade). This concern was emphasized by subsequent batch open cleaner testing on a sample from drill core in the southern part of the UCZ (SC11-072) that returned a Cu recovery of 66.7% at a concentrate grade of 10.2% Cu.

Mineralogy conducted on the UCZ-OA to examine the locking characteristics was undertaken as part of the assessment. The mineral fragmentation characteristics of the UCZ-OA composite were estimated at a sizing of P_{80} of 62 µm. The effect of the primary grind sizing on mineral liberation and the limiting grade recovery for Cu data is presented in Figure 13.4 and Figure 13.5 with the emphasis on a finer primary grind required to provide the necessary liberation of the copper sulphide.

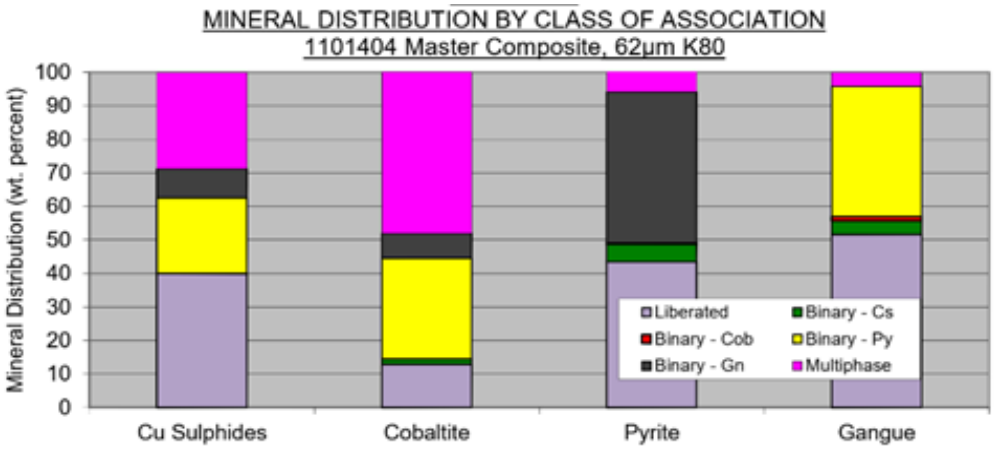


Figure 13.4 Mineral Distribution

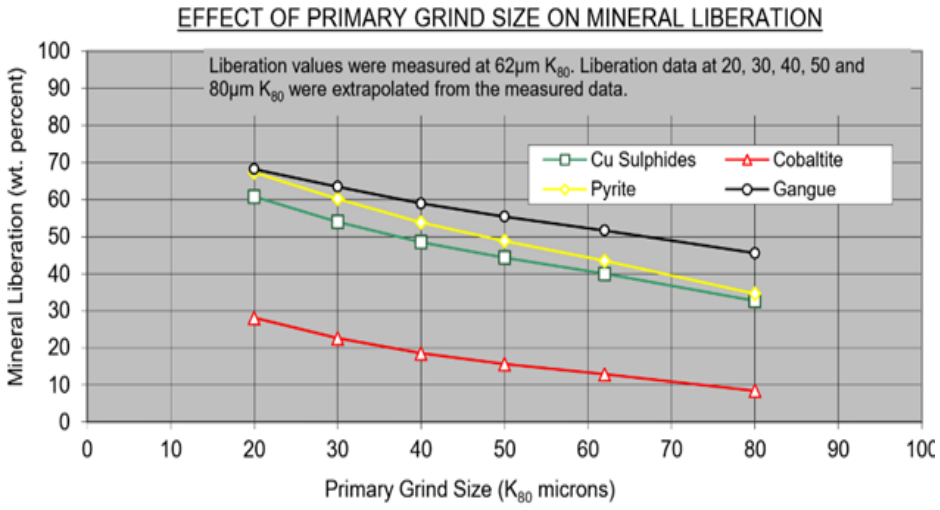


Figure 13.5 Effect of Primary Grind Size on Mineral Liberation

Figure 13.5 also indicates a significant amount of concentrate regrinding is required to liberate the copper sulphide minerals and generate a saleable concentrate from the UCZ ores.

Batch flotation testing conducted on the LCZ indicated excellent flotation response at much coarser grind size of 100 µm K_{80} generating a rougher recovery of 97.6% Cu, which was subsequently increased to 98.4% Cu recovery at a 53 µm K_{80} primary grind. Mineralogical analysis of the LCZ composite indicated that 88% of the chalcopyrite is liberated at a particle size passing 53 µm.

The LCT's of the LCZ-OA composite shown in Figure 13.6 followed a similar flowsheet to the UCZ, with the exception being the removal of the third cleaning stage and the pyrite flotation step.

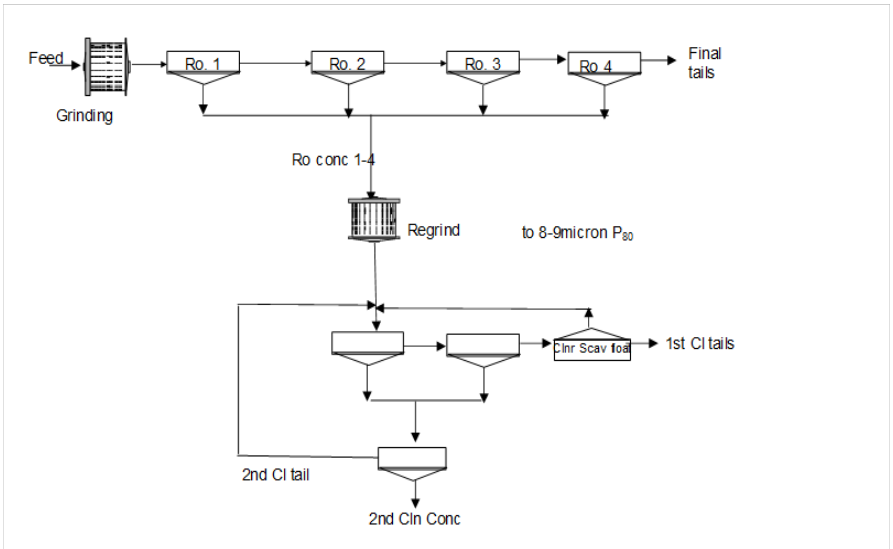


Figure 13.6 LCT Flowsheet for LCZ

The primary grind size was maintained at P_{80} of 38 µm as per UCZ LCT with results summarized in Table 13.2. The results for the LCZ indicate a significant difference in metallurgical response between the lower and upper zones. The LCZ provides a higher recovery with a cleaner concentrate.

Product	Weight		Assay			Distribution		
		Cu	Fe	%S	%C	Cu	Fe	%S
	(%)	(%)	(%)	(%)	(%)	(%)	(%)	(%)
Second Cleaner Concentrate	14.3	**26.95**	33.06	35.1	0.08	**96.6**	19.4	20.2
First Cleaner Scavenger Tails	7.6	0.63	29.81	30.8	0.18	1.2	9.3	9.4
Final Tails	78.1	0.11	22.16	22.4	0.07	2.2	71.3	70.4
Calc Head	100.0	3.98	24.30	24.9	0.08	100.0	100.0	100.0

Table 13.2 LCT Results – LCZ Composite

13.2.2 2016 PFS Test Work

SGS Mineral Services conducted a testing program consisting of batch flotation tests, mineralogical studies, and final flow sheet locked cycle testing as part of an unpublished 2016 PFS level study.

Development Composites						
			Upper Zone			Lower Zone
Element	Unit	Dev.1	Dev.2	Dev.3	Dev.4	Dev.5
Au	**g/t**	<0.01	<0.01	<0.01	<0.01	0.37
Ag	**g/t**	11.5	8.50	11.6	12.6	4.30
Cu	%	1.88	2.51	2.02	3.05	5.37
Fe	%	23.6	23.8	21.8	26.2	21.7
Co	%	0.16	0.14	0.09	0.11	0.04
S	%	26.4	27.0	29.0	33.9	23.9

Table 13.3 Head Assays for Development Composites (Dev Comp)

Five main development composites were prepared consisting of four composites for the UCZ and one composite for the LCZ. The composites examined the northern part of the UCZ covered in the Inspectorate PEA test work program using Development Composites (Dev Comp) 1 and 2, as well as an examination of the southern part of the orebody covered by Dev Comps 3 and 4, refer to Figure 13.7 for sample locations. The head analyses of the composites are given in Table 13.3.

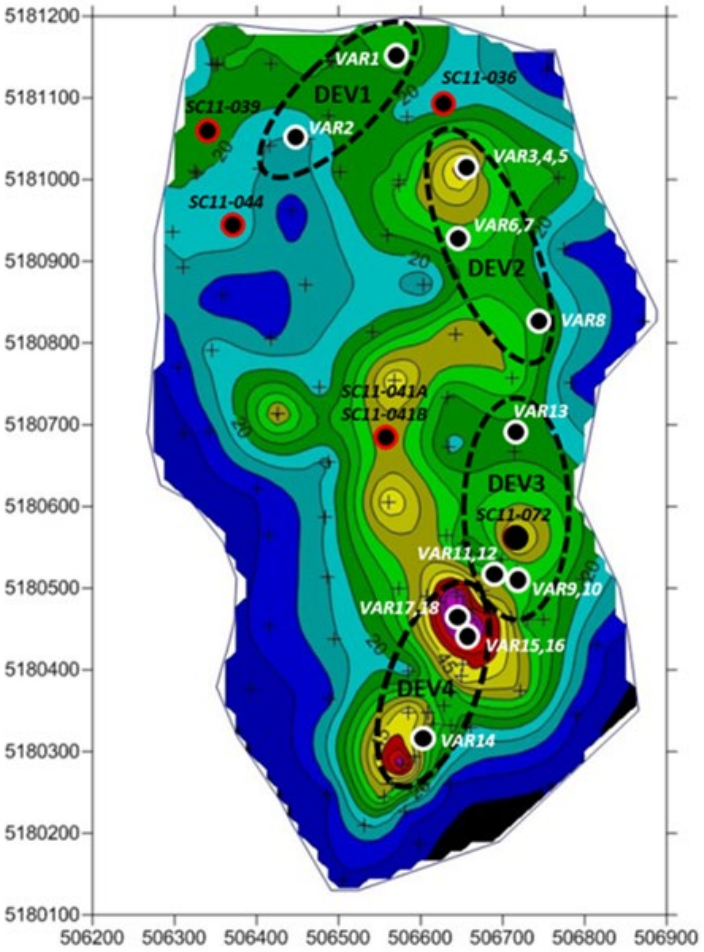


Figure 13.7 Development Composite Location Map

Following extensive rougher and cleaner batch test work, the final flowsheet for the locked cycle testing utilized a 30 µm P_{80} primary grind and a 10 µm P_{80} regrind of the rougher concentrate. The reagent suite was similar to that adopted in the 2012 PEA test work program, i.e. Lime, SIPX, Cyctec 3418A and MIBC. Sodium Cyanide (NaCN) was also used in some of the LCT's, as reagent was used within the optimization stages, but will not be available for use within Montana.

The LCT's conducted on UCZ development composites used two flowsheet options. These are shown in Figure 13.8.

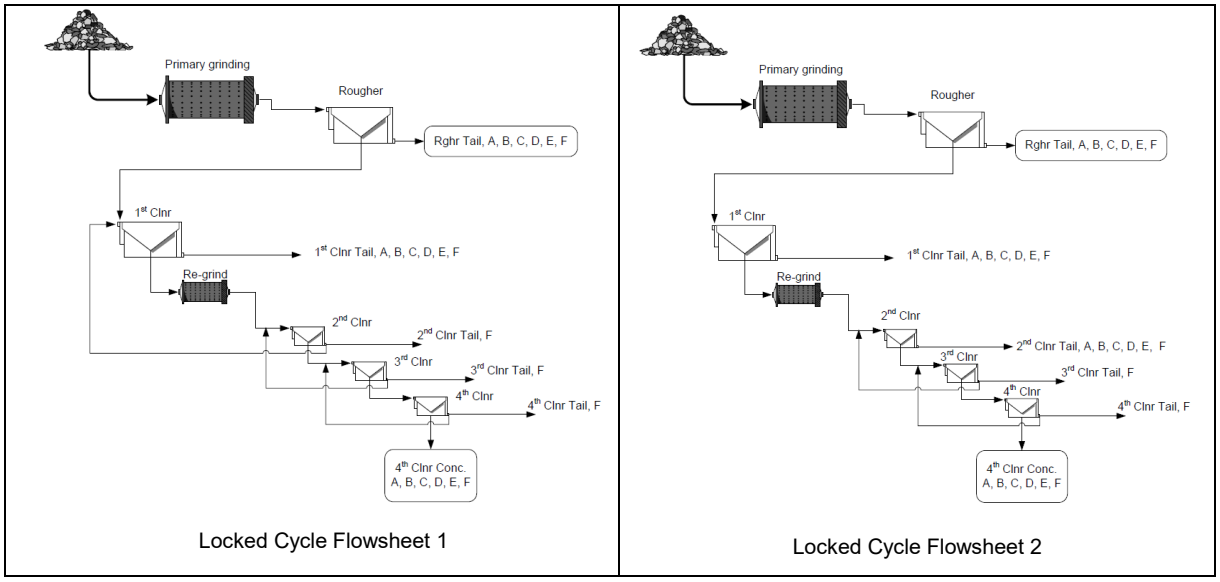


Figure 13.8 UCZ Development Composite LCT Flowsheets

Results from the LCT's showed considerable variation in both Cu grade and recovery to concentrate as reported in 0. Excluding the tests using NaCN addition to the cleaner circuit, the DEV1 – DEV4 LCT's showed Cu recoveries ranging from 61.9% to 91.2% at Cu concentrate grades of 18.5% to 24.5%. It is important to note the recoveries in the south of the orebody were significantly lower than those generated in the north of the orebody.

Test ID	Product	Weight	Grade %				Distribution, %			
		%	Cu	Fe	S	As	Cu	Fe	S	As
Dev 1-LCT1 FS1	**4th Clnr Conc.**	**6.19**	**24.5**	**33.2**	**37.7**		**84.1**	**8.5**	**8.8**	
	Head (calculated)		1.80	24.1	26.7		100	100	100	
Dev 1-LCT2 FS2	**4th Clnr Conc.**	**8.97**	**20.6**	**35.8**	**40.4**	**0.534**	**88.2**	**12.8**	**13.2**	-
	Head (calculated)		2.09	25.1	27.4	0.246	100	100	100	
Dev 2-LCT1 FS2	**4th Clnr Conc.**	**9.5**	**23.7**	**32.6**	**38.6**	**0.919**	**91.2**	**13.2**	**13.7**	
	Head (calculated)		2.45	23.3	26.60	0.28	100	100	100	
Dev 2-LCT2 FS2	**4th Clnr Conc.**	**8.8**	**24.1**	**32.7**	**37.7**		**86.7**	**12.4**	**12.8**	
	Head (calculated)		2.45	23.1	26.3		100	100	100	
Dev 2-LCT3 FS2	**4th Clnr Conc.**	**8.5**	**24.5**	**32.6**	**37.3**	**0.426**	**86.1**	**12**	**12.1**	**15.7**
	Head (calculated)		2.40	22.90	26.20	0.23	100	100	100	100
Dev 3-LCT1 FS2	**4th Clnr Conc.**	**8.4**	**18.5**	**33.1**	**39.9**		**76.5**	**12.7**	**11.8**	
	Head (calculated)		2.03	21.8	28.4		100	100	100	
Dev 3-LCT2 FS2	**4th Clnr Conc.**	**6.4**	**22.4**	**31.6**	**38.9**	**1.495**	**70.4**	**9**	**8.7**	**54.2**
	Head (calculated)		2.04	22.30	28.70	0.18	100	100	100	100
Dev 3-LCT3 FS2	**4th Clnr Conc.**	**6.1**	**23.1**	**32**	**38.8**	**1.455**	**70.3**	**8.9**	**8.3**	**49.8**
	Head (calculated)		2.01	22.00	28.50	0.18	100	100	100	100
Dev 3-LCT4 FS2	**4th Clnr Conc.**	**10.1**	**15.9**	**35.2**	**42.1**	**1.09**	**78.7**	**16.2**	**14.8**	**62.2**
	Head (calculated)		2.04	21.9	28.7	0.18	100	100	100	100
Dev 4-LCT1 FS2	**4th Clnr Conc.**	**8.3**	**22.7**	**31.9**	**39.1**	**1.533**	**61.9**	**10.1**	**9.8**	**53.2**
	Head (calculated)		3.05	26.2	33.3	0.24	100	100	100	100
Dev 4-LCT2 FS1	**4th Clnr Conc.**	**10.3**	**19.7**	**34.6**	**40.3**	**1.052**	**68.9**	**13.7**	**12.8**	**51.8**
	Head (calculated)		2.96	26.2	32.6	0.21	100	100	100	100
Dev 4-LCT3 FS1	**4th Clnr Conc.**	**9.7**	**22.8**	**33.2**	**40.1**	**1.315**	**73**	**12.4**	**11.6**	**53.1**
	Head (calculated)		3.04	26.0	33.6	0.24	100	100	100	100

Note: *Dev3-LCT3, Dev3-LCT4 and Dev4-LCT3 used NaCN in the cleaner stage.*

Table 13.4 Dev Comps 1 to 4 - LCT Summary

Detailed mineralogical investigations were conducted by SGS to determine the mineralogy of composite feed samples and flotation products using QEMSCAN. The study concluded that the upper zone material is fine-grained, massive sulphide, while the lower zone material is coarse-grained stock work mineralization.

Liberation of the chalcopyrite varies significantly across the composites, ranging from 59.7% to 76.3% liberated chalcopyrite at P_{90} 75 µm. The finer grind to P_{90} 38 µm produced a liberation range of 72.9% to 83.9%, with most of the locking mechanisms being chalcopyrite/pyrite and chalcopyrite/pyrite/gangue in the binary and tertiary composites (Table 13.5).

Mineral Name	DEV.COMP 1	DEV.COMP 1	DEV.COMP 2	DEV.COMP 2	DEV.COMP 3	DEV.COMP 3	DEV.COMP 4	DEV.COMP 4
	A	B	A	B	A	B	A	B
	90% 75 µm	90% 38 µm	90% 75 µm	90% 38 µm	90% 75 µm	90% 38 µm	90% 75 µm	90% 38 µm
Pure Chalcopyrite	54.2	67.0	58.8	70.7	56.4	63.1	42.6	56.3
Free Chalcopyrite	9.86	9.34	8.92	4.59	0.89	1.60	7.02	10.7
Lib Chalcopyrite	10.5	7.41	8.56	8.57	7.62	8.23	10.1	7.10
Chalcopyrite Pyrite	8.31	4.37	9.93	4.64	8.86	9.38	16.0	13.5
Chalcopyrite Tennantite	0.01	0.23	0.11	0.00	0.03	0.09	0.26	0.48
Chalcopyrite Cobaltite/Carrolite	0.00	0.00	0.00	0.00	0.01	0.00	0.00	1.00
Chalcopyrite: Other Cu Sulphides	0.57	0.70	0.33	0.18	0.15	0.32	0.53	0.34
Chalcopyrite Other Sulphides	0.00	0.00	0.01	0.00	0.00	0.00	0.00	0.00
Chalcopyrite: Quartz/Feldspars	5.36	3.75	4.40	5.90	5.07	4.73	1.54	1.89
Chalcopyrite: Barite	0.03	0.01	0.01	0.01	1.60	0.91	0.51	0.88
Chalcopyrite: Pyrite: Qtz/Feld	8.31	5.79	7.19	3.68	11.0	5.90	8.14	4.60
Chalcopyrite Pyrite: Tennantite	0.46	0.20	0.25	0.12	0.63	0.31	3.90	0.59
Complex	2.45	1.24	1.49	1.57	7.73	5.48	9.37	3.72
Total	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0
Liberated	**74.5**	**83.7**	**76.3**	**83.9**	**64.9**	**72.9**	**59.7**	**74.0**

Table 13.5 Chalcopyrite Liberation and Association (Dev 1-4)

There is a greater extent of chalcopyrite associated with pyrite in Dev Comps 3 and 4, and especially composite 4, than the other two composites. By looking at the association data, there is also more in the ternary particles and complex group than composites 1 and 2. Consequently, fine or ultra-fine grinding would be required to completely liberate the minerals.

Liberation of tennantite also varied across the development composites. At the grind size of P_{90} 75 µm, 32.9% to 63.3% of the tennantite was liberated. At the finer grind size of P_{90} 38 µm, 43.7% to 76.0% of the tennantite was liberated. As tennantite ($Cu_{12}As_4S_{13}$) contains As, the concentrates will have to be monitored for As concentration, as the ability to reject or depress the tennantite will be difficult with the chalcopyrite float and mineral association. The liberation and mineral association are reported in Table 13.6.

Mineral Name	DEV.COMP 1 A 90% 75 µm	DEV.COMP 1 B 90% 38 µm	DEV.COMP 2 A 90% 75 µm	DEV.COMP 2 B 90% 38 µm	DEV.COMP 3 A 90% 75 µm	DEV.COMP 3 B 90% 38 µm	DEV.COMP 4 A 90% 75 µm	DEV.COMP 4 B 90% 38 µm
Pure Tennantite	54.0	53.2	44.3	67.4	40.5	35.9	26.6	37.5
Free Tennantite	0.00	0.00	2.11	0.00	2.53	7.07	0.00	4.79
Lib Tennantite	9.28	10.6	10.8	8.65	1.85	14.0	6.30	1.39
Tennantite Pyrite	3.27	10.9	10.8	8.73	6.27	14.8	12.6	15.5
Tennantite Chalcopyrite	0.42	3.17	1.51	0.00	0.27	3.26	0.59	8.00
Tennantite Cobaltite/Carrolite	0.00	0.00	0.00	0.00	0.60	0.00	0.00	2.03
Tennantite: Other Cu Sulphides	1.42	0.00	0.93	0.55	0.31	0.00	0.54	1.68
Tennantite Other Sulphides	0.21	0.00	0.05	0.00	0.00	0.00	0.13	0.00
Tennantite Pyrite: Qtz/Feld	1.89	12.2	5.29	5.69	6.02	5.08	1.57	2.90
Tennantite Pyrite:Chalcopyrite	21.9	2.47	5.10	3.30	7.75	3.63	33.8	6.90
Complex	7.57	7.40	19.1	5.69	33.9	16.3	17.9	19.3
Total	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0
Liberated	63.3	63.9	57.2	76.0	44.8	56.9	32.9	43.7

Table 13.6 Tennantite Liberation and Association (Dev 1-4)

While the tennantite is between 0.24% to 1.07% of the sulphide distribution, it accounts for 15.9% to 17.2% of the Cu deportment and 59% to 62% of the As deportment in Dev Comps 3 and 4 which are associated with the southern zone of the orebody (Table 13.7 and Table 13.8). Depression of tennantite would be detrimental to the Cu recovery for both development composites, with the concentrate likely to contain elevated As concentrations.

Mineral Name	DEV.COMP 1	DEV.COMP 1	DEV.COMP 2	DEV.COMP 2	DEV.COMP 3	DEV.COMP 3	DEV.COMP 4	DEV.COMP 4
	A	B	A	B	A	B	A	B
	90% 75 µm	90% 38 µm	90% 75 µm	90% 38 µm	90% 75 µm	90% 38 µm	90% 75 µm	90% 38 µm
Pyrite	1.72	1.84	1.17	1.73	1.81	2.06	1.40	1.70
Chalcopyrite	88.7	90.4	84.2	84.5	75.9	74.6	74.8	76.3
Bomite	4.52	2.12	2.19	1.93	5.03	5.74	6.36	6.05
Tennantite	4.33	5.23	11.6	11.6	17.2	16.4	17.1	15.9
Cobaltite	0.28	0.32	0.12	0.17	0.02	0.01	0.05	0.03
Carrolite	0.01	0.03	0.07	0.02	0.05	0.07	0.01	0.05
Other Cu Sulphides	0.39	0.04	0.68	0.05	0.09	1.05	0.31	0.05
Other Sulphides	0.02	0.00	0.00	0.01	0.00	0.00	0.00	0.00
Total	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0

Table 13.7 Cu Deportment for (Dev 1-4)

Mineral Name	DEV.COMP 1	DEV.COMP 1	DEV.COMP 2	DEV.COMP 2	DEV.COMP 3	DEV.COMP 3	DEV.COMP 4	DEV.COMP 4
	A	B	A	B	A	B	A	B
	90% 75 µm	90% 38 µm	90% 75 µm	90% 38 µm	90% 75 µm	90% 38 µm	90% 75 µm	90% 38 µm
Pyrite	1.72	1.84	1.17	1.73	1.81	2.06	1.40	1.70
Chalcopyrite	88.7	90.4	84.2	84.5	75.9	74.6	74.8	76.3
Bomite	4.52	2.12	2.19	1.93	5.03	5.74	6.36	6.05
Tennantite	4.33	5.23	11.6	11.6	17.2	16.4	17.1	15.9
Cobaltite	0.28	0.32	0.12	0.17	0.02	0.01	0.05	0.03
Carrolite	0.01	0.03	0.07	0.02	0.05	0.07	0.01	0.05
Other Cu Sulphides	0.39	0.04	0.68	0.05	0.09	1.05	0.31	0.05
Other Sulphides	0.02	0.00	0.00	0.01	0.00	0.00	0.00	0.00
Total	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0

Table 13.8 As Deportment for (Dev 1-4)

A single LCT was conducted on the LCZ Dev Comp 5 using flowsheet 2. The final Cu concentrate grade achieved for this test was 30.8% Cu at 93.3% Cu recovery. The grade of Ag in the final concentrate for Dev Comp 5-LCT1 was 5.5 g/t Ag (Table 13.9).

Test ID	Product	Weight		Grade, % Ag g/t				Distribution, %		
		%	Cu	Fe	S	As	Ag	Cu	Fe	S
Dev 5-LCT1	**4th Clnr Conc.**	**16.8**	**30.8**	**31.0**	**34.1**	**0.021**	**5.50**	**93.3**	**24.0**	**24.5**
	1st Clnr Tail	11.9	1.48	30.8	34.0	-	-	30.2	16.9	17.3
	2nd Clnr Tail	14.8	0.67	39.9	44.8	-	-	1.8	27.1	28.2
	Ro Tail	56.5	0.17	12.3	12.4	-	-	1.7	32.0	29.9
	Head (calculated)		5.56	21.7	23.4	0.048	4.30			

Table 13.9 LCZ – LCT

The primary grind utilized in the LCZ was coarser than the UCZ at a P_{80} of 100 µm. The regrind requirements were also coarser at a P_{80} of 42 µm reflecting the high degree of liberation of the coarser mineralization.

Mineralogical analysis was also conducted on LCZ Dev Comp 5 using QEMSCAN to determine bulk modal analysis and liberation information in two size fractions. The main sulphide mineral is pyrite, which makes up approximately 34.5% of the mass of sulphide mineralogy. Chalcopyrite is the next most abundant, with almost 17.5% by mass. Traces of cobaltite 0.01% to 0.02% are also noted (Table 13.10).

Liberation of the chalcopyrite was 88.8% at the coarser grind of P_{90} 75 µm and increased to 89.7% at the finer grind of P_{90} 38 µm. There is little to no benefit in producing the finer grind for the LCZ mineralization based on the liberation achieved in the coarse grind. Pyrite liberation was 82.8% and 88.4% for the grind sizes of P_{90} 75 µm and 38 µm, respectively. Table 13.10 shows the chalcopyrite liberation and association at grind sizes P_{90} of 38 µm and 75 µm.

Mineral Name	DEV-COMP 5 A	DEV-COMP 5 B
	90% 75 µm	90% 38 µm
Pure Chalcopyrite	71.9	76.9
Free Chalcopyrite	9.23	5.52
Lib Chalcopyrite	7.68	7.25
Chalcopyrite Pyrite	4.74	4.06
Chalcopyrite:Tennantite	0.00	0.00
Chalcopyrite Cobaltite/Carrolite	0.00	0.00
Chalcopyrite: Other Cu Sulphides	0.00	0.00
Chalcopyrite Other Sulphides	0.00	0.00
Chalcopyrite: Quartz/Feldspars	2.00	2.08
Chalcopyrite: Barite	0.00	0.00
Chalcopyrite: Pyrite: :Qtz/Feld	1.38	0.77
Chalcopyrite: Pyrite: Tennantite	0.00	0.00
Complex	3.11	3.39
Total	**100.0**	**100.0**
Liberated	88.8	89.7

Table 13.10 Chalcopyrite Liberation and Association for Dev Comp 5

13.3 2019 Base Metal Laboratories FS Test Program – Upper Copper Zone

The FS work program was undertaken at Base Metal Laboratories (BML) in Kamloops BC. The program reported in BL02082 July 2019 and consisted of 21 variability samples taken across the UCZ of the deposit and one LCZ composite as illustrated in Figure 13.2. The program consisted of the following key flotation test stages:

- Rougher flotation variability testing;
- Kinetic cleaner tests;
- Upgrade cleaner tests;
- LCT's on six selected composites from the group of 21, based on variability in mineralogy across the orebody.
- Open cycle – cleaner flotation tests based on the final LCT flowsheet for each composite.

13.3.1 Samples

Following the results of the PFS metallurgical program, 86 drillholes were undertaken, and 21 holes designated BBFT01 to 21 were selected for metallurgical testing. Detailed mineralogical examinations were conducted prior to metallurgical testing to examine the variation in the mineralogical changes from the north to the south of the deposit. The drillholes were selected based on the mineralogical alterations across the deposit to allow for a comprehensive metallurgical evaluation and determination of grade and recovery relationships. Table 13.11 and Figure 13.9 detail the sample holes.

Metallurgical Composite	MODA Recovery %	Supergene Alteration	Cu Grade	Tennantite	% Cu in Tennantite
BBFT01	84%	TRACE	2.18%	0.30%	17%
BBFT02	97%	TRACE	1.71%	0.30%	10%
BBFT03	99%	NONE	3.36%	0.10%	3%
BBFT04	94%	NONE	3.37%	1.10%	17%
BBFT05	48%	TRACE	2.19%	0.40%	15%
BBFT06	86%	STRONG	2.71%	0.40%	13%
BBFT07	63%	NONE	1.45%	0.40%	15%
BBFT08	74%	TRACE	1.65%	0.40%	13%
BBFT09	74%	NONE	2.43%	0.40%	16%
BBFT10	91%	TRACE	3.16%	0.40%	8%
BBFT11	85%	TRACE	2.54%	0.20%	4%
BBFT12	78%	NONE	3.33%	1.60%	27%
BBFT13	86%	STRONG	2.16%	0.30%	22%
BBFT14	74%	TRACE	2.02%	0.20%	14%
BBFT15	98%	TRACE	3.69%	0.30%	9%

Metallurgical Composite	MODA Recovery %	Supergene Alteration	Cu Grade	Tennantite	% Cu in Tennantite
BBFT16	88%	NONE	1.95%	1%	25%
BBFT17	79%	NONE	2.63%	0.70%	14%
BBFT18	85%	TRACE	2.95%	0.40%	14%
BBFT19	86%	NONE	2.56%	Tr	1%
BBFT20	82%	NONE	2.37%	0.10%	5%
BBFT21	75%	NONE	2.45%	0.30%	7%

Table 13.11 Metallurgical Composite Table

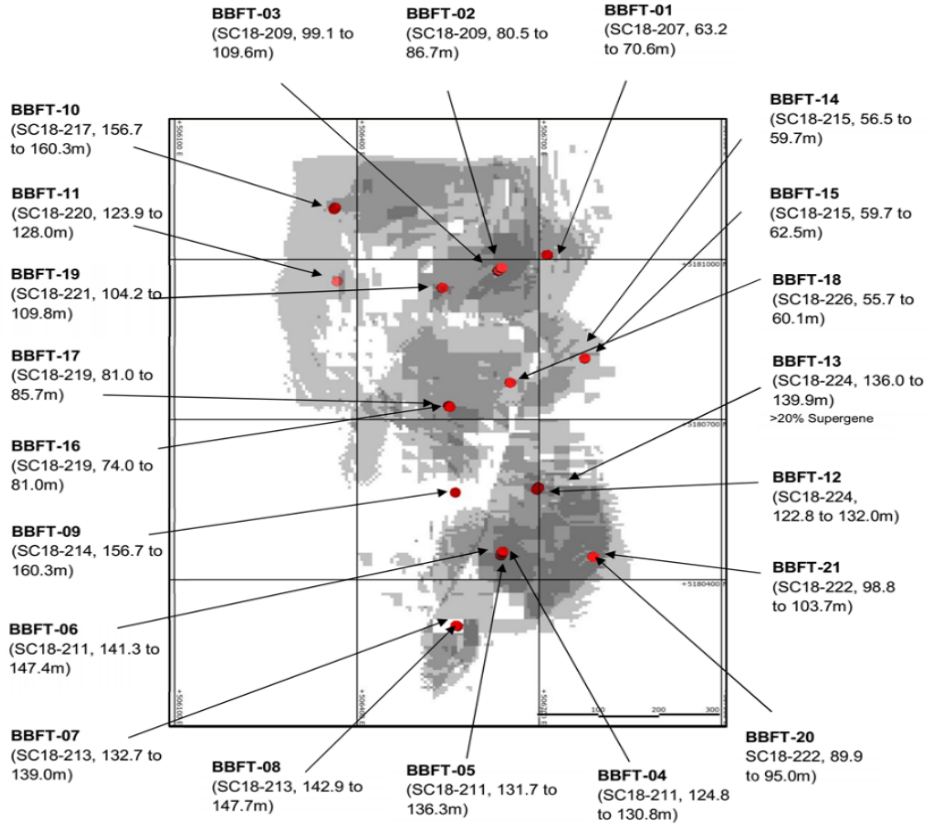


Figure 13.9 Metallurgical Sample – Drillhole Locations

13.3.2 Mineralogy

Metallurgical composites BBFT01 to 21 were submitted to McArthur Ore Deposit Assessments Pty Ltd (MODA) in 2017 for optical mineralogy. Liberation characteristics were completed using an arbitrary grind of 53 µm and measured by using a systematic objective technique, which enables a comparative "grade-recovery' assessment of the samples.

The MODA optical assessments (McArthur, 2019; 2018; 2017; and 2017a) verified the previous findings with significant variability in ore mineralogy, texture and grain size. Cu residence based on a SEM analysis indicates that Cu is divided up variably between Chalcopyrite (63% to 96%), Tennantite (<0.1% to 23%) and Bornite (0% – 16%). All three copper sulphide minerals exhibit significant variability in grain size.

A summary of the sample weighted compositions is reported in Table 13.12 and the liberation characteristics based on a grid count and 53 µm mask are reported in Table 13.13.

Metallurgical Composite	Py	Ma	Cp	Bn	Cv	Cc	Tn	Cz	Sg	Ga
BBFT01	37.98	0.93	2.91			0.01	0.28		0.1	57.79
BBFT02	33.42	1.7	5.61		0.13		0.42		0.4	58.31
BBFT03	48.17	0.94	7.27							43.62
BBFT04	45.02		7.8	0.01		0.03	1.34	0.01	0.03	45.77
BBFT05	62.64	0.01	2.47	0.34		0.01	0.37		0.25	33.91
BBFT06	58.04			0.18	0.01	1.56	0.75			39.46
BBFT07	73.3	0.1	3.27	0.11			0.55			22.66
BBFT08	57.15		3.26	0.22			0.32			39.05
BBFT09	59.32	1.95	3.43				0.55			34.75
BBFT10	37.02	3.47	7.63				0.26			51.61
BBFT11	48.52	3.45	7.37				0.31		0.17	40.18
BBFT12	47.52		6.31	0.06			1.82		0.27	44.03
BBFT13	58.44	0.01	0.03	0.07		0.69	0.46			40.32
BBFT14	28.81	0.19	2.71			0.01	0.04		0.7	67.54
BBFT15	28.29	0.42	6.17				0.05			64.86
BBFT16	31.24	0.03	5.77				0.59		0.23	62.14
BBFT17	62.93		7.16				0.74	0	0.03	29.13
BBFT18	35.65	0.38	5.37		0.01		0.47		0.85	57.42
BBFT19	48.65	0.01	6.34				0.02			44.97
BBFT20	58.75	0.07	3.21	0.02			0.21			37.72
BBFT21	55.93	0.09	5.69				0.49			37.80

Py = Pyrite, Ma =Marcasite, Cp=Chalcopyrite, Bn= Bornite, Cv=Covellite, Cc= Calcocite, Tn=Tennantite, Cz=Colusite,Sg= Cuprion Siegenite

Table 13.12 BBFT Series Weight % Composition by Mineral Species

Metallurgical Composite	Free	Binary with					Binary	Ternary	Quat'y	Free "Cu"
		Pyrite	Bornite	Tennantite	Ma/Sg	Gangue				
BBFT01	17	59	17	0	0	24	31	25	27	34
BBFT02	38	44	0	0	0	56	17	36	9	38
BBFT03	38	22	0	0	0	78	38	20	4	38
BBFT04	39	34	0	33	0	33	12	41	8	72
BBFT05	12	83	0	0	0	17	32	51	5	12
BBFT06	0	0	0	0	0	0	0	100	0	0
BBFT07	9	63	0	0	0	37	19	62	10	9
BBFT08	7	17	0	0	0	83	11	77	5	7
BBFT09	0	42	0	0	23	34	28	62	10	0
BBFT10	11	31	0	6	0	62	44	36	9	17
BBFT11	9	30	0	0	26	43	36	37	18	9
BBFT12	13	93	0	0	0	7	17	67	3	13
BBFT13	0	0	0	0	0	0	0	89	11	0
BBFT14	29	74	0	0	0	26	38	26	7	29
BBFT15	48	4	8	8	0	80	40	12	0	64
BBFT16	19	50	0	0	0	50	31	46	4	19
BBFT17	14	97	0	0	0	3	23	58	5	14
BBFT18	22	24	0	0	0	76	40	26	12	22
BBFT19	27	9	0	9	0	82	38	33	2	36
BBFT20	11	98	0	0	0	2	39	37	13	11
BBFT21	10	50	0	0	0	50	22	61	7	10

Free "Cu" is the total distribution of Free + (Binary Bornite and Tennantite)

Table 13.13 BBFT Series Chalcopyrite Liberation at 53 µm

Using the 53 µm mask, the free Chalcopyrite ranges from 0% to 48% across the set in Table 13.13, with a large variation in binary association with both Pyrite and Gangue and minor binary associations with Bornite, Tennantite and Marcasite.

The mineralogical test work identified that the primary liberation of chalcopyrite would be weak based on the 53 µm arbitrary grind utilized in the procedure. A finer primary grind would be required to increase the level of liberated chalcopyrite by reducing the amount of chalcopyrite-pyrite binaries, chalcopyrite-gangue binaries and pyrite-gangue-chalcopyrite ternary grains.

13.3.3 *Rougher Flotation Test Work*

Extensive rougher flotation test work was undertaken with the new composites to optimize on the previous PFS results. Cyanide was no longer used in the BML program. The program focused on the following key areas:

- Effects of primary grind on the rougher grade and recovery using mild steel media;
- Effects of reagent and pH changes on rougher kinetics;
- The use of monosodium phosphate (MSP) as pyrite depressant; and
- The use of dextrin to address carbonaceous pyrite.

Baseline testing was conducted on BBFT01 to 15 and BBFT17 composites (Figure 13.10), using a mild steel mill and mild steel rods to generate a nominally P_{80} of 53 µm. The baseline reagent scheme used Aero 3477, MIBC, lime for pH adjustment to a nominal 9.5 pH and no depressants. The results from the baseline rougher flotation tests showed a large variation in flotation response and grade/recovery across the orebody.

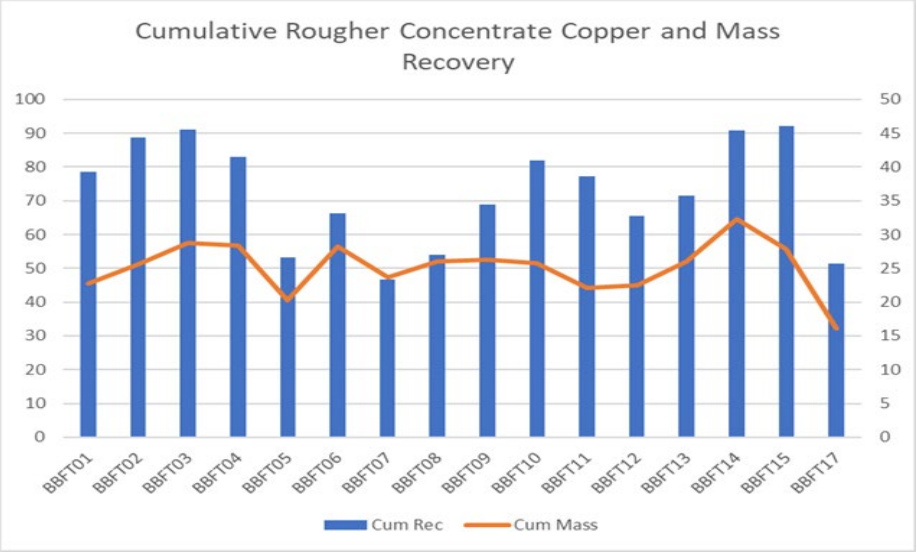


Figure 13.10 Cumulative Rougher Cu and Mass Recovery – Baseline Tests

The variance in performance was compared to the feed sample characteristics. The most significant relationship relating to rougher metallurgical performance was the pyrite content in the feed as defined by the mineralogical estimate from the MODA analysis. As the pyrite content in the feed increased, the metallurgical performance of Cu dropped as shown by Figure 13.11 and Figure 13.12.

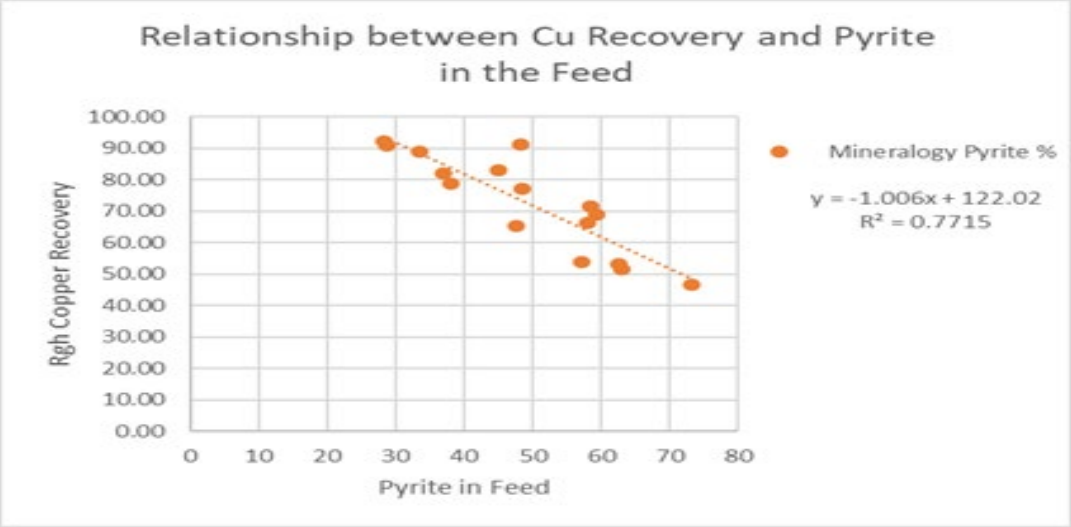


Figure 13.11 Cu Recovery to Pyrite in the Feed

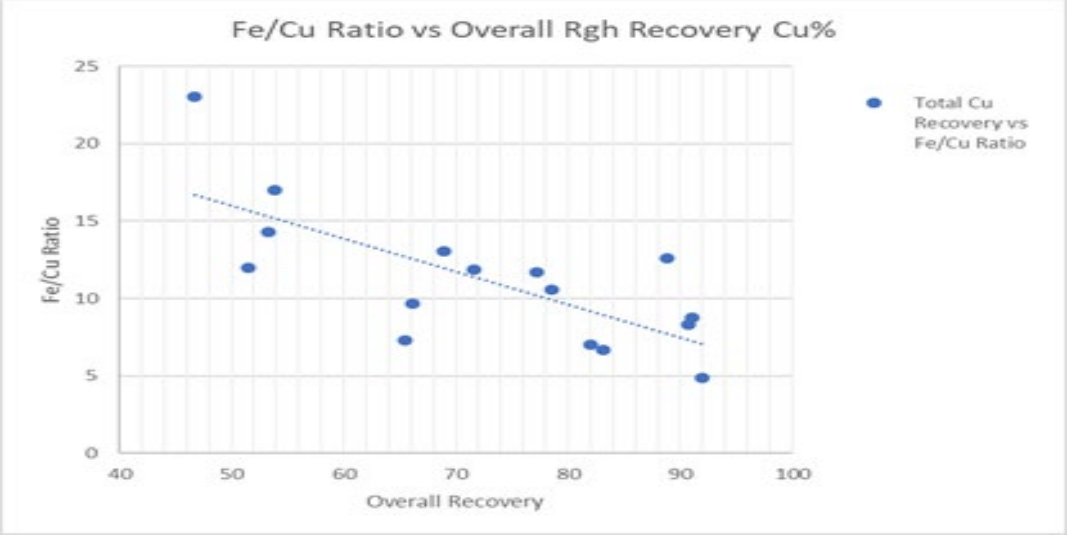


Figure 13.12 Fe/Cu Ratio to Rougher Cu Recovery

13.3.4 Rougher Flotation PE26 (CMC) Depressant

Carbonaceous pyrite has been described in the mineralogical reports as carbon rimmed pyrite. During the baseline rougher tests, an aeration stage was added after primary grinding to ensure the dissolved oxygen content in the flotation pulp was sufficient to promote copper sulphide flotation and good pyrite depression. During the aeration phase, the operators noted hydrophobic mineralization prior to reagent additions was occurring and this was likely due to organic carbon.

Carboxymethyl Cellulose (CMC) (PE26TM) was used to depress the carbonaceous material in dosages ranging from of 100 to 200 g/t prior to the addition of the Cu collector.

The results show an improvement in Cu rougher grade with a reduction in flotation mass recovery while maintaining similar recovery values to the baseline test in each case. Refer to Figure 13.13 and Figure 13.14 for Cu grade and mass recovery.

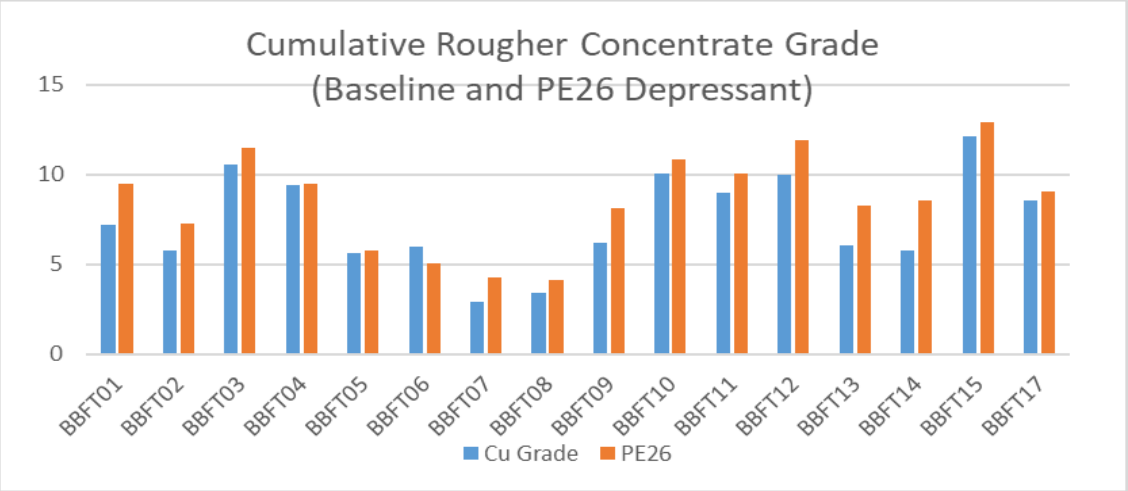


Figure 13.13 PE26 Depressant - Cu Rougher Concentrate Grade

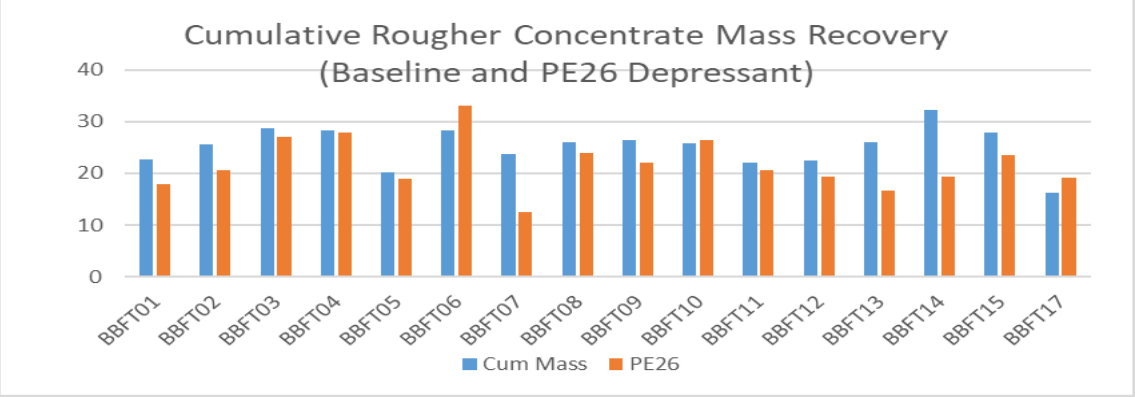


Figure 13.14 PE26 Depressant - Cu Rougher Mass Recovery

The addition of the depressant reduced the mass recovery to the rougher by 3.4% on average. The reduction of both sulphur and carbon recovery with the addition of the depressant, while maintaining Cu recovery, is indicative of a carbon/pyrite activation.

13.3.5 *Rougher Flotation – Monosodium Phosphate (MSP), Dextrin and PE26*

The testing scope of this phase of optimization was limited to starch type, dosage, and the effect of a chelating agent mono-sodium phosphate (MSP) on Cu metallurgy. The starches, Dextrin and CMC (PE26), were used to limit the hydrophobicity of carbon or mineral with embedded organic carbon. The chelating agent aids in the removal of unwanted ions (Pb and Cu) that may be on the surface of pyrite and are causing activation and flotation of undesirable minerals.

Testing was undertaken on composites BBFT01, BBFT03, BBFT07, BBFT08, BBFT09, BBFT13, and BBFT17 (Table 13.14). The samples were selected to cover a range of mineralogy types and metallurgical response. The MSP and dextrin tests had the reagent added to the grinding mill, while PE26 was added at the first stage of rougher flotation.

	Test	MSP	DEXTRIN	PE26	% Mass	Rgh %Cu Grade	Rgh %Cu Rec	Cu% Head	Fe% Head	Fe/Cu
BBFT01	Ro-31	80	320		20.64	8.76	82.60	2.19	21.81	9.96
	Ro-35	80		150	21.19	8.91	83.89	2.25	20.69	9.19
	Ro-41	40	160		18.59	9.27	78.48	2.19	20.08	9.15
BBFT03	Ro-32	80	320		23.43	13.21	93.74	3.30	28.35	8.59
	Ro-36	80		200	23.70	13.73	93.22	3.49	25.39	7.27
	Ro-42	160	640		25.17	12.35	92.84	3.35	27.15	8.11
	Ro-51	200	320		21.42	14.17	92.13	3.29	27.77	8.43
	Ro-52	80			25.74	12.42	94.93	3.37	29.20	8.67
	Ro-58	400	320		22.46	13.45	93.75	3.22	28.07	8.71
BBFT07	Ro-33	80	320		13.89	5.69	51.62	1.53	34.37	22.45
	Ro-37	80		200	12.95	5.16	46.02	1.45	31.01	21.37
	Ro-43	160	640		16.89	5.27	58.83	1.51	31.33	20.69
	Ro-53	200	320		15.50	5.18	53.66	1.50	34.01	22.74
	Ro-54	80			12.53	4.89	40.69	1.50	34.30	22.80
	Ro-59	400	320		14.00	4.88	46.87	1.46	32.85	22.55
BBFT08	Ro-34	80	320		19.23	5.31	58.71	1.74	29.21	16.78
	Ro-38	80		150	18.34	5.28	60.93	1.59	24.98	15.72
	Ro-44	160	640		25.70	4.39	67.54	1.67	26.53	15.88
BBFT09	Ro-39	80		150	19.23	8.50	74.31	2.20	27.83	12.65
	Ro-45	40		75	19.62	8.67	73.83	2.30	29.05	12.61
BBFT13	Ro-40	80	200		18.56	7.98	70.78	2.09	23.50	11.24
	Ro-46	80	320		18.95	7.56	70.20	2.04	27.33	13.40
	Ro-47	160	640		18.59	7.95	71.06	2.08	27.41	13.17
	Ro-55	200	320		16.88	8.38	68.03	2.08	24.25	11.66
	Ro-56	80			17.60	7.04	60.79	2.04	24.68	12.11
	Ro-60	400	320		23.88	6.57	76.02	2.06	24.42	11.83
BBFT17	Ro-50	80	320		20.44	9.74	72.28	2.76	31.29	11.36

Table 13.14 Depressant Performance

It became evident from the test work, reported in Table 13.14, that the combination of MSP and Dextrin improved flotation performance, with a lower mass recovery, higher rougher concentrate grade and maintained or improved the rougher Cu recovery.

MSP/dextrin was adopted as standard conditions for the rougher flotation as a result of the rougher optimization testing.

13.3.6 *Rougher Collector Tests*

Three tests were undertaken to investigate the effect of an alternative collector, Aero 3418A, a unique dithiophosphinate collector.

The 3477 produced a superior result when compared to 3418A as shown in Table 13.15. Hence, 3477 became the collector of choice for the remaining test work.

	Test	3477	3418A	MSP	Dextrin	PE26	% Mass	Rgh %Cu Grade	Rgh %Cu Rec	Cu% Head	Fe% Head	Fe/Cu
BBFT03	Ro-51	80		200	320		21.42	14.17	92.13	3.29	27.77	8.43
	Ro-61		80	400	320		27.09	11.18	93.47	3.24	28.08	8.67
BBFT07	Ro-53	140		200	320		15.50	5.18	53.66	1.50	34.01	22.74
	Ro-59	80		400	320		14.00	4.88	46.87	1.46	32.85	22.55
	Ro-62		80	400	320	50	16.91	3.05	37.80	1.36	33.07	24.27
BBFT13	Ro-55	90		200	320		16.88	8.38	68.03	2.08	24.25	11.66
	Ro-63		80	400	320	50	19.88	3.29	33.07	1.98	25.25	12.78

Table 13.15 Rougher Collector Tests

13.3.7 *Rougher Flotation – Primary Grind*

Three sets of composites were selected to further examine the effect of the primary grind size and liberation on rougher Cu recovery. The tests, conducted at 53 µm, used in the baseline tests were to be compared to a new 35 µm P_{80} primary grind size.

Flotation results indicated improvements in grade and recovery for BBFT03 and BBFT07 with a minor improvement in the performance of BBFT13 as detailed in Table 13.16 and discussed below.

	Test	K_{80}	3477	MSP	Dextrin	% Mass	Rgh %Cu Grade	Rgh %Cu Rec	Cu% Head	Fe% Head	Fe/Cu
BBFT03	Ro-58	53 µm K_{80}	80	400	320	22.46	13.45	93.75	3.22	28.07	8.71
	Ro-64	35 µm K_{80}	80	400	320	23.72	14.74	95.20	3.18	27.14	8.53
BBFT07	Ro-59	53 µm K_{80}	80	400	320	14.00	4.88	46.87	1.46	32.85	22.55
	Ro-65	35 µm K_{80}	80	400	320	15.26	5.24	56.38	1.42	33.01	23.28
BBFT13	Ro-60	53 µm K_{80}	80	400	320	23.88	6.57	76.02	2.06	24.42	11.83
	Ro-66	35 µm K_{80}	80	400	320	19.94	7.31	74.61	1.96	24.69	12.63

Table 13.16 Rougher Primary Grind Effects

- BBFT03 composite located in the northern ore zone, had 38% of the Chalcopyrite quantified as being liberated at 53 µm grind mask size. The effect of a finer grind is noticeable in the initial rougher stage with the latter slower floating Cu mineralization results being similar to the 53 µm grind;
- BBFT07 composite located in the southern ore zone, had 9% of the Chalcopyrite quantified as being liberated at 53 µm grind mask size, with 19% as binary and 62% as tertiary composites. There was a considerable improvement in rougher recovery at the finer grind on this sample, which has indicated significant interlocking with pyrite and multiphase particles;
- BBFT13 composite located in the southern ore zone, had no liberated and binary Chalcopyrite quantified at 53 µm grind mask size. Ternary made up 89% and Quaternary 11%. Overall results using the finer grind did not produce an improvement.

There was a marked improvement with the finer grind size of 35 µm. The mineral liberation characteristics of many of the samples indicated that further primary grinding could improve rougher metallurgical performance. The tests were restricted to 35 µm K_{80} due to the practical implication of equipment capital and the effect on the operating cost.

13.3.8 Cleaner Test Work – Overview

There were 97 open cleaner tests performed. Development of the cleaner circuit utilized several different cleaner flowsheets. A schematic of the test flowsheets used is displayed in Figure 13.15.

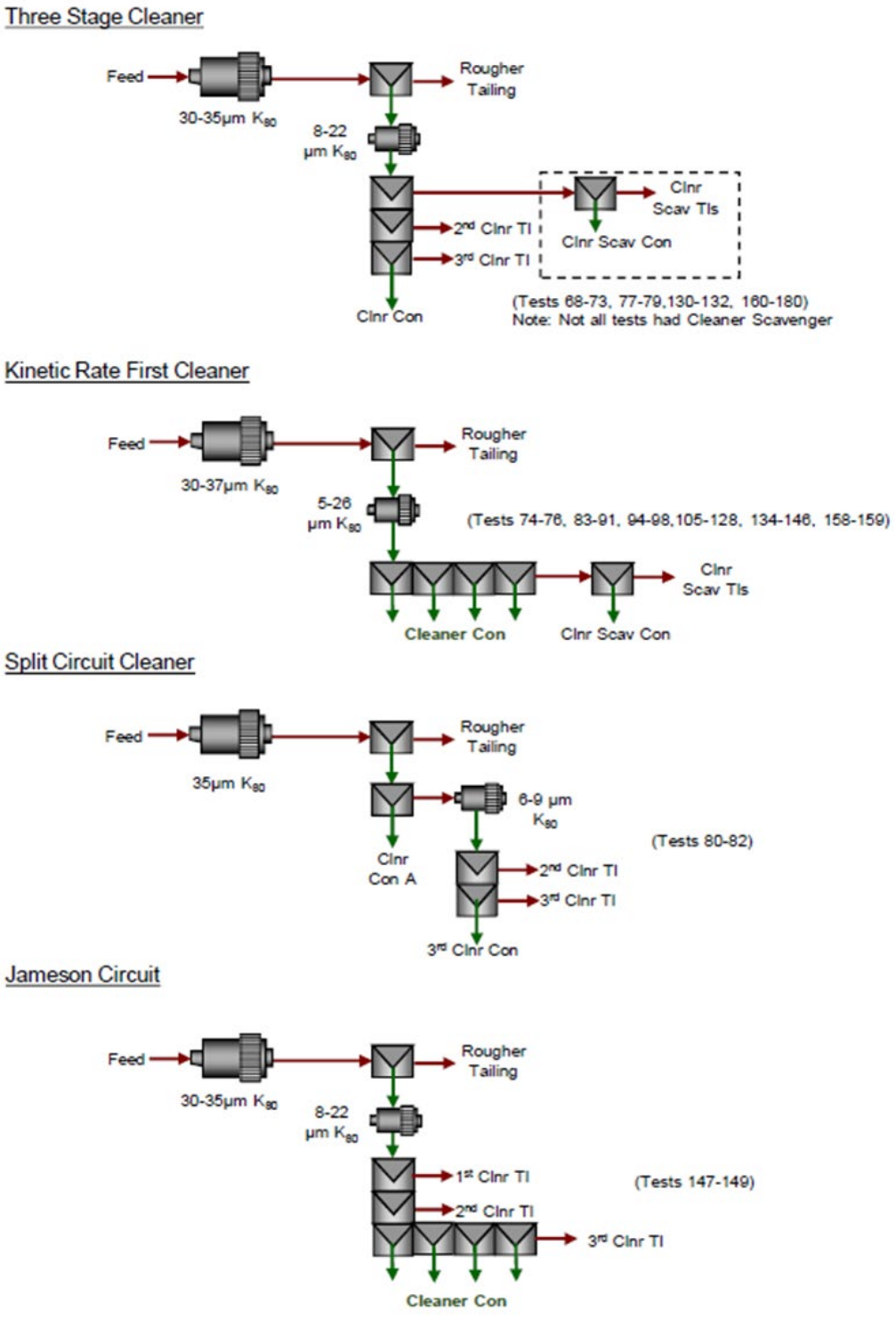


Figure 13.15 Cleaner Test Flowsheets

Flowsheet test conditions varied. However, the following is a basic summary of the key parameters:

- Primary grinding used a 20 kg rod charge in a mild steel grinding mill. Initial cleaner test work was undertaken with a mild steel charge prior to optimization using a stainless steel charge in the latter test series. The grinding size was nominally 30 - 35 µm P_{80};
- The rougher flotation was performed as the optimized conditions established in the rougher testing phase. This included the use of MSP/dextrin as depressant, pH of 9.5 modulated with lime and Aero 3477 as the collector;
- Regrinding was conducted in a stainless stirred mill with high density ceramic beads (3.5 mm);
- The kinetic rate first cleaner test flowsheet was used to assess specific parameters. The three stage cleaner test was used to estimate final concentrate grades, geometallurgical performance predictions and set up tests for locked cycle testing. The split circuit flowsheet was investigated as a plant configuration. Only a limited number of tests were performed with this flowsheet. Finally, the Jameson batch test was a cleaner test protocol for estimating the performance of a Jameson flotation cell.

Optimization of the cleaner circuit investigated several parameters, these included:

- Primary grind media type, mild steel or stainless;
- Regrind size of P_{80} of 5 to 25 µm;
- Cleaner flotation pH;
- Split flotation circuit Jameson tests;
- Site water.

Batch testing was conducted in a similar fashion as the rougher testing. Select composites were tested with the optimized protocols deployed on all the composites.

13.3.9 Cleaning Testing – Effect of pH

The effect of pH was briefly investigated on three samples; namely BBFT03, BBFT07 and BBFT16. The pH was maintained at 9.5 during the rougher flotation stage with additional lime added to the cleaner stage to modulate the cleaner pH to 10.5 and 11.5.

A three stage cleaner test was performed on BBFT03 and BBFT10 while a series of kinetic cleaner tests were conducted on BBFT16 for comparison. While results for BBFT03 and BBFT10 showed a benefit to the grade and recovery at an elevated pH of 10.5, results on BBFT16 showed the reverse trend with metallurgical performance reducing with the elevated pH.

Based on the mixed results, the cleaner pH was maintained at 9.5.

13.3.10 Cleaner Testing – Site Water

A series of cleaner tests was undertaken to assess differences between the Kamloops laboratory water and site water. Five samples were tested representing a range of geological lithologies. The tests were run in duplicates. The results indicated very similar metallurgical performance within experimental error for each water source. Therefore, the raw site water is not expected to cause any metallurgical performance issues.

13.3.11 Cleaner Testing – Split Cleaner Flowsheet

Due to the relatively high rougher mass recovery and fine regrind size needed for some of the samples, an alternative split regrind and cleaner flowsheet was investigated on three samples.

The objective of testing this process alternative was to minimize the generation and loss of fine liberated copper sulphide minerals that were observed in the cleaner tailings stream. The comparative test results are given in Figure 13.16.

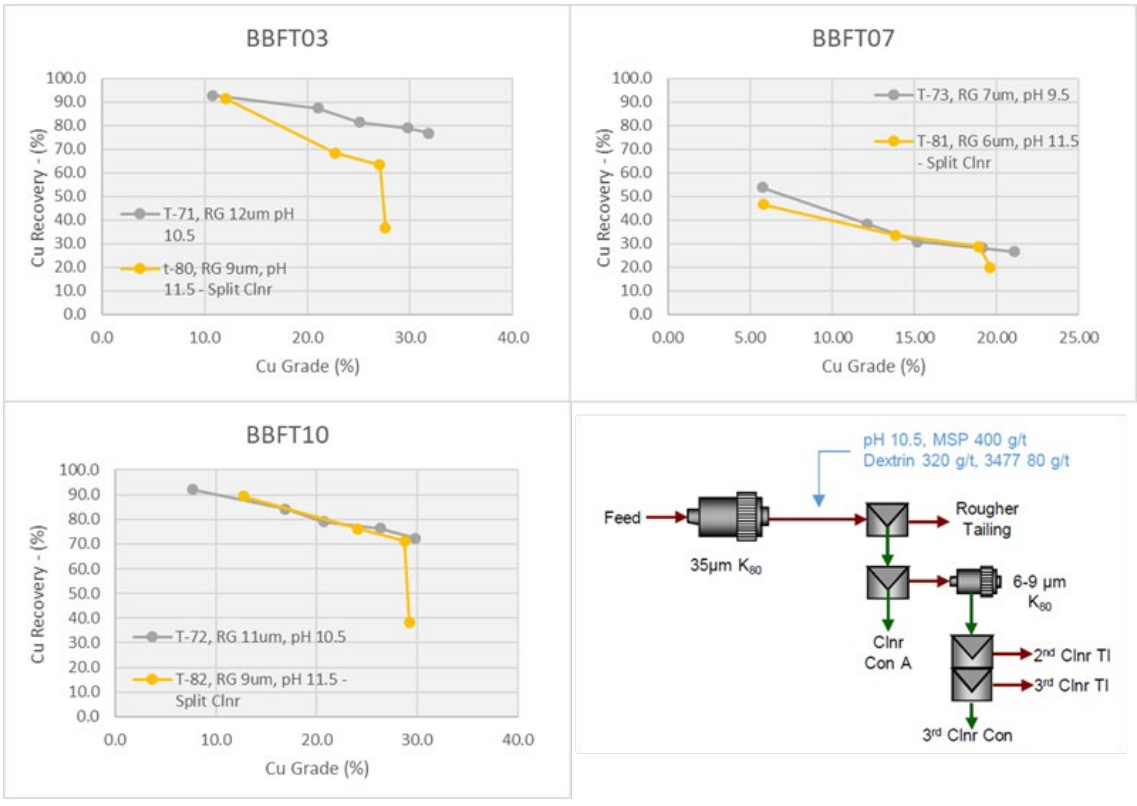


Figure 13.16 Cleaner Tests – Split Cleaners

Sample BBFT03 showed poorer Cu performance for the split circuit when compared to the standard three stage cleaner circuit. Samples BBFT07 and BBFT10 had very comparable performance. The cause of the poorer performance of BBFT03 was unknown, possibly related to collector dosage.

Given that two out of three samples showed very similar performance, but did not improve performance, the standard three stage flowsheet was recommended for the remaining testing.

There is still potential for this circuit to reduce the capital and operating cost of the process. A common configuration is to have a Jameson cell in the duty of the selective first cleaner.

13.3.12 Cleaner Testing – Primary Grind Media Effects

Initial tests conducted at a primary grind of P_{80} of 35 - 37 µm and at similar regrind targets for each composite showed improved metallurgical response between the mild steel rods and the stainless steel rod options. Additional test runs on using both a finer primary grind and regrind and a combination of mild steel against stainless steel, reduced the impact witnessed in the initial assessment.

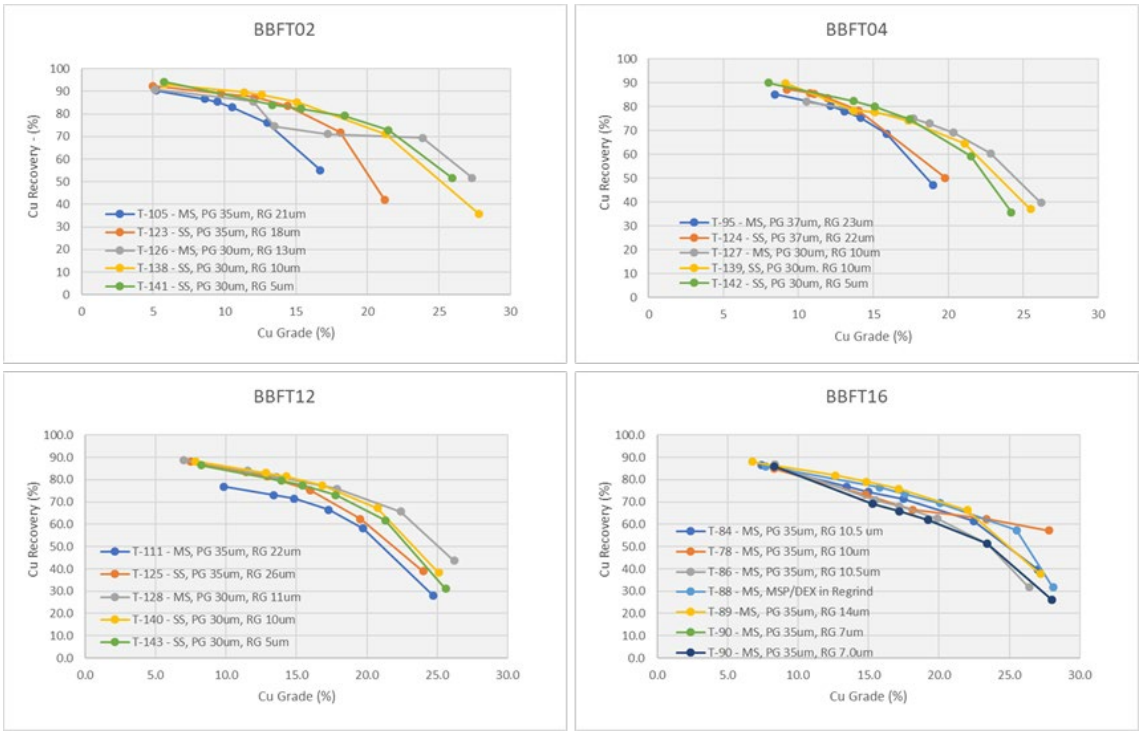


Figure 13.17 Cleaner Testing – Primary Grind Media

The test data, given in Figure 13.17, indicates media type did not have a significant impact on the metallurgical performance.

The use of stainless primary media was adopted following the initial assessment for the new standard conditions and to reduce the amount of iron in solution associated with mild steel media.

13.3.13 Cleaner Testing – Regrind Size

The regrind size was also examined in the same series of tests used to determine media type, with reference to Figure 13.17. The test series included the composites from BBFT02, BBFT04, BBFT12 and BBFT16.

The optimal regrind size was about P_{80} of 10 µm for most of the composites.

At regrind sizes coarser than P_{80} of 10 µm, Cu metallurgical performance degraded, presumable due to insufficient liberation. At regrind targets finer than P_{80} of 10 µm, Cu performance also decreased. The loss in performance was due to higher losses in the cleaner tailing stream. Hence, additional collector was utilized to account for the increased surface area at the finer particle size, thereby improving recovery but having detrimental effect on concentrate grade.

Hence, a regrind target of nominally 10 µm K_{80} was selected for the final standard conditions.

13.3.14 Locked Cycle Tests – Variability Samples

Two sets of LCT's were performed on a subset of samples from the UCZ using the flowsheet shown in Figure 13.18. The samples were selected by SRA technical staff and best represent the UCZ from a geographical, geological and grade perspective. An additional sample BBFT12 was added to the final test series.

The first set of LCT's used a mild steel primary grinding media and a target regrind sizing of nominally P_{80} of 15 µm. After continuing optimization batch tests, the second set of LCT's were performed using a stainless steel primary grind charge and a finer regrind target of P_{80} of 10 µm.

The adjustment in conditions for the second set of LCT's improved the performance for all of the composites as shown in 0. The stainless grinding charge and finer regrind size recovered 80.6% of the Cu into a concentrate grading 23.5% Cu.

Composite	Test	Primary Grind µm K_{80}	Overall Cleaner Conditions				Mass	Assay - Percent				Distribution - Percent			
			g/tonne		pH	Rgd									
			MSP/ Dextrin	3477		µm P_{80}	%	Cu	Fe	S	C	Cu	Fe	S	C
BBFT-1	99	36.7	100/75	50	9.5	13.7	6.6	**21.0**	31.2	39.9	0.37	**67.4**	9.9	10.8	7.9
	150	34.0	100/75	45	9.5	8.9	5.9	**25.5**	28.8	38.4	0.40	**71.6**	**8.1**	**8.7**	**7.0**
BBFT-3	100	35.0	100/75	60	9.5	16.3	11.1	**24.2**	31.6	37.8	0.29	**83.6**	12.9	12.6	5.4
	151	35.0	100/75	65	9.5	10.7	11.4	**27.1**	31.8	36.1	0.05	**90.1**	12.4	12.5	1.4
BBFT-4	101	53.0	100/75	40	9.5	19.8	12.3	**19.3**	32.6	40.1	0.28	**74.4**	18.1	16.1	10.7
	152	37.0	100/75	43	9.5	9.3	12.6	**18.9**	33.0	40.6	0.10	**75.1**	19.3	17.3	8.3
BBFT-10	102	37.7	100/75	60	9.5	15.0	11.2	**23.1**	30.5	36.5	0.41	**83.4**	16.1	14.8	8.4
	153	35.0	100/75	65	9.5	8.5	10.4	**25.6**	30.5	35.8	0.20	**86.3**	14.5	13.6	5.7
BBFT-12	154	35.0	100/75	65	9.5	8.8	10.7	**21.8**	28.8	38.0	0.20	**70.6**	13.6	12.2	6.8
BBFT-13	103	35.0	100/75	45	10.3	15.0	9.9	**13.9**	32.7	43.6	0.68	**66.8**	12.8	13.4	3.4
	155	35.0	100/75	45	9.5	8.9	10.0	**14.1**	30.9	40.4	0.40	**69.8**	13.6	13.3	2.2
BBFT-16	92	35.0	100/75	45	9.5	15.0	8.9	**19.5**	30.4	38.6	0.30	**83.2**	16.9	15.4	9.3
	93	35.0	100/75	18	9.5	15.0	6.9	**22.1**	29.1	38.9	0.28	**81.1**	27.1	11.6	4.6
BBFT-20	104	36.7	100/75	45	9.7	14.0	10.0	**15.2**	35.2	41.2	0.16	**63.3**	11.7	11.0	5.1
	156	35.0	100/75	50	9.5	9.5	10.5	**16.9**	31.8	41.2	0.14	**78.0**	12.0	13.0	4.0

Note: *The primary grinds for tests 99-104 were conducted with mild steel rods, while tests 150-156 were done with stainless steel rods.*

Table 13.17 LCT Summary

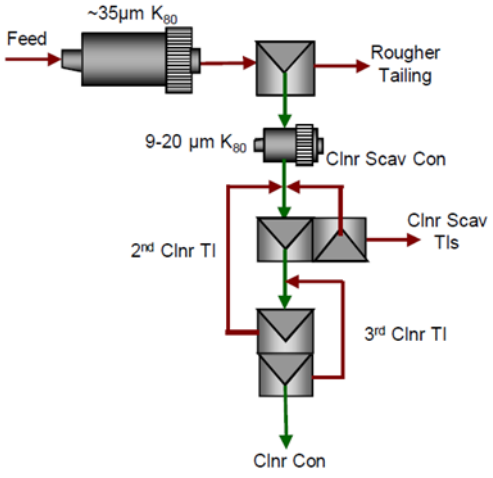
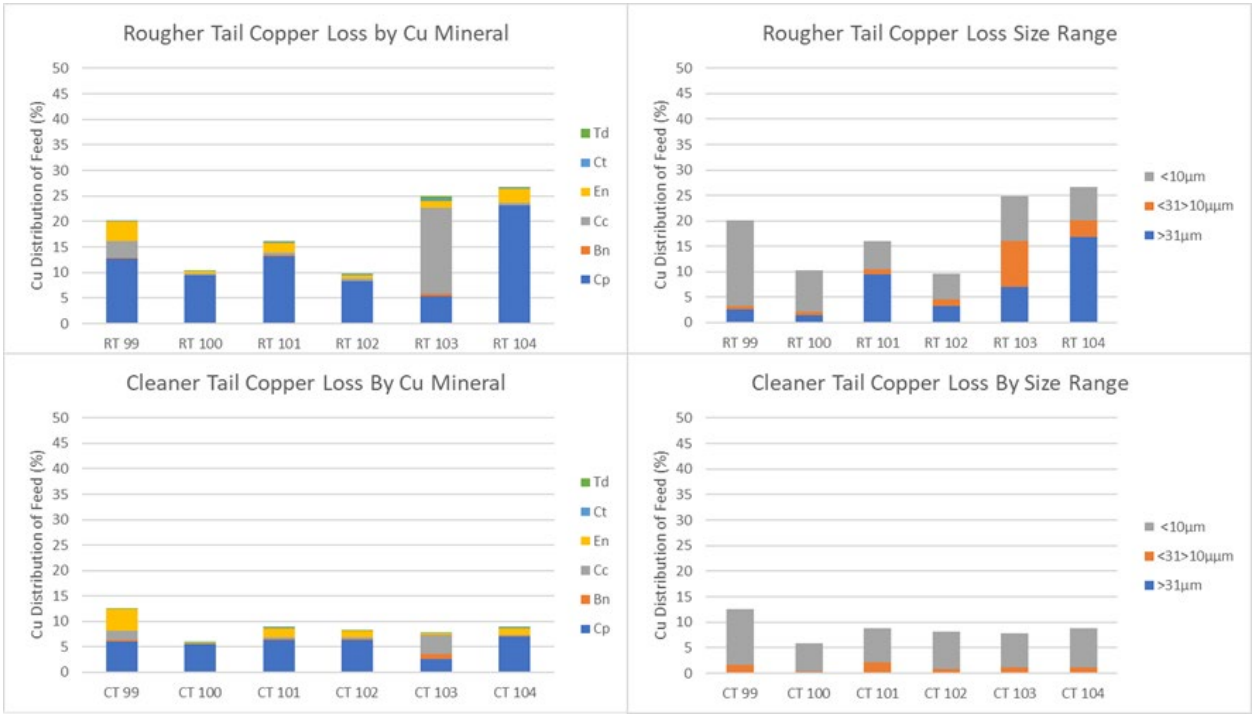


Figure 13.18 LCT's Flowsheet

13.3.15 *Locked Cycle – Mineralogy Analysis of Tailings*

After the first set of LCT's were performed, the tailings streams of the LCT's were analyzed to assess the status of Cu losses. Each sample was sized into three size fractions, and the size fractions were assayed for the key elements. Each size fraction was analyzed by QEMSCAN using the PMA protocol. This protocol quantitatively determines the mineral content and liberation characteristics. A summary of the Cu loss by species and size range are displayed in Figure 13.19.



Note: *Cp-Chalcopyrite, Bn-Bornite, Cc-Chalcosite/Covellite, En-Enargite/Tennantite, Ct-Carrollite, Td-Tetrhedrite*

Figure 13.19 QEMSCAN – Mineral Identification – Rougher and Cleaner Tails

Most of the Cu losses were in the form of chalcopyrite, except for sample BBFT-13. This sample contained mostly secondary Cu mineralization, so it follows that most of the losses occurred as chalcocite/covellite.

The losses from the cleaner tailings stream averaged 8% and were relatively consistent. Conversely, the losses from the rougher tailings were much more variable, ranging from 10% to 25%

Cu losses by three size ranges were examined. The rougher tailings samples 101 to 104 had Cu losses equally distributed in the fine and coarse size fractions. For tests 99 and 100, the rougher tailings Cu losses were mainly in the fine fractions and more liberated.

The loss of Cu in the cleaner tailings streams was mostly less than 10 µm and reasonably well liberated. This is a typical profile for cleaner tailings and it would be difficult to recover further Cu without risk of pyrite flotation to the concentrate.

13.3.16 *Locked Cycle – Minor Element Analysis*

Concentrates from the LCT's were subjected to minor element analysis (Table 13.18).

Analyte Symbol	Unit Symbol	Detection Limit	Analysis Method	BBFT-03 T151	BBFT-04 T152	BBFT-10 T153	BBFT-12 T154	BBFT-13 T155	BBFT-20 T156
Ag	ppm	0.2	AR-AAS	9	14	14	12	44	53
Al	%	0.01	FUS-Na_2O_2	0.05	0.07	0.12	0.07	0.03	0.08
As	ppm	5	FUS-MS-Na_2O_2	3090	9300	8740	17500	5270	4280
Au	g/mt	0.07	FA-ICP	<0.07	<0.07	<0.07	<0.07	<0.07	<0.07
Ba	ppm	3	FUS-MS-Na_2O_2	119	>10000	4580	>10000	7100	7220
Bi	ppm	2	FUS-MS-Na_2O_2	414	71	746	388	91	7220
C	%	0.02	IR-Leco	0.03	0.09	0.32	0.21	0.49	0.14
Ca	%	0.01	FUS-Na_2O_2	0.05	0.03	0.05	<0.01	0.42	0.06
Co	ppm	0.2	FUS-MS-Na_2O_2	686	1660	1880	1780	764	1310
Cs	ppm	0.1	FUS-MS-Na_2O_2	0.5	1.1	0.5	0.9	1.2	0.7
Cu	%	0.02	AR-AAS	26.9	18.9	25.6	21.5	14.2	16.9
Fe	%	0.01	FUS-ISE	<0.01	0.01	<0.01	<0.01	<0.01	<0.01
Fe	%	0.1	AR-AAS	31.6	33.0	30.3	28.8	30.7	31.8
Ga	ppm	0.2	FUS-MS-Na_2O_2	25.7	13.2	16.8	27.3	5.4	17.6
Ge	ppm	0.7	FUS-MS-Na_2O_2	70.0	62.2	71.1	61.1	17.1	64.9
Hg	ppm	1	AR-ICP	1	<1	3	4	9	<1
In	ppm	0.2	FUS-MS-Na_2O_2	10.1	7.7	18.1	9.7	2.9	17.7
Mg	%	0.01	FUS-Na_2O_2	0.02	0.01	0.02	0.01	0.01	0.01
Ni	ppm	10	FUS-MS-Na_2O_2	280	760	1120	1030	210	340
Pb	ppm	0.8	FUS-MS-Na_2O_2	402	1170	660	1200	2570	3200
S	%	0.1	IR-Leco	36.3	40.0	35.6	39.0	41.2	41.3
Sb	ppm	2	FUS-MS-Na_2O_2	331	506	422	539	217	284
Sn	ppm	0.5	FUS-MS-Na_2O_2	24.9	7.7	7.2	14.4	1.1	5.4
U	ppm	0.1	FUS-MS-Na_2O_2	0.2	0.3	1.3	0.9	0.4	0.4
Zn	ppm	30	FUS-MS-Na_2O_2	690	380	400	980	350	460

Table 13.18 Minor Element Analysis

The assay data indicates the following:

- As levels were elevated and as high as 1.7% As, which will attract a smelter penalty. The primary As bearing mineral was tennantite. Due to the floatability of tennantite, flotation options to reduce As recovery to the concentrate are limited;
- The levels of As did vary significantly sample to sample, so blending control may be the most practical option to control As levels in the concentrate. Prediction and modelling of As will be required to execute mining control strategy;
- The concentrates also contained elements that may have some payment potential. These elements are cobalt, germanium, gallium, indium and silver.

13.3.17 Upper Copper Zone Concentrate Dewatering

Bulk concentrate from the UCZ was used to assess the dewatering properties for the concentrate. The tests included raked static graduated cylinders and pressure filtering tests. The concentrate tested was a slurry from bulk Test 157.

Settling tests were conducted using three commercial flocculants, namely Magnafloc 351, 155 and 10. From the initial screening tests, Magnafloc 10 was selected, providing the best overflow clarity and fastest free settling rate (Table 13.19 and Figure 13.20).

Test	Test Type	Floc Type	Dosage (g/t)	Free Settling Rate (mm/sec)	Final Density (% solids)	Final Supernatant Clarity
S1	Screen	MF 315	15.0	2.6	56	Turbid
S2	Screen	MF 10	15.0	7.8	52.1	Turbid
S3	Screen	MF 155	15.0	3.4	44.7	Very Turbid
S4	Screen	MF 10	10.0	8.5	50.7	Turbid
S5	Screen	MF 10	12.5	15.9	52.1	Clear
S6	Raked 2L	MF 10	10.0	3.5	67.6	Clear
S7	Raked 2L	MF 10	13.0	4.6	64.7	Turbid
S8	Raked 2L	MF 10	15.0	5.2	66.8	Turbid
S9	Raked 2L	MF 10	20.0	5.8	67.6	Clear

Table 13.19 Settling Tests

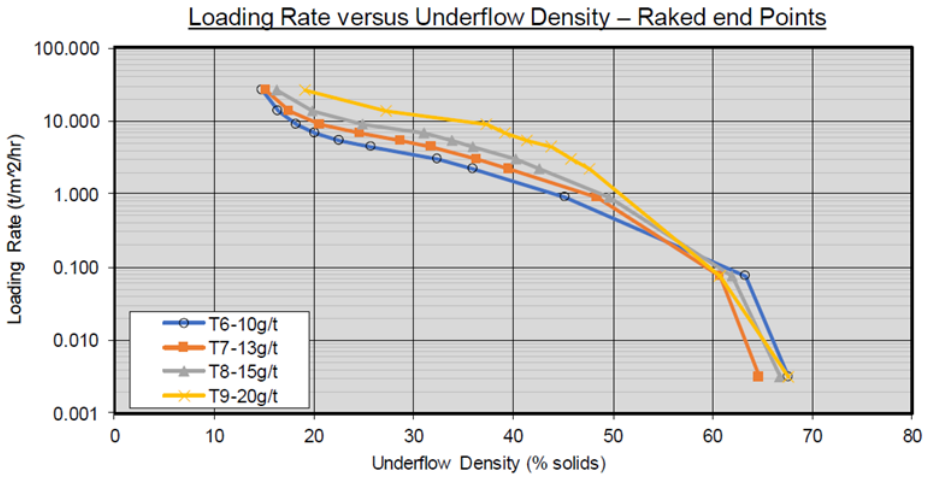


Figure 13.20 Loading Rates vs Underflow Density

The tests were repeated in larger two litre graduated cylinders. The cylinders were raked at one hour and at 24 hours to better estimate the output densities.

A pressure filtration test T-157 was undertaken utilizing a Micronics pressure filter unit. The apparatus simulates a plate and frame style pressure filter, with the three cycles, namely fill, press to 15 Bar with a rubber membrane and air blow drying. The filter press was filled with a diaphragm pressure pump and the filtrate was assessed for total suspended solids. The variables investigated included chamber thickness and feed pump pressure. Blow times of one minute and three minutes were assessed for each test.

The concentrate was fed at a nominal 61% solids by weight. The concentrate SG was measured at 3.96 and the particle size was P_{80} of 15.4 µm.

Test	Chamber Thickness mm	Feed Pressure psi	Cake Moisture %		Filtration Rate - kg/m²h	
			Blow 1 min	Blow 3 min	Blow 1 min	Blow 3 min
F1	35	60	14.3	10.8	666	526
F2	25	75	11.6	9.0	620	465
F3	25	90	12.0	9.2	636	475
F4	25	75	10.8	8.8	566	432
F5	40	75	13.0	10.6	831	644
F6	35	60	12.4	10.1	744	576
F7	35	75	12.4	9.8	758	581
F8	35	90	12.9	10.3	816	635

Table 13.20 Pressure Filtration Results

The test results, summarized in Table 13.20, indicated that thinner chamber thicknesses and longer blow times resulted in the lowest cake moistures, generally less than 10%. These parameters resulted in lower cake production rates.

13.3.18 Upper Copper Zone Tailings Dewatering – Paterson & Cooke

A bulk tailings composite developed by BML was shipped to Paterson & Cooke for dewatering tests and paste generation. A full report by Paterson & Cooke (Bachman et al, 2020), reference GTB-31-0923 can be found in Appendix H. The sample consisted of two separate streams, namely rougher tailings and a cleaner scavenger tailings for recombination to produce a final fails for dewatering work. (Table 13.21).

Paterson & Cooke sized the relevant streams with the following reported data:

Parameter	Rougher Tailings	Cleaner Tailings	Calculated Blended Tailings
d_{90} particle (µm)	67.5	16.8	60.4
d_{50} particle (µm)	22.8	5.1	13.7
% passing 20 µm	45.9	93.4	58.7

Table 13.21 Tailings Recombined Final Tail Size

- Flocculant screening tests were conducted for the selection of flocculants and if required a coagulant. The outcome from the tests indicated that the material is colloidally unstable (settles naturally). Hence it did not require a coagulant. Nalco 9602 flocculant was selected as the optimum tested flocculant in regard to the achieved underflow density;
- Dynamic thickener test results shown in Table 13.22 were conducted in batch mode using a range of flocculant dosages to select an optimum dosage, followed by a range of solids loading rates to assess the impact of thickener loading on the thickener performance.

Test	Unit	1	2	3	4
Solids Loading Rate	t/m^2h	0.477	0.305	0.928	0.773
% Solids - Thickener U/F	% Solids	64.0%	63.7%	66.0%	64.8%
Flocculant Dosage	g/t	47.5	43.8	38.6	41.2
Clarity of Thickener O/F (Clarity Wedge)	NTU	38	31	131	14
U/F Un-Sheared Vane Yield Measurement	Pa	25	28	13	32

Table 13.22 Dynamic Thickener Testing – Final Tails

- A design valve for solids loading rate of 0.7 t/m^2h was adopted for the design criteria with a lower design solids underflow at 60% solids;

- Vacuum filtration tests were conducted to replicate vacuum disk filtration technology. These were conducted over three feed solids densities, using standardized total cycle times to develop the cake filtration rates in Table 13.23 and Figure 13.21.

Based on the thickener underflow design criteria of 60% solids and average filtration rate of 430 kg/m^{2}h, the design rate is discounted to 75% of the test value at 322 kg/m^{2}h. This discount is applied to allow for cloth blinding by fines not experienced in the lab tests.

Feed Density % Solids	54.7%			60.1%			69.5%		
Cycle Time (sec)	Average Loading Rate (kg/m^2 h)	Cake Moisture (% solids)	Cake Thickness (mm)	Average Loading Rate (kg/m^2 h)	Cake Moisture (% solids)	Cake Thickness (mm)	Average Loading Rate (kg/m^{2}h)	Cake Moisture (% solids)	Cake Thickness (mm)
60	405	16.2%	3.27	565	16.1%	4.62	769	15.7%	6.50
90	325	16.1%	3.75	456	15.8%	5.49	591	15.7%	7.78
120	220	16.6%	4.90	269	15.7%	6.64	411	15.3%	10.14

Table 13.23 Final Tailings Filtration Rates – kg/m^{2}h

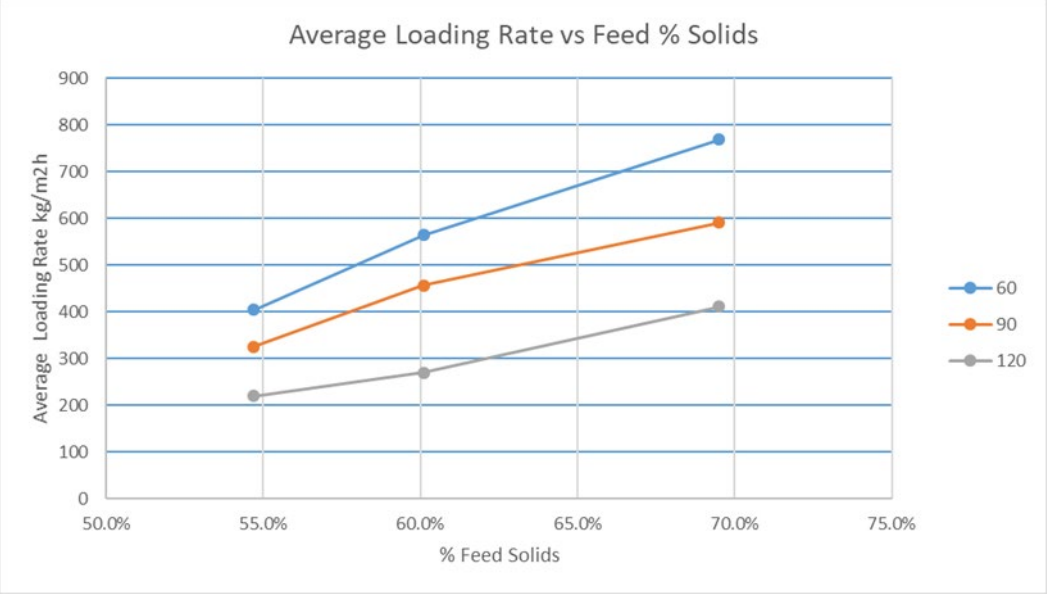


Figure 13.21 Average Loading Rate vs Feed % Solids

Rheology testing was also conducted on the final tails sample to determine the static yield stress of the paste. The thickened underflow sample was blended with 5% Lafarge Portland Cement and tested over a range of thickened % solids (Table 13.24 and Figure 13.22).

Mass Concentration	Boger Slump (78 mm Cyl) (mm)	Static Yield Stress (Pa)
81.47%	12	658
81.18%	15	531
80.54%	20	391
79.88%	25	279
78.74%	33	163
77.58%	46	98

Table 13.24 Rheology Testing on Final Tails Blend

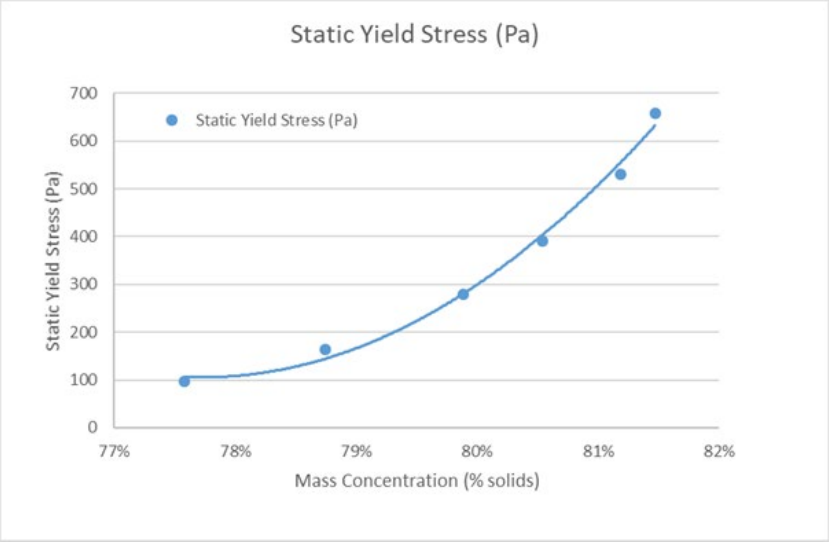


Figure 13.22 Static Yield Stress – Final Tails Paste

Pump requirements for paste backfill would target a yield stress of less than 200 Pa, hence a selected paste generation below 79.5% will be adequate for design considerations.

13.3.19 Upper Copper Zone Tails Dewatering – Andritz

Andritz was approached to complete solid-bowl and Belt Filter Press (BFP) test work to produce suitable cake for paste generation. The testing was completed on a recombined rougher and cleaner tail sample to produce a plant final tailing.

Andritz completed an independent flocculant screening to determine that SNF EM-430 would aid in dewatering and reduce the effluent solids concentrations for both the Centrifuge and Belt Filter Press (BFP) technologies. Both tests were undertaken at flotation tailings density of 26% solids with no pre-thickening.

Decanter centrifuge bench testing indicated that cake solids in the range of 17% to 20% moisture were achievable. Solids capture was found to be 98% ±1%, while using a flocculant dosage of 110 to 160 g/t.

Decanter tests indicated similar cake moisture development was achieved at 1,500 to 2,000 g-force. Results of the tests are shown in Figure 13.23 and Figure 13.24. Results are displayed as a function of the spin retention time at the relevant g-force and final cake height mm as Spin (g.sec/mm).

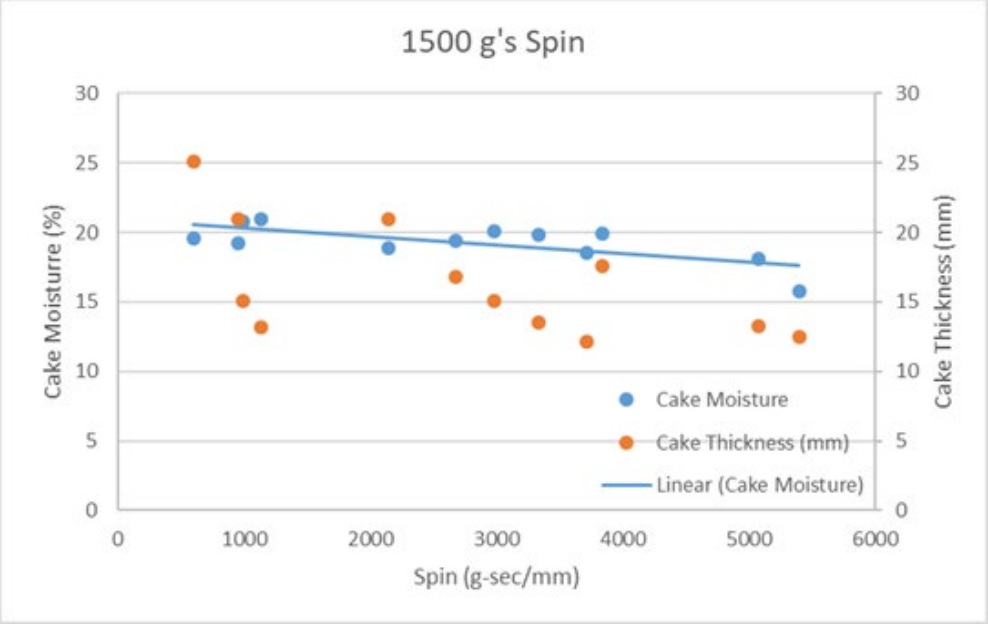


Figure 13.23 Centrifuge 1,500 g's Spin

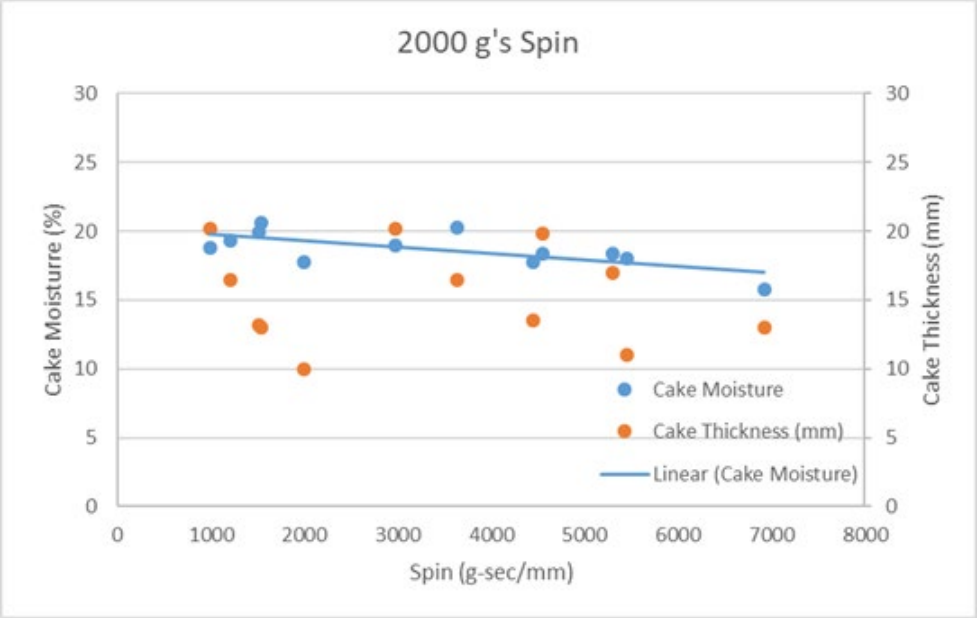


Figure 13.24 Centrifuge 2,000 g's Spin

The concentrate collected from the centrifuge tests contained 0.2% to 0.7% solids and corresponds to a capture rate of 98% ±1%.

BFP testing also developed cake moistures between 17% to 19% based on a 14 mm cake thickness. Flocculant dosage was considerably higher at 230 g/t with the possibility of this being reduced with the BFP using feed thickener underflow at a higher solids loading. Solids capture results were again similar to the centrifuge at 97% ±1%.

13.4 2019 Base Metal Laboratories FS Test Program – Lower Copper Zone

The FS test work program was undertaken at BML in Kamloops BC. The program reported in BL02082 July 2019, consisted of a limited amount of LCZ test. As previously identified the lower zone material has significantly different mineralogical characteristics and has a very good metallurgical response to flotation. Some ore hardness testing, preliminary assessments, and blend testing with the bulk UCZ sample were conducted in this phase of the study.

13.4.1 Lower Copper Zone – Comminution Testing

Three samples were provided for LCZ comminution testing with results summarized in Table 13.25. SMC and Bond ball mill testing were performed on each sample. The SMC tests were performed first. The rejects of the SMC test were crushed to a nominal 3.35 mm, and a BBWi test was conducted on each of the samples. The hardness values are typical for this style of mineralization.

Composite	SMC			BBMWi kWh/t
	A x b	A Dwi kWh/m³x b	SCSE kWh/t	75µm CSS
BBLZ Ore	46.1	6.6	9.93	15.6
BBLZ Waste	81.9	4.8	7.37	14.3
BBUZ Waste	45.6	8.3	9.91	17.0

Table 13.25 LCZ – Comminution Test Results

13.4.2 Lower Copper Zone – Flotation Response

Two tests, a rougher test T-129 and a kinetic cleaner test T-133, were performed on the LCZ sample using the flowsheet and conditions developed for the UCZ.

The rougher test work generated a 19.3% Cu concentrate at an overall recovery of 97.1%, using a coarse primary grind of P_{80} of 53 µm. The test also used both a mild steel mill and media.

Table 13.26 displays the results from the kinetic cleaner test (T-133). The test used a primary grind of 53 µm, using a mild steel mill and stainless steel rod charge. The regrind was conducted in a stirred ceramic bead mill to a P_{80} of 23.9 µm.

The test result confirmed the good overall performance of the composite. No locked cycle testing was conducted on this sample.

Product	Weight		Assay - Percent or g/t				Distribution - Percent			
	%	grams	Cu	Fe	S	C	Cu	Fe	S	C
Cu Con A	8.4	83.9	28.8	28.6	37.7	0.16	33.1	10.9	11.7	1.7
Cu Con A-B	16.9	169.4	29.2	28.4	37.3	0.16	67.8	21.9	23.3	3.3
Cu Con A-C	22.2	223.2	28.7	28.5	37.2	0.18	87.7	28.9	30.6	4.8
Cu Con A-D	24.6	246.9	26.8	28.5	37.1	0.20	90.7	32.0	33.8	6.0
Cu Con A-Scav	25.7	258.0	26.1	28.5	37.1	0.20	92.3	33.5	35.3	6.8
Cu Rougher Con	48.1	483.5	14.6	28.4	37.2	0.43	96.5	62.5	66.4	25.2
Cu Rougher Tail	51.9	521.0	0.5	15.8	17.5	1.18	3.5	37.5	33.6	74.8
Recalc. Feed	100.0	1004.5	7.26	21.9	27.0	0.82	100	100	100	100

Table 13.26 LCZ - Cleaner Flotation Test 133

13.5 FS Upper Copper Zone – Lower Copper Zone Blend Lock Cycle Test

The primary purpose of generating the LCZ composite was to determine the effect of blending UCZ and LCZ mineralization on Cu metallurgy. A blend of 76:24 of the UCZ - LCZ was created for additional testing. A single batch open cleaner test, followed by LCT's, were performed using the UCZ process flowsheet and reagents developed. (Figure 13.25).

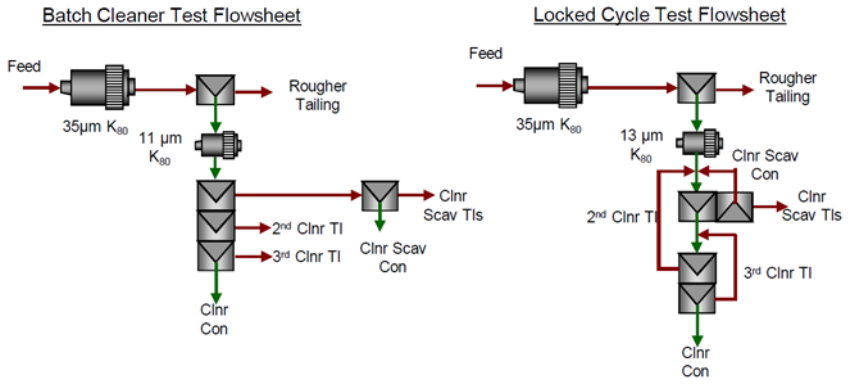


Figure 13.25 UCZ-LCZ Flotation Flowsheet Development

The performance of the composite was good and compares to the earlier test work on the UCZ. Overall, a 20% Cu concentrate at 70% Cu recovery was achievable in the batch open cleaner test (Figure 13.26).

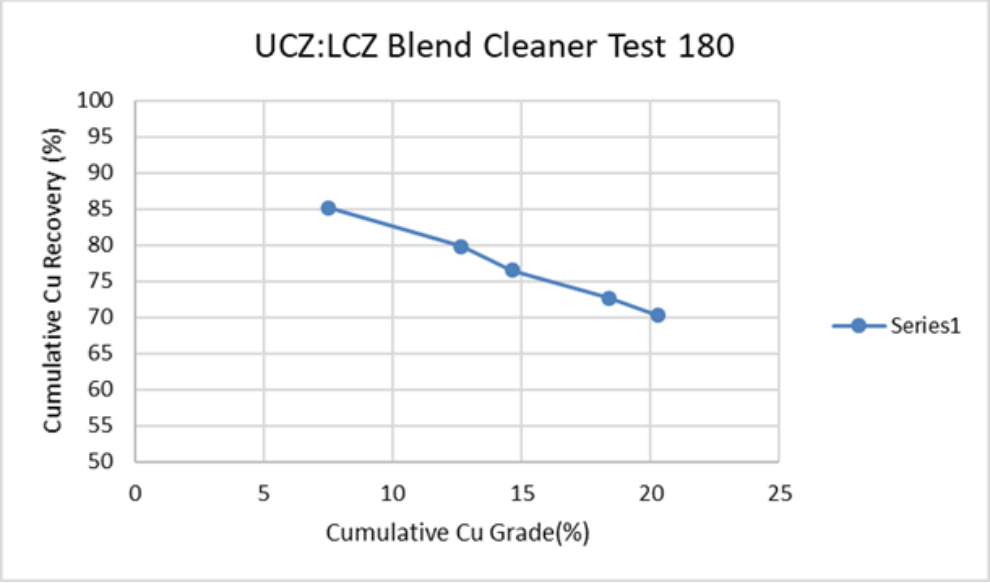


Figure 13.26 UCZ - LCZ Batch Cleaner Test (T-180)

The LCT results, given in Table 13.27, produced a 21.5% Cu concentrate at 93% Cu recovery.

Product	Weight	Assay - Percent				Distribution - Percent			
	%	Cu	Fe	S	C	Cu	Fe	S	C
Cycles E+F									
Feed	100.0	4.28	24.4	29.9	0.35	100.0	100.0	100.0	100.0
Cu Con	18.5	**21.50**	33.4	40.6	0.15	**93.2**	25.4	25.2	7.8
Cu Clnr Tail	18.6	0.79	29.3	35.2	0.38	3.5	22.4	21.9	20.5
Cu Rougher Tl	62.8	0.23	20.2	25.1	0.40	3.4	52.2	52.9	71.8

Table 13.27 LCT Performance

Blending in the LCZ material to the UCZ resulted in a substantial improvement in Cu recovery for the process over treating only UCZ material. This was the expected result and indicated no negative synergies of mixing the two feed sources with vastly different metallurgical performance.

13.5.1 ISA Mill Test Work – Concentrate Regrind

A bulk rougher concentrate for IsaMill™ signature plot test was developed by BML from a global upper zone composite. The composite was floated using the approved locked cycle procedure to produce a rougher concentrate for regrind test work. The testing was carried out at SGS Lakefield labs in Ontario, Canada (Davies and Imeson, 2019).

The sample had a SG of 4.07 g/cm^3 and an F_{80} of 26 µm. The IsaMill™ test was conducted at ~45% solids, and it was shown that a specific energy requirement of 19.6 kWh/t was required to generate a product with a P_{80} of 10 µm. (Table 13.28 and Figure 13.27).

Pass #	Net kW	% Solids	M (t/h)	Cumul. E (kWh/t)	P_{80} (µm)	P_{98} (µm)
Feed	-	-	-	-	26.0	68.3
1	0.57	44.4%	0.142	4.0	18.5	46.5
2	0.58	44.7%	0.145	8.0	14.6	37.8
3	0.57	44.6%	0.143	12.0	12.3	28.7
4	0.58	44.7%	0.142	16.1	10.7	25.3
5	0.58	44.5%	0.139	20.2	9.8	23.1
Specific Energy Requirement @ P_{80}-of 10 µm:			**19.6 kWh/t**			

Table 13.28 IsaMill Signature Test Results

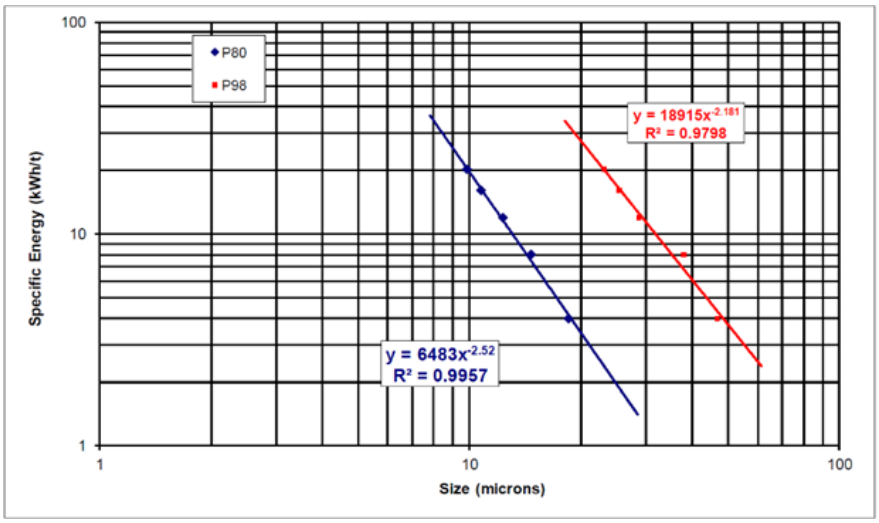


Figure 13.27 Black Butte – T157 – IsaMill Signature Plot

13.6 Geometallurgy

13.6.1 Upper Copper Zone

The UCZ has highly variable ore characteristics that have returned significant variability in Cu recoveries during metallurgical testing. Comparison between mineragraphy observations and metallurgical test results indicated that, once flotation chemistry was optimized, UCZ Cu recovery variability is largely a function of copper sulphide liberation.

Attempts to model the ore characteristics using macro-scale physical observation (categorized proportional geometallurgical logging) showed that physical characteristics define broad trends when plotted against Cu recovery estimates, but these trends were not precise enough to allow for Cu recovery estimation.

Geochemical discrimination, using comprehensive analytical data, between samples with different Cu recovery characteristics was not possible as there is significant geochemical overlap between composites with different Cu recoveries The lack of correlation between metallurgical recovery and ore chemistry supports the interpretation that UCZ Cu recovery is primarily a function of copper sulphide liberation.

Systematic mineragraphy for the 21 metallurgical composites (BBFT01-21) has been completed by Gary McArthur. The mineralogically estimated pyrite content and chalcopyrite liberation characteristics of these composites correlate with cleaner Cu recovery from open circuit variability tests conducted on 19, non-supergene altered metallurgical composites (Figure 13.28). A regression-based formula allows estimation of variability test cleaner Cu recovery using the mineragraphy derived metrics.

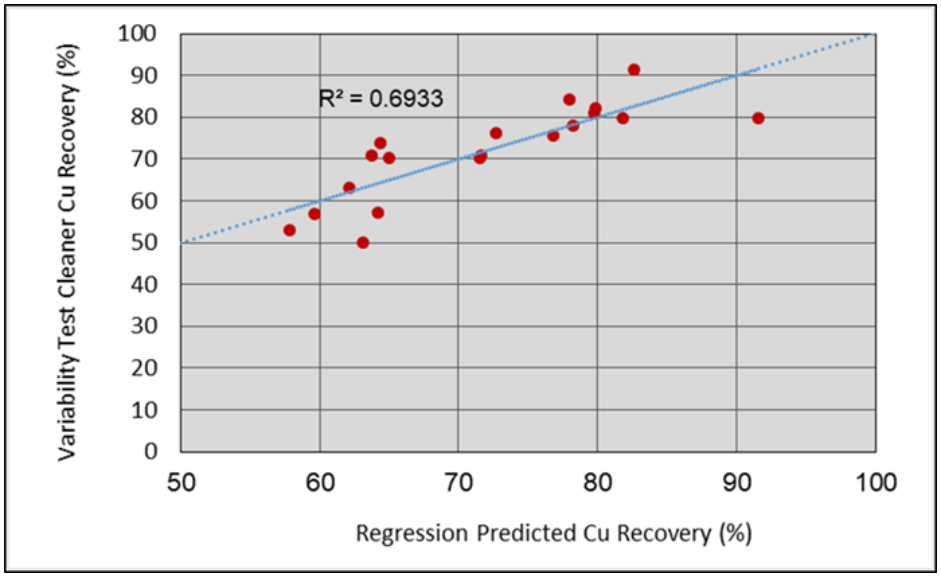
Variability Test Cu Cleaner Recovery = 94.144 + (0.10615*(A+B)) + (-0.28667*(C+D)) + (-0.26708*E)

- A = % Chalcopyrite Interlocked with Marcasite/Siegenite
- B = % Chalcopyrite Interlocked with Gangue
- C = % Chalcopyrite in Ternary Grains
- D = % Chalcopyrite in Quaternary Grains
- E = % Pyrite

There is a robust linear correlation (Figure 13.29) between the variability test cleaner Cu recovery and LCT cleaner recovery for the six non-supergene altered metallurgical composites.

Locked Cycle Test Cu Cleaner Recovery = (0.6619 * Variability Test Cu Cleaner Recovery) + 31.231

Using the two formulae above, systematic mineragraphy metrics from 113 non-supergene altered UCZ composites, spaced throughout the UCZ (both laterally and vertically) have been converted into LCT cleaner Cu recovery estimates. IDW2 interpolation of these Cu recovery estimates has been completed to create a geometallurgical Cu recovery block model (Figures 13.30 to Figure 13.33). The geological model boundaries (wireframes) used to constrain this model are identical to those of the UCZ Mineral Resource estimate, allowing integration of the geometallurgical and Mineral Resource models to develop “Recovery” and “Recoverable Cu” Grade fields for the Mineral Resource estimate.



Note: *Data set does not include supergene altered BBFT composites*

Figure 13.28 Cu Cleaner vs Regression Cu Recovery

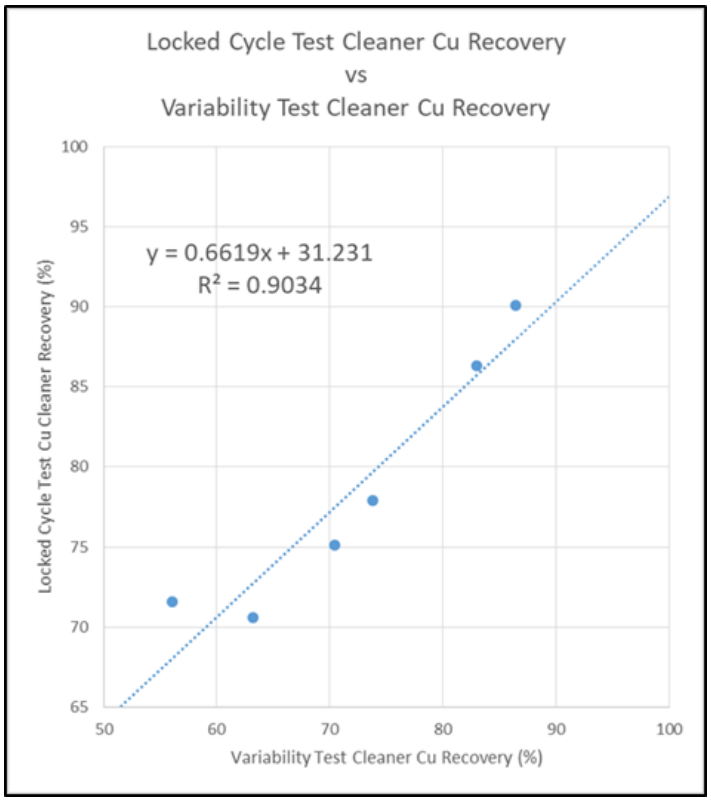


Figure 13.29 LCT Cleaner Recovery vs Variability Test Cleaner Recovery

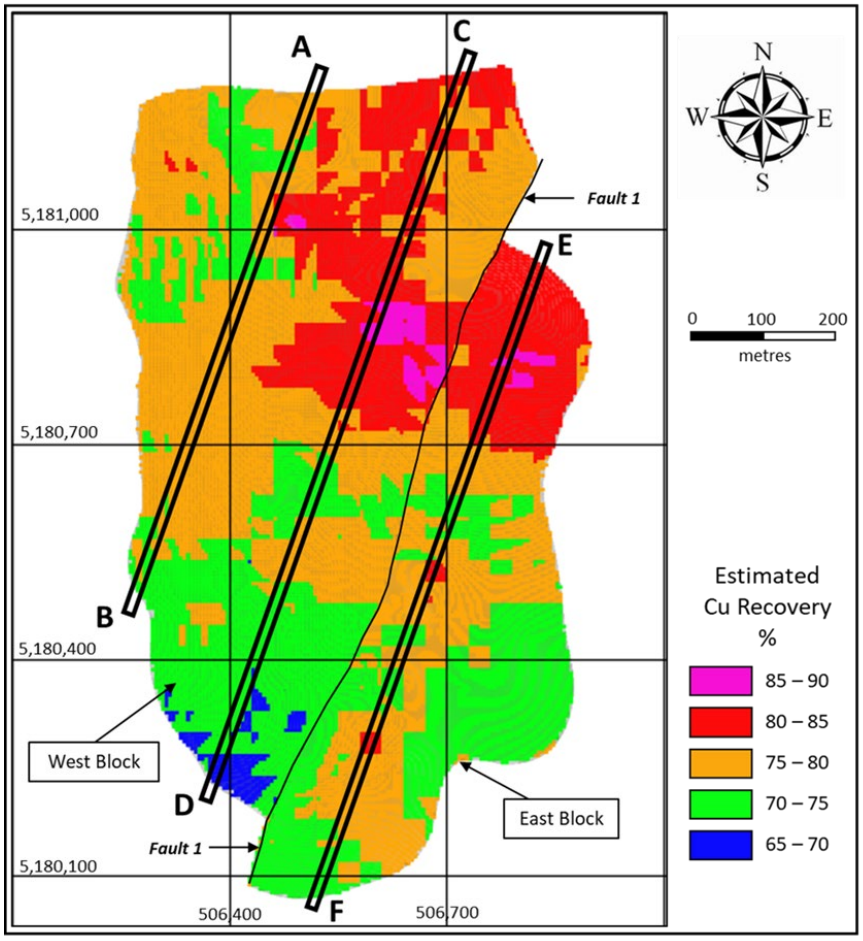


Figure 13.30 Geometallury Block Model

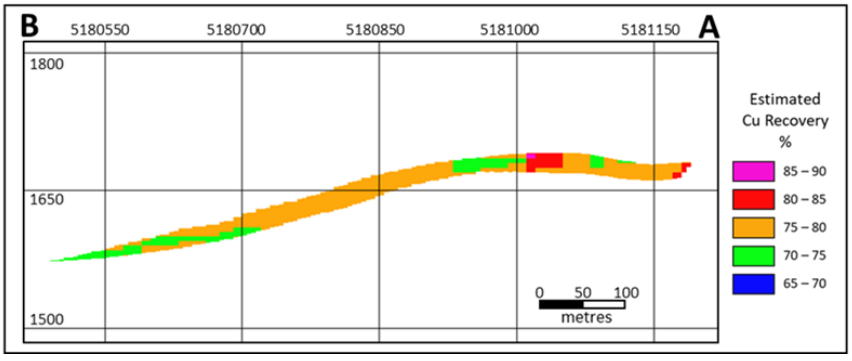


Figure 13.31 Cross-section A to B

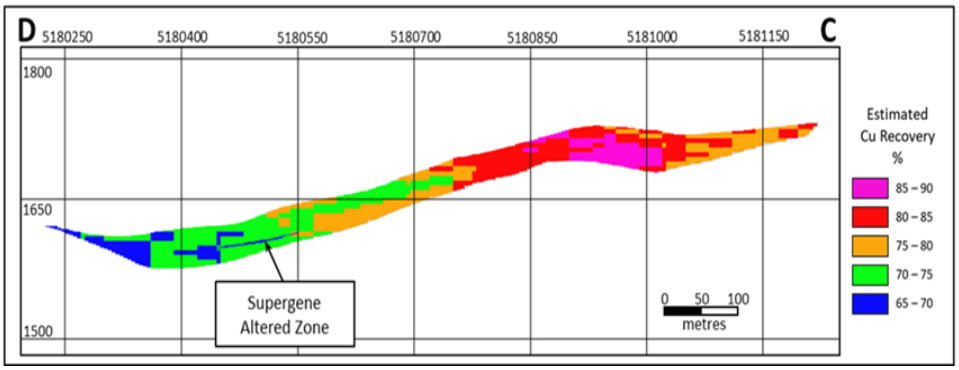


Figure 13.32 Cross-section C to D

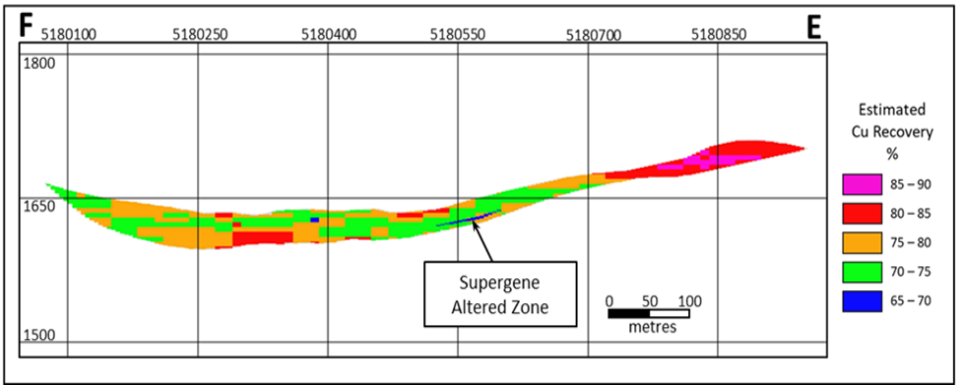


Figure 13.33 Cross-section E to F

With the exception of a small portion of the NE corner of the UCZ the surficial regolith profile does not intersect the UCZ. The portion of the UCZ that is surficially oxidised has been excluded from the UCZ Mineral Resource estimate. Structurally controlled supergene alteration of copper sulphide minerals, below the level of surficial oxidation in the UCZ, occurs along a bedding sub-parallel brittle-ductile shear zone. In this supergene alteration zone, primary copper sulphides (chalcopyrite, tennantite and primary bornite) have been altered to chalcocite, covellite and secondary bornite. The amount of supergene alteration of copper sulphides along the brittle-ductile shear zone is low (generally <5%) and does not materially impact the Cu recovery as estimated using mineragraphy based cumulative yield curves. Where the brittle-ductile shear zone intersects, and is offset by Fault 1, the amount of supergene alteration of copper sulphides exceeds 20% and the impact on cumulative yield curves is significant. Two geological wireframes have been constructed to encapsulate the zone of >20% supergene alteration, see Figure 13.32 and Figure 13.33, one for the West block and one for the East block.

Two open-circuit variability metallurgical tests were performed on composites of >20% supergene altered material (BBFT06 and BBFT13). The two composites returned similar Cu recoveries that were both markedly lower than those of the surrounding non-supergene altered or weakly supergene altered metallurgical composites. A LCT on metallurgical composite BBFT13 returned a cleaner Cu recovery of 69.8%. This Cu recovery estimate has been applied to the modelled >20% supergene alteration zones in the geometallurgical Cu recovery model.

The potentially deleterious concentrations of As in UCZ flotation test work concentrates correlate with the percentage of tennantite that occurs in the metallurgical composite feed. A robust relationship exists between tennatite in feed versus LCT cleaner concentrate As grade Figure 13.34.

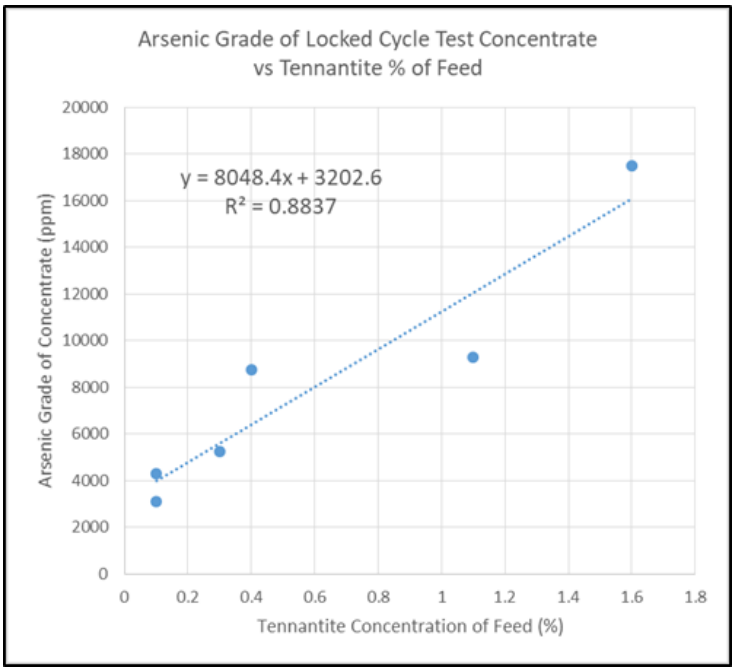


Figure 13.34 As grade (LCT) vs Tennantite % in Feed

The formula below has been used to convert mineragraphy determined tennantite percentages for the 131 UCZ mineralogy composites to estimated grade in LCT cleaner concentrate.

Locked Cycle Test Cleaner Concentrate As Grade (ppm) = (8048.4 * Tennantite %) + 3202.6

IDW2 interpolation has been used to populate the geometallurgical block model with a field containing the estimated As grades expected in concentrate. This data has been integrated with the Cu recovery estimate and the Mineral Resource estimate to form a single block model that is visualized in Figures 13.35 through Figure 13.38.

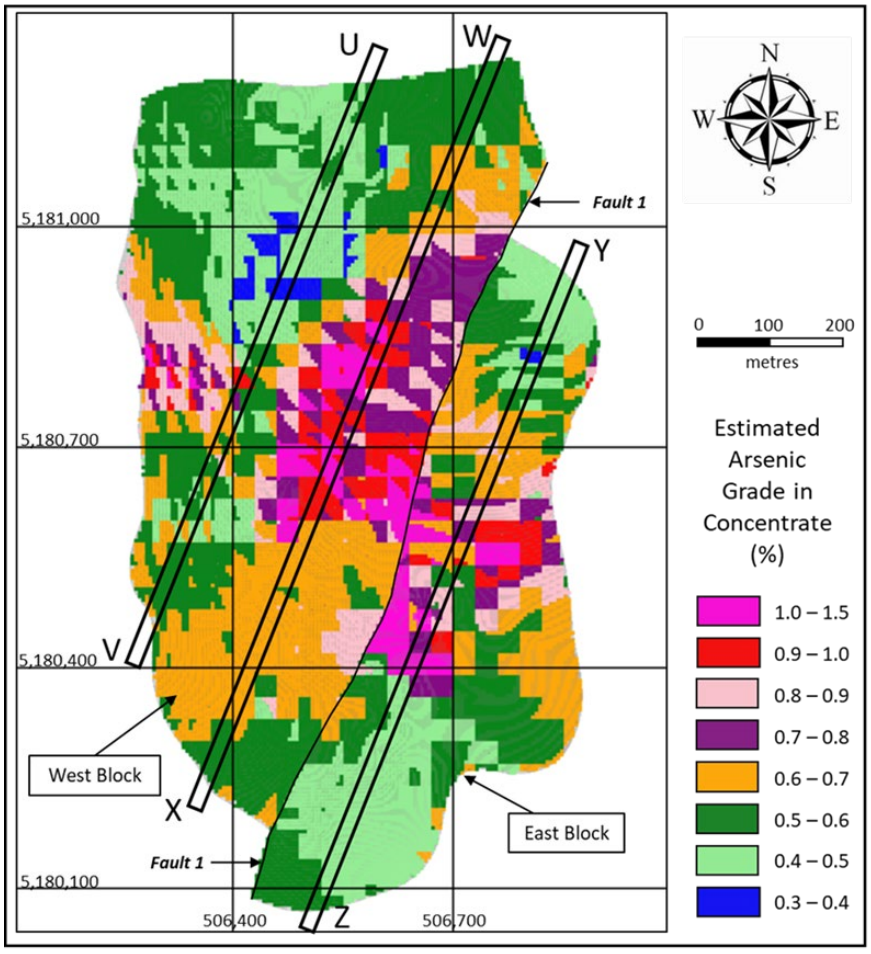


Figure 13.35 Geometallurgy Block model – As Grade in Concentrate

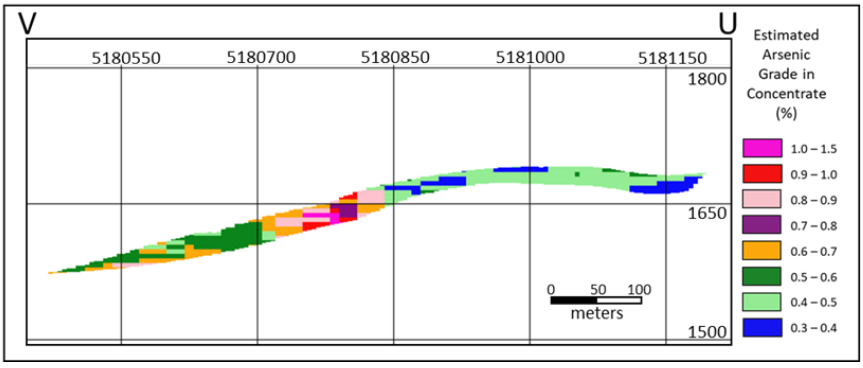


Figure 13.36 Cross-section U-V

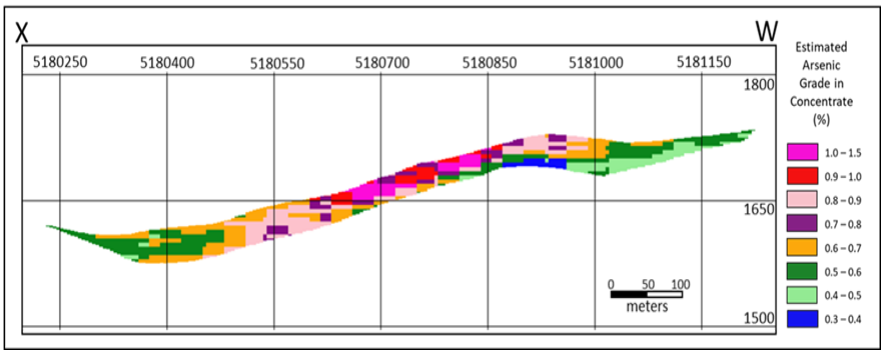


Figure 13.37 Cross-section W-X

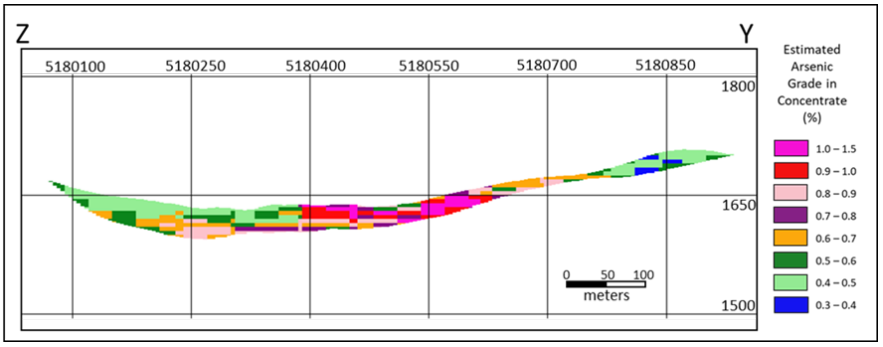


Figure 13.38 Cross-section Y-Z

13.6.2 Lower Copper Zone

Mineralogical investigation of the LCZ composites has shown copper sulphide mineralogy and liberation to be relatively homogenous. Historic and current metallurgical testing has shown metallurgical recovery to be similarly homogenous. Metallurgical test work undertaken for the FS has estimated a 94% Cu recovery for the LCZ using the flow sheet that was developed for the UCZ. A global 94% Cu recovery has been used for the LCZ in the geometallurgical block model.

Copper sulphide mineralization in the LCZ is almost entirely chalcopyrite, on rare occasions trace amounts (<0.1%) of tennantite are observed. The LCZ ore contains 200 – 900 ppm of As, lower than the UCZ which contains 900 – 5,800 ppm As. Locked cycle testing of a LCZ composite blend during the PEA work produced a composite containing 230 ppm As. The low As reporting to concentrate is consistent with the very low tennantite concentrations and with mineralogical observations associating the As in LCZ samples occurring in relict primitive pyrite. For the purpose of the geometallurgical model a global 230 ppm As grade in concentrate has been assigned to the LCZ.

13.6.3 Lowry Zone

Limited metallurgical testing has historically been undertaken on the Lowry deposit. A single batch cleaner test on a Lowry mineralized metallurgy composite, undertaken for the 2013 PEA, recovered 84% of the Cu to a concentrate grading 24.9%.

10 Lowry drillhole composites were submitted for systematic mineralogical analyses (McArthur, 2019). Using the regression derived relationship between mineralogy derived copper sulphide liberation metrics and LCT's for the Johnny Lee UCZ, recoveries at Lowry are expected to range between 78.0 – 91.4% with an average of 86.1%. A global 96% recovery has been used for the Lowry geometallurgical model.

Tennantite concentrations at Lowry are very low, ranging from 0 – 0.1% (McArthur, 2019). Given the similar concentrations to the Johnny Lee LCZ a global As in concentrate value of 230 ppm has been assigned to the Lowry geometallurgical model.

13.7 Concentrate Specification

The process design concentrate values for treatment of UCZ ore were generated from weighted LCT results. The final concentrate conditions yielded a Cu grade of 23.5% and Cu recovery of 80.8%. These figures have been utilized in the process design criteria and mass balance. The weighted As grade was 0.72% As.

The bulk blended UCZ at 76% and LCZ at 24% discussed in Section 13.5 was also sent for final concentrate specification analysis. Key concentrate figures are reported in Table 13.29.

Sample	%Cu	%Fe	%S	%As	Pb ppm	Sb ppm	Zn ppm	Ag ppm	Ba ppm	Bi ppm
UCZ/LCZ Blend	21.8	32.8	37.6	0.514	815	215	548	17	151	314

Table 13.29 UCZ/LCZ – Concentrate Specifications

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14. MINERAL RESOURCE ESTIMATES

The Mineral Resource modelling and estimation for the Johnny Lee and Lowry deposits were reviewed and accepted by Mr. Erik Ronald, P.Geo (APGO #3050) of SRK acting as QP for Mineral Resources. SRK's 2019 Mineral Resource block model and estimation for the Johnny Lee deposit of the Black Butte Copper Project was completed by SRA staff using Datamine Studio RM™ under the guidance of the QP. The final Mineral Resource classification and calculations were performed by Mr. Erik Ronald, P.Geo of SRK using Maptek's Vulcan™ software.

The 2020 Mineral Resource block model and estimation for the Lowry deposit was completed by Mr Erik Ronald, P.Geo of SRK using Geovariance's Isatis™ software for data analysis and variography and Maptek's Vulcan™ software for block modelling, estimation, validation, and reporting of Mineral Resources. Domaining of Cu mineralization was performed by SRA staff using Leapfrog Geo™ software and reviewed by the QP.

14.1 Drillhole Database

The Johnny Lee and Lowry deposit drilling data were extracted from the Sandfire Resources Ltd Structured Query Language (SQL) database as seven comma-separated value (.CSV) files with collar, survey, geology, assay, density, recovery, and structure information exported. These were subsequently imported into Datamine Studio RM™ version 1.4.132.0.

A total of 188 drillholes have been used to inform the 2019 Johnny Lee Mineral Resource estimate including historic drilling and more recent drilling between 2010 and present (Table 14.1). A total of 51 drillholes have been used for the 2020 Lowry Mineral Resource estimate (Table 14.2). A full description of drilling procedures, sample preparation, sample analysis and QA/QC is presented in Sections 10 and 11 of this Report. Information relating to data management and the validation of data is presented in Section 12.

Drilling Campaign	No. of Holes	Lode intercept metres
Historic Drilling	31	498
SRA 2010	5	99
SRA 2011	60	1,612
SRA 2012	44	1,185
SRA 2014	11	307
SRA 2018 Phase 1 & 2018/19 Phase 2	37	943

Table 14.1 Drillholes informing the 2019 Johnny Lee Mineral Resource estimate by drilling campaign

Drilling Campaign	No. of Holes
Historic Drilling	16
SRA 2010	1
SRA 2011	17
SRA 2012	17

Table 14.2 Drillholes informing the 2020 Lowry Mineral Resource estimate by drilling campaign

14.1.1 Geometallurgical data

A total of 125 UCZ composites, covering the entire UCZ, have been investigated using systematic particle mineralogy (McArthur, 2017, 2017a; 2018; 2019). Mineralogical observations from these composites allowed the >20% supergene altered areas to be modelled. Mineralogically-determined copper sulphide liberation estimates (from non-supergene altered UCZ ore) were used, along with the metallurgical test data from variability and LCT work to develop a regression-based relationship between liberation and expected recovery.

After removal of the >20% supergene altered mineralogy composites and compositing six pairs of narrow-interval composites (that occur adjacent to each other in the same drillholes), a total of 113 mineralogy composites are available for Cu recovery estimation. Using the regression-based relationship, fresh rock UCZ Cu recoveries range from 68.2% to 87.9% with an average recovery of 77.4%. A recovery of 94% has been applied for the LCZ.

Ten mineralogical composites from the LMCZ have been investigated (McArthur, 2019). Using the regression-based relationship derived for the UCZ, an average Cu recovery of 86% is estimated for the Lowry deposit in both the LMCZ and the LLCZ.

14.2 Geologic Model

A 3-D lithostratigraphic and structural framework model of the Johnny Lee deposit has been developed. Within the framework of that model, Resource wireframes have been constructed for the UCZ and LCZ using Seequent's Leapfrog Geo™ v4.4 software.

14.2.1 *Lithostratigraphic and Structural Framework Model*

The development of a 3-D lithostratigraphic and structural model is an important element in ensuring that Resource wireframes are consistent with the geological framework within which the orebodies occur. A framework model is also a valuable input into geotechnical studies, where different rock types (such as those in the Johnny Lee deposit) exhibit various geomechanical characteristics and where structural elements (folds, faults, shear zones) locally modify those characteristics. In mining areas where rock types outside of the orebodies show markedly different densities, a framework model is used to assign densities to different lithotypes such that more accurate development and haulage metrics can be estimated.

As detailed in Section 7.2.2, a 3-D lithostratigraphic and structural framework model has been developed for the Johnny Lee area using Leapfrog Geo™. This model encompasses all areas of proposed mine development as outlined in this FS associated with the Johnny Lee deposit.

The Lowry deposit contains no Mineral Reserves, therefore does not include mining as part of the FS and has a much lower density of drilling than the Johnny Lee deposit. A lithostratigraphic and structural framework model has not been developed for the Lowry deposit.

14.2.2 *Johnny Lee Upper Copper Zone*

The approach utilized for modelling the UCZ is to use the Cu analyses as the principle wireframe constraint but to ensure that the wireframes respect the geological framework and local lithological logging (shale and conglomerate Lenses). Multiple mineralogical investigations were conducted to determine whether variations in Cu-bearing minerals and their respective properties could be beneficially modelled. Ultimately, it was concluded that total Cu grade provided the most robust volume for estimation purposes due to the variable nature of Cu mineralization within the UCZ. Multiple Cu values were trialled based on natural breaks in slope or data with a 1.2% Cu shell ultimately providing appropriate mineralization breaks and continuity of mineralization within the UCZ. At grade shells <1.2% Cu grade, mineralization appears discontinuous and incorporates discontinuous pods of potentially uneconomic material <0.5% Cu, therefore the basis for the UCZ was selected at 1.2% Cu.

Various compositing methods and intervals were investigated with straight composites exhibiting minimal to no dilution and were the preferred method selected. The raw >1.2% Cu data showed the same overall zonation as the composited data but allowed accurate termination of the modelled wireframes at lithological boundaries.

The mineralized zones were modelled using the raw analytical data at a 1.2% Cu boundary cut-off grade. In order to create realistic geological wireframes using this approach, it was essential to incorporate intervals of <1.2% Cu mineralization within the wireframe. Strict rules regarding inclusion of certain maximum widths of internal low-grade material have not be applied during wireframe generation but rather, geologically realistic solids have been modelled that incorporate variable amounts of <1.2% Cu mineralization.

Modelling of the >1.2% Cu mineralized zones was conducted using Interval Selection and the Vein System modelling protocols in Leapfrog Geo™. Mineralized intervals >1.2% Cu were assigned a Vein ID (Interval Selection). The Vein System modelling protocol uses Radial Base Function (RBF), implicit interpolation to create a vein solid by linking HW and FW contacts from intervals that have been assigned the same Vein ID. Vein terminations occur where data–defined trends of HW and FW contacts cause the surfaces to cross or where drilling data shows that the >1.2% zone of mineralization pinches out (i.e., a Vein ID has not been interval selected in a drillhole and the pinch-out function of the Vein System protocol forces the HW and FW contacts to cross, using data defined trends, between the drillholes that contain the Interval selected Vein and the one that does not). Limited use was made of polylines (strings and points) to refine the Vein solids.

The >1.2% Cu mineralized zones (Veins) within the UCZ are contained within an envelope of >0.25% Cu Halo mineralization. Modelling of the Halo solids was completed in Leapfrog Geo™ by using points to define contacts for the HW and FW of the Halo and then using RBF implicit modelling to construct HW and FW surfaces. Where data defined trends of HW and FW surfaces did not produce geologically realistic terminations, polylines (strings) were used to pinch-out the Halo zone.

Both host lithostratigraphy and mineralization are truncated in the north by the Volcano Valley HW Fault (Figure 14.1). A master fault plane was constructed for the Volcano Valley HW Fault using points and implicit modelling. This surface was used to truncate both lithostratigraphy and Resource wireframes (Vein and Halo solids).

The UCZ is transected and offset by the N/NE striking, steeply west dipping Fault 1, the master fault plane has been modelled in a similar manner to that of Volcano Valley HW Fault (Figure 14.1). Fault 1 has been used to divide the lithostratigraphic model into East and West blocks. Lithostratigraphic units and Resource wireframe domains in the East and West blocks have been modelled separately and are truncated by Fault 1 (Figure 14.1 and Figure 14.2).

The UCZ Vein models for the East and West blocks are entirely encapsulated within the Halo solids (Figure 14.2). The Halo solids are mostly encapsulated within the USZ but >0.25% Cu mineralization, and the Halo solids that enclose it, does transgress the upper and lower contacts of the USZ into the shale unit that occurs in the HW and FW (Figure 14.1 and Figure 14.2). The >1.2% Cu Vein solids are entirely contained within the USZ.

A base of oxidation surface has been constructed (using points selected on drillholes and implicit modelling in Leapfrog Geo™) for the area included in the lithostratigraphic model (Figure 14.3). The base of oxidation was defined as the base of any recognizable oxidation associated with surficial weathering, so includes any material that is at least partially weathered. The extreme NE corner of the UCZ is oxidized to a depth of 13 m below surface. The portion of the UCZ that is surficially oxidized has been excluded in the Mineral Resource estimate presented here.

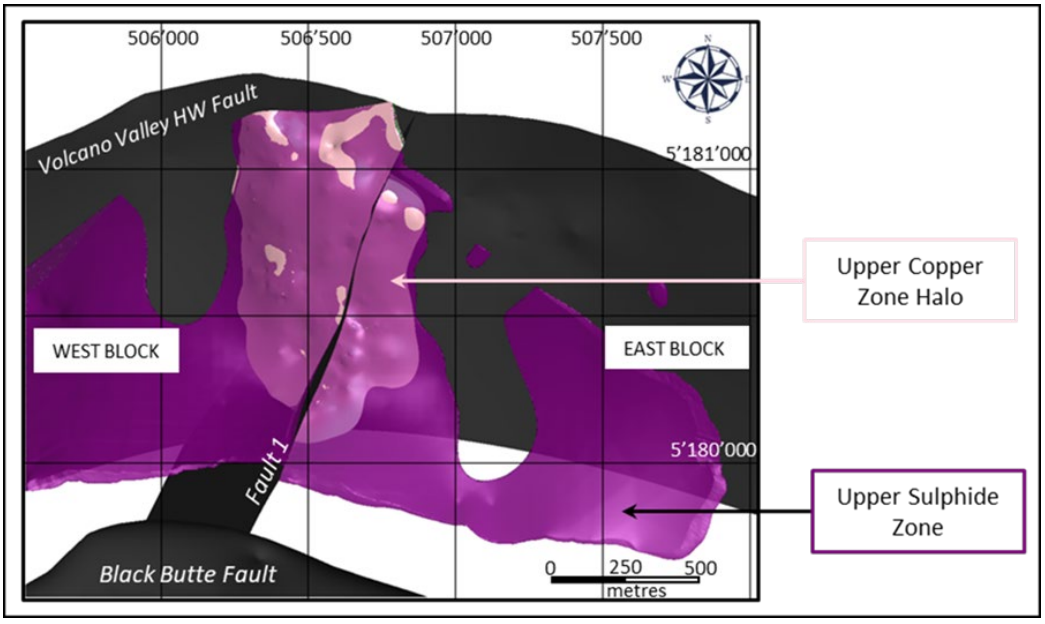


Figure 14.1 Plan view of the USZ (semi-transparent) and UCZ Halo models for the East and West blocks showing faults that truncate and offset the UCZ

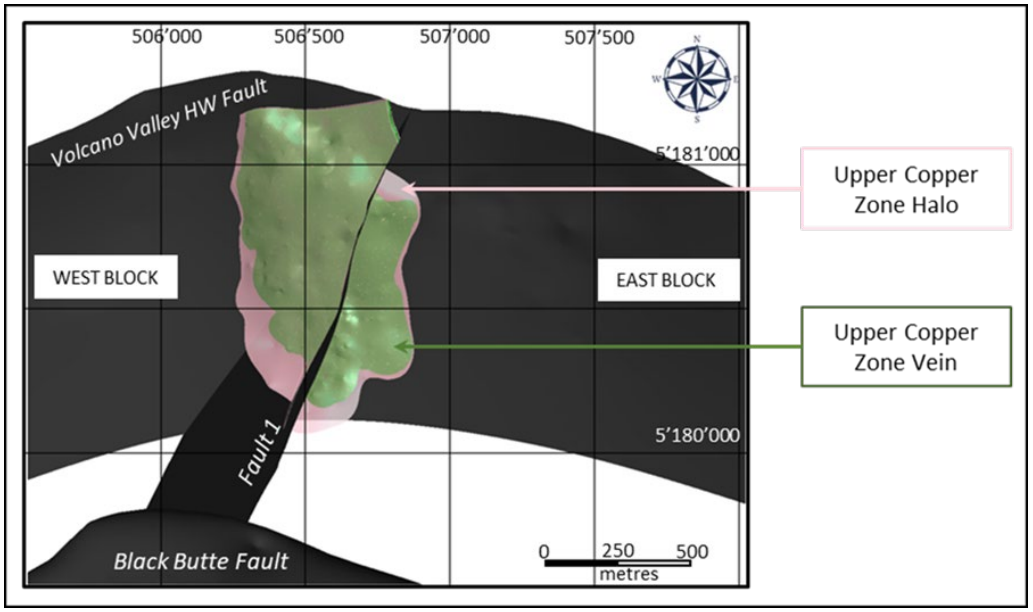


Figure 14.2 Plan view of the UCZ Halo (semi-transparent) and Vein models for the East and West blocks

Although the majority of the UCZ is unaffected by surficial oxidation, localized supergene alteration of copper sulphides in the UCZ has occurred at depth where structures have allowed acidic waters to penetrate the orebody (see section 7.3.3). Whilst the degree of supergene alteration is generally low (<5%), there is one area where it exceeds 20% and it materially impacts the metallurgical behavior of the mineralization as demonstrated during flotation test work. The area where this occurs is at the intersection of a bedding parallel, brittle-ductile shear zone and Fault 1. Two supergene altered solids have been modelled that encapsulate the zone of >20% supergene alteration on either side of Fault 1 (Figure 14.3).

The surfaces and solids used to directly constrain the Mineral Resource estimate for the UCZ are listed in Table 14.3.

File Name	Solid/ Surface	Description
190427_UCZ_E_Block_Halo_PH	Solid	Encapsulates zone of >0.25% Cu Halo mineralization in East Block
190427_UCZ_EBlock_Veins_PH	Solid	Multiple solids encapsulating zones of >1.2% Cu mineralization in East Block
190427_UCZ_W_Block_Halo_PH	Solid	Encapsulates zone of >0.25% Cu Halo mineralization in West Block
190427_UCZ_W_Block_Veins_PH	Solid	Multiple solids encapsulating zones of >1.2% Cu mineralization in West Block
190427_VVF_HW_PH	Surface	Master fault plane that truncates UCZ in the north
190427_Fault_1_PH	Surface	Master fault plane that separates the East Block from the West Block
190514_UCZ_E Block Supergene 1_PH	Solid	Zone of >20% supergene alteration of Cu Sulphides in East Block
190514_UCZ_W_Block Supergene 1_PH	Solid	Zone of >20% Supergene alteration of Cu Sulphides in West Block
190427_Quaternary_Alluvium_PH	Solid	Zone of surficial Quaternary sediment
190427_Base_of_Oxidation_PH	Surface	Base of surficial oxidation

Table 14.3 File names of solids and surfaces utilized for the constraint of the UCZ Mineral Resource estimate

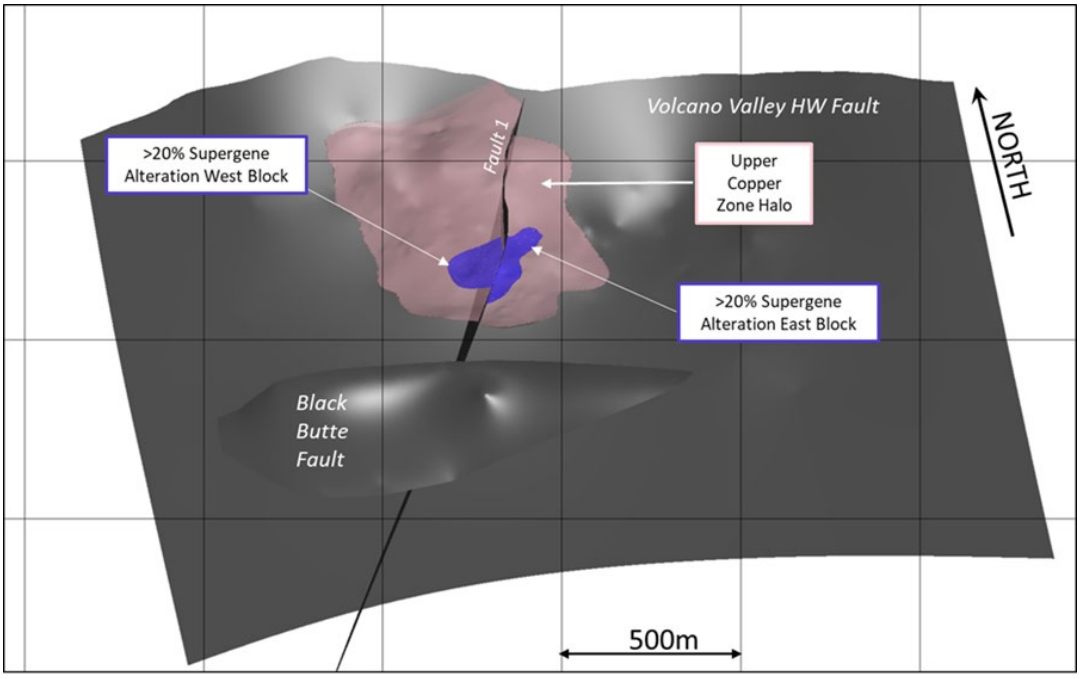


Figure 14.3 Isometric view, looking N/NE along Fault 1, of the UCZ showing the two zones of supergene alteration modelled at the junction of a brittle-ductile shear zone and Fault 1

14.2.3 Johnny Lee Lower Copper Zone

The LCZ, although having more visible copper sulphides than the UCZ, also has portions where fine-grained copper sulphide results in economically significant intersections having little visual contrast with the surrounding lithofacies. A similar modelling protocol to that of the UCZ was adopted for the LCZ. Geological continuity of the LCZ was best achieved using a >2% Cu outer cut-off rather than the >1.2% Cu that was used for the UCZ.

The >2% Cu mineralization shell was used to define the intervals selected for modelling as Veins in Leapfrog Geo™ software. Data-defined trends were utilized to create pinch-outs at Vein terminations. Limited use was made of polylines, in areas of lower drill-density, to ensure Veins followed trends consistent with that of the lithostratigraphic host units.

Three discrete >2% Cu mineralized zones, termed the LCZ West, Central, and East Lenses (Figure 14.4, Figure 14.5, Figure 14.6) were constructed. All Lenses are predominantly enclosed within a bedded pyrite, massive sulphide unit referred to as the LSZ although locally the Lenses extend into the overlying clastic sediment unit (Figure 14.4 and Figure 14.5). The LSZ and adjacent clastic sediment lithofacies were modelled as part of the 3-D lithostratigraphic model.

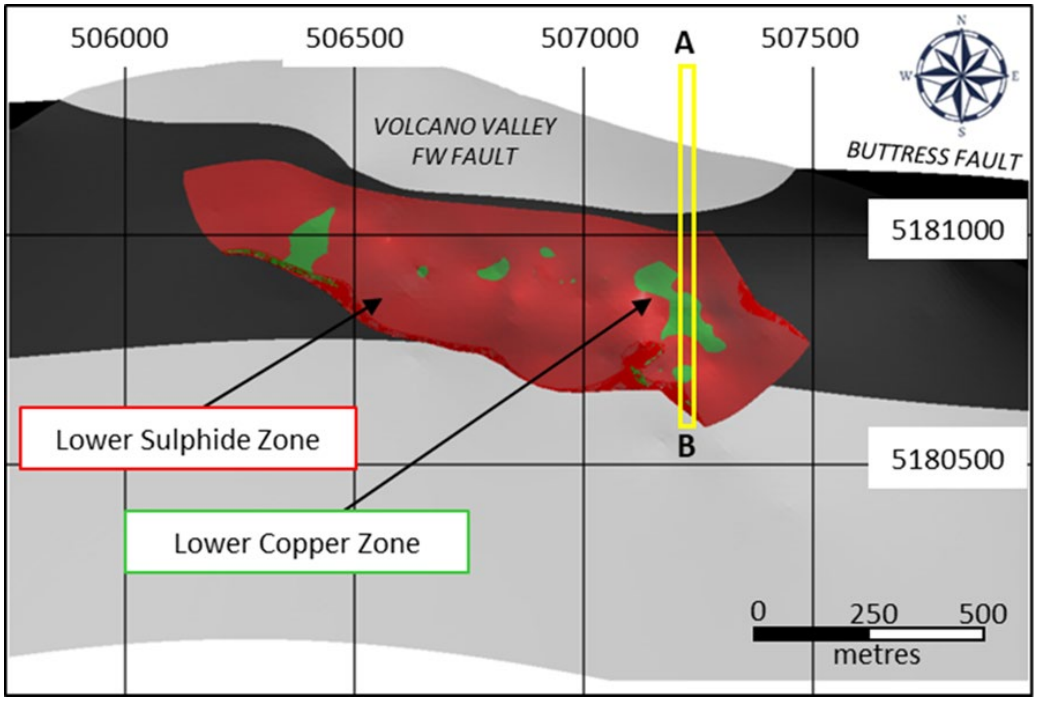


Figure 14.4 *Plan view of the models of the LSZ, LCZ and bounding faults. The VVF has been rendered semi-transparent.*

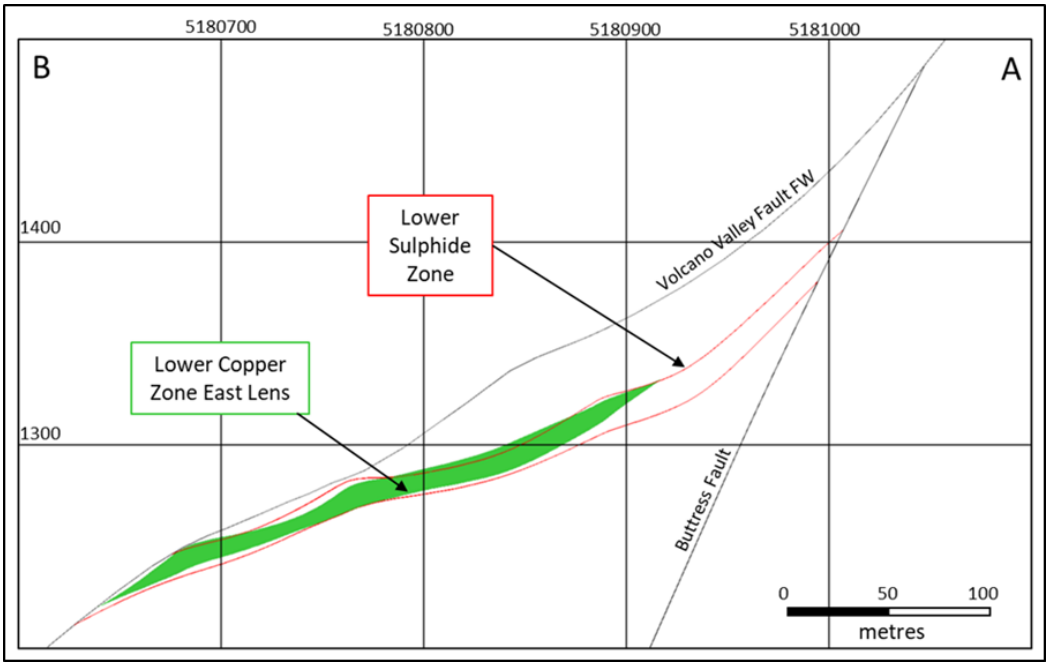


Figure 14.5 *Cross-section from A – B (see Figure 14.4) looking west of the LCZ*

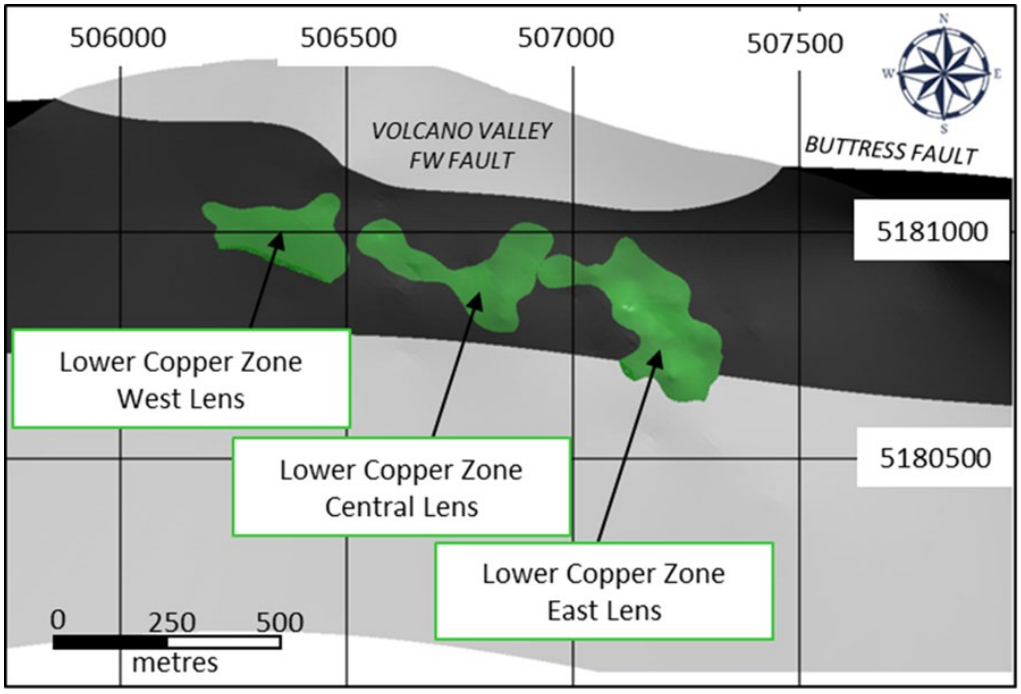


Figure 14.6 Plan view of the three Lenses that form the LCZ. The VVF is rendered semi-transparent

The LSZ is truncated in the north at the Buttress Fault and in the south by the Volcano Valley FW Fault (Figure 14.6). The > 2% Cu mineralization shell that forms the LCZ Lenses does not extend to the Buttress Fault but, in places, does extend south to the Volcano Valley FW Fault, where it is truncated. The master fault planes of the Buttress Fault and Volcano Valley FW Fault were modelled in Leapfrog Geo™ by creating points at drillhole fault intersections and creating a fault surface using radial base function, implicit modelling.

Minor, fault-controlled supergene alteration of the LCZ occurs adjacent to the Buttress and Volcano Valley FW Fault. This supergene alteration does not materially affect metallurgical recoveries.

The surfaces and solids used to constrain the Mineral Resource estimate for the LCZ are listed in Table 14.4.

File Name	Solid/ Surface	Description
190502_LCZ_West_PH	Solid	>2% Cu Mineralization in Lower Copper Zone West Lens
190502_LCZ_Central_PH	Solid	>2% Cu Mineralization in Lower Copper Zone Central Lens
190502_LCZ_East_PH	Solid	>2% Cu Mineralization in Lower Copper Zone East Lens
190502_LCZ_MASU1_PH	Solid	Lower Sulphide Zone
190423_Buttress_Fault_PH	Surface	Master fault plane that truncates USZ in the north
190423_VVF_Footwall_PH	Surface	Master fault plane that truncates USZ and UCZ in the south

Table 14.4 File names of solids and surfaces used for the constraint of the LCZ Mineral Resource estimate

14.2.4 *Lowry Middle Copper Zone*

Drillhole intersection spacing in the Lowry deposit's LMCZ ranges from 40 – 100 m.

The LMCZ is hosted by a succession of ferruginous sediment (massive sulphide and ferruginous shales) with interbedded conglomerate, carbonaceous shale and shale lithofacies. The host unit that contains the majority of the mineralization in the LMCZ has been modelled as a solid (File Name: 191211_MCZ_Host).

Entirely encapsulated within the host unit are three bedding sub-parallel zones of >1.2% Cu mineralization, analogous to the Veins in the Johnny Lee UCZ. These Veins have been modelled as three solids (File Names: 191211_MCZ_Vein_1; 191211_MCZ_Vein_2; 191211_MCZ_Vein_3).

The >1.2% Cu Veins are surrounded by a zone of >0.25% Cu mineralization referred to as Halo mineralization. The Halo mineralization is largely confined to the host unit but does transgress the HW and FW contacts in places. The Halo mineralization has been modelled as a solid (File Name: 191211_MCZ_Halo).

Figure 14.7, Figure 14.8 and Figure 14.9 show the solids that form the LMCZ model. In plan view, the host unit and mineralization are truncated along their eastern margins by the HW Fault of the VVFZ. Figure 14.10 shows a cross-section through the LMCZ model.

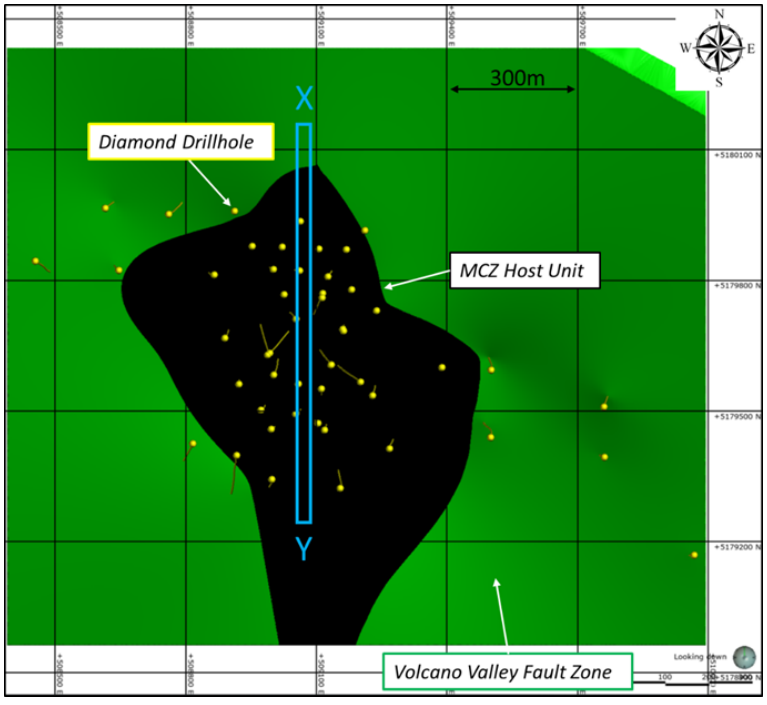


Figure 14.7 Plan View of the LMCZ Model showing the VVFZ and the LMCZ Host Unit. The position of the cross-section X-Y (Figure 14.10) is indicated.

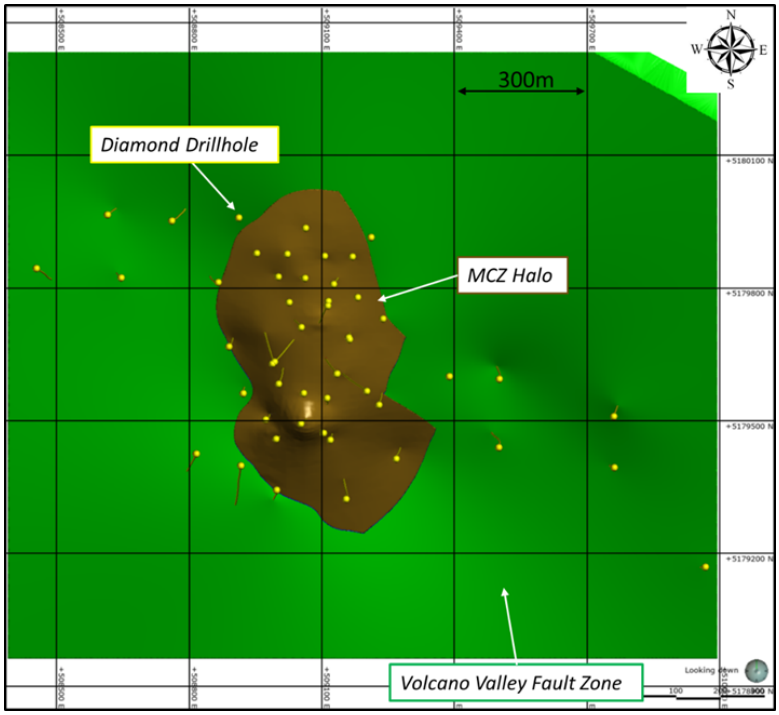


Figure 14.8 Plan view of the LMCZ model showing the VVFZ and the LMCZ Halo

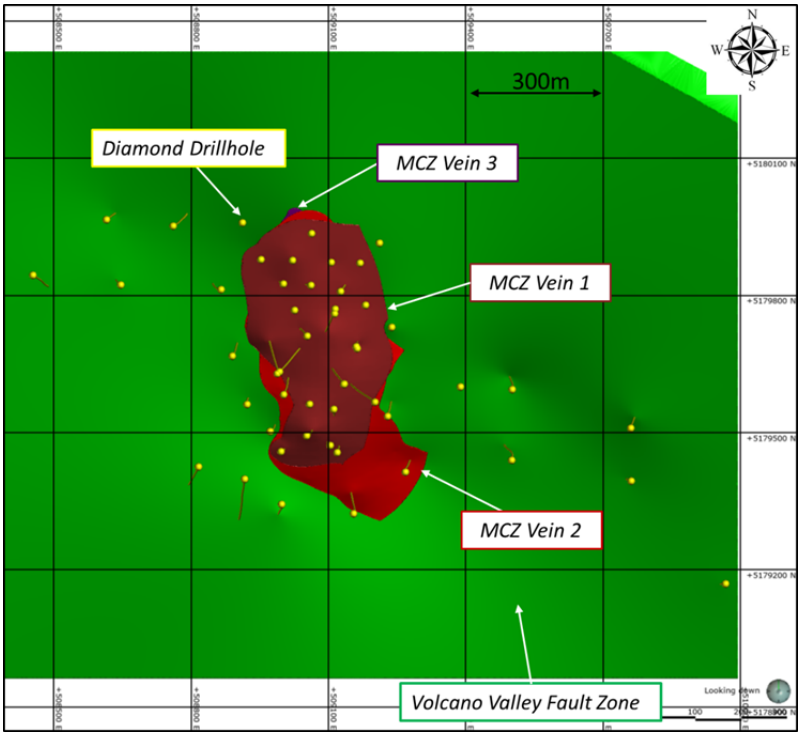


Figure 14.9 Plan view of the LMCZ model showing LMCZ Veins 1 – 3

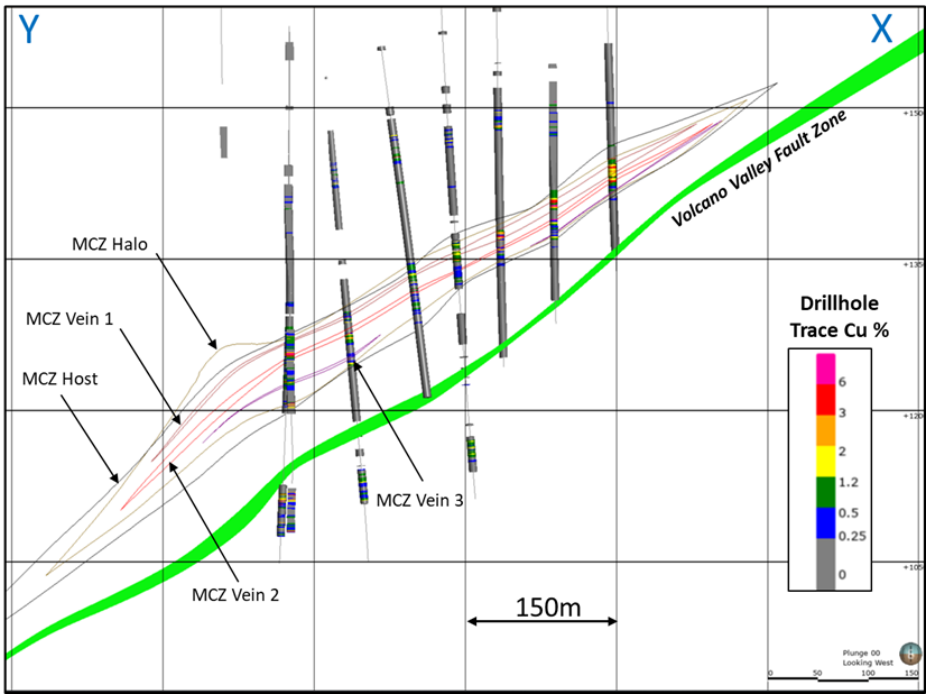


Figure 14.10 Cross-section of the LMCZ model from X to Y (Figure 14.7) looking west

14.2.5 Lowry Lower Copper Zone

Many of the drillholes that intersected the LMCZ were stopped short of the LLCZ, consequently drillhole spacing in the LLCZ is larger than that of the LMCZ, ranging from 60 – 200 m. The LLCZ occurs in the FW of the VVFZ and both the host and mineralization are truncated in the SE by the Volcano Valley FW Fault.

The LLCZ is hosted by a succession of ferruginous sediment (massive sulphide and ferruginous shales) with interbedded shale and conglomerate lithofacies. The host unit that contains the majority of the mineralization in the LLCZ (Figure 14.11) has been modelled as a solid (File Name: 191211_LCZ_Host).

Two bedding sub-parallel zones of >1.2% Cu mineralization, encapsulated within the LLCZ Host unit, have been modelled for the LLCZ (Figure 14.12 and Figure 14.13). The lowermost (File Name: 191211_LCZ_Vein_1) is continuous, the upper (File 191211_LCZ_Vein_1) forms three discrete Lenses.

Although some Halo style mineralization is developed in the LLCZ, the drilling to date has shown this to be localized. No Halo mineralization solids have been modelled for the LLCZ.

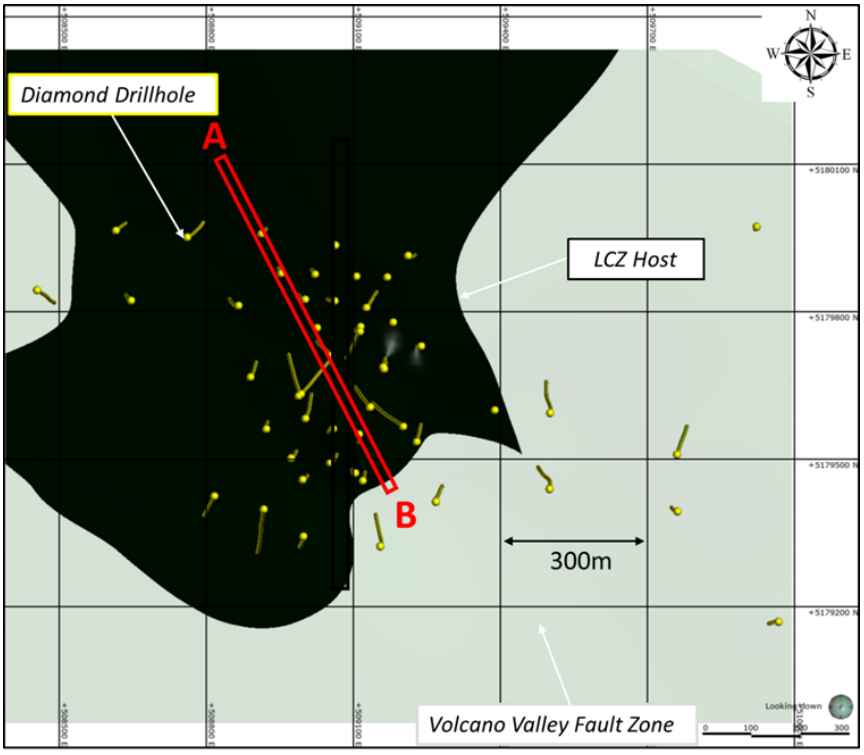


Figure 14.11 Plan View of the LLCZ Model showing the VVFZ and the LLCZ Host Unit. The VVFZ is rendered transparent such that the underlying LLCZ Host solid is visible. The position of the cross-section A-B (Figure 14.13) is indicated.

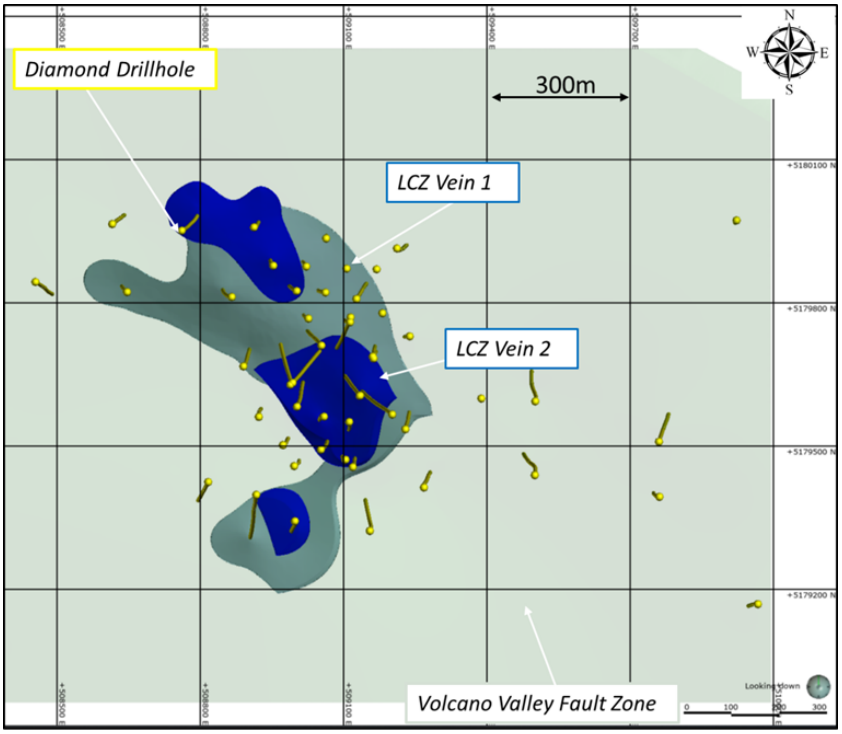


Figure 14.12 *Plan View of the LLCZ Model showing the VVFZ and LLCZ Veins 1 - 2. The VVFZ is rendered transparent such that the underlying Vein solids are visible.*

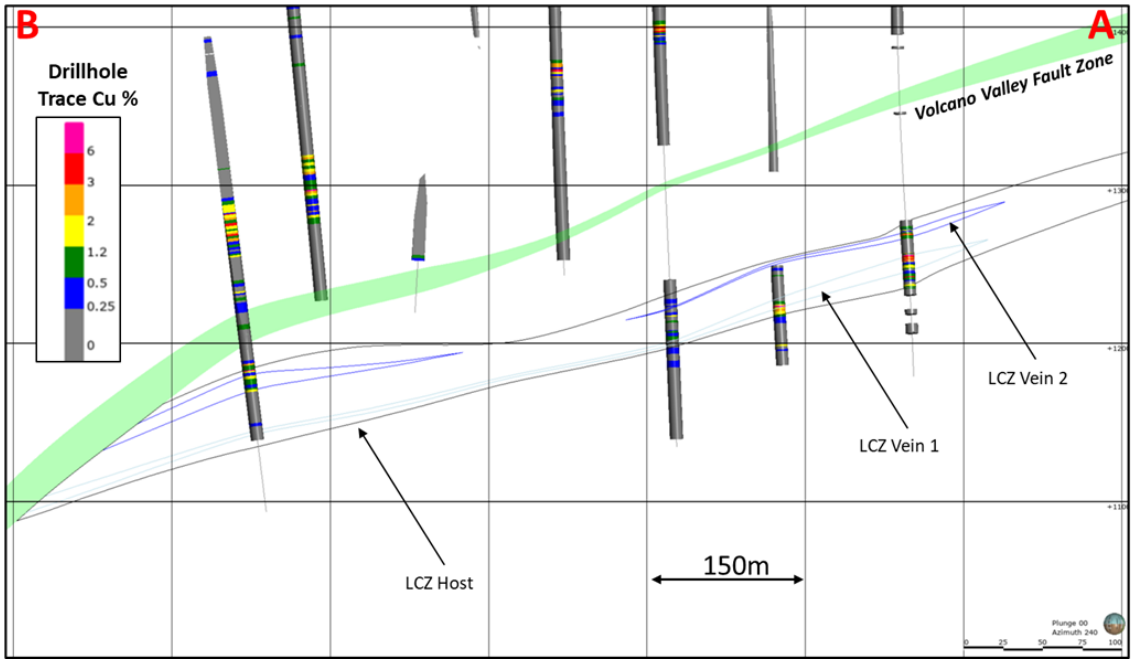


Figure 14.13 *Cross-section of the LLCZ Model along line A-B (Figure 14.11) looking west*

14.3 Density

14.3.1 Johnny Lee Upper Copper Zone

Specific Gravity (SG) displays variability within both strongly (Vein) and weakly (Halo) Cu mineralized material within the UCZ. In particular, SG of the potentially economic mineralization in the northern part of the UCZ was generally lower (by approximately 0.8) than that of the southern part of the UCZ.

Figure 14.14 and Figure 14.15 show material types in diamond drill core in the northern and southern parts of the UCZ. Although the presence of quartz-carbonate Veining, fracturing and oxidation locally contribute to SG variability, the principal driver to the steady increase in SG to the south is the decrease in clastic sediment content of the USZ, the ferruginous unit that hosts the UCZ mineralization.

In the northern part of the USZ, the massive sulphide (for 2018 logging purposes this was defined as >50% sulphide over a >30 cm width) is characterized by an increased proportion of narrow shale and siltstone intercalations within the massive sulphide, compared to the southern part of the USZ.

Historic geological logging has lacked the detail required for domaining of the USZ/UCZ into zones with particular SG characteristics.

Due to the variable nature of density in the UCZ and the fact that the UCZ Vein and Halo domains are not suitable to constrain density data for estimation, it was determined that an appropriate approach is to estimate density constrained by the USZ unit. This allows for a robust geologic framework and density dataset for estimation. The density estimate is split into USZ east and USZ west.

Where UCZ mineralization transgresses the upper and lower contacts of the USZ into the shale unit, a mean SG of 3.3 is assigned based on review of local density data points.

14.3.2 Johnny Lee Lower Copper Zone

In contrast to the UCZ, the LCZ has relatively consistent SG values with the only major difference being between the density of the portions of the Lenses (East, Central and West) that occur within the lower LSZ and the portions that transgress into the overlying clastic sediment. Similar to the approach taken for estimation of density in the USZ, density is estimated into the LSZ. Where LCZ Lenses locally extend into the overlying clastic sediment unit, a mean SG of 3.3 is assigned based on review of local density data points.

Figure 14.14 *Core photographs for SC18-238 (northern part of the UCZ) showing top and base of Halo and >1.2% Cu Veins. Resource sample intervals are labelled with SG measurements. Although robust massive sulphide units have an SG that exceeds 4.0, most of the SG's are in the range of 3.1 to 3.7.*

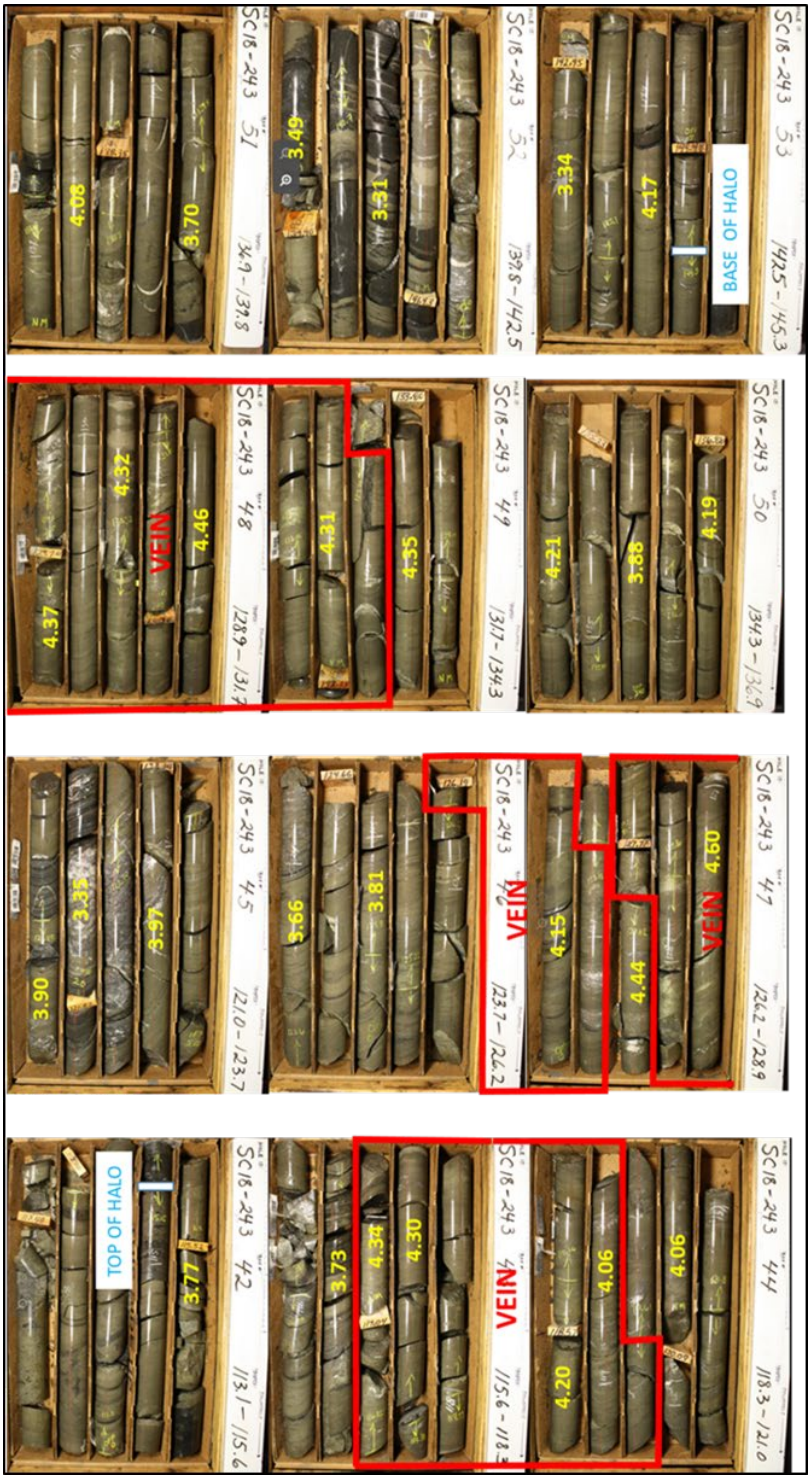


Figure 14.15 *Core photographs for SC18-245 (southern part of the UCZ) showing top and base of Halo and >1.2% Cu Veins. Resource sample intervals are labelled with SG measurements. Much of the UCZ in this area is comprised of robust massive sulphide units that have an SG in the range of 3.7 to 4.4.*

14.3.3 Lowry Middle Copper Zone and Lowry Lower Copper Zone

The host units for the LMCZ and LLCZ comprise variable proportions of interlayered lithotypes (massive sulphide and ferruginous shales with interbedded shale and conglomerate lithofacies) resulting in locally variable SG. Density data within the LMCZ and LLCZ have been estimated in a similar manner to that of the Johnny Lee deposit. Where LMCZ or LLCZ Halo mineralization transgresses the upper and lower contacts of the host unit into the surrounding clastic sediment unit, a mean SG of 3.3 is assigned based on review of local density data points.

14.4 Compositing, Summary Statistics, and Capping

14.4.1 Johnny Lee Compositing

Historic sample intervals range in length from 0.1 to 2.2 m. In recent drilling of 2018 and 2019, sample intervals have been restricted to a range between 0.3 and 1.3 m with the overall sample length distribution presented in Figure 14.16 for the UCZ and Figure 14.17 for the LCZ. After reviewing compositing at average 1.5 and 2.0 m lengths, it was determined that 1.5 m was most appropriate considering the data and geological model. Using Datamine Studio RM™ version 1.4.132.0, data was composited at 1.5 m, with density weighting where data was available, and downhole intervals constrained by domain boundaries. Summary composite length histograms are presented in Figure 14.8 by estimation domain. In each histogram, the small intervals represent residual lengths cut by domain boundaries. Where SG data has been measured on a sample interval basis, the data has been composited as above and then merged with the historic point SG data for estimation (See Section 12.1.6 on verification of point SG data).

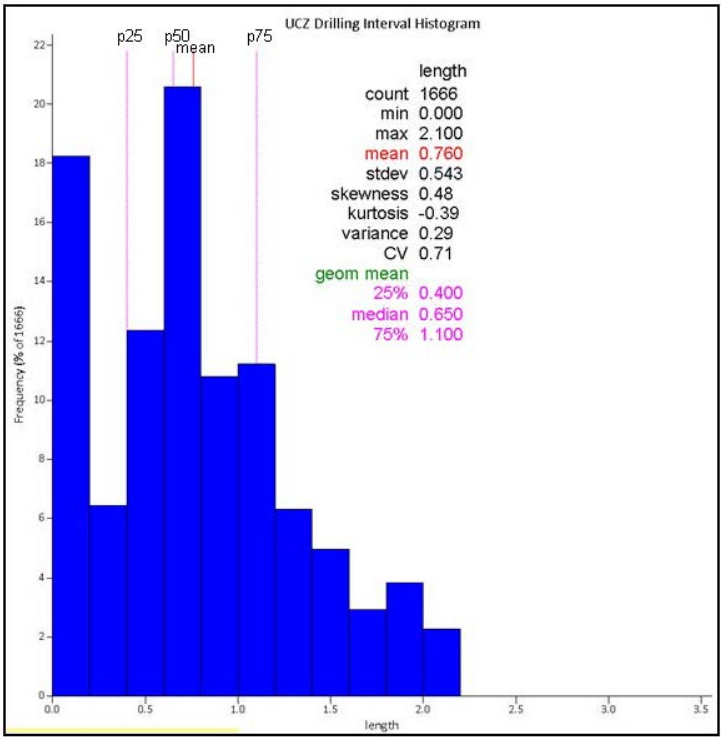


Figure 14.16 Drilling Sample Lengths in the UCZ

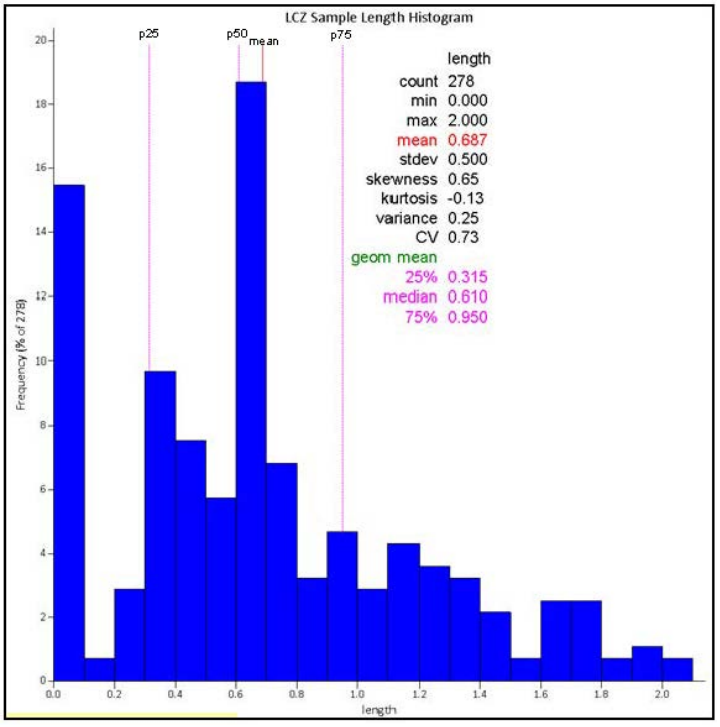


Figure 14.17 Drilling Sample Lengths in the LCZ

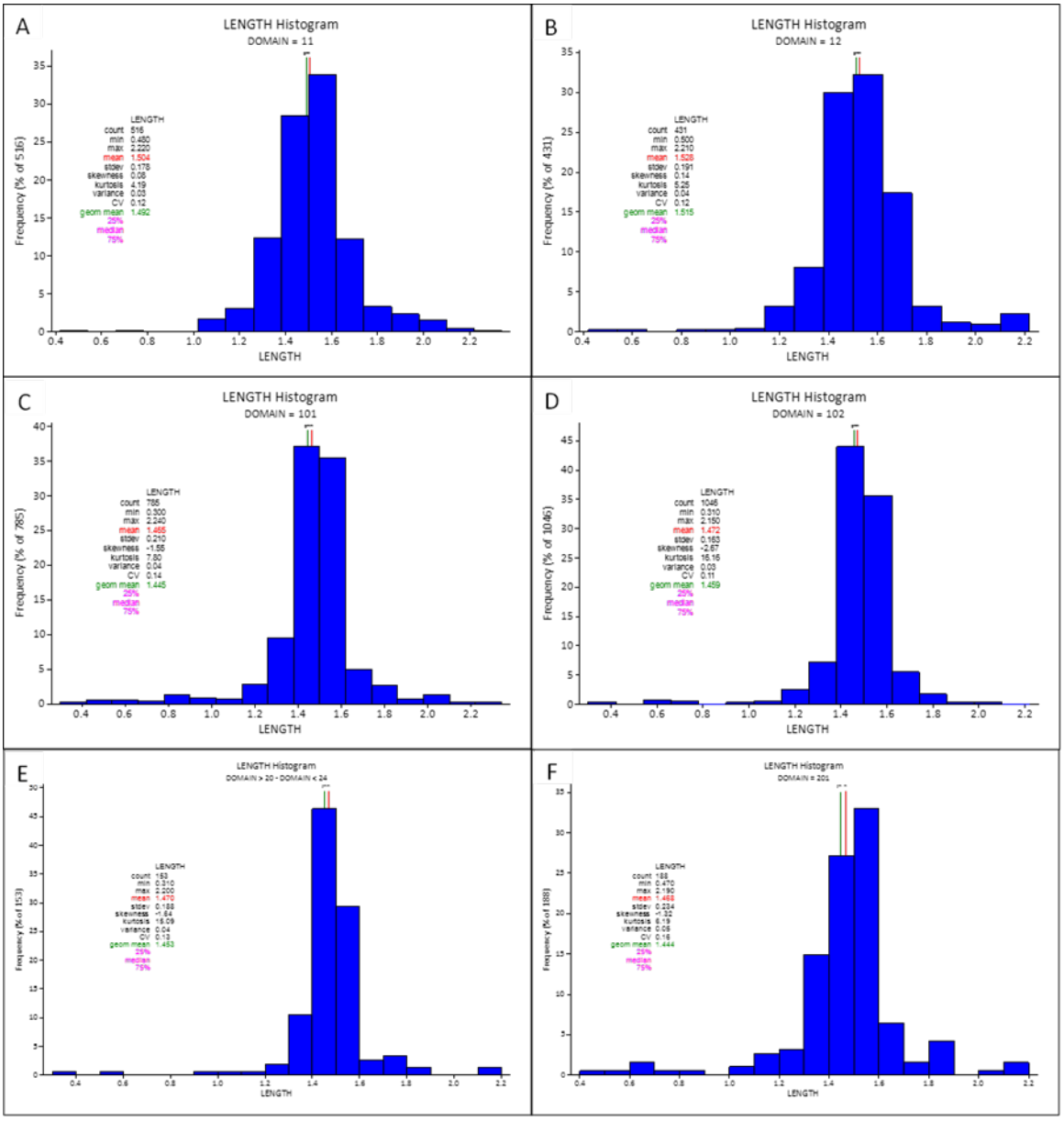


Figure 14.18 Composite Lengths for each of the Johnny Lee Cu Domains. A = UCZ Vein East, B = UCZ Vein West, C = UCZ Halo East, D = UCZ Halo West, E = LCZ Vein and F = LSZ

14.4.2 *Lowry Compositing*

Straight and run-length compositing methods were trialled with a 1.5 m run-length compositing method determined to be preferred. Compositing size proved challenging due to the majority of sample intervals being at either 1.5 or 2 m length but many high-grade (>1.2% Cu) zones being less than 2 m thickness. Compositing was performed using Maptek™ Vulcan™ software, v. 11.0.4. run-length composites up to 1.5 m were broken and flagged by bounding wireframes of the primary mineralization zones at Lowry so composites do not cross modelled boundaries. Intervals less than 0.25 m were merged with the previous interval. The resultant 1.5 m run-length compositing provides 5,027 composites with a mean length of 1.491 m.

A Composite Length Analysis (CLA) was performed on the raw drilling data, straight composites, and run-length composites to determine the preferred composting methodology and length appropriate for the Lowry data and deposit characteristics.

Raw drilling data was analysed with a histogram of sample lengths presented in Figure 14.19. The mean length is just under 1.5 m with a significant percentage of data collected at 2 m. The relation of Cu grades to sampling intervals is tabulated in Table 14.5.

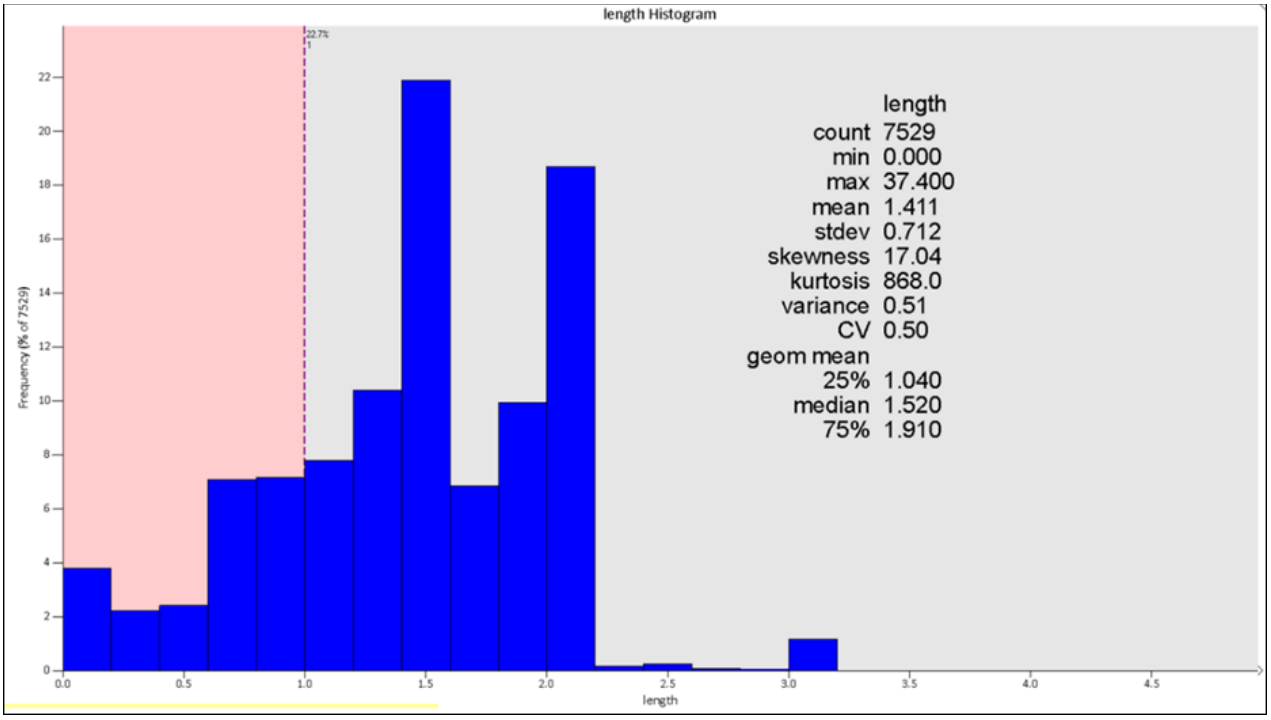


Figure 14.19 Histogram of Sample Lengths at the Lowry deposit

Column	_Filter	ColFilter	Count	Min	Max	Mean	Variance	StDev	CV	Median
Cu_pct		0.001	5089	0.01	16.95	0.618	1.99	1.412	2.29	0.10
Cu_pct	length <1	0.001	1467	0.01	16.95	1.349	4.52	2.125	1.58	0.54
Cu_pct	length <2	0.001	4253	0.01	16.95	0.706	3001	2.31	1.519	2.15
Cu_pct	length <1.5	0.001	2604	0.01	16.95	0.963	2508	3.15	1.775	1.84
Cu_pct	length >= 1.5	0.001	2485	0.01	14.20	0.256	637	0.53	0.725	2.83
length			7529	0.00	37.40	1.411	10624	0.51	0.712	0.50

Table 14.5 Summary Cu Grade by Sample Length

Using run-length compositing at 1.5 m results in a more consistent support size with 98.33% of composites at 1.5 m in size. Though sampling intervals greater than 1.5 m are broken into multiple smaller intervals, it is the opinion of the QP that a consistent and appropriate composite size that aligns with the geometry and thin nature of high grade (>1.2% Cu) mineralization at Lowry provides the most appropriate methodology. A histogram of the final composite lengths is provided in Figure 14.20 with the distribution of Cu grades by composite size presented in Figure 14.21. The summary statistics for length and Cu grade of composites is tabulated in Table 14.6.

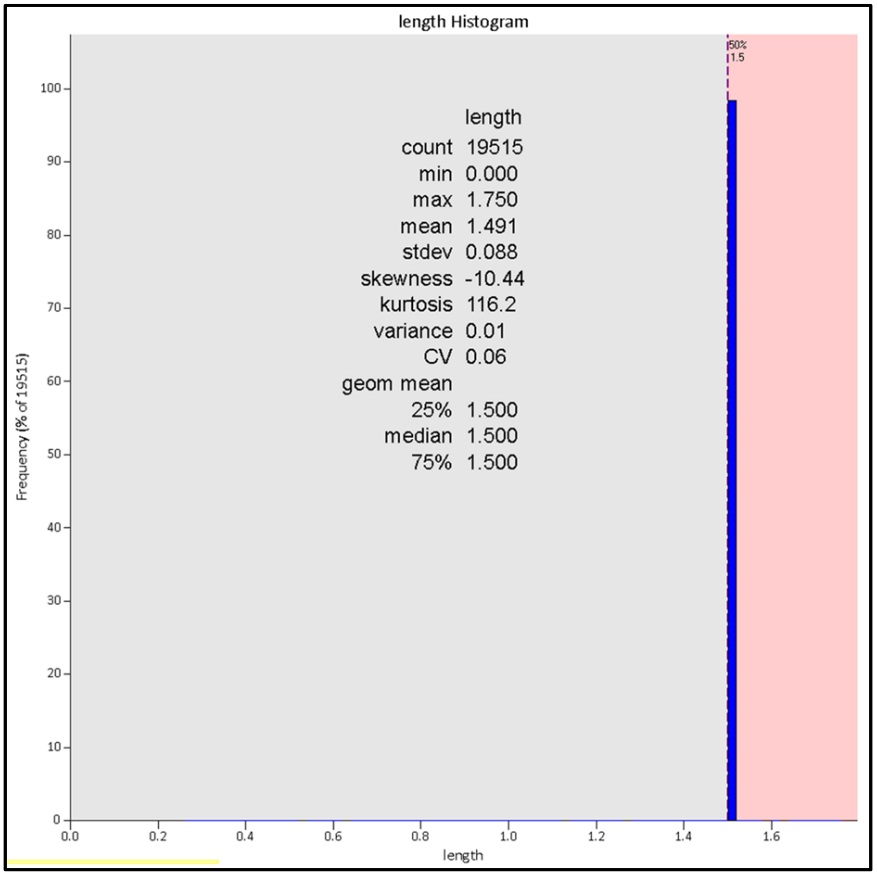


Figure 14.20 Histogram of Composite Lengths using 1.5 m run-length methodology

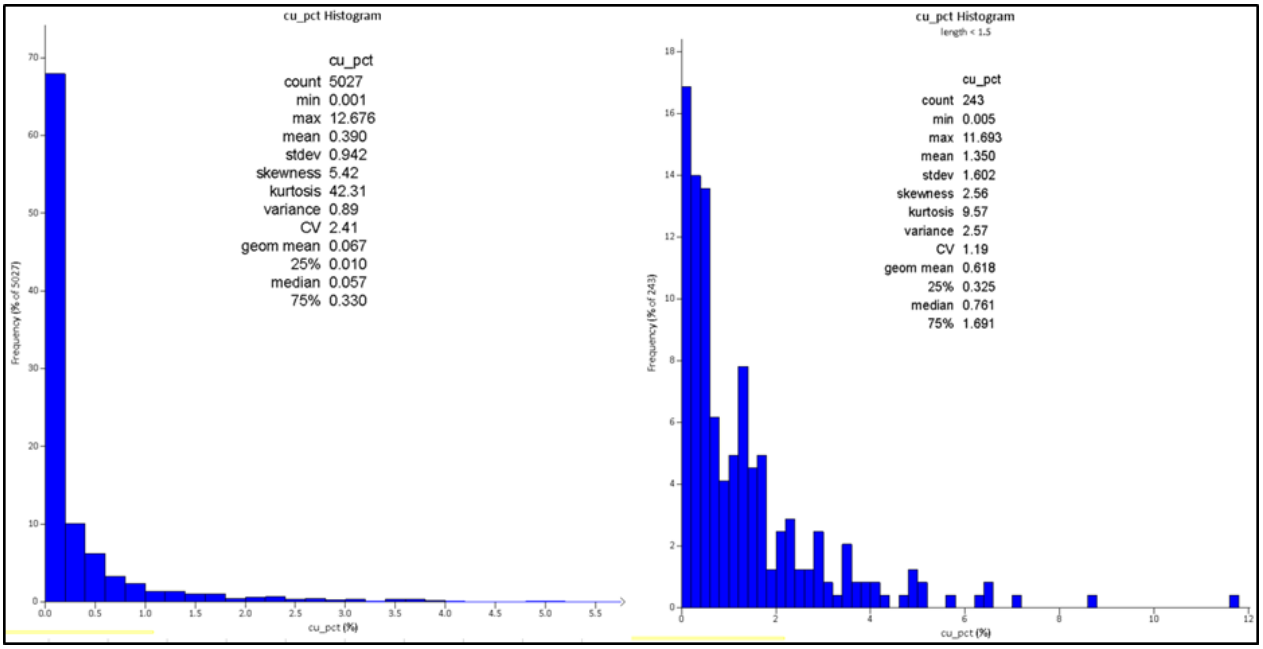


Figure 14.21 Histograms of Cu Grade Distribution by (left) All Composites and (right) Composites Less than 1.5 m

Column	_Filter	Min filter	Count	Min	Max	Mean	Variance	StDev	CV
length			19515	0.000	1.75	1.49	0.01	0.088	0.06
Cu_pct		0.001	5027	0.001	12.68	0.39	0.89	0.942	2.41
Cu_pct	length <1.5	0.001	243	0.005	11.69	1.35	2.57	1.602	1.19

Table 14.6 Summary Cu Grade by Composite Size

14.4.3 Johnny Lee Summary Statistics

Summary descriptive statistics for Cu were calculated from composited data whereas for SG statistics were calculated from the merged dataset (Table 14.7 and Table 14.8). All domains show consistent variance and Coefficient of Variation (CV) except for the LSZ which has a higher CV due to the nature of the domain and presence of grade outliers. Analysis suggests that a stationarity assumption is reasonable for the style of deposit and linear estimation of Cu and SG using modelled domains as appropriate. Log normal histograms for Cu and histograms for SG by domain are presented in Figure 14.22 and Figure 14.23.

Cu	UCZ Vein East	UCZ Vein West	UCZ Halo East	UCZ Halo West	LCZ Vein	LSZ
Composites	516	431	778	1008	153	188
Average Cu %	2.45	2.33	0.55	0.45	6.56	0.24
Min Cu %	0.296	0.222	0.005	0.004	0.003	0.001
Max Cu %	9.99	8.53	2.16	3.59	18.57	5.15
Variance	2.20	2.27	0.09	0.11	23.50	0.40
Std. Deviation	1.48	0.30	0.30	0.33	4.85	0.63
CV	0.61	0.13	0.55	0.73	0.74	2.68

Table 14.7 Pre-Capping Composite Statistics by Domain for Cu

Specific Gravity	USZ East	USZ West	LSZ
Composites	702	644	213
Average SG	4.0	3.6	3.6
Min SG	2.68	2.27	2.64
Max SG	4.71	4.55	4.43
Variance	0.1	0.2	0.2
Std. Deviation	0.4	0.4	0.4
CV	0.1	0.1	0.1

Table 14.8 Pre-Capping Composite Statistics by Domain For SG

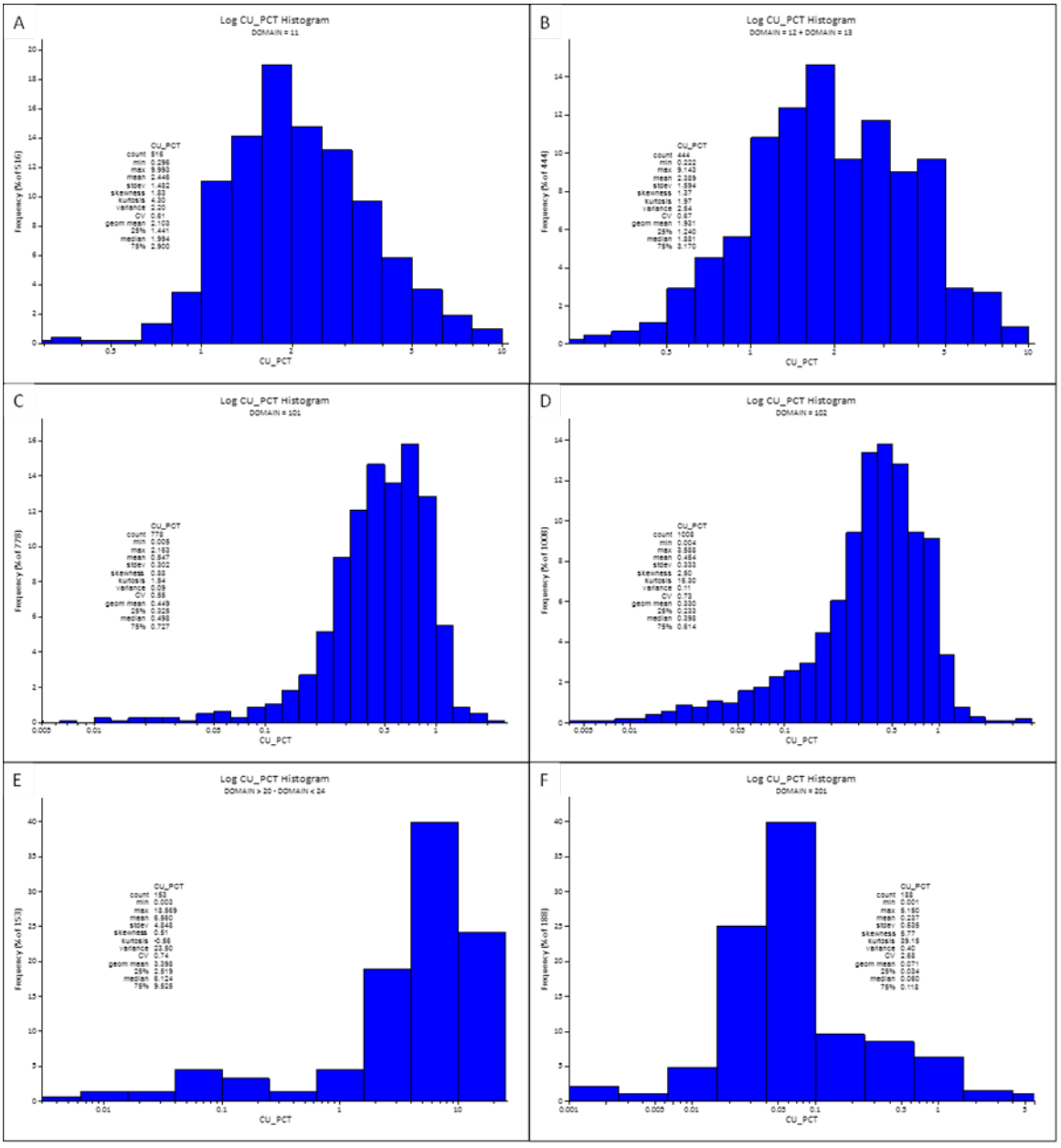


Figure 14.22 Log Normal Cu Histogram Distributions by Domain. A = UCZ Vein East, B = UCZ Vein West, C = UCZ Halo East, D = UCZ Halo West, E = LCZ Vein and F = LSZ

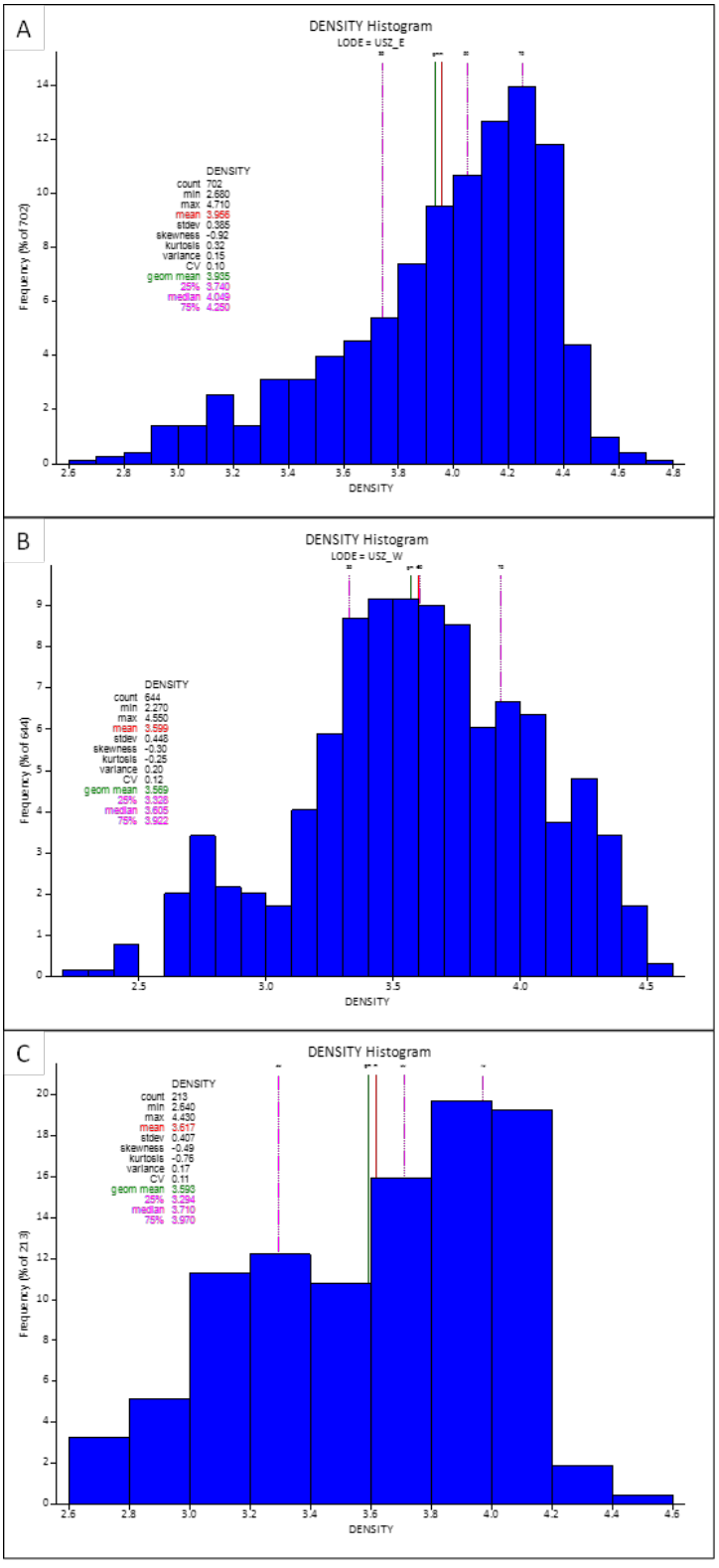


Figure 14.23 SG Histogram Distributions by Domain. A = USZ East, B = USZ West and C = LSZ

14.4.4 Lowry Summary Statistics

Exploratory Data Analysis (EDA) was calculated on composited Cu, Ag, Co, Au, As, Ba, and SG by domain with Cu being the key economic variable of interest. A series of statistical and graphical outputs were calculated including descriptive univariate statistics, bivariate statistics, histograms, and box and whisker plots. Summary descriptive statistics for key variables by domain are provided in Table 14.9. Figure 14.24 illustrates composited Cu grade distribution by modelled domain at Lowry in a combined log histogram. Lower grade zones such as the LMCZ Halo and the LLCZ Veins are evident from their position on the graph. Figure 14.25 through Figure 14.29 provide box and whisker plots for major variables of interest broken by domain, showing the key domains containing significant metal content in addition to zones of elevated As concentrations.

Many modelled domains exhibit skewed distributions evident by long tails shown as whiskers in the plots below. These near log-normal distributions are expected given the presence of limited higher-grade samples locally distributed within the broader high-grade (> 1.2% Cu) modelled zones. The implications of this type of skewed distribution is commonly observed through difficulty obtaining a robust modelled variogram for spatial continuity and a reduction in mean grade from composites to estimated blocks.

Variable	Domain	Count	Mean	Min	Max	Median	Variance	StDev	CV	25%	50%	75%	95%	97.50%	99%	IQR	Outlier
Cu_pct	All	5027	0.39	0.00	12.68	0.057	0.89	0.942	2.41	0.01	0.057	0.33	1.987	3.067	4.659	0.32	0.81
Cu_pct	lcz_Vein_1	60	2.30	0.27	8.69	1.909	2.52	1.588	0.69	1.21	1.909	2.966	5.533	6.217	7.364	1.756	5.6
Cu_pct	lcz_Vein_2	47	1.78	0.17	4.60	1.42	1.49	1.223	0.69	0.844	1.42	2.402	3.815	3.936	4.297	1.558	4.739
Cu_pct	mcz_Halo	565	0.49	0.01	3.83	0.43	0.18	0.421	0.87	0.158	0.43	0.685	1.158	1.437	1.918	0.527	1.4755
Cu_pct	mcz_Vein_1	145	2.55	0.02	12.68	1.982	3.64	1.907	0.75	1.382	1.982	3.083	6.358	7.36	9.733	1.701	5.6345
Cu_pct	mcz_Vein_2	140	2.88	0.18	12.37	2.232	4.94	2.223	0.77	1.42	2.232	3.775	7.086	8.831	11.63	2.355	7.3075
Cu_pct	mcz_Vein_3	40	2.15	0.41	7.63	1.618	2.35	1.532	0.71	1.246	1.618	2.525	5.636	6.357	7.119	1.279	4.4435
Cu_pct	ucz_Vein_1	13	2.52	0.26	5.03	2.055	1.44	1.201	0.48	1.77	2.055	3.33	4.037	4.535	4.835	1.56	5.67
Cu_pct	ucz_Vein_2	13	3.14	0.85	11.69	1.985	7.96	2.822	0.9	1.577	1.985	3.368	7.364	9.529	10.827	1.791	6.0545
Ag_ppm	All	7504	7.013	0.05	78.92	3.156	76.37	8.739	1.25	0.648	3.156	10.862	24.207	29.893	37.792	10.214	26.183
Ag_ppm	lcz_Vein_1	60	8.678	0.746	32.328	6	69.38	8.33	0.96	2.961	6	11.055	27.67	29.457	31.554	8.094	23.196
Ag_ppm	lcz_Vein_2	47	7.357	0.5	59.113	4	98.99	9.949	1.35	1	4	10.009	21.909	22.817	42.064	9.009	23.5225
Ag_ppm	mcz_Halo	567	9.1	0.3	39.2	8.4	41	6.4	0.7	4.3	8.4	13.0	21.3	24.2	26.1	8.7	26.0
Ag_ppm	mcz_Vein_1	145	13.6	0.5	42.7	11.6	59	7.7	0.6	8.3	11.6	18.0	27.3	29.5	34.9	9.7	32.6
Ag_ppm	mcz_Vein_2	140	17.6	0.7	37.1	17.9	51	7.2	0.4	12.4	17.9	22.2	29.6	33.3	35.7	9.7	36.8
Ag_ppm	mcz_Vein_3	40	15.8	0.5	44.8	13.2	86	9.3	0.6	11.1	13.2	22.0	33.0	33.9	40.4	10.9	38.4
Ag_ppm	ucz_Vein_1	13	13.0	3.2	25.5	11.5	63	7.9	0.6	4.0	11.5	17.9	25.1	25.3	25.4	13.9	38.8
Ag_ppm	ucz_Vein_2	13	20.3	4.9	59.2	16.1	212	14.5	0.7	11.2	16.1	23.4	44.1	51.7	56.2	12.3	41.9
Co_ppm	All	7485	159.1	0.2	5249.0	35.2	115562.0	339.9	2.1	10.0	35.2	150.0	705.8	1048.0	1592.0	140.0	360.0
Co_ppm	lcz_Vein_1	60	491.0	10.4	2575.6	374.1	205564.0	453.4	0.9	170.5	374.1	668.0	1163.4	1523.4	2070.1	497.5	1414.3
Co_ppm	lcz_Vein_2	47	403.8	33.0	1671.9	166.4	187144.0	432.6	1.1	80.0	166.4	605.5	1217.5	1229.5	1464.2	525.5	1393.8
Co_ppm	mcz_Halo	567	401.0	5.9	2137.9	330.1	126686.0	355.9	0.9	113.0	330.1	598.0	1026.0	1176.1	1751.1	485.0	1325.4
Co_ppm	mcz_Vein_1	145	1020.8	30.0	5249.0	655.8	1134196.0	1065.0	1.0	375.1	655.8	1202.7	3614.0	4277.2	4926.3	827.6	2444.0
Co_ppm	mcz_Vein_2	140	1174.4	35.4	4756.8	1065.5	591422.0	769.0	0.7	672.3	1065.5	1423.9	2518.9	3218.2	4217.5	751.6	2551.4
Co_ppm	mcz_Vein_3	40	900.8	108.0	1810.5	991.4	176102.0	419.6	0.5	495.8	991.4	1174.3	1363.7	1793.8	1803.8	678.4	2191.9
Co_ppm	ucz_Vein_1	13	744.9	251.8	1604.0	676.2	164944.0	406.1	0.6	311.0	676.2	926.0	1326.4	1465.2	1548.4	615.0	1848.5
Co_ppm	ucz_Vein_2	13	778.8	360.9	1590.0	636.4	167569.0	409.4	0.5	425.8	636.4	961.8	1491.5	1540.7	1570.3	536.0	1765.8
As_ppm	All	7504	302.1	1.0	15,812.4	83.3	463,984	681.2	2.3	25.0	83.3	290.0	1,213.3	1,763.5	2,897.6	265.0	687.5

Variable	Domain	Count	Mean	Min	Max	Median	Variance	StDev	CV	25%	50%	75%	95%	97.50%	99%	IQR	Outlier
As_ppm	lcz_Vein_1	60	1,323.8	70.0	6,488.9	985.7	1,709,654	1,307.5	1.0	372.5	985.7	1,605.0	3,174.2	5,312.8	6,311.8	1,232.5	3,453.7
As_ppm	lcz_Vein_2	47	676.6	58.9	2,871.4	361.8	477,745	691.2	1.0	179.7	361.8	949.0	2,055.6	2,520.3	2,747.6	769.2	2,102.8
As_ppm	mcz_Halo	567	696.1	25.0	4,735.1	564.4	381,842	617.9	0.9	246.7	564.4	957.7	1,762.6	2,519.7	3,017.3	711.0	2,024.1
As_ppm	mcz_Vein_1	145	1,987.2	25.0	15,812.4	1,141.4	5,199,779	2,280.3	1.2	716.5	1,141.4	2,259.5	7,106.0	8,532.7	10,048.0	1,543.1	4,574.2
As_ppm	mcz_Vein_2	140	1,969.7	46.1	8,721.7	1,470.0	2,342,745	1,530.6	0.8	941.5	1,470.0	2,449.1	5,272.0	6,138.5	7,174.5	1,507.6	4,710.5
As_ppm	mcz_Vein_3	40	1,085.6	115.3	2,216.1	1,090.0	227,405	476.9	0.4	767.9	1,090.0	1,356.4	1,730.9	2,053.2	2,151.0	588.5	2,239.2
As_ppm	ucz_Vein_1	13	1,587.7	440.0	3,235.5	1,231.4	1,082,669	1,040.5	0.7	549.8	1,231.4	2,409.8	3,082.2	3,158.9	3,204.9	1,860.0	5,199.9
As_ppm	ucz_Vein_2	13	2,217.9	420.8	10,480.5	1,317.5	7,127,050	2,669.7	1.2	613.7	1,317.5	2,606.1	5,794.3	8,137.4	9,543.2	1,992.4	5,594.8
Au_ppb	All	7505	7	3	2687	3	1719	41	6.33	3	3	6	16	22	31	3	10.5
Au_ppb	lcz_Vein_1	60	7	3	42	3	47.99	7	0.94	3	3	9	18	19	29	6	18
Au_ppb	lcz_Vein_2	47	38	3	777	4	18492	136	3.56	3	4	13	94	468	667	10	28
Au_ppb	mcz_Halo	567	10	3	2039	3	7775	88	8.41	3	3	5	17	25	40	2	8
Au_ppb	mcz_Vein_1	145	8	3	121	3	206.2	14	1.86	3	3	7	20	29	87	4	13
Au_ppb	mcz_Vein_2	140	6	3	34	3	38.55	6	0.96	3	3	7	22	26	27	4	13
Au_ppb	mcz_Vein_3	40	6	3	29	3	32.35	6	0.98	3	3	5	16	17	24	2	8
Au_ppb	ucz_Vein_1	13	4	3	12	3	6.05	2	0.63	3	3	3	7	10	11	0	3
Au_ppb	ucz_Vein_2	13	5	3	11	3	7.47	3	0.58	3	3	6	10	10	11	3	10.5
Ba_ppm	All	7504	14,791	10	440,691	620	1,830,818,509	42,788	3	310	620	3,643	102,563	153,908	233,307	3,333	8,643
Ba_ppm	lcz_Vein_1	60	8,575	19	115,536	524	421,166,134	20,522	2	217	524	7,132	34,392	72,653	96,954	6,915	17,505
Ba_ppm	lcz_Vein_2	47	463	25	6,825	257	977,006	988	2	142	257	433	1,108	1,287	4,228	291	870
Ba_ppm	mcz_Halo	567	12,158	37	341,220	421	1,480,134,415	38,473	3	213	421	2,349	79,545	132,138	212,019	2,136	5,553
Ba_ppm	mcz_Vein_1	145	411	25	10,702	160	1,462,295	1,209	3	109	160	284	704	1,696	7,170	175	546
Ba_ppm	mcz_Vein_2	140	3,033	25	245,000	182	474,040,184	21,772	7	119	182	326	732	21,780	53,245	207	637
Ba_ppm	mcz_Vein_3	40	535	57	2,401	383	259,013	509	1	170	383	556	1,318	1,810	2,165	386	1,136
Ba_ppm	ucz_Vein_1	13	13,452	108	120,516	230	1,210,326,874	34,790	3	183	230	787	73,641	97,078	111,141	604	1,694
Ba_ppm	ucz_Vein_2	13	392	160	861	302	44,529	211	1	233	302	524	733	797	835	291	961

Table 14.9 Summary Descriptive Statistics for Composited Data at the Lowry deposit

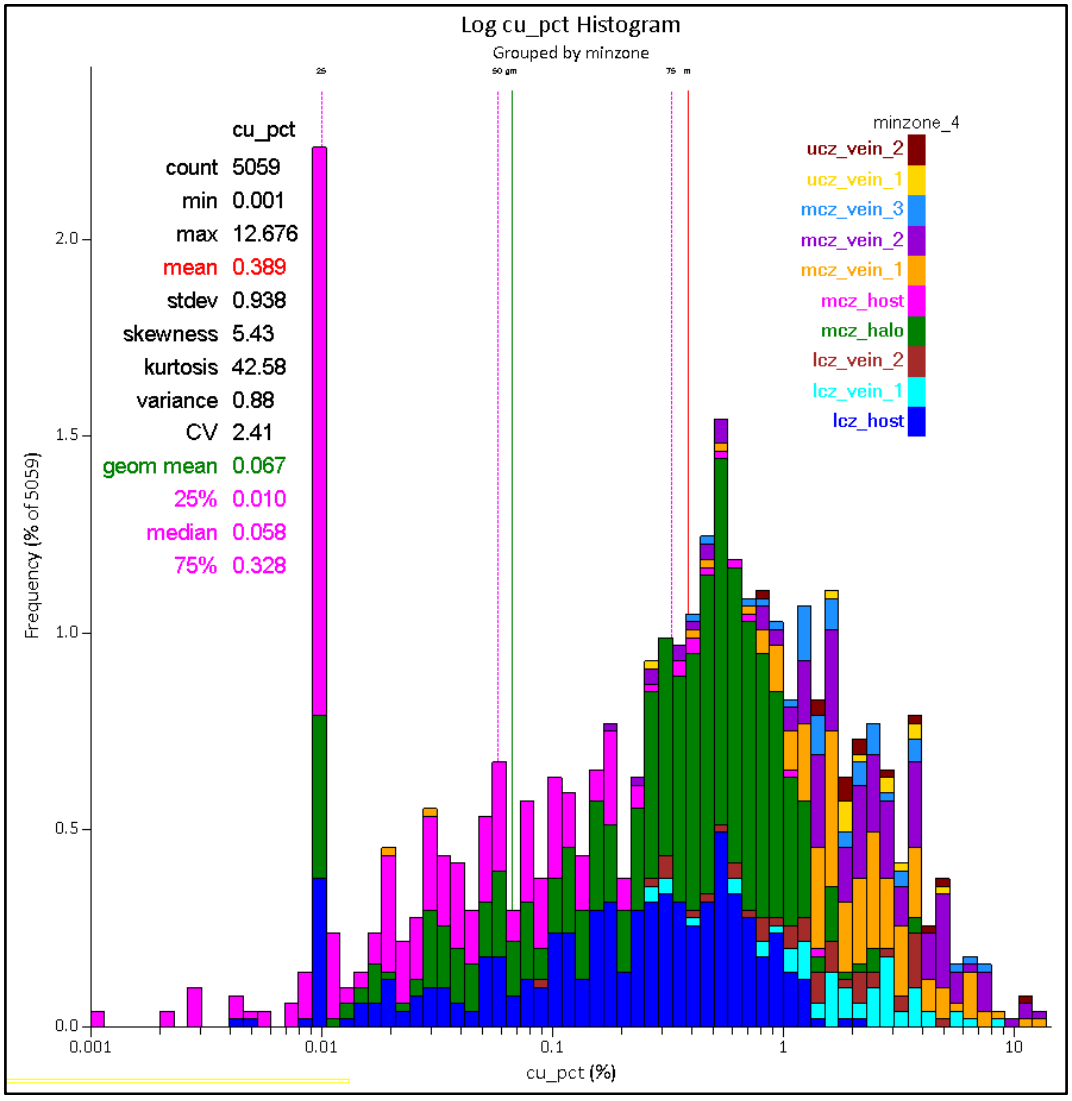


Figure 14.24 Grouped Log Histogram of Cu Distribution by Domain in the Lowry deposit

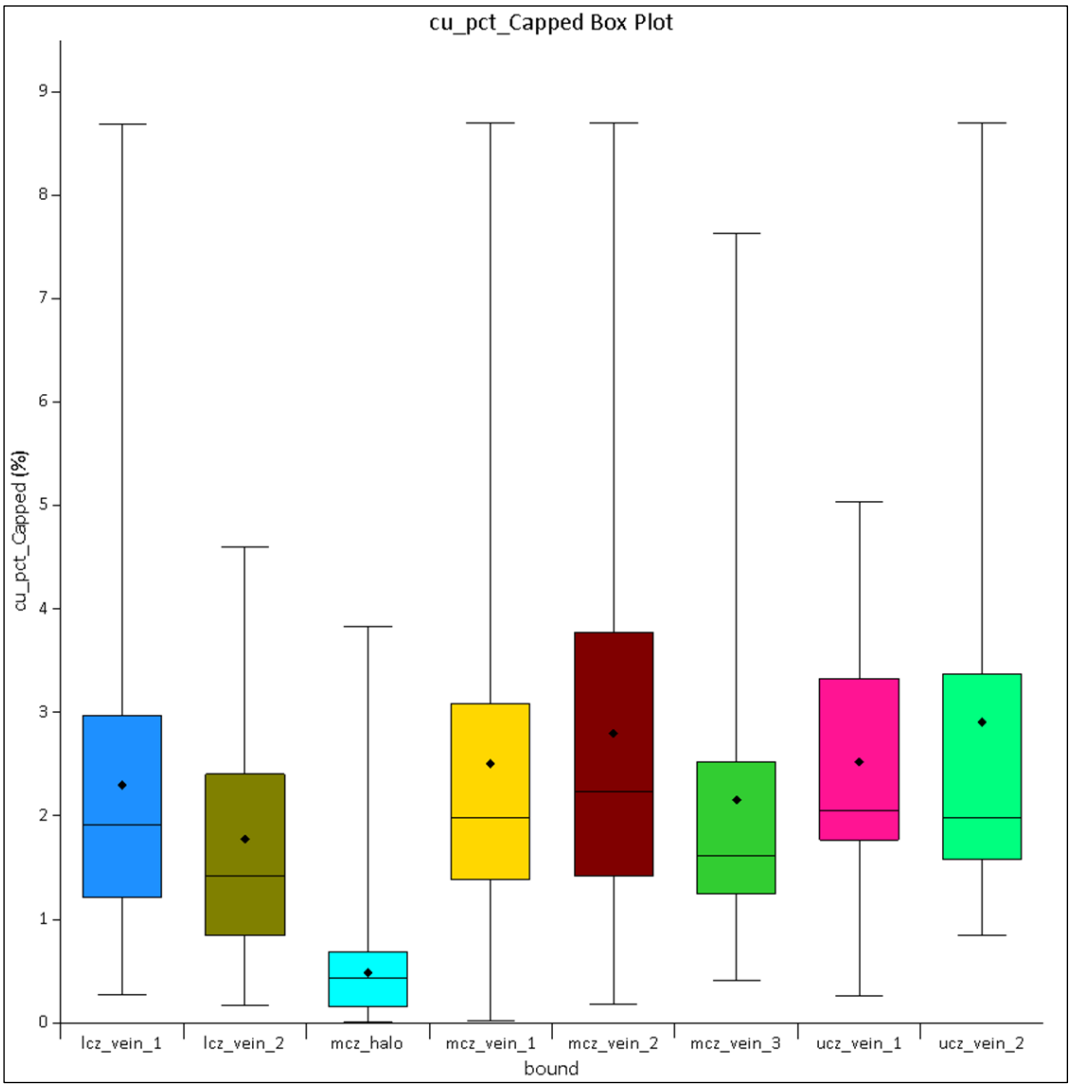


Figure 14.25 Box and Whisker Plot of Cu (capped) by Domain in the Lowry deposit

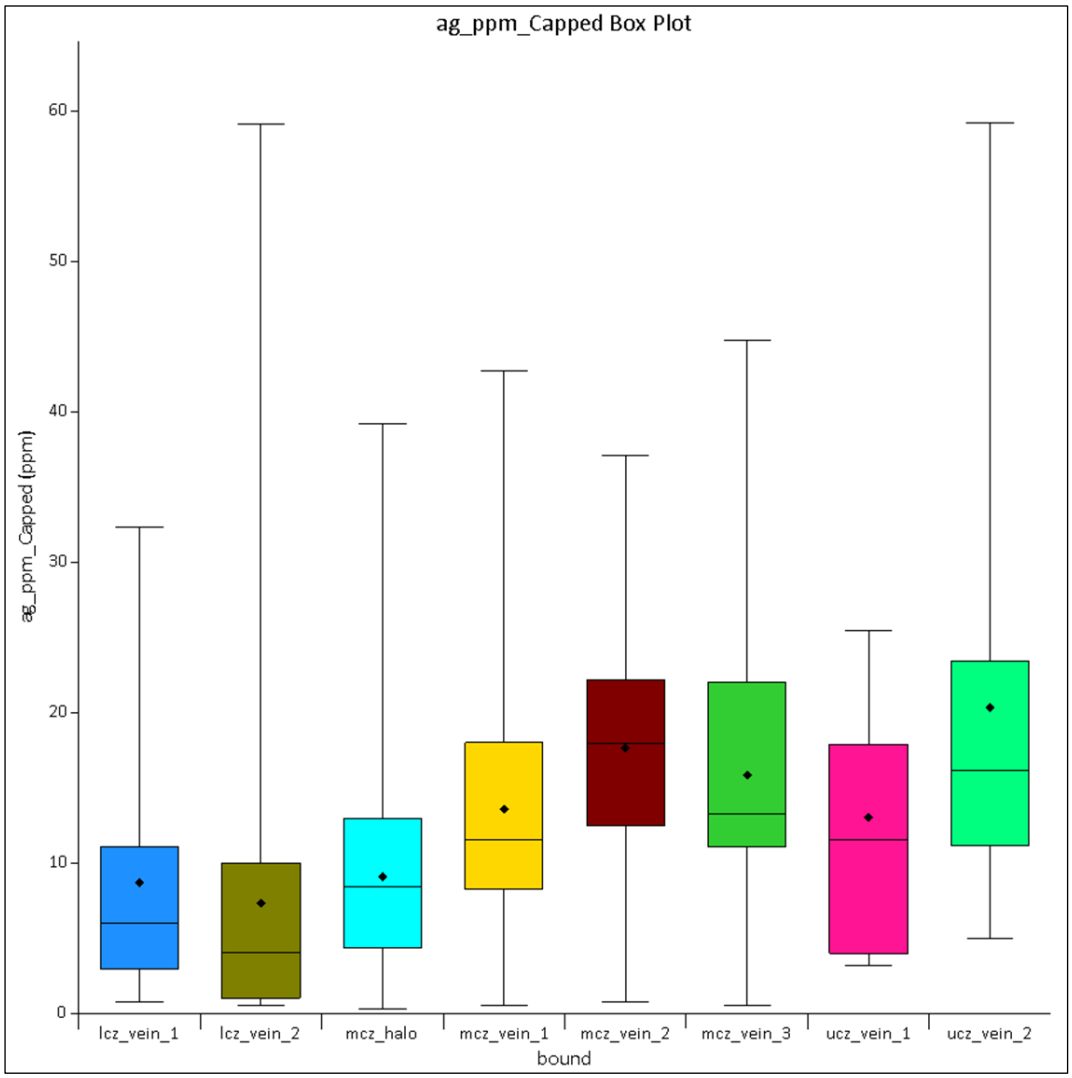


Figure 14.26 Box and Whisker Plot of Ag (capped) by Domain in the Lowry deposit

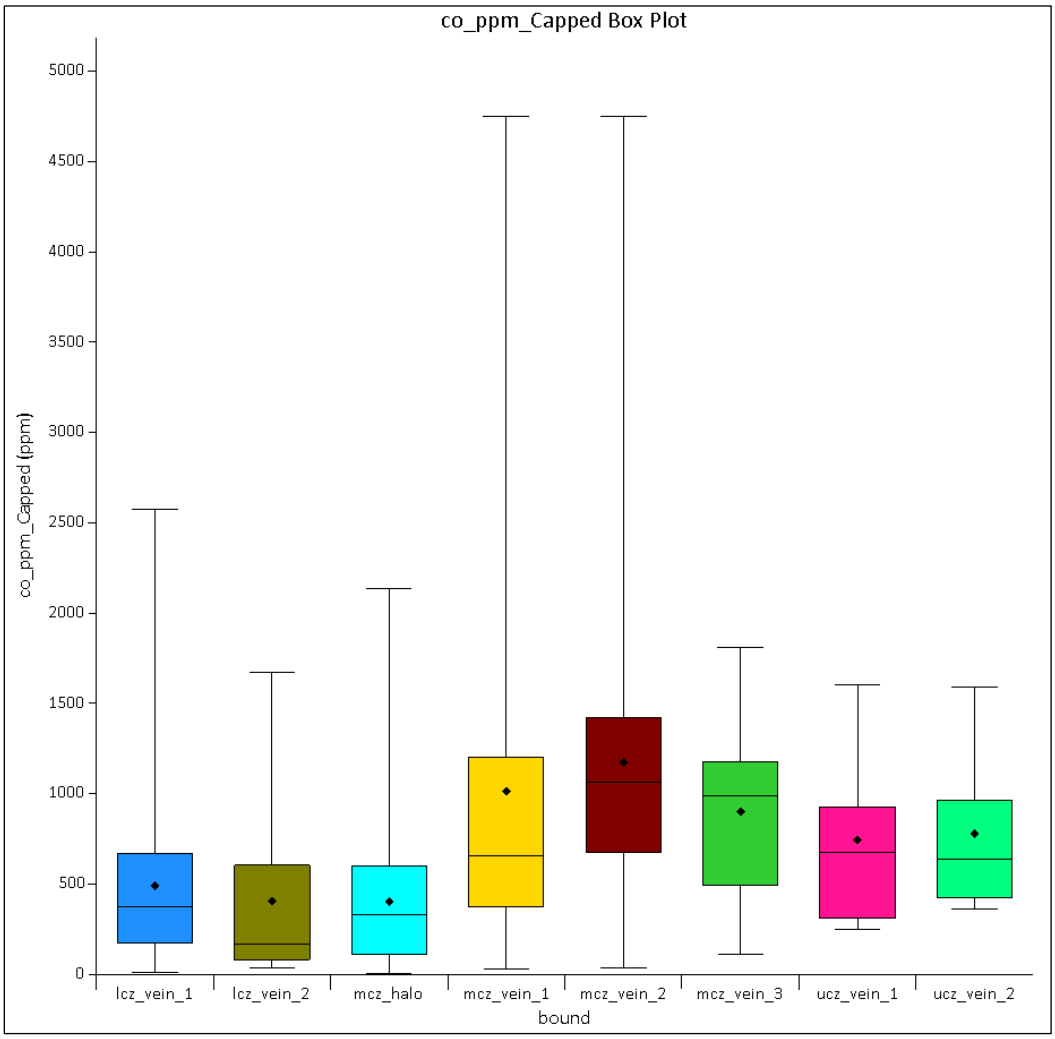


Figure 14.27 Box and Whisker Plot of Co (capped) by Domain in the Lowry deposit

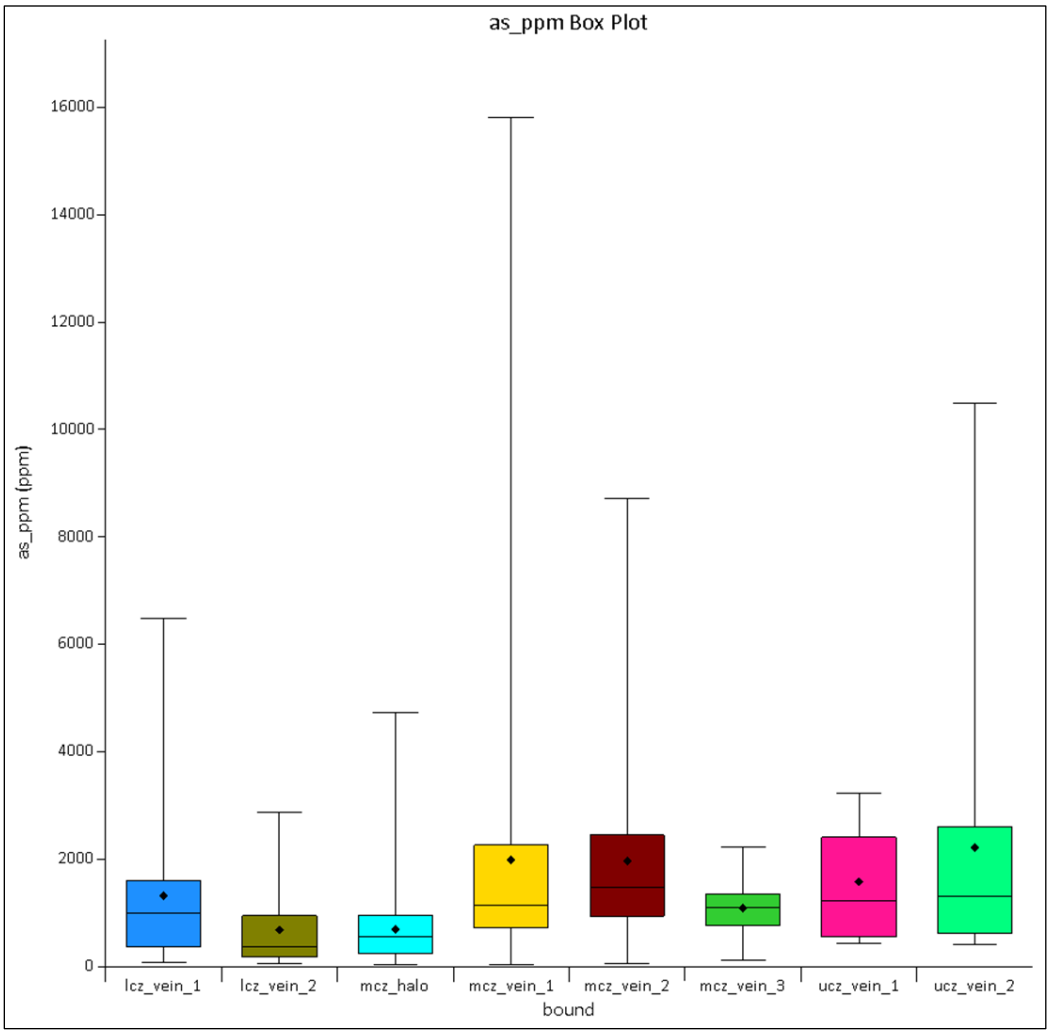


Figure 14.28 Box and Whisker Plot of As by Domain in the Lowry deposit

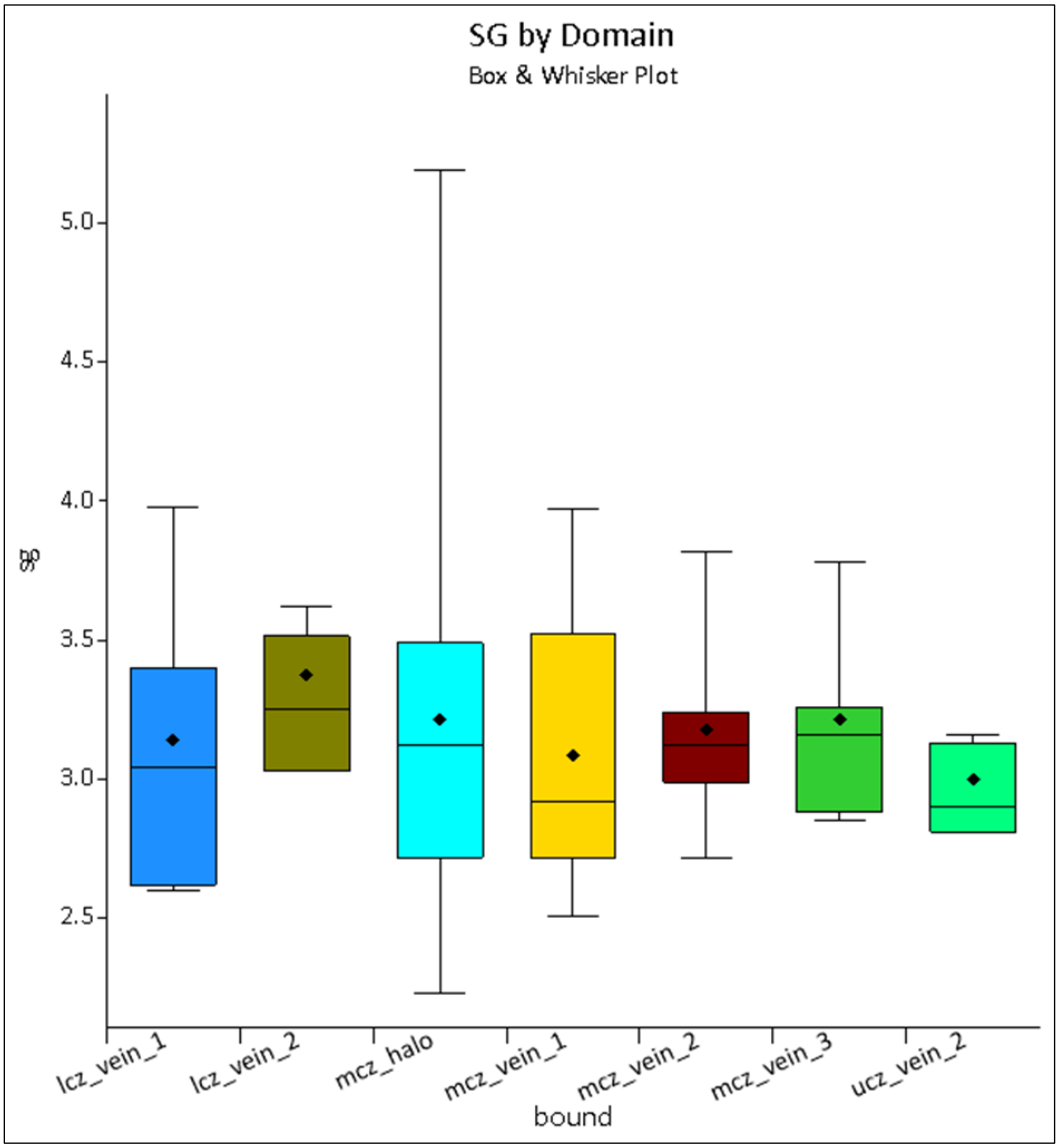


Figure 14.29 Box and Whisker Plot of SG by Domain in the Lowry deposit

14.4.5 Bivariate Statistics

Bivariate correlations were calculated for all key variables with summary values presented in Table 14.10. The highest directly correlated elements are Co-As-Bi, likely from these elements being contained within pyrite, Ag-Pb, and Cu-As-Bi-Co also likely associated with pyrite and marcasite. An X-Y scatterplot of Co-As is presented in Figure 14.30.

Variable	Ag_ppm	As_ppm	Au_ppb	Ba_ppm	Bi_ppm	Co_ppm	Cu_%	Fe_%	Pb_ppm	S_pct	Zn_ppm	Cu_pct_ Capped	Ag_ppm_ Capped	Co_ppm_ Capped
Ag_ppm	1	0.36	0.01	0.3	0.19	0.37	0.24	0.58	0.66	0.59	-0.01	0.24	1	0.37
As_ppm	0.36	1	0.03	-0.05	0.77	0.84	0.6	0.15	0.33	0.25	0.07	0.61	0.36	0.84
Au_ppb	0.01	0.03	1	0.05	0.02	0.03	0.02	-0.01	-0.01	0.01	-0.01	0.02	0.01	0.03
Ba_ppm	0.3	-0.05	0.05	1	-0.06	-0.08	-0.12	0.21	0.1	0.23	-0.03	-0.12	0.3	-0.08
Bi_ppm	0.19	0.77	0.02	-0.06	1	0.71	0.64	0.05	0.09	0.12	0	0.64	0.19	0.71
Co_ppm	0.37	0.84	0.03	-0.08	0.71	1	0.6	0.16	0.3	0.28	-0.02	0.61	0.37	1
Cu_%	0.24	0.6	0.02	-0.12	0.64	0.6	1	0.03	0.09	0.15	0.04	0.99	0.24	0.6
Fe_%	0.58	0.15	-0.01	0.21	0.05	0.16	0.03	1	0.6	0.82	0.09	0.03	0.58	0.16
Pb_ppm	0.66	0.33	-0.01	0.1	0.09	0.3	0.09	0.6	1	0.6	0.33	0.09	0.66	0.3
S_pct	0.59	0.25	0.01	0.23	0.12	0.28	0.15	0.82	0.6	1	0.11	0.16	0.59	0.28
Zn_ppm	-0.01	0.07	-0.01	-0.03	0	-0.02	0.04	0.09	0.33	0.11	1	0.04	-0.02	-0.02
Cu_pct_Capped	0.24	0.61	0.02	-0.12	0.64	0.61	0.99	0.03	0.09	0.16	0.04	1	0.24	0.61
Ag_ppm_Capped	1	0.36	0.01	0.3	0.19	0.37	0.24	0.58	0.66	0.59	-0.02	0.24	1	0.37
Co_ppm_Capped	0.37	0.84	0.03	-0.08	0.71	1	0.6	0.16	0.3	0.28	-0.02	0.61	0.37	1

Table 14.10 Correlation Coefficients for Key Variables in the Lowry deposit

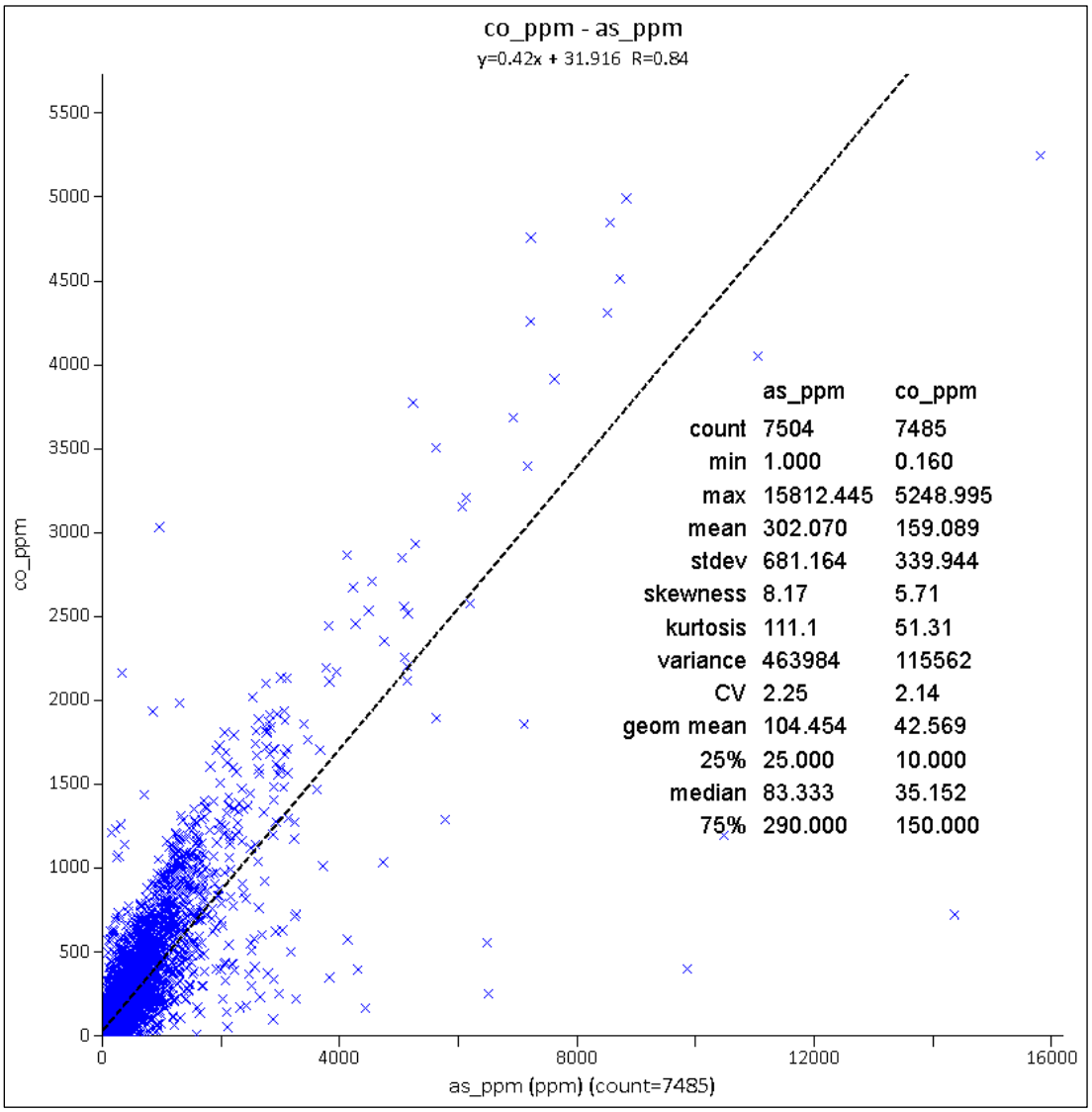


Figure 14.30 X-Y Scatterplot of Co-As Relationship in the Lowry deposit

14.4.6 Johnny Lee Outliers and Capping

After compositing, a capping of Cu values was applied to the UCZ Vein East, UCZ Vein West, LCZ Vein, and LSZ domains based on disintegration analysis of Cu log probability plots (Figure 14.31 and summarized in Table 14.11). No capping or cutting of data was applied to the UCZ Halo East or West for Cu, nor was capping applied to SG. The capping was selected to minimize the effect of isolated high-grade outliers, without materially reducing metal or modifying a material proportion of data. Capped values by domain are presented in Table 14.11.

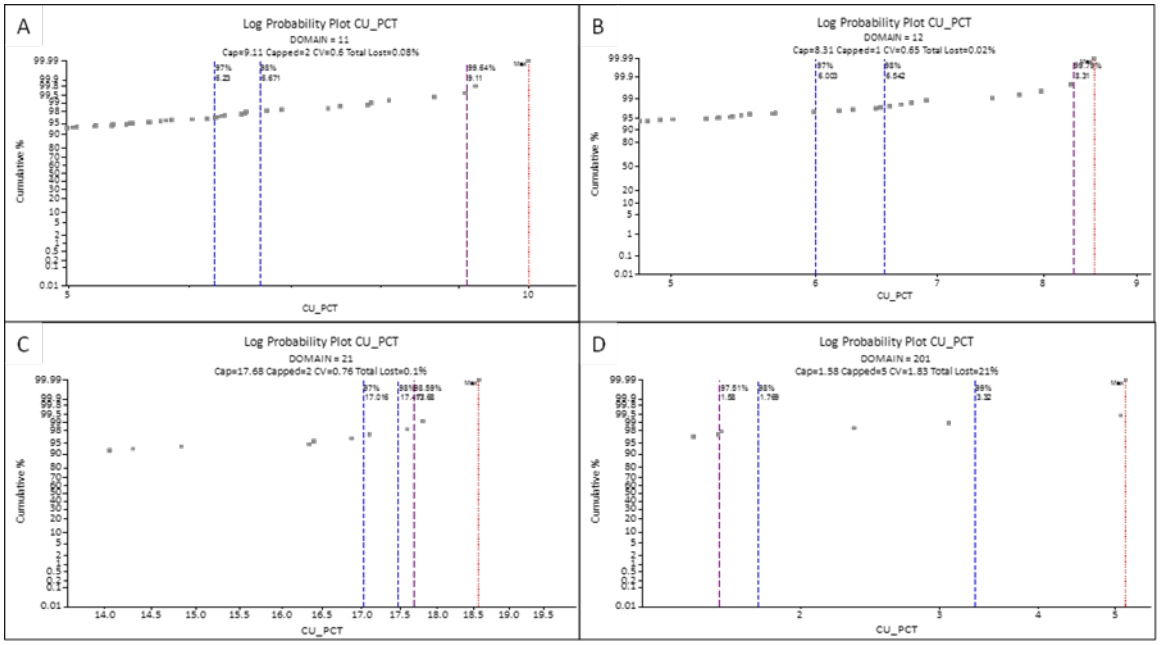


Figure 14.31 Cu Log Probability Plots. A = UCZ Vein East, B = UCZ Vein West, C = LCZ Vein and D = LSZ.

Domain	Cap	Capped	Percentile	Capped%	Lost	Min	Max	Mean	Variance	CV
UCZ Vein East	9.11	2	99.64%	0.40%	0.08%	0.296	9.11	2.444	2.17	0.60
UCZ Vein West	8.31	1	99.79%	0.20%	0.02%	0.222	8.31	2.331	2.27	0.65
LCZ Vein	17.68	2	99.59%	1.80%	0.10%	0.027	17.68	6.379	23.37	0.76
LSZ	1.58	5	97.51%	2.70%	21%	0.001	1.58	0.187	0.12	1.83

Table 14.11 Summary of Cu (%) capping applied by domain in the Johnny Lee deposit

14.4.7 Lowry Outlier and Capping

Composited values of potentially key economic variables of Cu, Ag, Co, and As along with SG were analyzed for high-yield outliers that may negatively impact statistical evaluations and block estimation. The capping analysis consisted of determination of statistically defined outliers, log probability plots testing various capping limits, and the assignment of capping values for each variable. Summary applied capping values are presented in Table 14.12

SRK reviewed capping limits applied in the previous model version from 2013 and the 2020 updated capped values are similar for Cu and Co. Material differences from the 2013 model include a previous cap of 30 ppm (g/t) for Ag, while SRK has applied a 67.2 ppm Ag cap and capping was not applied for As or SG in 2013. It is the opinion of SRK that the 30 g/t cap applied in 2013 was too low at 97.5 percentile resulting in a 3.2% loss of metal.

Variable	Capped	Unit
Cu_%	8.7	%
Ag_ppm	67.2	ppm
Co_ppm	4,750	ppm
As_ppm	7,200	ppm
SG	5.0	n/a

Table 14.12 Summary Compositing Capping Values for the Lowry deposit

Log probability charts were used to select various potential capping limits based on a combination of data disintegration, change in slope, and the resultant statistical change to the data population. Cu capping analysis is presented in Figure 14.32 and Figure 14.33. The selection of 8.7% Cu as a cap is at the 99.8 percentile representing a one percent loss of metal while reducing seven values of outlier high-grade back to 8.7% Cu.

Ag values were capped back to 67.2 ppm by reducing the highest six samples. This is at the 99.92 percentile with only minor changes in lost total metal and a slight reduction in CV. Similarly, Co was capped at 4,750 ppm reducing four samples. As cap was set at 7,200 ppm and SG at 5.0. In all cases, capping reassigns the upper values to the cap limit but does not exclude those samples. Summary data and charts by variable are presented below.

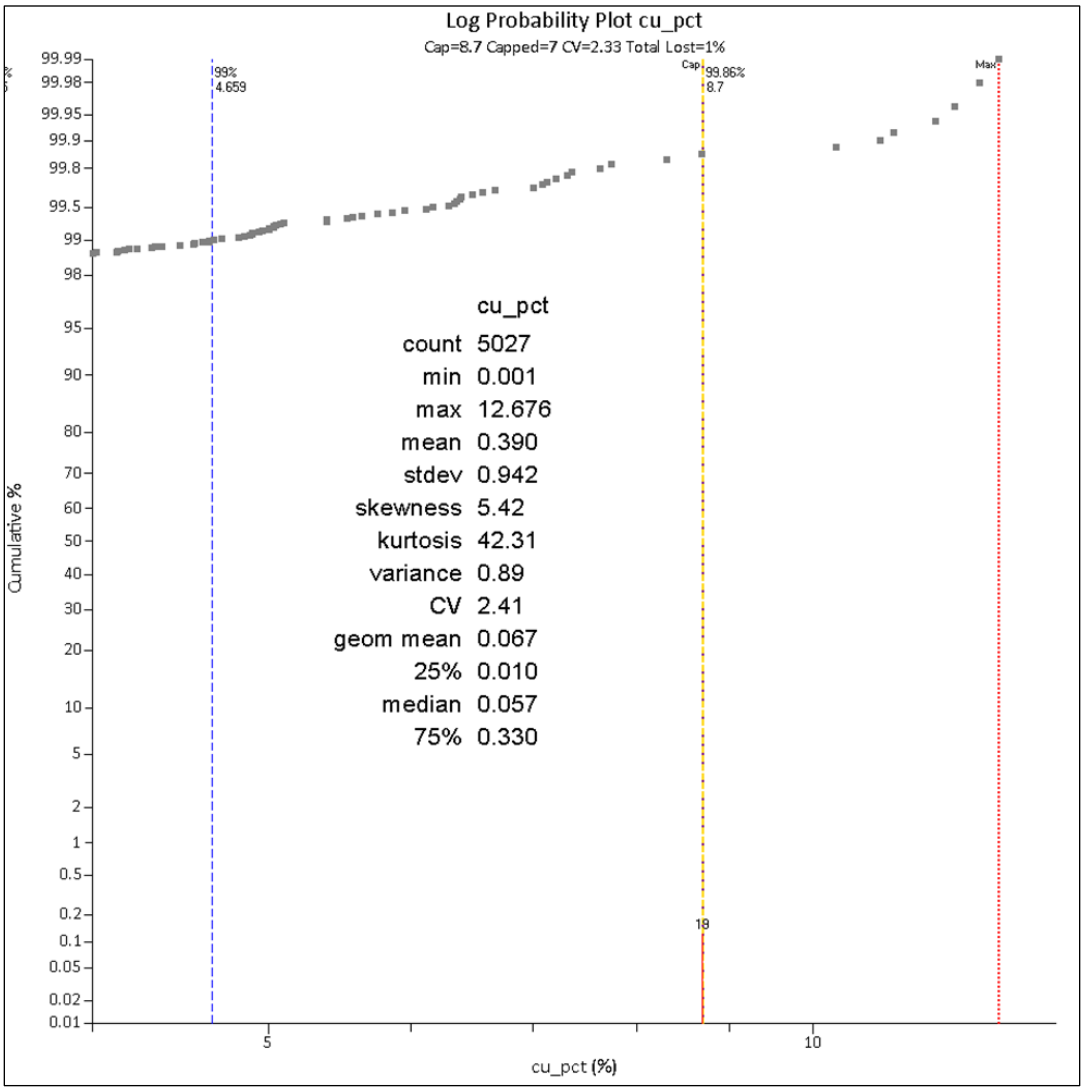


Figure 14.32 Log Probability Plot for Cu with assigned Cap at 8.7% Cu in the Lowry deposit

Column	_Filter	Cap	Capped	Percentile	Capped%	Lost	Lost	Count	Min	Max	Mean	Total	Variance	CV
						Total%	CV%							
Cu_%								5027	0.001	12.68	0.39	1961	0.89	2.41
Cu_%		8.7	7	99.86%	0.10%	1%	3.60%	5027	0.001	8.7	0.386	1941	0.81	2.33
Cu_%	Cu_% >8.7							7	10.3	12.676	11.574	81.02	0.72	0.07
Cu_$	Cu_% <= 8.7							5020	0.001	8.685	0.375	1880	0.71	2.25

Table 14.13 Summary Capping Analysis for Cu in the Lowry deposit

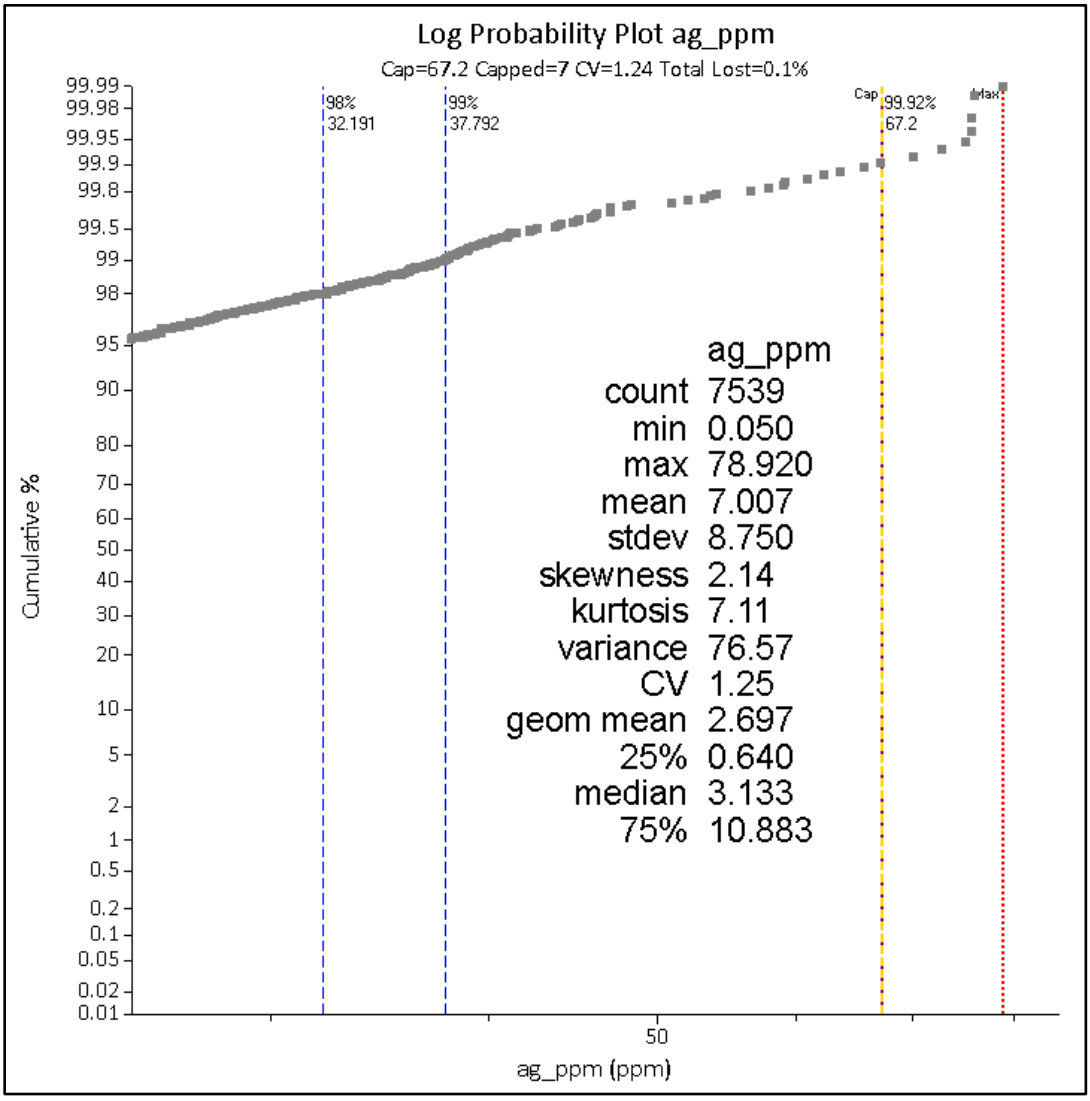


Figure 14.33 Log Probability Plot for Ag with Assigned Cap at 67.2 ppm in the Lowry deposit

Column	_Filter	Cap	Capped	Percentile	Capped%	Lost	Lost	Count	Min	Max	Mean	Total	Variance	CV
						Total%	CV%							
Ag_ppm								7504	0.05	78.92	7.013	52628	76.37	1.25
Ag_ppm		67.2	6	99.92%	0.10%	0.10%	0.50%	7504	0.05	67.20	7.007	52577	75.50	1.24
Ag_ppm		53.997	16	99.80%	0.20%	0.40%	1.60%	7504	0.05	54.00	6.987	52431	73.45	1.23
Ag_ppm		48.329	22	99.72%	0.30%	0.60%	2.20%	7504	0.05	48.33	6.972	52321	72.16	1.22
Ag_ppm	ag_ppm >67.2							6	72.73	78.92	75.690	454	3.92	0.03
Ag_ppm	ag_ppm <= 67.2							7498	0.05	67.17	6.958	52174	72.66	1.22

Table 14.14 Summary Capping Analysis for Ag in the Lowry deposit

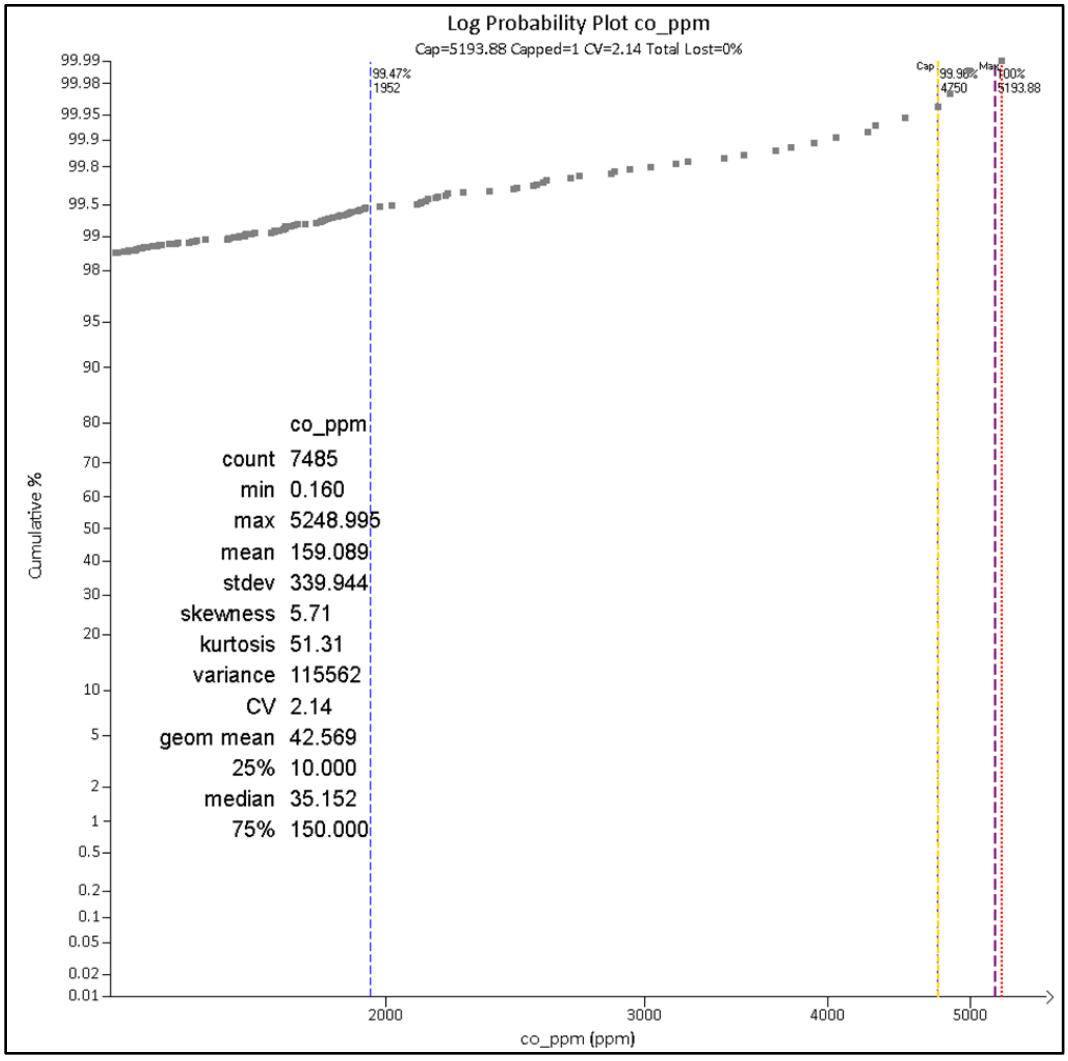


Figure 14.34 Log Probability Plot for Co with assigned Cap at 4,750 ppm Co in the Lowry deposit

Column	_Filter	Cap	Capped	Percentile	Capped%	Lost	Lost	Count	Min	Max	Mean	Variance	CV
						Total%	CV%						
Co_ppm								7485	0.16	5249	159.1	115562	2.14
Co_ppm		5193.88	1	100%	0.01%	0%	0.03%	7485	0.16	5194	159.1	115488	2.14
Co_ppm		4750	4	99.96%	0.10%	0.07%	0.40%	7485	0.16	4750	159.0	114484	2.13
Co_ppm		1952	40	99.47%	0.50%	3.70%	12%	7485	0.16	1952	153.3	83489	1.89
Co_ppm	Co_ppm >4750							4	4757	5249	4961	46057	0.04
Co_ppm	Co_ppm <= 4750							7481	0.16	4513	156.5	103268	2.05

Table 14.15 Summary Capping Analysis for Co in the Lowry deposit

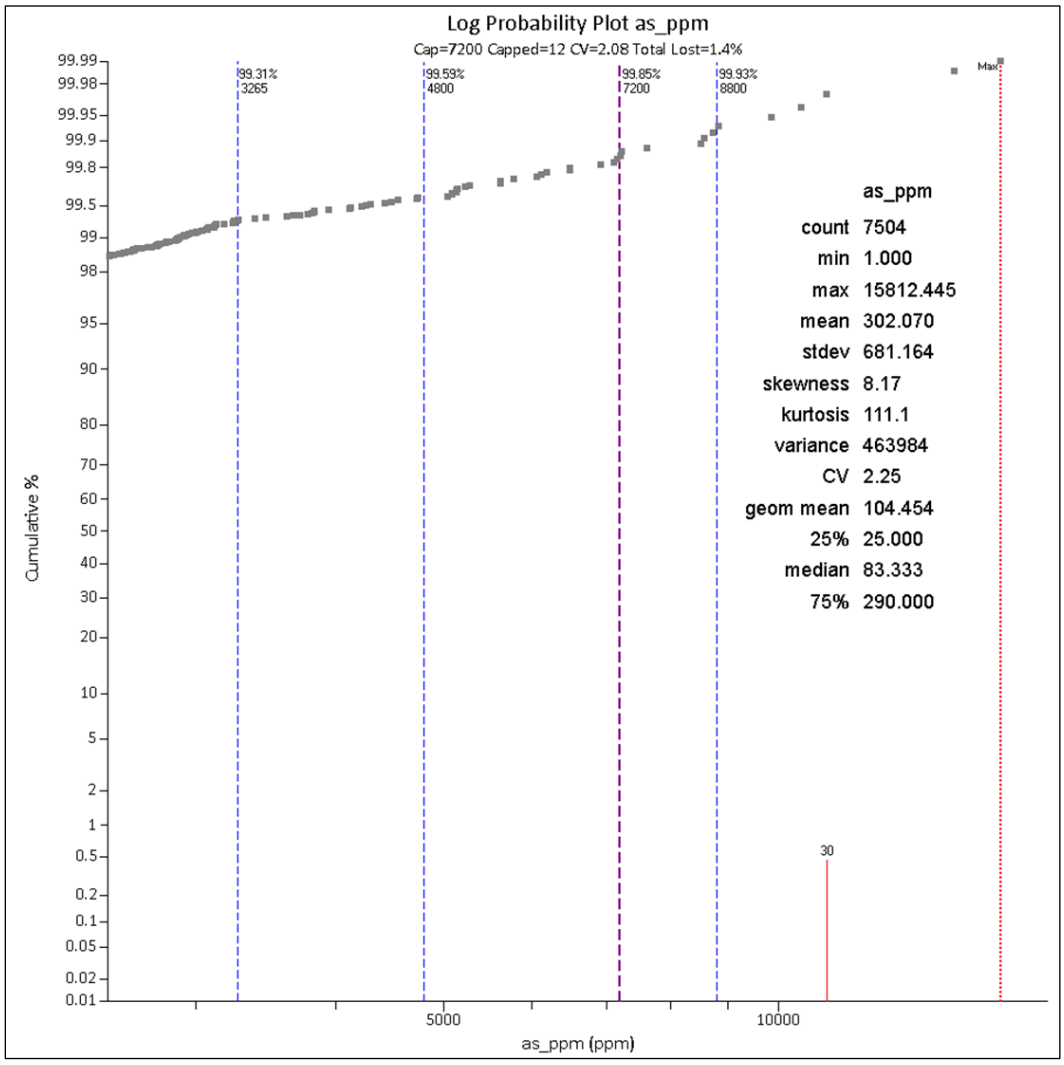


Figure 14.35 Log Probability Plot for As with assigned Cap at 7,200 ppm As in the Lowry deposit

Column	_Filter	Cap	Capped	Percentile	Capped%	Lost	Lost	Count	Min	Max	Mean	Variance	CV
						Total%	CV%						
As_ppm								7504	1	15812	302.1	463984	2.25
As_ppm		8800	6	99.93%	0.10%	0.80%	5%	7504	1	8800	299.7	412264	2.14
As_ppm		7200	12	99.85%	0.20%	1.40%	7.8%	7504	1	7200	297.8	383197	2.08
As_ppm		4800	31	99.59%	0.40%	3.50%	15%	7504	1	4800	291.4	311969	1.92
As_ppm		3265	52	99.31%	0.70%	6.30%	22%	7504	1	3265	282.9	249902	1.77
As_ppm	As_ppm >7200							12	7213	15812	9853	7485820	0.28
As_ppm	As_ppm <= 7200							7492	1	7168	286.8	307373	1.93

Table 14.16 Summary Capping Analysis for As in the Lowry deposit

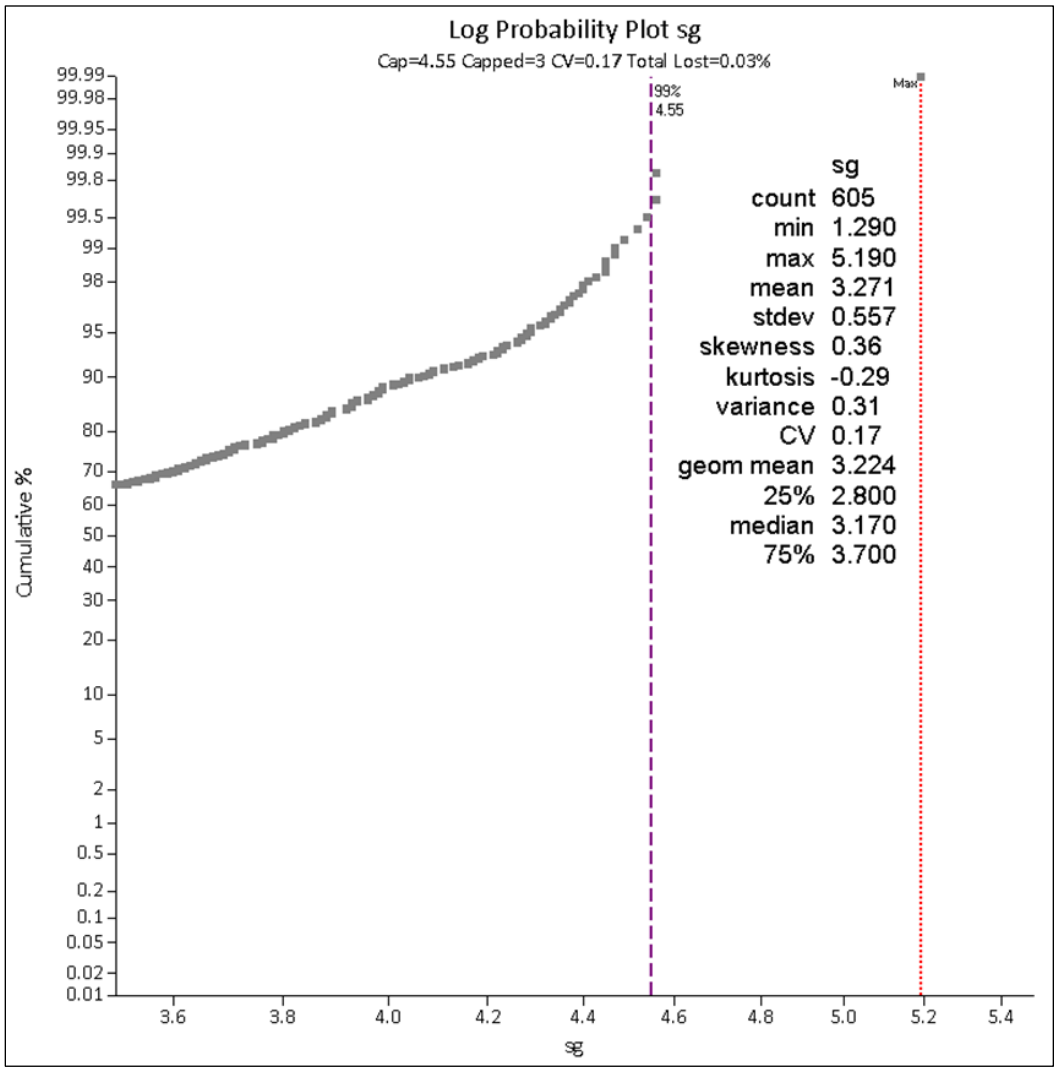


Figure 14.36 Log Probability Plot for SG with Assigned Cap at 4.55 in the Lowry deposit

Variable	_Filter	ColFilter	Count	Min	Max	Mean	Total	Variance	CV
SG		0	605	1.29	5.19	3.271	1979	0.31	0.17
SG	SG >4.55	0	3	4.56	5.19	4.770	14.31	0.13	0.08
SG	SG <= 4.55	0	602	1.29	4.54	3.264	1965	0.30	0.17

Table 14.17 Summary Capping Analysis for SG in the Lowry deposit

Summary descriptive statistics are provided in Table 14.18 for comparison of uncapped and capped composited data for Cu, Ag, and Co. Overall, capping has reduced the highest yield outlier data while not materially affecting the population of data. It is the opinion of the QP that capping has reduced the effect of high-yield outlier values while not materially reducing overall metal content in the Lowry deposit resulting in improved composited database appropriate for estimation.

Variable	Domain	Count	Mean	Min	Max	Median	Variance	StDev	CV	25%	50%	75%	95%	97.50%	99%
Cu_%_Capped	All	5027	0.386	0.001	8.7	0.057	0.81	0.899	2.33	0.01	0.057	0.33	1.987	3.067	4.659
Cu_%_Capped	lcz_Vein_1	60	2.299	0.27	8.685	1.909	2.52	1.588	0.69	1.21	1.909	2.966	5.533	6.217	7.364
Cu_%_Capped	lcz_Vein_2	47	1.782	0.174	4.597	1.42	1.49	1.223	0.69	0.844	1.42	2.402	3.815	3.936	4.297
Cu_%t_Capped	mcz_Halo	565	0.485	0.01	3.831	0.43	0.18	0.421	0.87	0.158	0.43	0.685	1.158	1.437	1.918
Cu_%_Capped	mcz_Vein_1	145	2.507	0.02	8.7	1.982	2.96	1.721	0.69	1.382	1.982	3.083	6.358	7.36	8.523
Cu_%_Capped	mcz_Vein_2	140	2.801	0.183	8.7	2.232	3.79	1.946	0.69	1.42	2.232	3.775	7.086	8.031	8.7
Cu_%_Capped	mcz_Vein_3	40	2.154	0.412	7.627	1.618	2.35	1.532	0.71	1.246	1.618	2.525	5.636	6.357	7.119
Cu_%_Capped	ucz_Vein_1	13	2.518	0.261	5.034	2.055	1.44	1.201	0.48	1.77	2.055	3.33	4.037	4.535	4.835
Cu_%_Capped	ucz_Vein_2	13	2.909	0.849	8.7	1.985	4.38	2.094	0.72	1.577	1.985	3.368	6.316	7.508	8.223
Variable	**Domain**	**Count**	**Mean**	**Min**	**Max**	**Median**	**Variance**	**StDev**	**CV**	**25%**	**50%**	**75%**	**95%**	**98%**	**99.00%**
Cu_%	All	5027	0.39	0.001	12.676	0.057	0.89	0.942	2.41	0.01	0.057	0.33	1.987	3.067	4.659
Cu_%	lcz_Vein_1	60	2.299	0.27	8.685	1.909	2.52	1.588	0.69	1.21	1.909	2.966	5.533	6.217	7.364
Cu_%	lcz_Vein_2	47	1.782	0.174	4.597	1.42	1.49	1.223	0.69	0.844	1.42	2.402	3.815	3.936	4.297
Cu_%	mcz_Halo	565	0.485	0.01	3.831	0.43	0.18	0.421	0.87	0.158	0.43	0.685	1.158	1.437	1.918
Cu_%	mcz_Vein_1	145	2.55	0.02	12.676	1.982	3.64	1.907	0.75	1.382	1.982	3.083	6.358	7.36	9.733
Cu_%	mcz_Vein_2	140	2.879	0.183	12.371	2.232	4.94	2.223	0.77	1.42	2.232	3.775	7.086	8.831	11.63
Cu_%	mcz_Vein_3	40	2.154	0.412	7.627	1.618	2.35	1.532	0.71	1.246	1.618	2.525	5.636	6.357	7.119
Cu_%	ucz_Vein_1	13	2.518	0.261	5.034	2.055	1.44	1.201	0.48	1.77	2.055	3.33	4.037	4.535	4.835
Cu_%	ucz_Vein_2	13	3.139	0.849	11.693	1.985	7.96	2.822	0.9	1.577	1.985	3.368	7.364	9.529	10.827
Variable	**Domain**	**Count**	**Mean**	**Min**	**Max**	**Median**	**Variance**	**StDev**	**CV**	**25%**	**50%**	**75%**	**95%**	**98%**	**99.00%**
Ag_ppm_Capped	All	7504	7.007	0.05	67.2	3.156	75.5	8.689	1.24	0.648	3.156	10.86	24.21	29.89	37.79
Ag_ppm_Capped	lcz_Vein_1	60	8.678	0.746	32.33	6	69.38	8.33	0.96	2.961	6	11.06	27.67	29.46	31.55
Ag_ppm_Capped	lcz_Vein_2	47	7.357	0.5	59.11	4	98.99	9.949	1.35	1	4	10.01	21.91	22.82	42.06
Ag_ppm_Capped	mcz_Halo	567	9.087	0.3	39.21	8.439	40.51	6.365	0.7	4.332	8.439	12.98	21.25	24.2	26.15
Ag_ppm_Capped	mcz_Vein_1	145	13.58	0.5	42.67	11.56	58.53	7.651	0.56	8.257	11.56	18	27.35	29.55	34.86
Ag_ppm_Capped	mcz_Vein_2	140	17.63	0.722	37.07	17.93	51.21	7.156	0.41	12.45	17.93	22.18	29.63	33.3	35.68
Ag_ppm_Capped	mcz_Vein_3	40	15.85	0.499	44.77	13.24	85.71	9.258	0.58	11.08	13.24	22	33	33.91	40.43
Ag_ppm_Capped	ucz_Vein_1	13	13.05	3.203	25.47	11.55	62.78	7.923	0.61	3.968	11.55	17.89	25.12	25.29	25.4
Ag_ppm_Capped	ucz_Vein_2	13	20.34	4.942	59.21	16.15	211.5	14.54	0.72	11.17	16.15	23.45	44.12	51.66	56.19
Variable	**Domain**	**Count**	**Mean**	**Min**	**Max**	**Median**	**Variance**	**StDev**	**CV**	**25%**	**50%**	**75%**	**95%**	**97.50%**	**99%**
Ag_ppm	All	7504	7.013	0.05	78.92	3.156	76.37	8.739	1.25	0.648	3.156	10.86	24.21	29.89	37.79
Ag_ppm	lcz_Vein_1	60	8.678	0.746	32.33	6	69.38	8.33	0.96	2.961	6	11.06	27.67	29.46	31.55
Ag_ppm	lcz_Vein_2	47	7.357	0.5	59.11	4	98.99	9.949	1.35	1	4	10.01	21.91	22.82	42.06
Ag_ppm	mcz_Halo	567	9.087	0.3	39.21	8.439	40.51	6.365	0.7	4.332	8.439	12.98	21.25	24.2	26.15
Ag_ppm	mcz_Vein_1	145	13.58	0.5	42.67	11.56	58.53	7.651	0.56	8.257	11.56	18	27.35	29.55	34.86
Ag_ppm	mcz_Vein_2	140	17.63	0.722	37.07	17.93	51.21	7.156	0.41	12.45	17.93	22.18	29.63	33.3	35.68
Ag_ppm	mcz_Vein_3	40	15.85	0.499	44.77	13.24	85.71	9.258	0.58	11.08	13.24	22	33	33.91	40.43
Ag_ppm	ucz_Vein_1	13	13.05	3.203	25.47	11.55	62.78	7.923	0.61	3.968	11.55	17.89	25.12	25.29	25.4
Ag_ppm	ucz_Vein_2	13	20.34	4.942	59.21	16.15	211.5	14.54	0.72	11.17	16.15	23.45	44.12	51.66	56.19
Variable	**Domain**	**Count**	**Mean**	**Min**	**Max**	**Median**	**Variance**	**StDev**	**CV**	**25%**	**50%**	**75%**	**95%**	**97.50%**	**99%**
Co_ppm_Capped	All	7485	159	0.16	4750	35.15	114484	338.4	2.13	10	35.15	150	705.8	1048	1592
Co_ppm_Capped	lcz_Vein_1	60	491	10.36	2576	374.1	205564	453.4	0.92	170.5	374.1	668	1163	1523	2070
Co_ppm_Capped	lcz_Vein_2	47	403.8	32.97	1672	166.4	187144	432.6	1.07	79.99	166.4	605.5	1217	1229	1464
Co_ppm_Capped	mcz_Halo	567	401	5.858	2138	330.1	126686	355.9	0.89	113	330.1	598	1026	1176	1751
Co_ppm_Capped	mcz_Vein_1	145	1015	30	4750	655.8	1088596	1043	1.03	375.1	655.8	1203	3614	4277	4750
Co_ppm_Capped	mcz_Vein_2	140	1174	35.41	4750	1065	591073	768.8	0.65	672.3	1065	1424	2519	3218	4217
Co_ppm_Capped	mcz_Vein_3	40	900.8	108	1811	991.4	176102	419.6	0.47	495.8	991.4	1174	1364	1794	1804
Co_ppm_Capped	ucz_Vein_1	13	744.9	251.8	1604	676.2	164944	406.1	0.55	311	676.2	926	1326	1465	1548
Co_ppm_Capped	ucz_Vein_2	13	778.8	360.9	1590	636.4	167569	409.4	0.53	425.8	636.4	961.8	1491	1541	1570
Variable	**Domain**	**Count**	**Mean**	**Min**	**Max**	**Median**	**Variance**	**StDev**	**CV**	**25%**	**50%**	**75%**	**95%**	**97.50%**	**99%**
Co_ppm	All	7485	159.1	0.16	5249	35.15	115562	339.9	2.14	10	35.15	150	705.8	1048	1592
Co_ppm	lcz_Vein_1	60	491	10.36	2576	374.1	205564	453.4	0.92	170.5	374.1	668	1163	1523	2070
Co_ppm	lcz_Vein_2	47	403.8	32.97	1672	166.4	187144	432.6	1.07	79.99	166.4	605.5	1217	1229	1464
Co_ppm	mcz_Halo	567	401	5.858	2138	330.1	126686	355.9	0.89	113	330.1	598	1026	1176	1751
Co_ppm	mcz_Vein_1	145	1021	30	5249	655.8	1134196	1065	1.04	375.1	655.8	1203	3614	4277	4926
Co_ppm	mcz_Vein_2	140	1174	35.41	4757	1065	591422	769	0.65	672.3	1065	1424	2519	3218	4217
Co_ppm	mcz_Vein_3	40	900.8	108	1811	991.4	176102	419.6	0.47	495.8	991.4	1174	1364	1794	1804
Co_ppm	ucz_Vein_1	13	744.9	251.8	1604	676.2	164944	406.1	0.55	311	676.2	926	1326	1465	1548
Co_ppm	ucz_Vein_2	13	778.8	360.9	1590	636.4	167569	409.4	0.53	425.8	636.4	961.8	1491	1541	1570

Table 14.18 Summary Descriptive Statistical Comparison of Capped Values in the Lowry deposit

14.5 Variogram Analysis and Modelling

14.5.1 Johnny Lee deposit

The UCZ is transected and offset by Fault 1 into the Eastern and Western blocks. Due to the significant offset, data from the east and west are treated as independent domains for the purposes of variography calculations.

The UCZ Vein and Halo east Cu variograms were modelled using an azimuth of 170°, dip of 5° and a plunge trending south and SW, respectively. The UCZ Vein and Halo west Cu variograms were modelled using an azimuth of 170°, dip of 10° and a high-grade plunge trending south and SE, respectively. The nugget for the UCZ Vein east and west domains is 31% of the total sill and for the Halo domains 20% and 25%, respectively. The nuggets are determined from downhole variograms and are consistent with Cu variability observed in drillholes through the UCZ sequence. Variograms for both the UCZ east and west exhibit minor D1/D2 anisotropy and strong D1/D3 anisotropy in the nearest lags.

The USZ east and west SG variograms were modelled using an azimuth of 170°, dip of 5° and 10° respectively and plunge trending SE. The USZ has nuggets of 18% of the total sill for the east and 14% for the west with minor D1/D2 anisotropy and strong D1/D3 anisotropy in the early lags.

The LCZ is split into three separate Lenses (West, Central, and East) with the majority of drilling occurring in the East Vein. Due to the lower drill density in the West and Central Lenses and discontinuous Cu mineralization between the Lenses, it was deemed appropriate that variography be conducted using data from the East Lens only.

The LCZ Vein Cu variogram was modelled using an azimuth of 75°, dip of 20° and plunge trending south. The LSZ Cu variogram was modelled using an azimuth of 75°, dip of 20° and an isotropic trend. Both have relatively low nugget values with 16% of the total sill for the LCZ Vein and 12% for LSZ consistent with low intrinsic variability in Cu data. The LCZ Vein variogram displays minor D1/D2 anisotropy and strong D1/D3 anisotropy in the nearby lags. The LSZ variogram is isotropic in directions X and Y (near horizontal) with strong D1/D3 anisotropy.

The LSZ SG variogram was modelled using an azimuth of 75°, dip of 20° and W/SW trend with a 9% nugget minor D1/D2 anisotropy and strong D1/D3 anisotropy in the nearby lags.

Experimental and model variograms for the UCZ and LCZ for Cu and density are summarized in Table 14.19 and presented below in Figure 14.37 to Figure 14.45 with all structures being spherical.

		Rotation			Nugget	Sill 1	Structure 1 (m)			Sill 2	Structure 2 (m)		
Variable	Domain	Azimuth	Dip	Pitch			Major	Semi Major	Minor		Major	Semi Major	Minor
Cu	UCZ_Vein_East	170	5	0	0.306	0.528	75	60	6	0.166	140	90	10
Cu	UCZ_Vein_West	170	10	0	0.314	0.353	120	100	2	0.333	210	130	10
Cu	UCZ_Halo_East	170	5	-130	0.2	0.45	75	70	2	0.35	260	210	10
Cu	UCZ_Halo_West	170	10	150	0.25	0.45	80	65	4	0.3	230	150	10
SG	USZ_East	170	5	-25	0.175	0.364	90	60	2.5	0.461	250	180	15
SG	USZ_West	170	10	140	0.14	0.39	100	80	4	0.47	200	160	15
Cu	LCZ_Vein	75	20	-70	0.16	0.33	85	75	4	0.51	155	130	10
Cu	LSZ	75	20	-50	0.12	0.05	90	90	5	0.83	200	200	10
SG	LSZ	75	20	-160	0.094	0.313	80	80	4	0.593	180	120	10

Table 14.19 Summary of variogram parameters for the Johnny Lee deposit

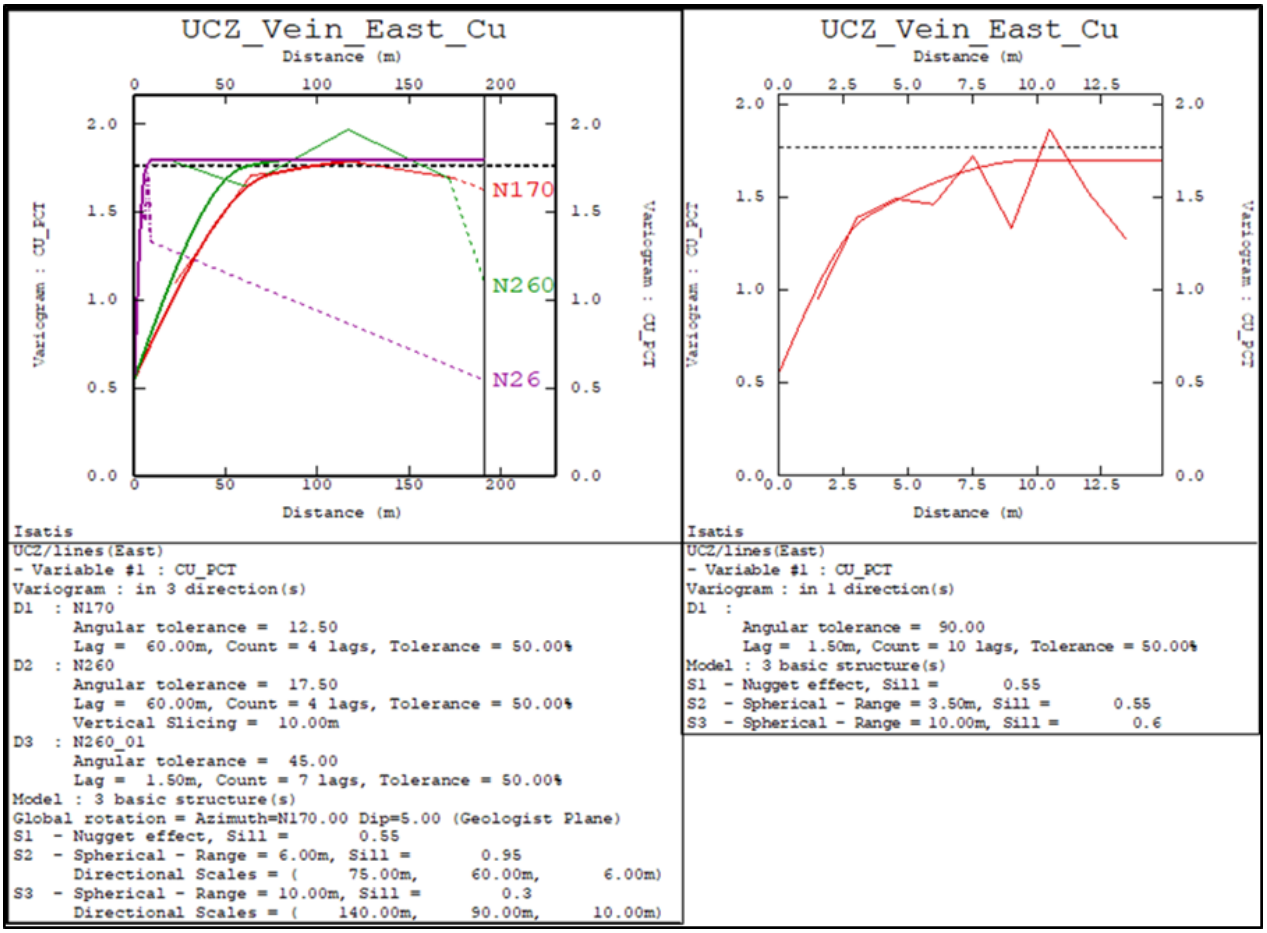


Figure 14.37 Experimental and Modelled Variogram for UCZ Vein East Cu. Directional variogram on the left and downhole variogram on the right

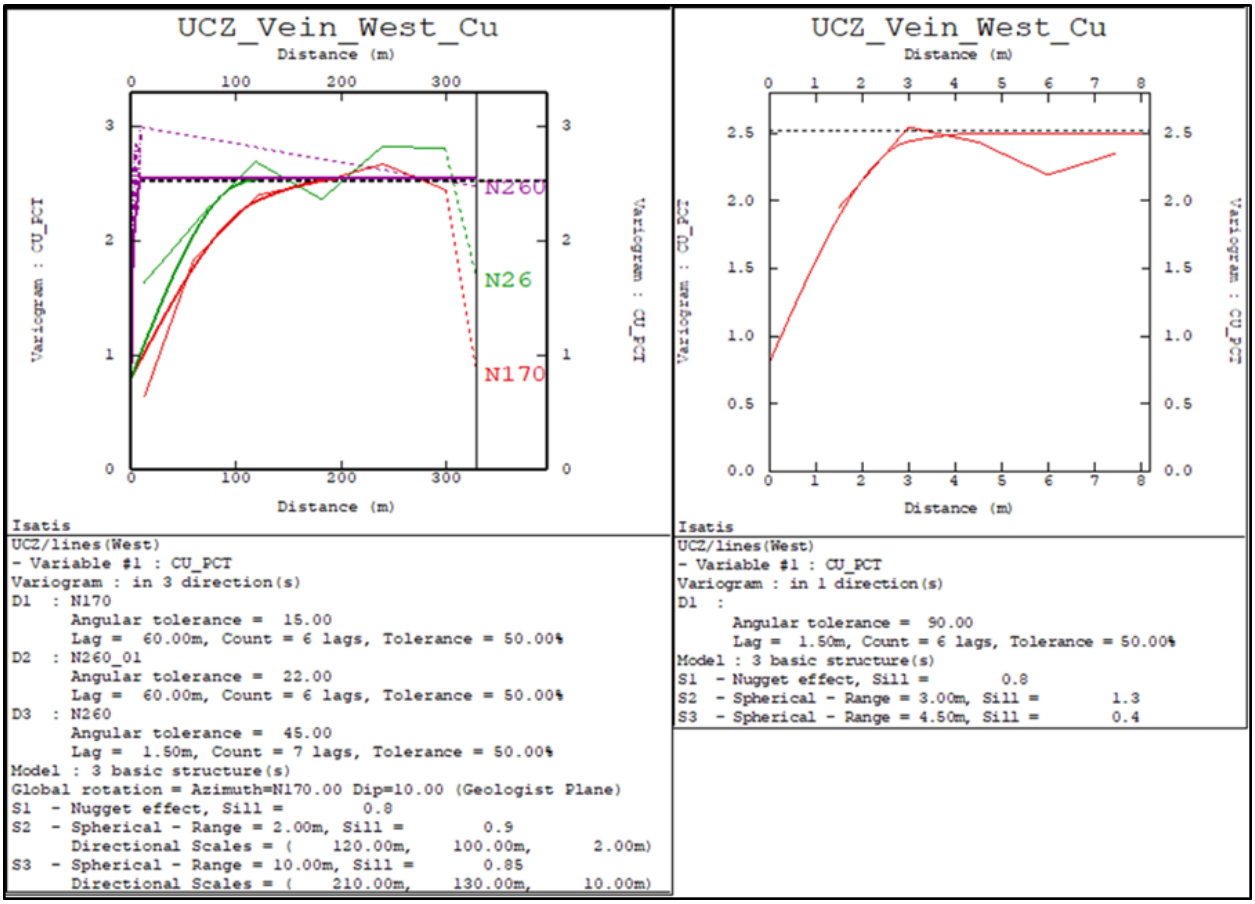
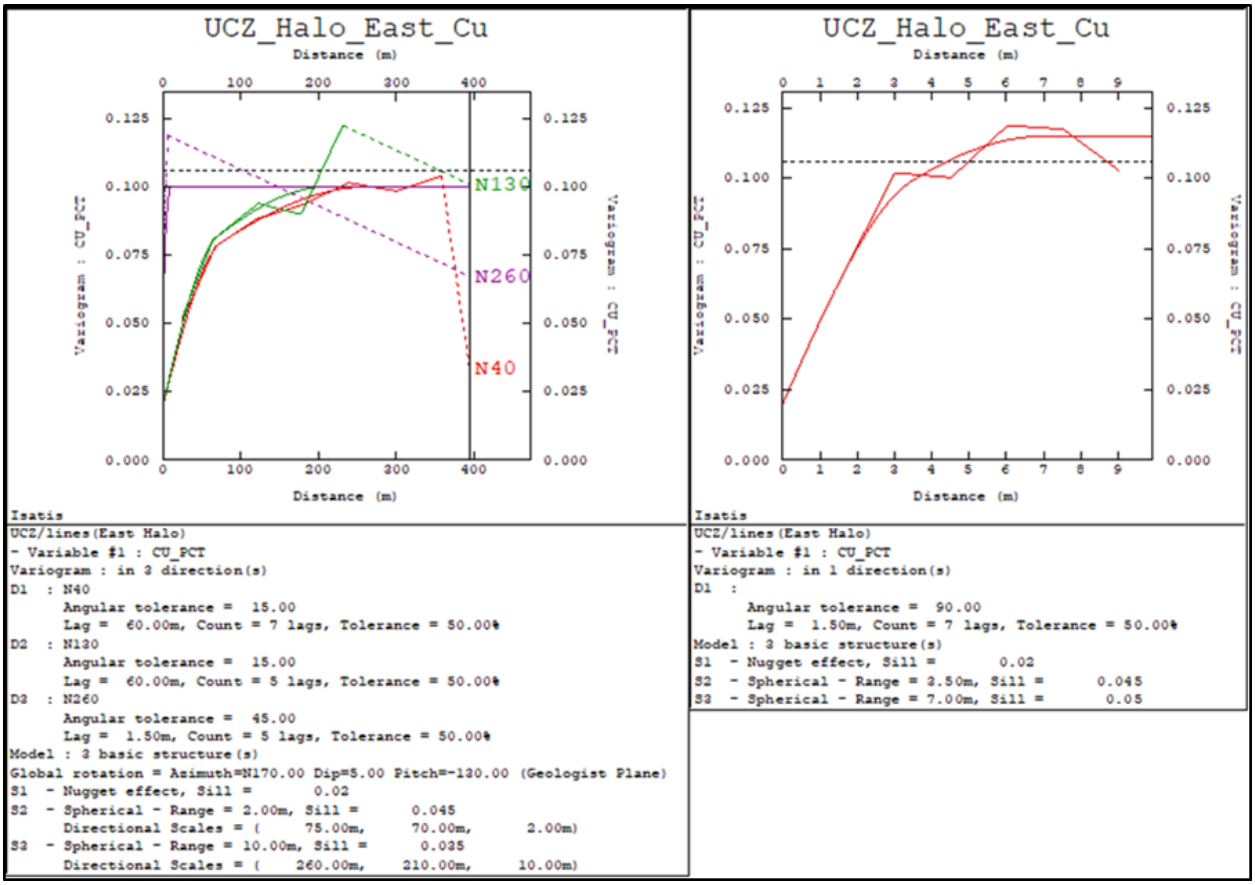


Figure 14.38 Experimental and Modelled Variogram for UCZ Vein West Cu. Directional variogram on the left and downhole variogram on the right



UCZ/lines(East Halo)
- Variable #1 : CU_PCT
Variogram : in 3 direction(s)
D1 : N40
Angular tolerance = 15.00
Lag = 60.00m, Count = 7 lags, Tolerance = 50.00%
D2 : N130
Angular tolerance = 15.00
Lag = 60.00m, Count = 5 lags, Tolerance = 50.00%
D3 : N260
Angular tolerance = 45.00
Lag = 1.50m, Count = 5 lags, Tolerance = 50.00%
Model : 3 basic structure(s)
Global rotation = Azimuth=N170.00 Dip=5.00 Pitch=-130.00 (Geologist Plane)
S1 - Nugget effect, Sill = 0.02
S2 - Spherical - Range = 2.00m, Sill = 0.045
Directional Scales = (75.00m, 70.00m, 2.00m)
S3 - Spherical - Range = 10.00m, Sill = 0.035
Directional Scales = (260.00m, 210.00m, 10.00m)

UCZ/lines(East Halo)
- Variable #1 : CU_PCT
Variogram : in 1 direction(s)
D1 :
Angular tolerance = 90.00
Lag = 1.50m, Count = 7 lags, Tolerance = 50.00%
Model : 3 basic structure(s)
S1 - Nugget effect, Sill = 0.02
S2 - Spherical - Range = 3.50m, Sill = 0.045
S3 - Spherical - Range = 7.00m, Sill = 0.05

Figure 14.39 Experimental and Modelled Variogram for UCZ Halo East Cu. Directional variogram on the left and downhole variogram on the right

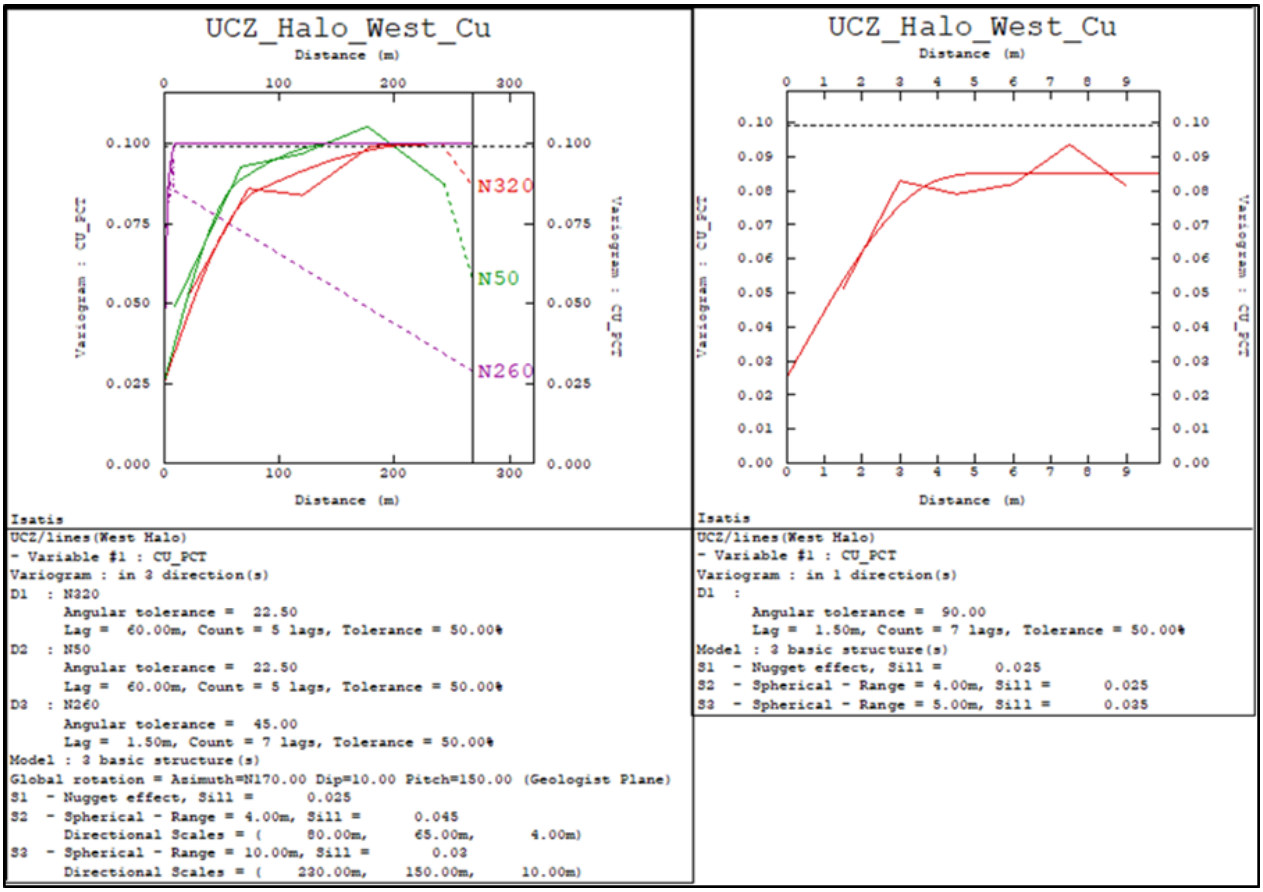


Figure 14.40 Experimental and Modelled Variogram For UCZ Halo West Cu. Directional variogram on the left and downhole variogram on the right

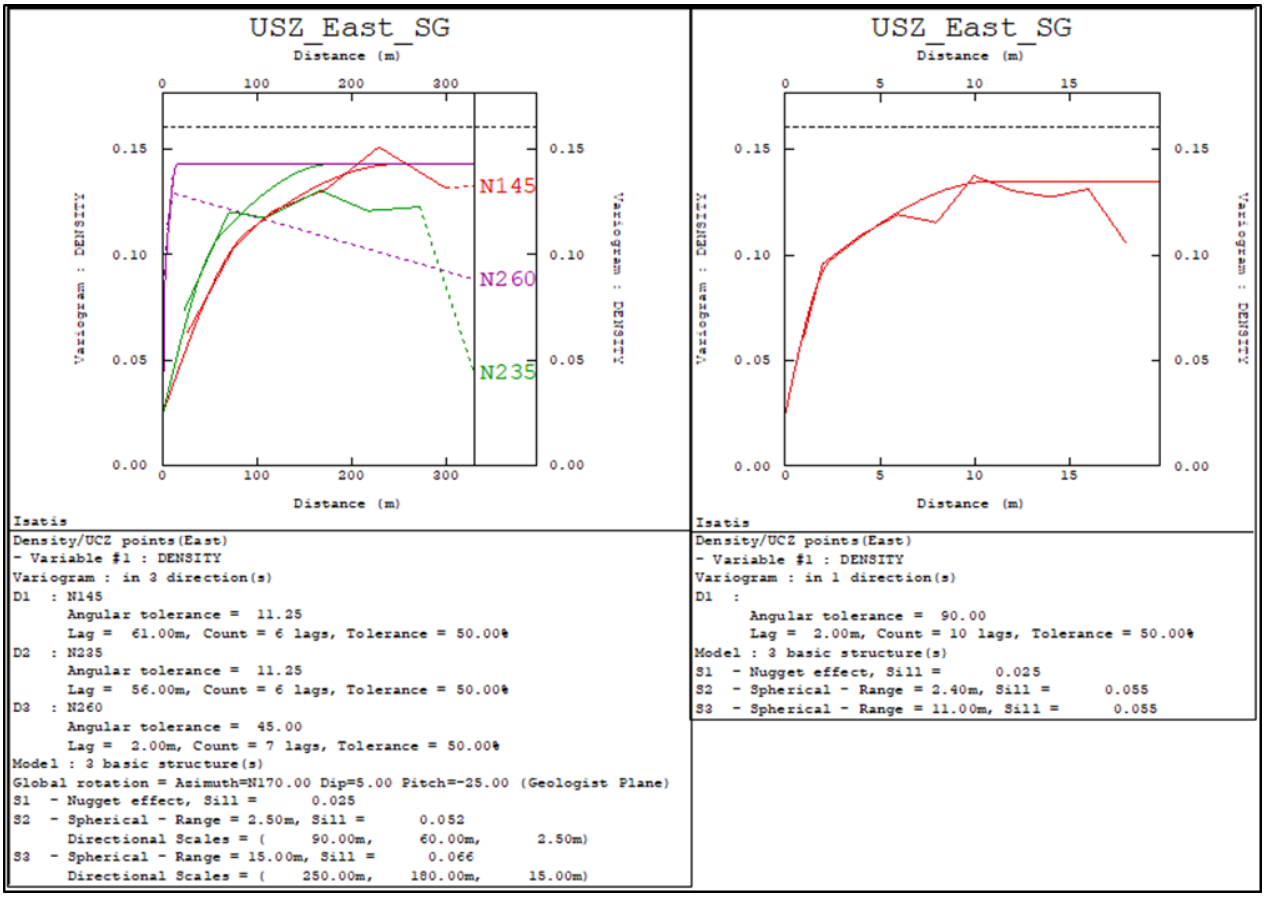


Figure 14.41 Experimental and Modelled Variogram for USZ East SG. Directional variogram on the left and downhole variogram on the right

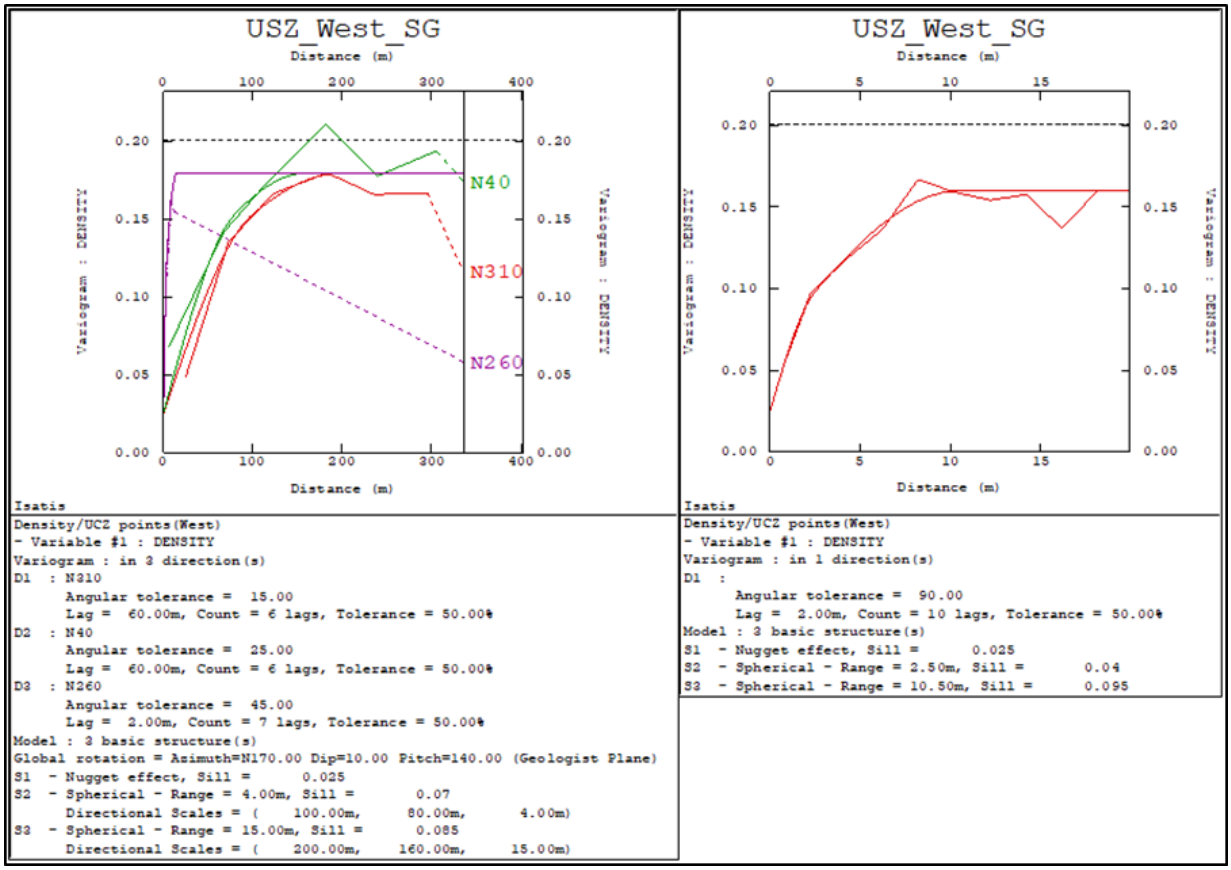


Figure 14.42 Experimental and Modelled Variogram For USZ West SG. Directional variogram on the left and downhole variogram on the right

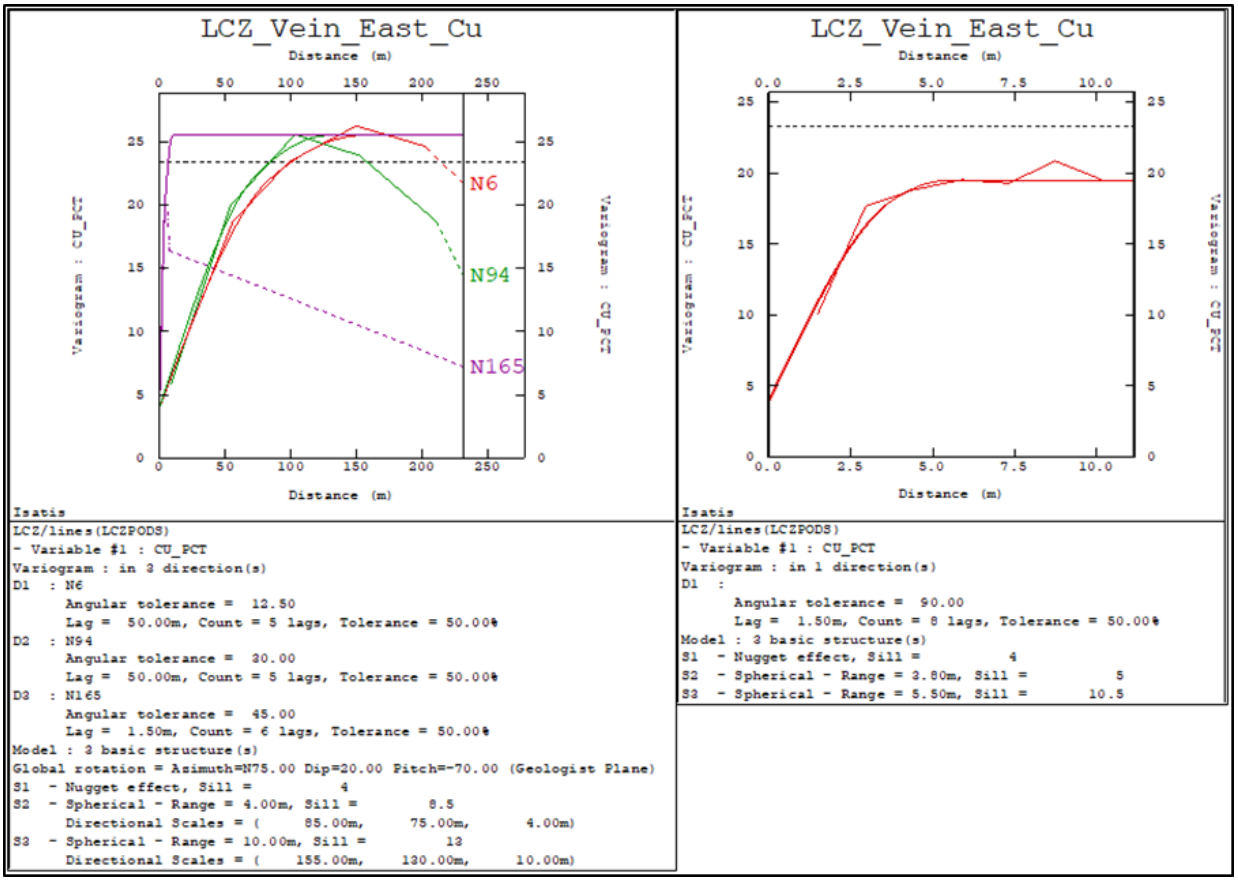


Figure 14.43 Experimental and Modelled Variogram for LCZ Vein East Cu. Directional variogram on the left and downhole variogram on the right

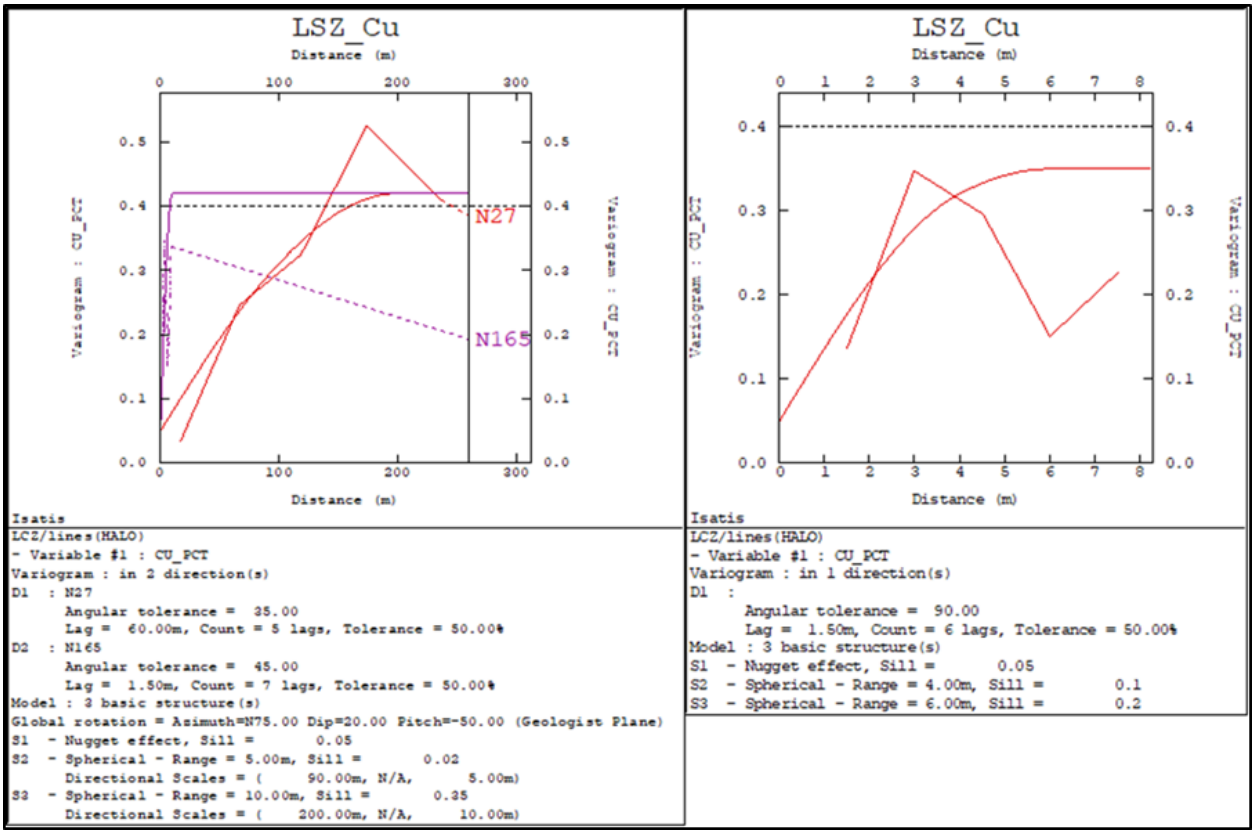


Figure 14.44 Experimental and Modelled Variogram for LSZ Cu. Directional variogram on the left and downhole variogram on the right

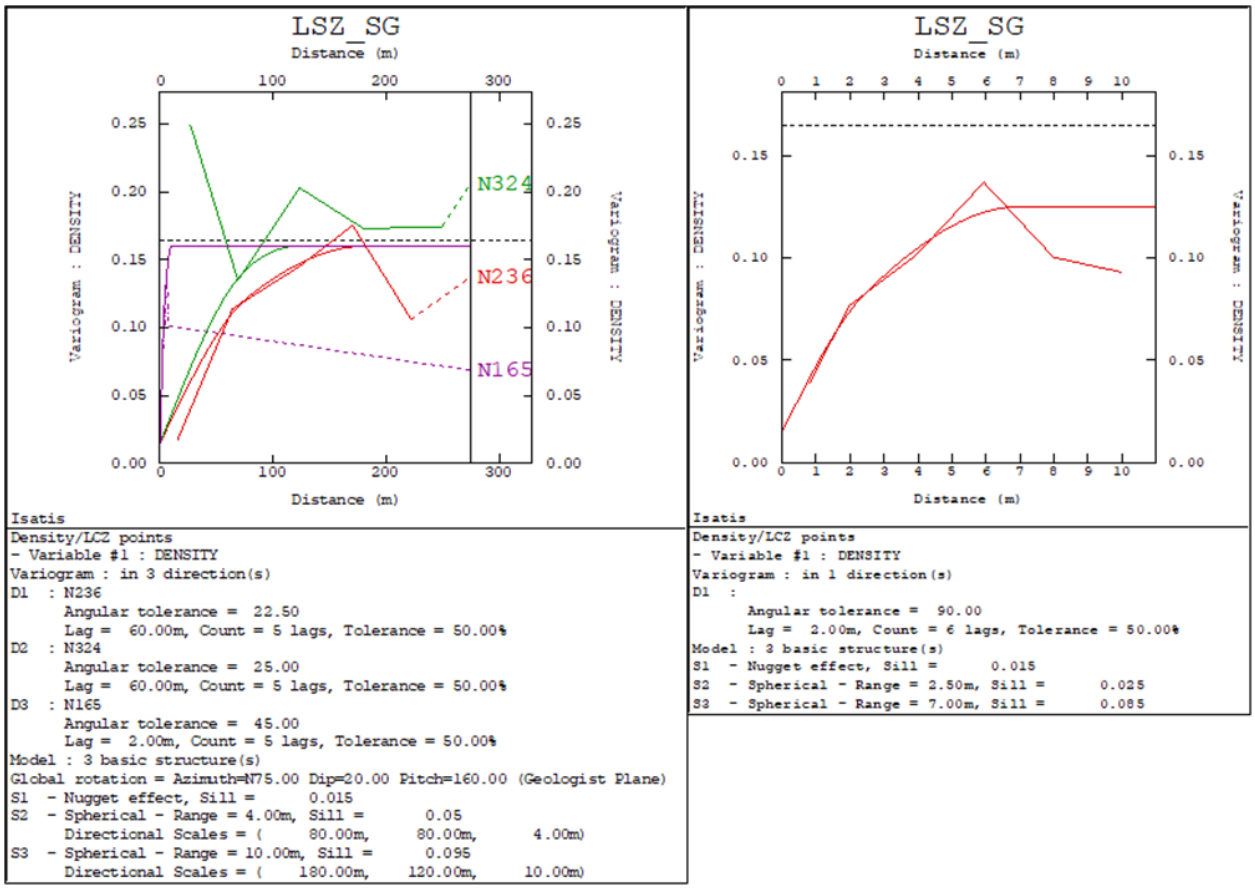


Figure 14.45 Experimental and Modelled Variogram for LSZ SG. Directional variogram on the left and downhole variogram on the right

14.5.2 Lowry deposit

Spatial analysis at the Lowry deposit was calculated for the key economic variable Cu by domain using variography. All variography maps, experimental variogram, downhole variograms, and modelled variograms were calculated using Geovariances' Isatis™ software, version 2018.3. Variography was not calculated for the secondary variables.

Variogram maps were generated for Cu by domain to determine a predominant orientation of continuity within each wireframed domain. Outputs of the variogram map analysis show the primary direction of spatial continuity aligns with the major geological direction, typically down dip along each mineralized horizon or Vein. The dip direction or bearing was typically nearly north – south or between 350° and 015° azimuth (Figure 14.46).

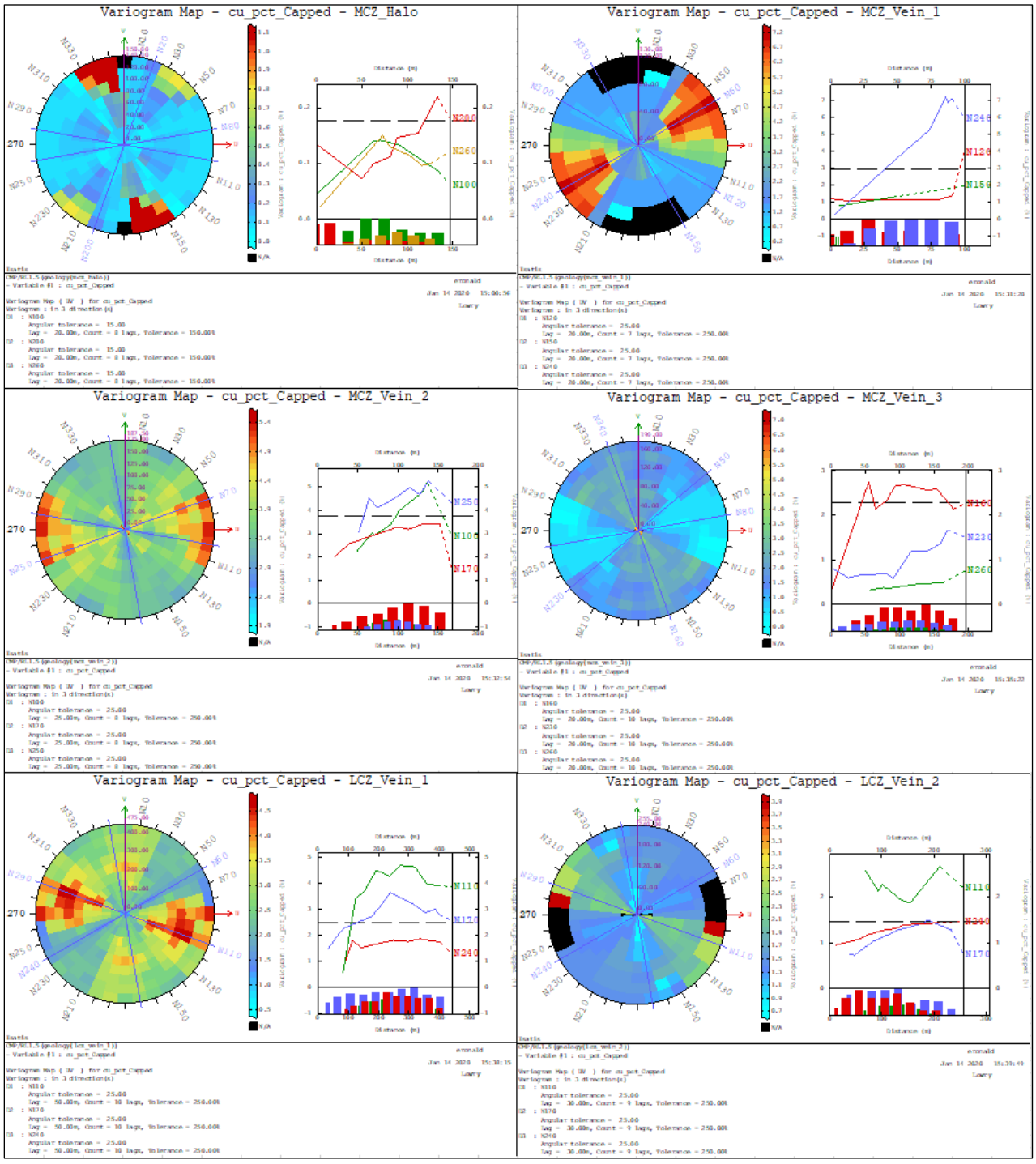


Figure 14.46 Multiple Variogram Maps for the Various Mineralized Domains at the Lowry deposit

In some cases, no robust variogram could be calculated due to poor continuity and/or low data population for particular domains. Omni-directional variography was determined to provide the most appropriate fit (e.g. Figure 14.47), except for Cu in the LMCZ Vein 2 domain where a directional modelled variogram could be calculated (Figure 14.48). The following figures show modelled variography for Cu in domains in which it could be determined (Figure 14.49 to Figure 14.51). Summary variography data is provided in Table 14.20.

Spatial continuity was calculated only for Cu by domain as the primary economic variable of interest. Experimental variograms were calculated with acceptable structure in the LMCZ Vein 1, LMCZ Vein 2, LMCZ Halo, LLCZ Vein 1, and the LLCZ Vein 2 domains. Due to poor structure and a lack of samples, acceptable variograms could not be calculated for Cu in the LMCZ Vein 3 and LLCZ Vein 2 domains. A directional variogram was determined for Cu in the LMCZ Vein 2 while all other variograms for Cu in the other domains were determined from omni-directional variography.

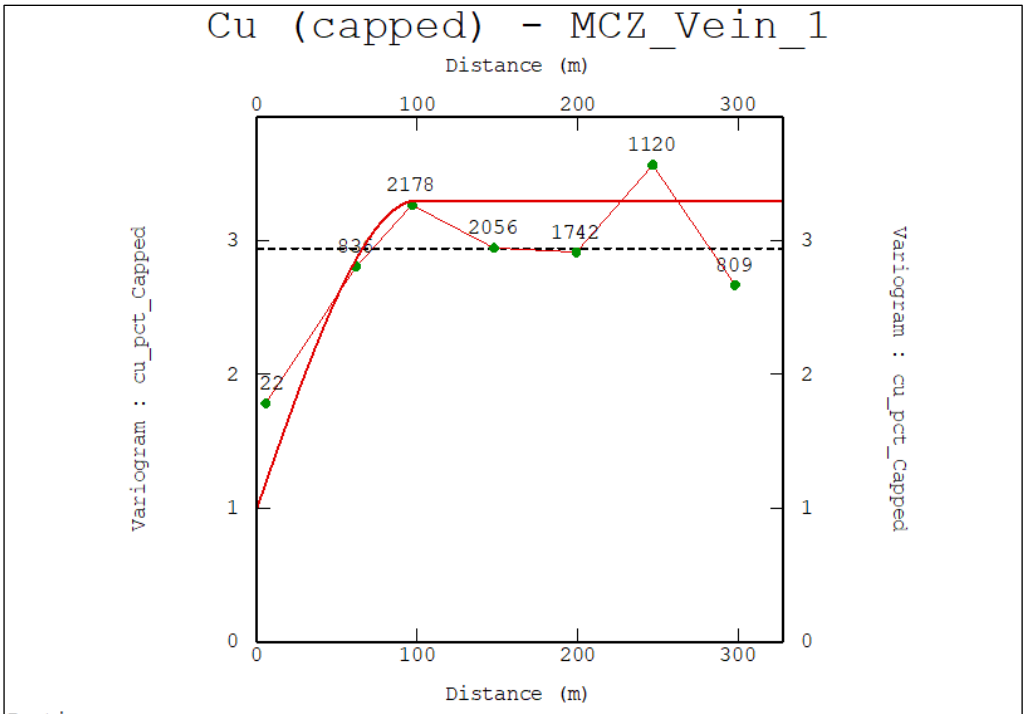


Figure 14.47 Omni-directional Modelled Variogram for Cu in the LMCZ Vein 1 Domain

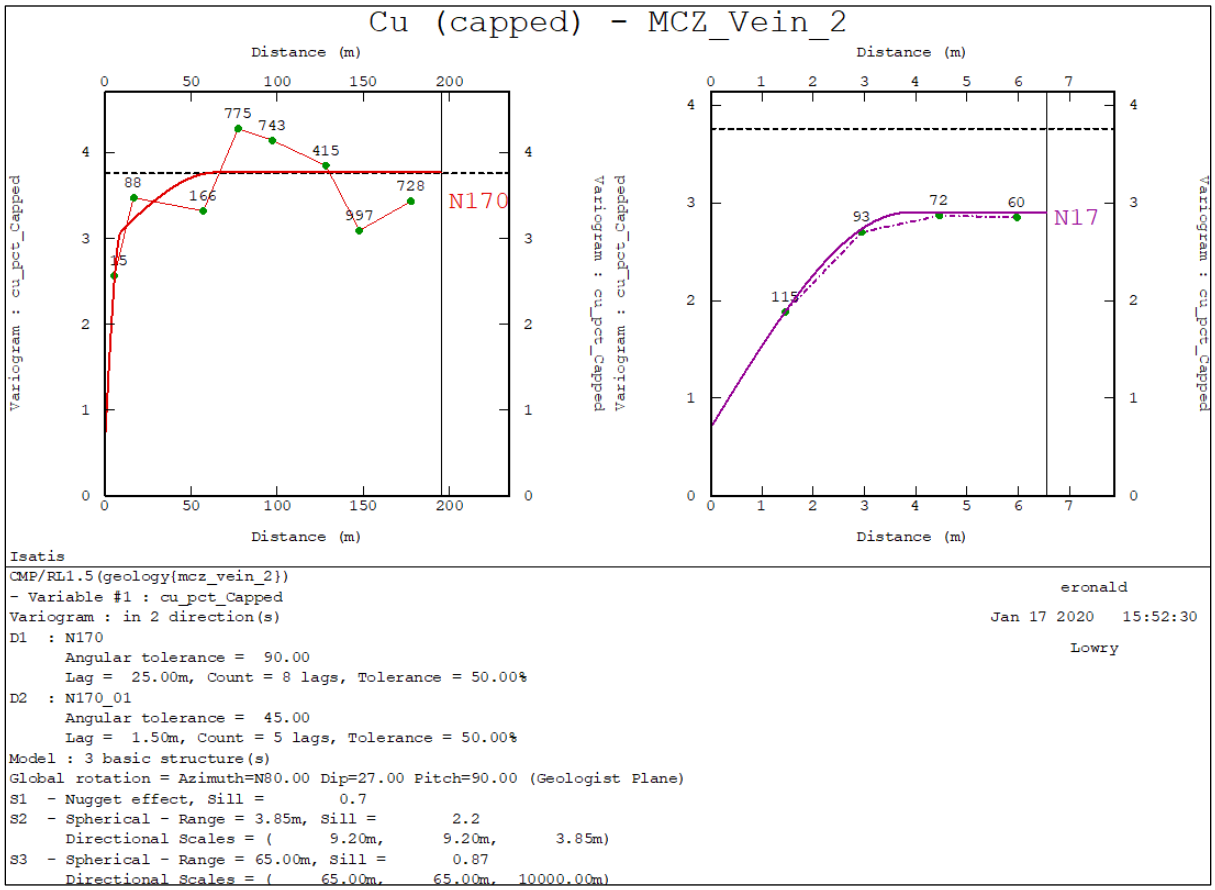


Figure 14.48 Directional Modelled Variography for Cu in the LMCZ Vein 2 Domain

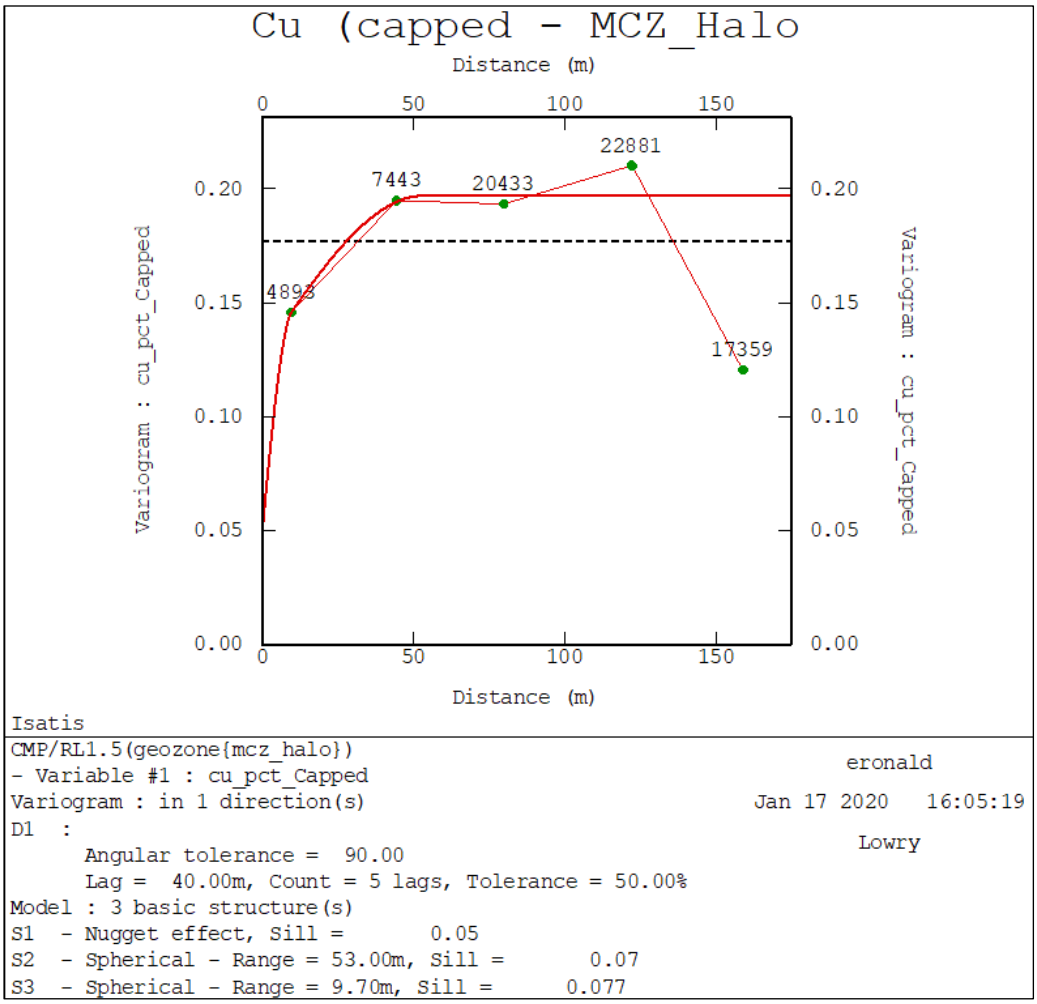


Figure 14.49 Omni-directional Modelled Variography for Cu in the LMCZ Halo Domain

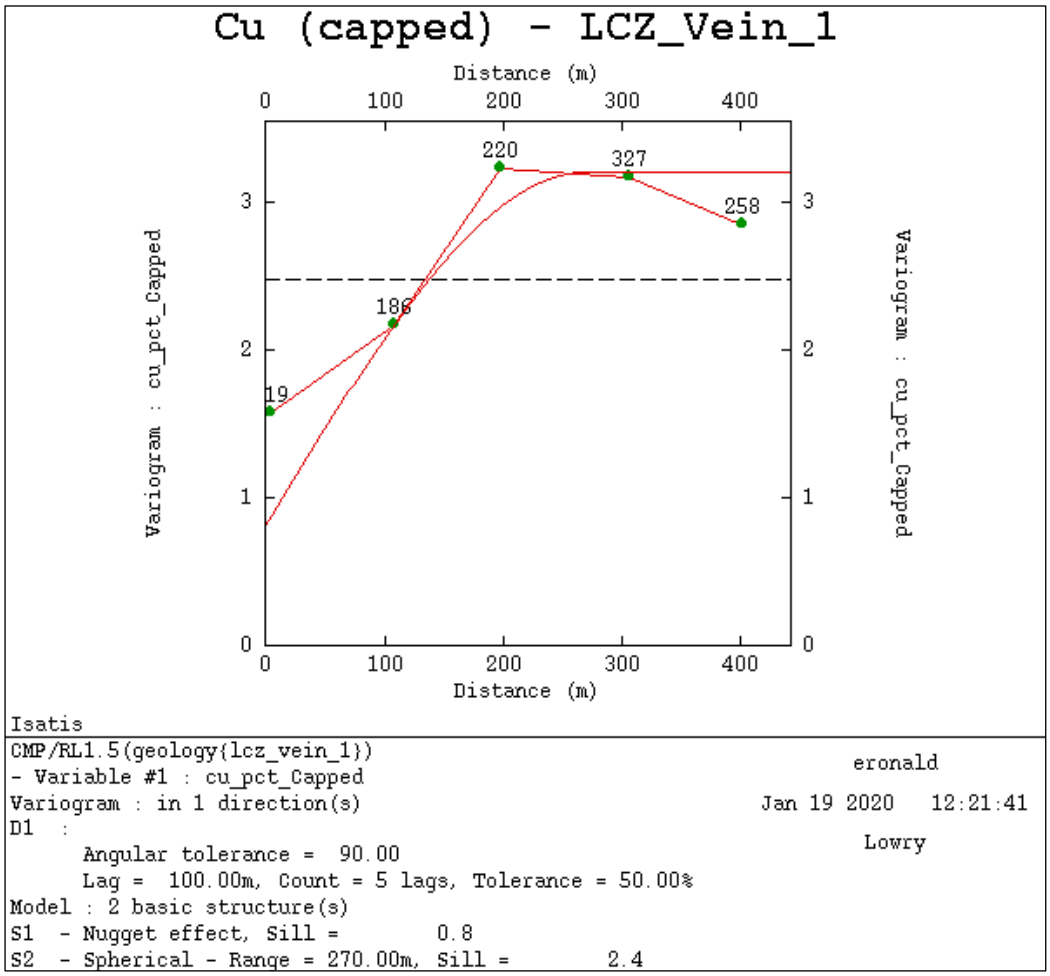


Figure 14.50 Omni-directional Modelled Variography for Cu in the LLCZ Vein 1 Domain

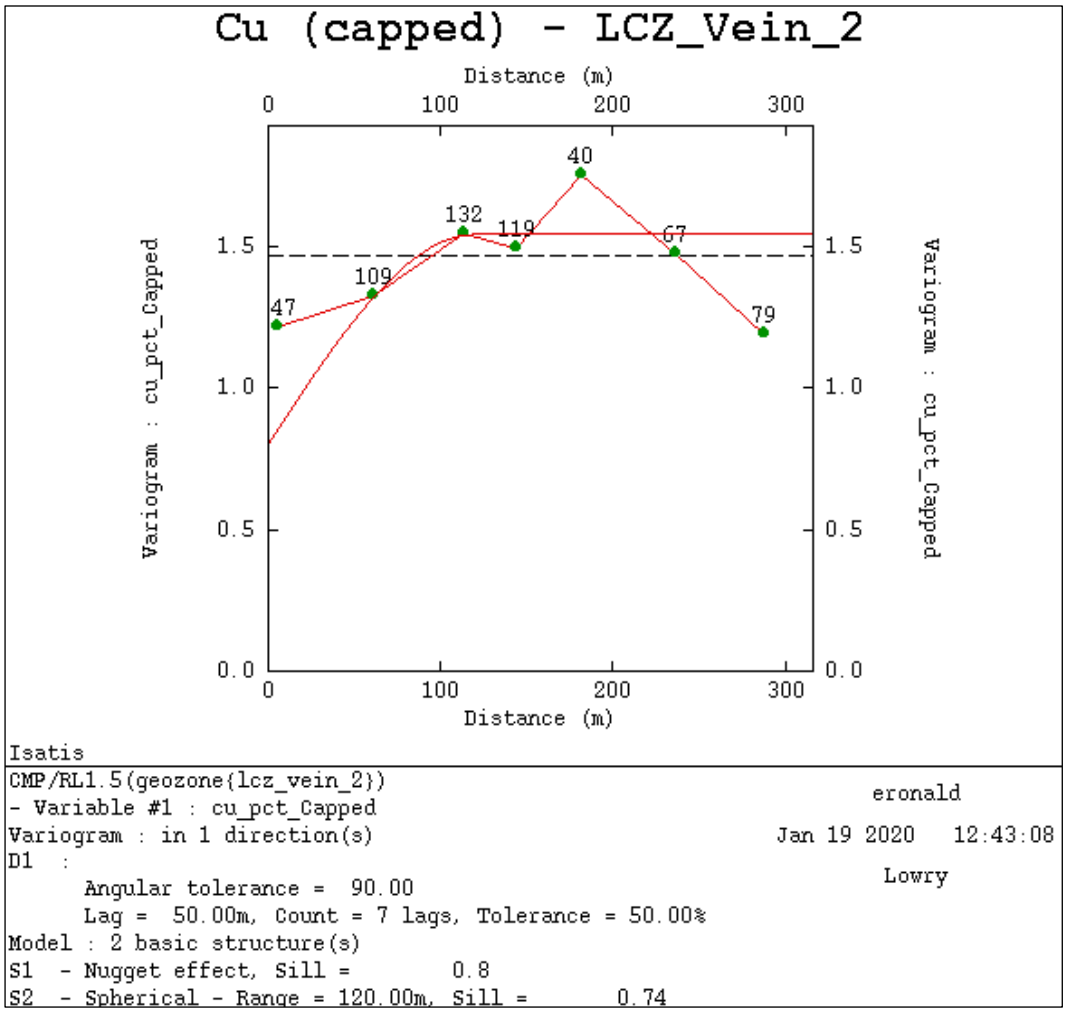


Figure 14.51 Omni-directional Modelled Variography for Cu in the LLCZ Vein 2 Domain

Domain	Variable	Nugget	Structures	Structure 1		Rotation			Range (ft)			Structure 2		Rotation			Range (ft)		
				Type	Sill	Bearing	Plunge	Dip	Major	Semi-major	Minor	Type	Sill	Bearing	Plunge	Dip	Major	Semi-major	Minor
LMCZ_Vein_1	Cu	1	1	spherical	2.3	90	0	0	100	100	100								
LMCZ_Vein_2	Cu	0.7	2	spherical	2.2	10	27	0	9.2	9.2	3.85	spherical	0.89	10	27	0	65	65	10000
LMCZ_Vein_3	Cu	no variogram due to insufficient samples																	
LMCZ_Halo	Cu	0.05	2	spherical	0.07	90	0	0	53	53	53	spherical	0.077	90	0	0	9.7	9.7	9.7
LLCZ_Vein_1	Cu	0.8	1	spherical	2.4	90	0	0	270	270	270								
LLCZ_Vein_2	Cu	no variogram due to insufficient samples																	

Table 14.20 Summary Variography for Cu at the Lowy deposit

14.6 Block Model

14.6.1 Johnny Lee deposit

The resource block models for the UCZ and LCZ were built and estimated using Datamine Studio RM™ version 1.4.132.0. A summary of model geometries is presented in Table 14.21. The block models are defined in the NAD83/UTM Zone 12 North coordinate system. Due to the different supports for drilling data and geometallurgical data, each block model has an optimal parent block size based on data spacing and is estimated independently.

UCZ and LCZ Cu and SG			
	Northing	**Easting**	**Elevation**
Minimum Coordinates	506,100	5,180,000	1,100
Maximum Coordinates	507,500	5,181,300	1,802
Parent Block Size	25 m	25 m	3 m
Minimum Sub-cell	5 m	5 m	1 m
UCZ Recovery and As in Concentrate			
	Northing	**Easting**	**Elevation**
Minimum Coordinates	506,100	5,180,000	1,100
Maximum Coordinates	507,500	5,181,300	1,820
Parent Block Size	30 m	30 m	5 m
Minimum Sub-cell	5 m	5 m	1 m

Table 14.21 Johnny Lee Model Geometries

For the UCZ, once both models are complete the function slimod in Datamine Studio RM™ allows for the IJK (index value giving the location of the parent cell) of the Cu and SG model to be adjusted to match the prototype of the recovery and As in concentrate model and allow the models to be combined into a single resource block model. Waste blocks encompass both the UCZ and LCZ and are assigned 0.01% Cu, 3.04 SG and 0% recovery. Block model attributes are presented in Table 14.22.

Block Model Attributes	Description
IJK	Parent cell address of the respective blocks
XC	Parent/sub-cell center coordinates in the X direction
YC	Parent/sub-cell center coordinates in the Y direction
ZC	Parent/sub-cell center coordinates in the Z direction
XINC	Increment of cell in metres in the X direction
YINC	Increment of cell in metres in the Y direction
ZINC	Increment of cell in metres in the Z direction
XMORIG	Origin X coordinate - Lower left center of parent cell
YMORIG	Origin Y coordinate - Lower left center of parent cell
ZMORIG	Origin Z coordinate - Lower left center of parent cell
NX	Number of parent cells in the X direction
NY	Number of parent cells in the Y direction
NZ	Number of parent cells in the Z direction
CU	Estimated Cu grade in%
DOMAIN	Numeric Code for Lode (11 – UCZ Vein east, 12/13 – UCZ Vein west, 101 – UCZ Halo east, 102 – UCZ Halo west, 103 – USZ east, 104 – USZ west, 21/22/23 LCZ east, central and west Veins, 201 – LSZ, 1000 – Waste)
DENSITY	Estimated Density in g/cm3
CLASS	Classification (Mea = Measured, Ind = Indicated and Inf = Inferred)
T REC	Estimated Total Recovery for UCZ. Supergene zones applied 69.8% Recovery (LCZ 94%).
CU R	Recoverable Cu grade = CU * (T_REC/100)
TENNAN	Estimated Tennantite ppm (As in concentrate) for UCZ (LCZ 230ppm)

Table 14.22 Johnny Lee Model Attributes

It is the opinion of the QP that the Johnny Lee deposit block model provides an acceptable representation of interpreted 3-D wireframes domaining the primary mineralized zones. The parent block size was selected to allow for appropriate estimation of quality variables into a volume given the data spacing, composite size, and assumed mining method. The estimated variables are acceptable for use in economic and engineering analyses.

14.6.2 Lowry deposit

The Lowry deposit resource block model was created using Maptek™ Vulcan™ software, version 11.0.4. The extents were selected to encompass the majority of Lowry drilling and the primary mineralized zones as illustrated in plan view in Figure 14.52. The model was created in Universal Transverse Mercator (UTM) coordinate system, Zone 12N. As per Vulcan format, the model origin is located at the bottom left corner of the model box with origin coordinates and offset extents:

X: 508550.0 East with offset of 1,300 m;
Y: 5179150.0 North with offset of 960 m; and
Z: 800.0 Elevation with offset of 1,000 m.

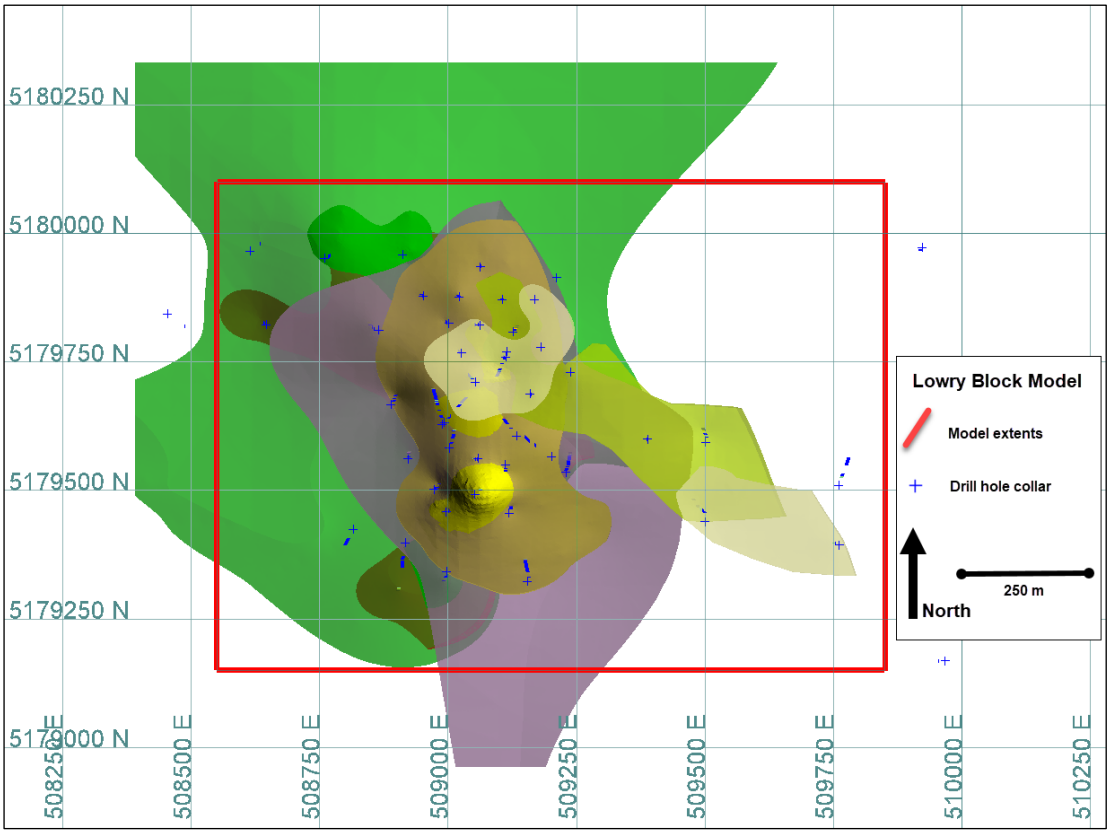


Figure 14.52 Lowry deposit Block Model Extents

The block model is a sub-blocked model with parent blocks at 20 m (X) by 20 m (Y) by 5 m (Z) and sub-blocks minimal size of 5 m (X) by 5 m (Y) by 1 m (Z). The model is not rotated with origination of 90,0,0 in the Vulcan rotation scheme (Table 14.23). The model parent block size was selected based upon the nominal drilling data spacing between 40 m and 100 m within the Lowry deposit. The sub-block size was selected in order to provide an accurate volumetric representation of the modelled wireframes that accounts for the thin nature of mineralization in some areas of the deposit.

Origin		Rotation	
X	508550	Bearing	90
Y	5179150	Plunge	0
Z	800	Dip	0
Block Size	**Parent**	**Sub-block**	
X	20	5	
Y	20	5	
Z	5	1	

Table 14.23 Summary Input Parameters for the Lowry Block Model

Multiple 3-D wireframes were used to code the block model with topography, host rocks to mineralization, low-grade "Halo" (<0.25% Cu) mineralization, and higher-grade (>1.2% Cu) mineralized Veins. The wireframes were constructed by SRA geological staff and reviewed by the QP for validity and validation against raw drilling data. Details of the wireframe modelling is provided in Section 14.2.

Variables estimated in the Lowry deposit block model include numeric and alphanumeric variables. Summary variables are tabulated in Table 14.24.. The key economic and potentially economic quality variables include Cu (Cu_pct), Ag (Ag_ppm), Co (Co_ppm), and As (As_ppm). SG has also been estimated. Alphanumeric variables include land ownership, Cu zone, mineralization zone (minzone), and mineral resource classification (class).

Multiple estimation quality variables were calculated that allow for quantitative assessment of block quality using a variety of data to aid in Mineral Resource classification. These variables have the prefix "est" and include outputs such as: Kriging efficiency, number of samples, number of drill holes, average distance to samples, and estimation pass. Lastly, three variables are coded in the model that provide local anisotropy used in the search neighborhood to provide a unique neighborhood that aligns with the geometry of the modelled domain. These variable search anisotropy variables include the prefix "LVA" with bearing, plunge, and dip.

Variables	Default	Type	Description
land	none	alphanumeric	land parcel ownership
copperzone	None	alphanumeric	Zone (LUCZ, LMCZ, LLCZ)
minzone	none	alphanumeric	mineralized horizon/vein
class	none	alphanumeric	resource classification (mea, ind, inf, none)
Cu_pct	-99	numeric	copper (%) – OK and IDW
Ag_ppm	-99	numeric	silver (g/t)
Co_ppm	-99	numeric	cobalt (ppm)
As_ppm	-99	numeric	arsenic (ppm)
SG	-99	numeric	specific gravity

Variables	Default	Type	Description
est_cu_nn	-99	numeric	copper (%) - NN
est_nsam	-99	numeric	number of samples
est_ndh	-99	numeric	number of drill holes
est_avgdist	-99	numeric	average distance to samples
est_ke	-99	numeric	kriging efficiency
est_run	-99	numeric	estimation run pass
lva_bear	-99	numeric	LVA bearing
lva_plunge	-99	numeric	LVA plunge
lva_dip	-99	numeric	LVA dip

Table 14.24 Summary of Variables in the Lowry Block Model

It is the opinion of the QP that the Lowry deposit block model provides an acceptable representation of interpreted 3-D wireframes domaining the primary mineralized zones. The parent block size was selected to allow for appropriate estimation of quality variables into a volume given the data spacing and composite size. The estimated variables are acceptable for use in economic and engineering analyses.

14.7 Estimation Methodology

14.7.1 Johnny Lee deposit

The primary Cu and SG domains were estimated using Ordinary Kriging (OK). Variography and search neighborhoods (Table 14.25) were defined by the geometry of the orebody and kriging parameters were optimized using Kriging Neighborhood Analysis (KNA). The estimation was performed using two search volumes: the first search is restricted to approximately 90-95% of the variogram range based on review of variograms for each domain which estimates the majority of blocks. The second search is five times the search distances to estimate remaining peripheral blocks that typically contain reduced data support. Due to undulations in the mineralization geometry for the UCZ and LCZ, dynamic anisotropy is employed to control searches and improve the local estimate.

Variable	Domain	Rotation			Maximum Search Distances			Block Discretization			Optimum Samples	Minimum Samples	Angular Sectors
		Azimuth	Dip	Pitch	U	V	W	X	Y	Z			
CU	UCZ_Vein_East	Dynamic Anisotropy			75	50	10	2	2	2	12	6	1
CU	UCZ_Vein_West	Dynamic Anisotropy			125	80	10	2	2	2	12	6	1
CU	UCZ_Halo_East	Dynamic Anisotropy			190	154	15	2	2	2	12	6	1
CU	UCZ_Halo_West	Dynamic Anisotropy			150	98	15	2	2	2	12	6	1
SG	USZ_East	Dynamic Anisotropy			130	104	15	2	2	2	12	6	1
SG	USZ_West	Dynamic Anisotropy			140	101	15	2	2	2	12	6	1
CU	LCZ_Vein	Dynamic Anisotropy			90	75	10	2	2	2	12	6	1
CU	LSZ	Dynamic Anisotropy			140	140	15	2	2	2	10	6	1
SG	LSZ	Dynamic Anisotropy			120	80	15	2	2	2	12	6	1

Table 14.25 Search parameters for Cu and SG by Domain for Johnny Lee deposit

The estimation of recovery for the UCZ via OK was not suitable due to insufficient data support. In order to convert this data into a 3-D geometallurgical model, various interpolation protocols were trialled, including: Radial Base Function isotropic, Radial Base Function anisotropic, IDW2 and Inverse Distance Weighted Cubed (IDW3) (Hilliard, 2019). The best correlation between block estimates and the source data, geology, and geometallurgical logging was obtained using IDW2 (see Section 13 for additional information).

The estimation of recovery in concentrate was constrained by the all-encompassing UCZ Halo East and West inclusive of UCZ Vein East and West. The estimation is run in two search volumes, increasing by two times the search distances for the second pass. Dynamic anisotropy is utilized to control the search and optimize the local estimate. Neighborhood search parameters are presented in Table 14.26.

For the LCZ, recovery is assigned a mean value of 94%. It was determined that estimation of block recovery was not required due to the low variability of LCZ recovery testing and consistency in this variable across the LCZ.

14.7.2 Lowry deposit

Resource estimation was performed for the Lowry deposit by SRK using Maptek™ Vulcan™ software (version 11.0.4). The focus of estimation was on Cu as the key economic variable of interest. OK was selected as the preferred method for block estimation in domains where acceptable variography was calculated. A multi-pass method was used such that the initial pass of well-informed blocks was estimated using OK to a search ellipsoid size informed by the variogram range for Cu within the domain. In order to estimate remaining blocks, IDW2 was selected as the preferred method for moderately to poorly informed blocks. IDW2 provides a reasonable representation of quality variables while not being overly smoothed or locally biased. The search pass number is stored in the est_run variable to aid in assessing block confidence and classification.

In areas of limited data that did not meet the minimum criteria for estimation in the final pass, a scripted value was assigned to the block variable by domain. The scripted value assigned is the variable mean from capped composites by domain. A limited number of blocks in the Lowry block model met this criterion and were located primarily in the UCZ domains.

SG was estimated in the block model using a two-pass method of IDW2 with varying search neighborhoods by domain. As with quality variables, blocks not estimated in the last pass were scripted a mean value based on composite data. As there is less SG data compared to quality analytical variables, a greater number of blocks were scripted with the domain mean.

Each mineralized domain has a unique search neighborhood based on the Cu variogram, mineralization thickness, and data spacing within the domain. For most domains, the directionality of the search ellipsoid was varied by block based on the average orientation of the domain's modelled wireframe. This was performed using Vulcan's Locally Varying Anisotropy (LVA) feature based upon a constructed top and bottom wireframe surface representing the domain. A summary table of each domain's search neighborhood is provided in Table 14.26.

A KNA was not performed due to the relatively poor quality of variography, thin nature of domains, and limited data per many domains. Typically, KNA provides indications for optimized search neighborhood based on the variography. In the case of the Lowry deposit, the thin horizons of mineralization constrain the estimate along with data population per domain being limited. It is the opinion of the QP that KNA should be reviewed if additional data is acquired and a more robust variogram is calculated in future studies of the Lowry deposit.

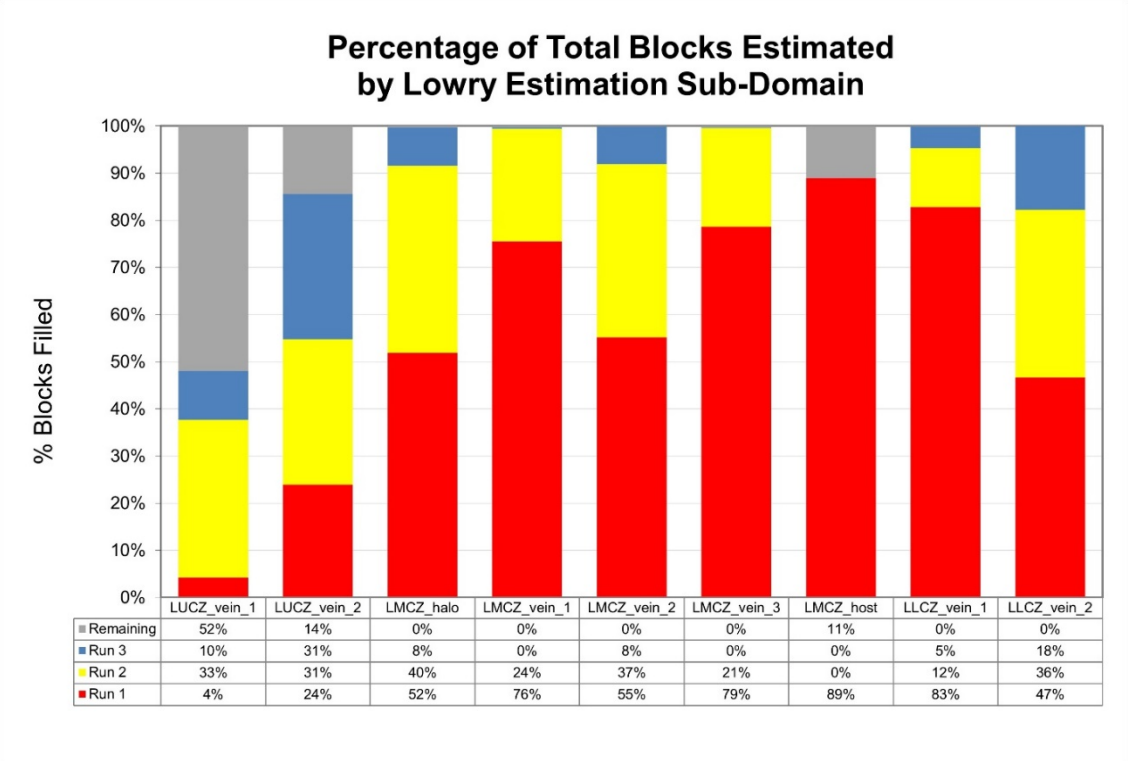
Domain	Est. Sub-Domain	Variable	Pass	Estimation method	Major Axis (m)	Semi-Major Axis (m)	Minor Axis (m)	Discretization Steps	Min Samples	Max Samples	Max Samplers per DH	Comp. Length weighted	Ellipsoid Orientation[1]	Unfolding Method[2]	Octant Restrictions
LLUZ	LUCZ_Vein_1	Cu	1	OK	100	100	10	5,5,2	3	5	2	Yes	15,30,0	n/a	No
LLUZ	LUCZ_Vein_1	Cu	2	IDW2	200	200	20	5,5,2	3	5	2	Yes	15,30,0	n/a	No
LLUZ	LUCZ_Vein_1	Cu	3	IDW2	400	400	20	5,5,2	2	5	n/a	Yes	15,30,0	n/a	No
LLUZ	LUCZ_Vein_2	Cu	1	IDW2	100	100	10	5,5,2	3	5	2	Yes	15,30,0	LVA	No
LLUZ	LUCZ_Vein_2	Cu	2	IDW2	200	200	20	5,5,2	3	5	2	Yes	15,30,0	LVA	No
LLUZ	LUCZ_Vein_2	Cu	3	IDW2	400	400	20	5,5,2	2	5	n/a	Yes	n/a	LVA	No
LMCZ	LMCZ_Vein_1	Cu	1	OK	75	75	10	5,5,2	3	6	2	Yes	n/a	LVA	No
LMCZ	LMCZ_Vein_1	Cu	2	OK	150	150	20	5,5,2	3	6	2	Yes	n/a	LVA	No
LMCZ	LMCZ_Vein_1	Cu	3	IDW2	300	300	20	5,5,2	3	8	n/a	Yes	n/a	LVA	No
LMCZ	LMCZ_Vein_2	Cu	1	OK	75	75	10	5,5,2	3	6	2	Yes	n/a	LVA	No
LMCZ	LMCZ_Vein_2	Cu	2	OK	150	150	20	5,5,2	3	6	2	Yes	n/a	LVA	No
LMCZ	LMCZ_Vein_2	Cu	3	IDW2	300	300	20	5,5,2	2	8	n/a	Yes	n/a	LVA	No
LMCZ	LMCZ_Vein_3	Cu	1	IDW2	100	100	10	5,5,2	3	6	2	Yes	n/a	LVA	No
LMCZ	LMCZ_Vein_3	Cu	2	IDW2	200	200	20	5,5,2	3	6	2	Yes	n/a	LVA	No
LMCZ	LMCZ_Vein_3	Cu	3	IDW2	300	300	20	5,5,2	2	6	n/a	Yes	n/a	LVA	No
LMCZ	LMCZ_halo	Cu	1	IDW2	75	75	10	5,5,2	3	6	2	Yes	n/a	LVA	No
LMCZ	LMCZ_halo	Cu	2	IDW2	150	150	20	5,5,2	3	6	2	Yes	n/a	LVA	No
LMCZ	LMCZ_halo	Cu	3	IDW2	300	300	20	5,5,2	2	5	n/a	Yes	n/a	LVA	No
LMCZ	LMCZ_host	Cu	1	IDW2	400	400	20	5,5,2	3	5	2	Yes	350,30,0	n/a	No
LLCZ	LLCZ_Vein_1	Cu	1	OK	175	175	10	5,5,2	3	6	2	Yes	n/a	LVA	No
LLCZ	LLCZ_Vein_1	Cu	2	OK	250	250	20	5,5,2	3	6	2	Yes	n/a	LVA	No
LLCZ	LLCZ_Vein_1	Cu	3	IDW2	350	350	20	5,5,2	2	5	n/a	Yes	n/a	LVA	No
LLCZ	LLCZ_Vein_2	Cu	1	OK	100	100	10	5,5,2	3	6	2	Yes	n/a	LVA	No
LLCZ	LLCZ_Vein_2	Cu	2	OK	200	200	20	5,5,2	3	6	2	Yes	n/a	LVA	No
LLCZ	LLCZ_Vein_2	Cu	3	IDW2	300	300	20	5,5,2	2	6	n/a	Yes	n/a	LVA	No
LLCZ	LLCZ_host	Cu	1	IDW2	400	400	20	5,5,2	3	5	2	Yes	15, 30, 0	n/a	No

1 – Ellipsoid orientation is described as bearing (z), plunge (y), and dip (x) as per Vulcan software standard rotation scheme.

2 – LVA refers to Vulcan software locally varying anistropy created from top and bottom wireframesurfaces of the domain

Table 14.26 Search Neighborhood Parameters for Lowry deposit

The results of the Lowry deposit block model estimation are provided in Figure 14.53 showing the percentage of blocks estimated in the various passes for Cu by domain. The primary mineralized domains of LMCZ and LLCZ show that the majority of blocks were populated in the first pass and no blocks were scripted with mean composite values. In the LUCZ, due to limited data, the percentage of blocks estimated in the first few passes show that portions of the domain exhibit limited confidence in estimated quality. In addition, large portions of the LUCZ blocks are low confidence and were populated in either a large search pass or with scripted mean values. Cu grade distribution by estimation domain is provided in Figure 14.54 through Figure 14.59.



	LUCZ_vein_1	LUCZ_vein_2	LMCZ_halo	LMCZ_vein_1	LMCZ_vein_2	LMCZ_vein_3	LMCZ_host	LLCZ_vein_1	LLCZ_vein_2
Remaining	52%	14%	0%	0%	0%	0%	11%	0%	0%
Run 3	10%	31%	8%	0%	8%	0%	0%	5%	18%
Run 2	33%	31%	40%	24%	37%	21%	0%	12%	36%
Run 1	4%	24%	52%	76%	55%	79%	89%	83%	47%

Note: *See Table 14.26 for complete list of estimation sub-domains for each of the three primary mineralized domains*

Figure 14.53 Summary Lowry Block Estimation by Estimation Pass per Estimation Sub-Domain

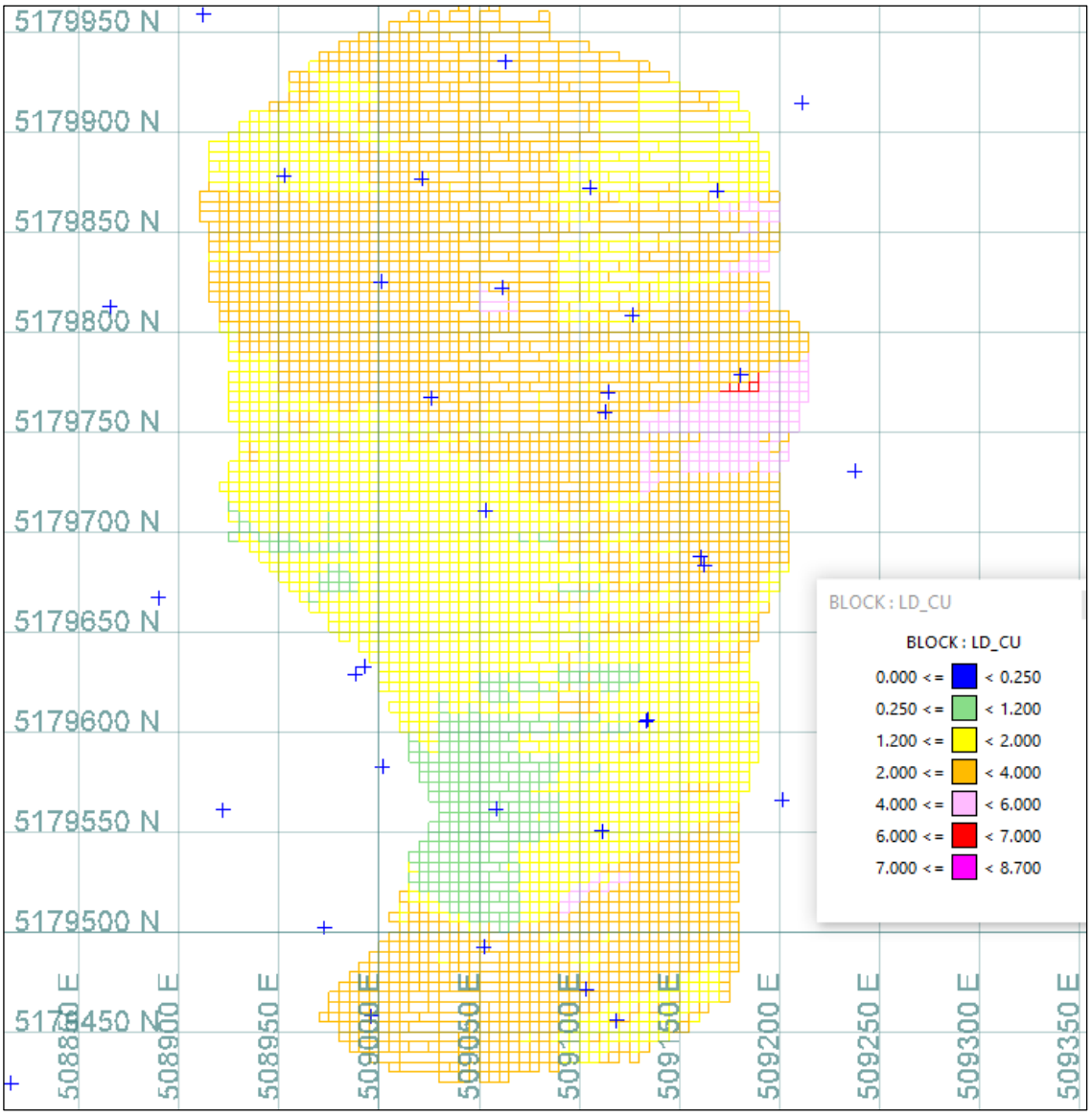


Figure 14.54 Cu Grade Distribution in LMCZ_Vein_1 Estimation Sub-Domain

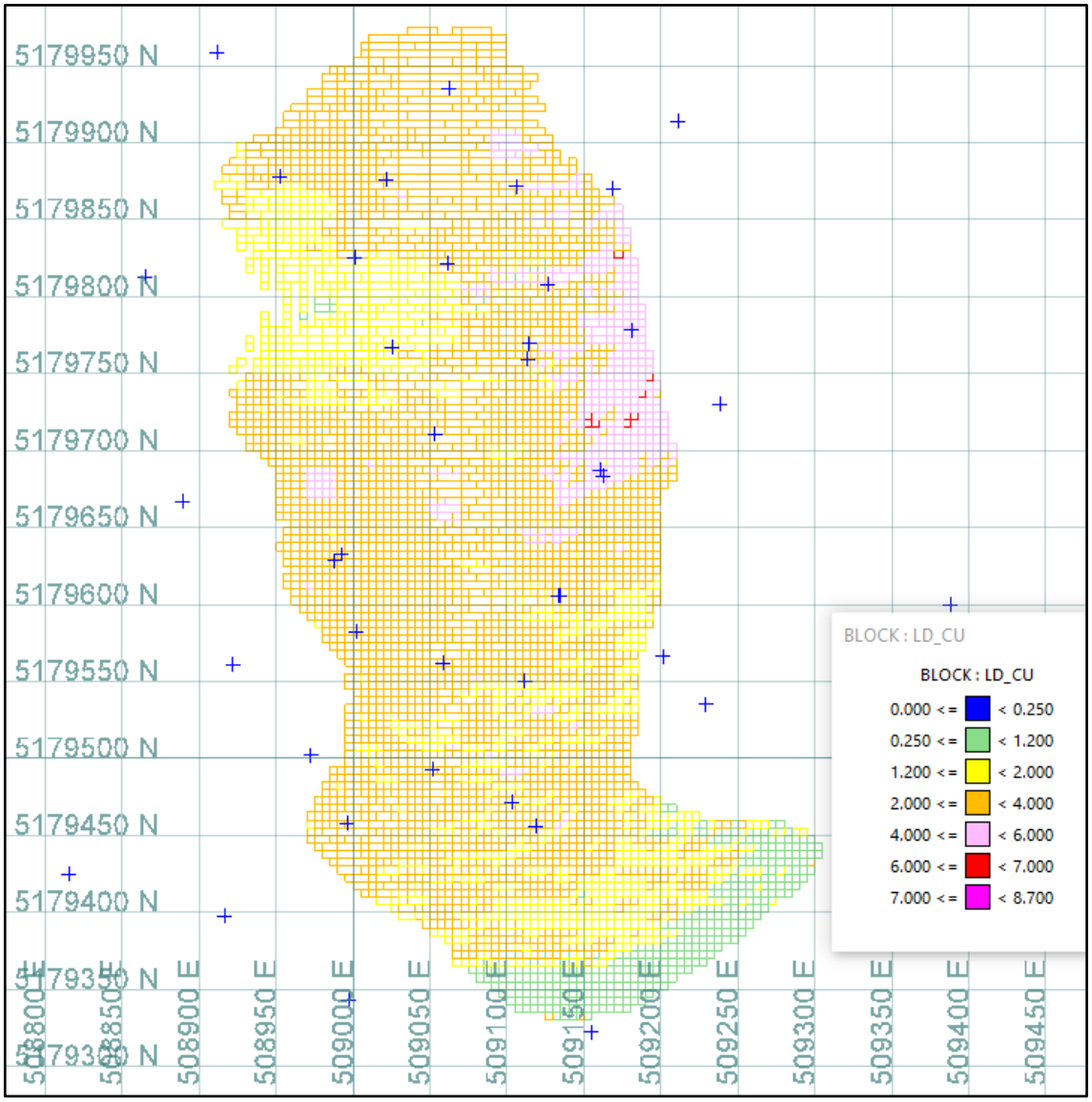


Figure 14.55 Cu Grade Distribution in the LMCZ_Vein_2 Estimation Sub-Domain

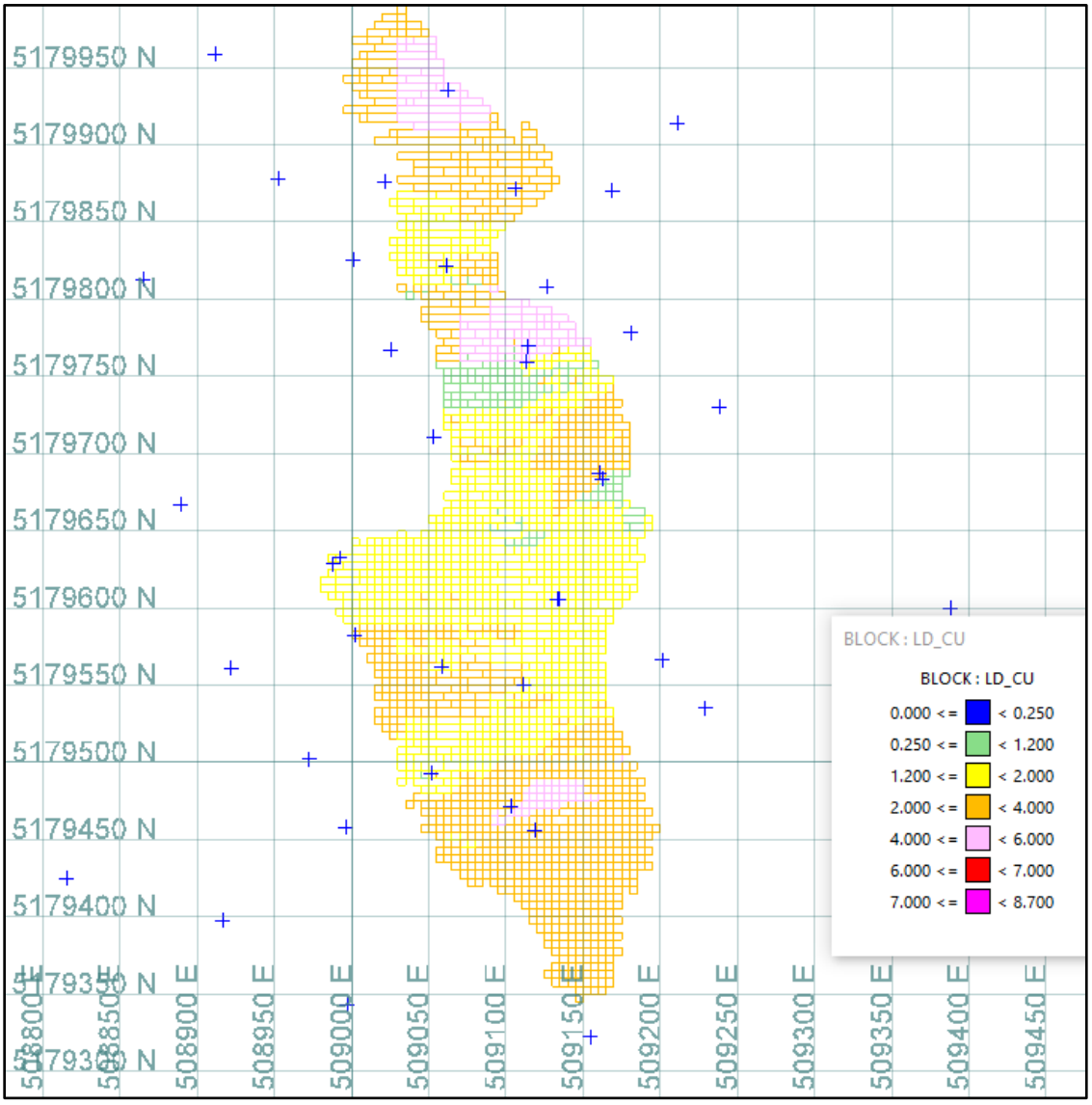


Figure 14.56 Cu Grade Distribution in the LMCZ_Vein_3 Estimation Sub-Domain

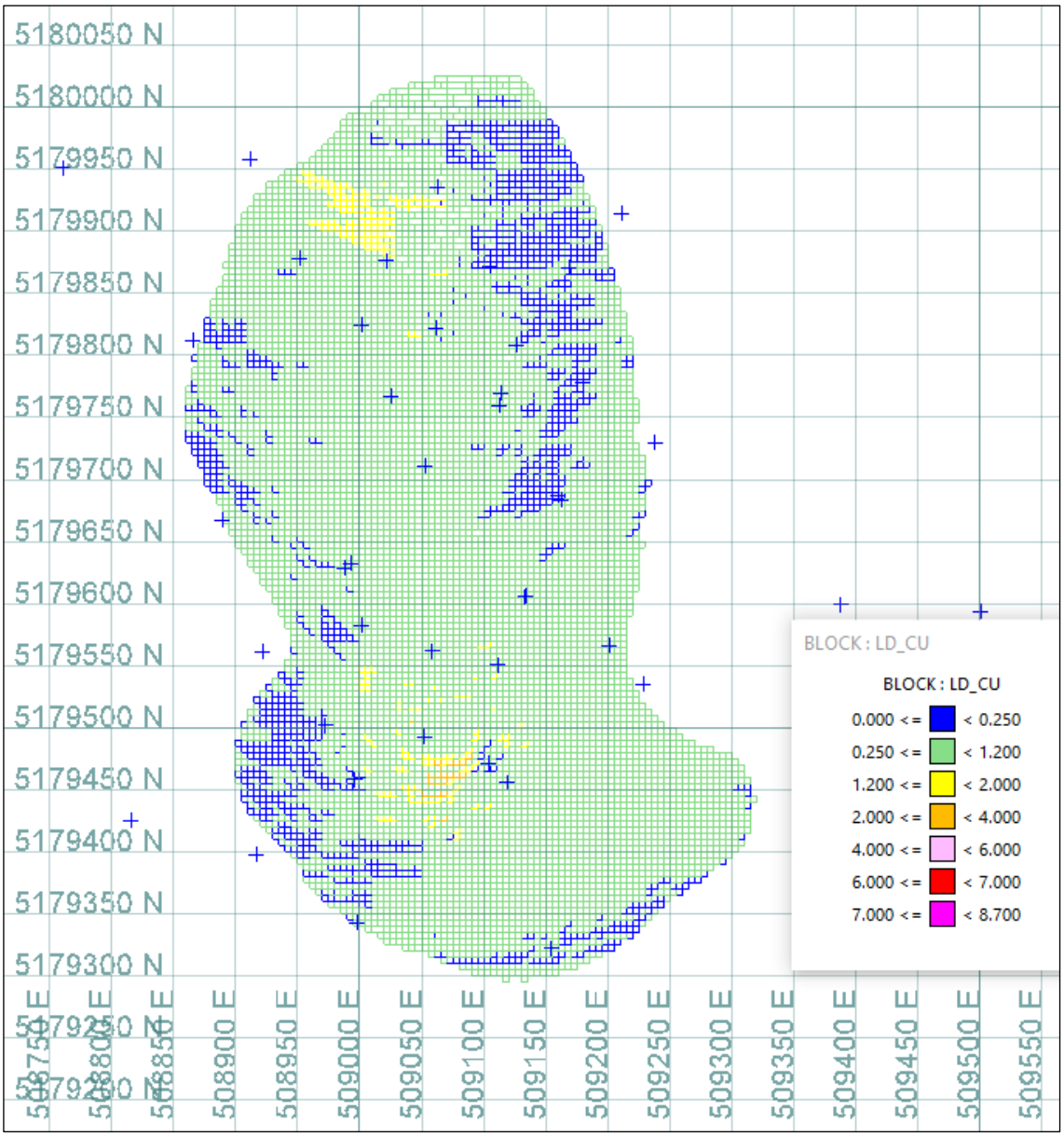


Figure 14.57 Cu Grade Distribution in the LMCZ_Halo (Low-Grade) Estimation Sub-Domain

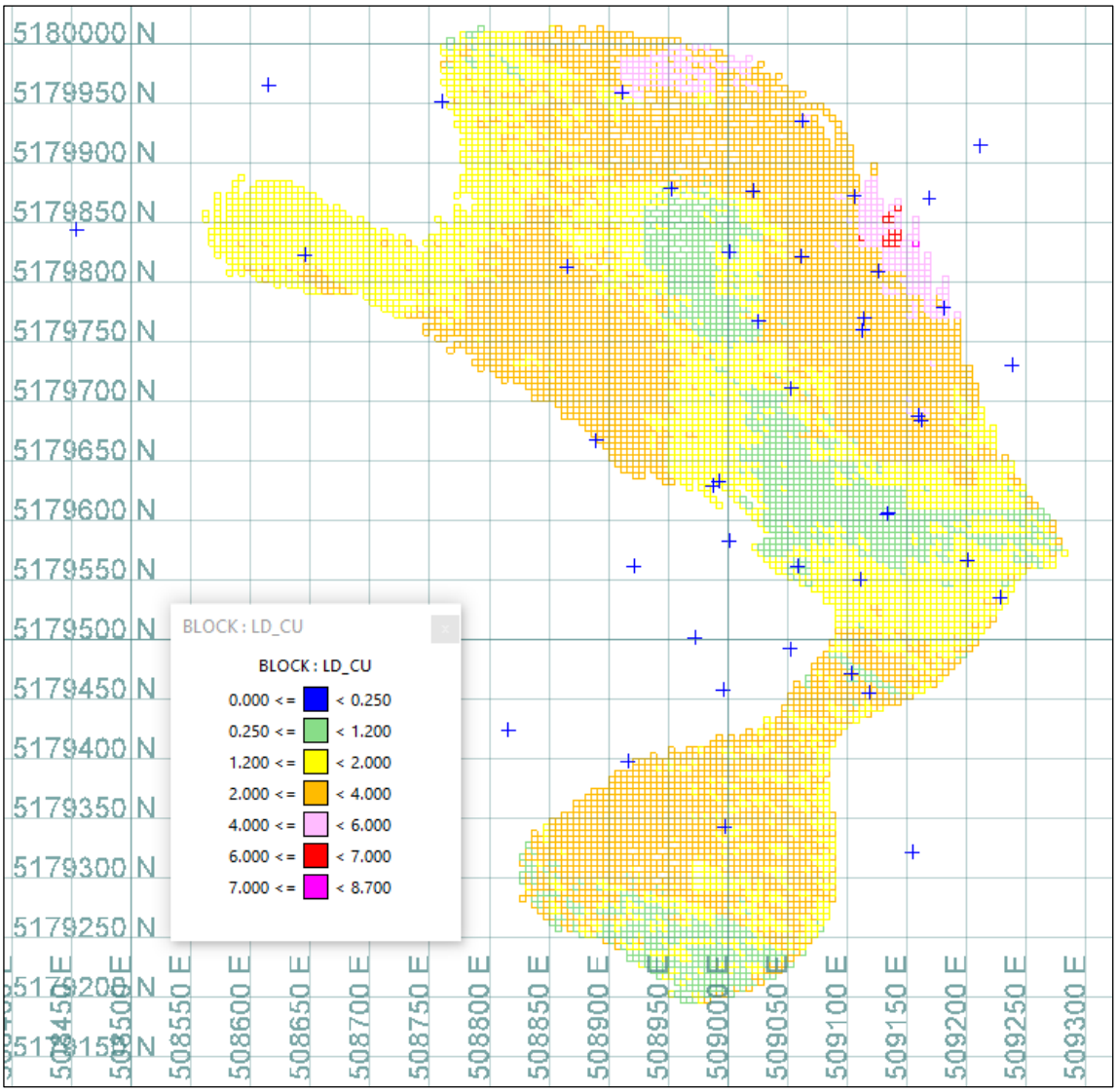


Figure 14.58 Cu Grade Distribution in the LLCZ_Vein_1 Estimation Sub-Domain

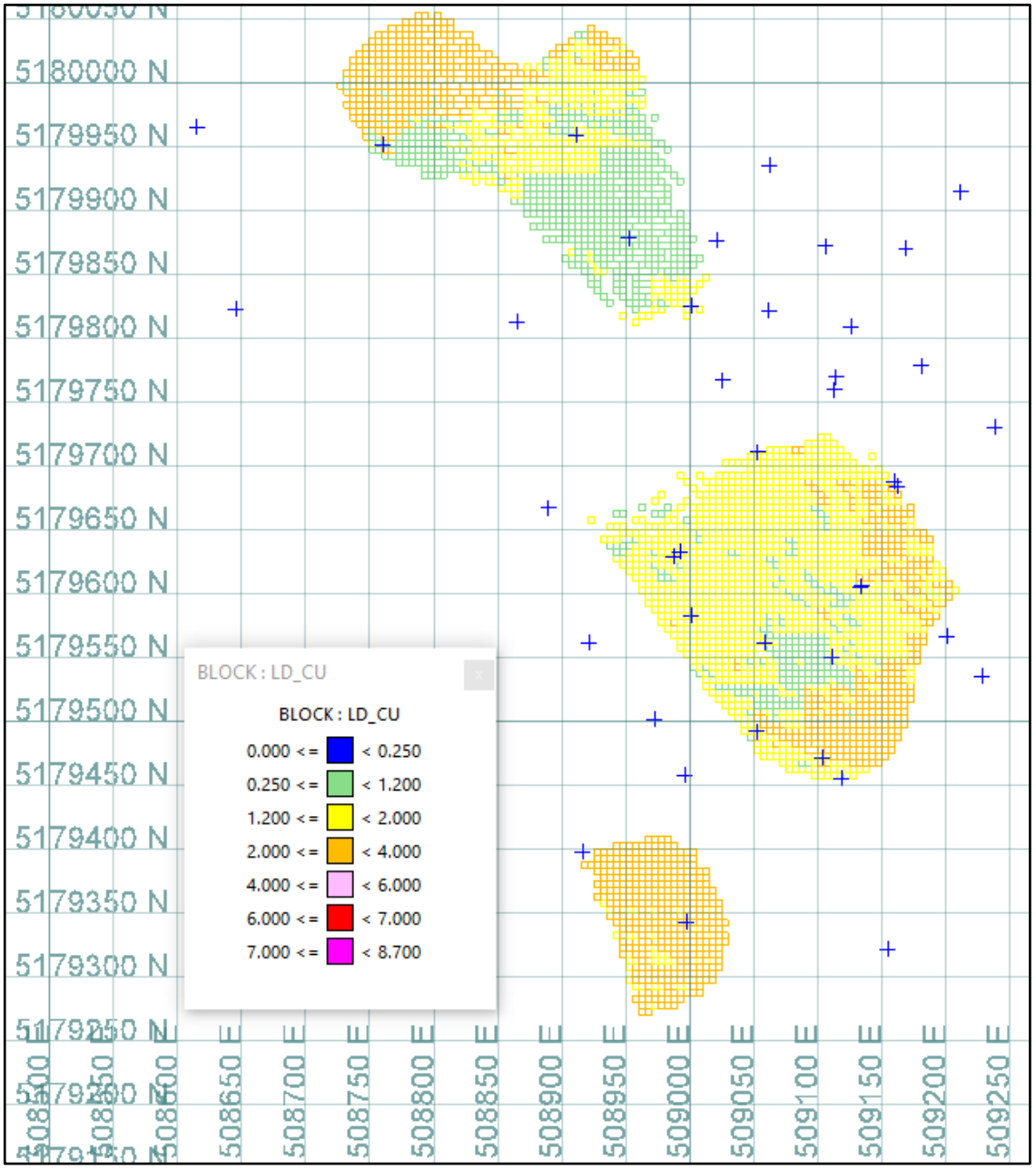


Figure 14.59 Cu Grade Distribution in the LLCZ_Vein_2 Estimation Sub-Domain

It is the opinion of the QP that the estimation for Cu, Ag, Co, As, and SG in the Lowry block model is appropriate given the data spacing, geological model, and data variability per domain. Some domains contain limited data, therefore were estimated using a simplified neighborhood and estimation method such as IDW2. In domains that are better informed with drilling data, OK was used when an acceptable variogram was calculated. It is recommended that the estimation methodology and search neighborhood be revisited upon the acquisition of additional data that may provide an improved estimate in the future.

14.8 Model Validation

14.8.1 Johnny Lee deposit

The 2019 Johnny Lee UCZ and LCZ block models were validated using multiple techniques including: visual validation of block grades with raw drilling data and a statistical comparison of composites and block grades through swath plots and mean comparison by domain. In summary, it is the opinion of the QP that both models validate well against raw and composited data and are both suitable for use in the determination of Mineral Resources as per CIM guidelines and standards.

Visual Comparison

Representative cross-sections through the UCZ and LCZ were produced for visual validation of block model Cu grade against composite Cu grade. Examples of these cross-sections are presented below in Figure 14.60 to Figure 14.63 that illustrate the block model grades provide an acceptable representation of composite data.

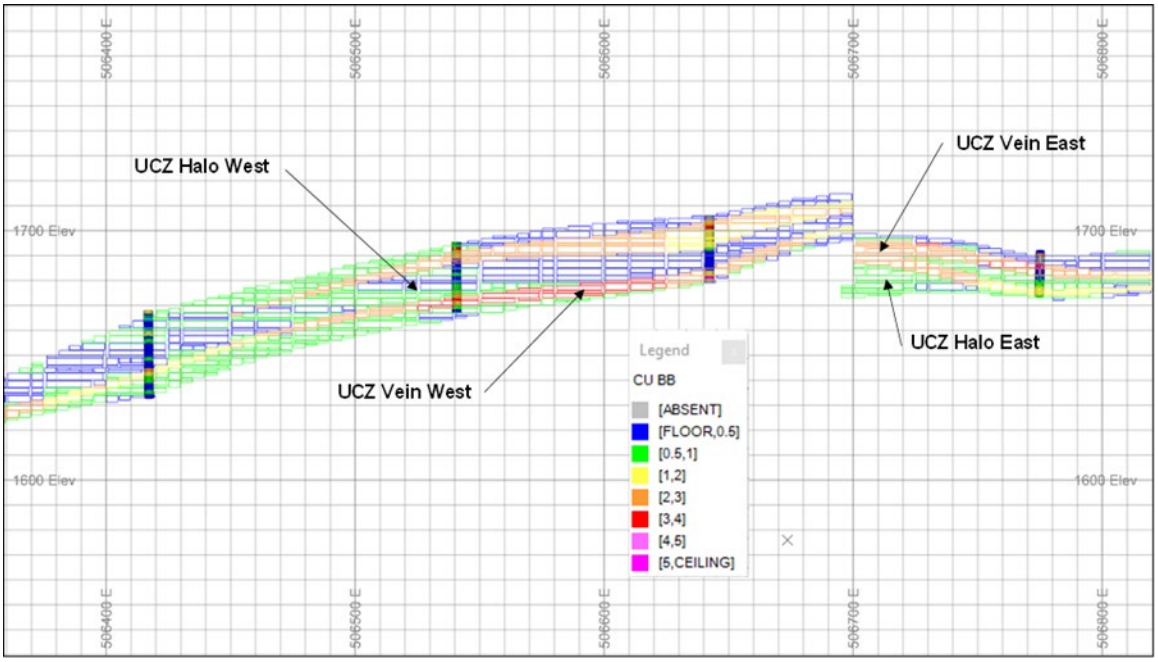


Figure 14.60 Section N5180805 Looking North – Cross-section through UCZ showing UCZ East and West Cu Block Model versus Cu Composite Data

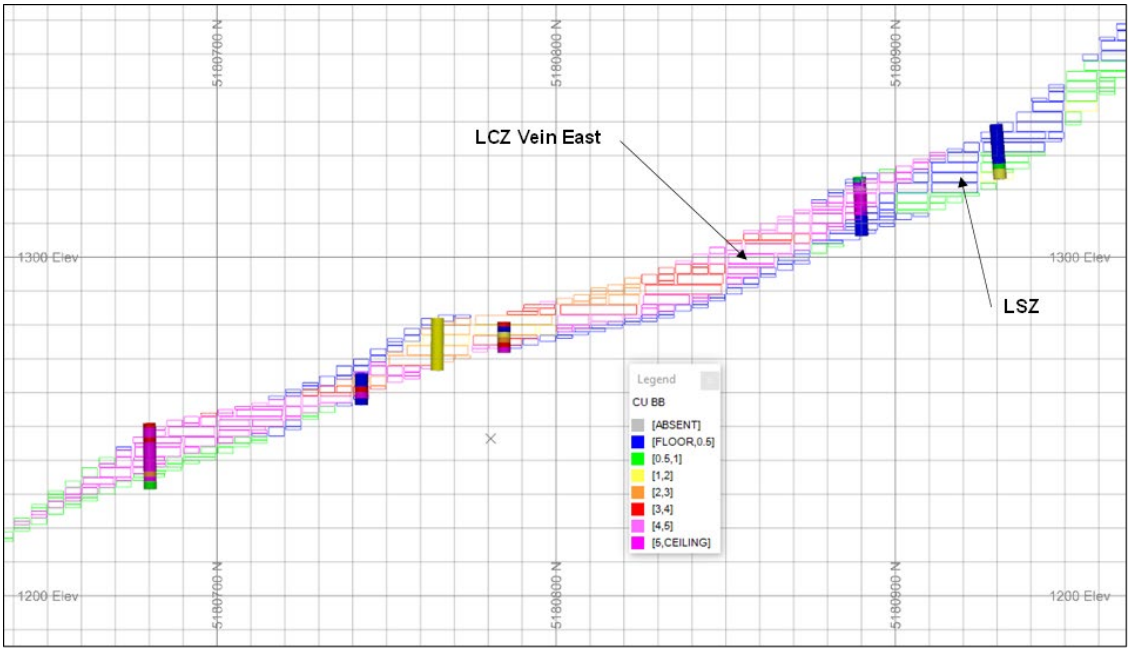


Figure 14.61 Section E507210 Looking West - Cross-section through LCZ Vein East/LSZ showing Cu Block Model versus Cu Data

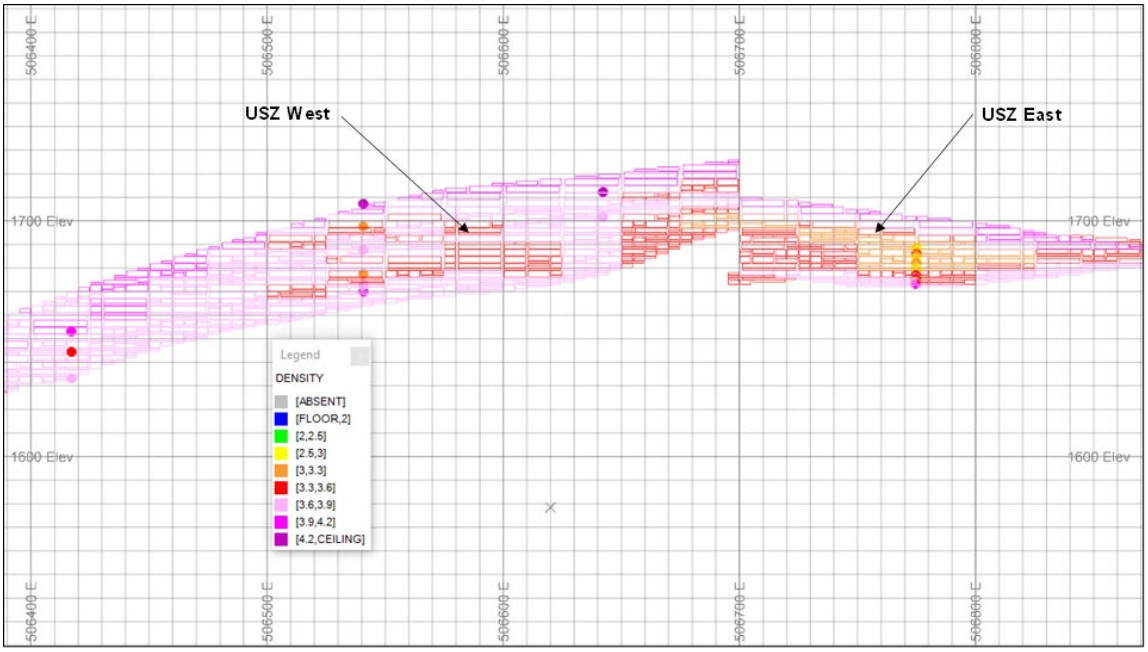


Figure 14.62 Section N5180805 Looking North – Cross-section through USZ showing USZ East and West SG Block Model versus SG Data

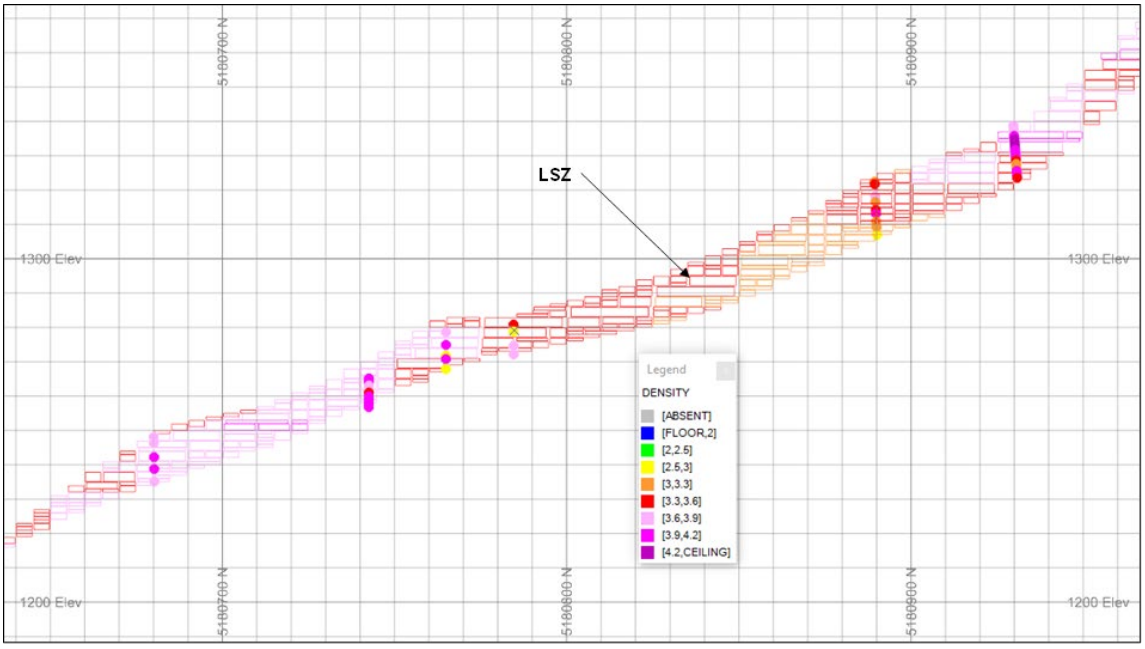


Figure 14.63 Section E507210 Looking West - Cross-section through LSZ Showing SG Block Model versus SG Data

Comparative Statistics

Frequency log histograms of capped composite and block grade distributions were produced to compare and validate the estimation of each domain (Figure 14.64 to Figure 14.72), and are summarized in Table 14.27 and Table 14.28 for Cu and SG respectively. The block grade histograms are a reasonable reflection of the composite grade histogram after smoothing of low and high-grade tails.

Data Type	Metric	UCZ Vein East	UCZ Vein West	UCZ Halo East	UCZ Halo West	LCZ Vein	LSZ
Composite	Count	516	431	778	1008	153	188
Block	Count	47623	85547	127022	282903	12810	89051
Composite	Mean	2.44	2.33	0.55	0.45	6.55	0.19
Block	Mean	2.31	2.27	0.51	0.46	6.73	0.20
Composite	Minimum	0.296	0.222	0.005	0.004	0.003	0.001
Block	Minimum	0.97	0.72	0.03	0.03	0.98	0.01
Composite	Maximum	9.11	8.31	2.16	3.59	17.68	1.58
Block	Maximum	6.78	4.93	1.27	1.63	14.90	1.15
Composite	Variance	2.17	2.27	0.09	0.11	23.28	0.12
Block	Variance	0.47	0.42	0.03	0.02	7.24	0.03
Composite	Std Deviation	1.47	0.30	0.30	0.33	4.83	0.34
Block	Std Deviation	0.68	0.19	0.19	0.16	2.69	0.18

Data Type	Metric	UCZ Vein East	UCZ Vein West	UCZ Halo East	UCZ Halo West	LCZ Vein	LSZ
Composite	CV	0.60	0.13	0.55	0.73	0.74	1.83
Block	CV	0.30	0.08	0.36	0.34	0.40	0.89

Table 14.27 Summary Statistics for Johnny Lee Cu Composite Grade (Capped) and Block Grade by Domain

Data Type	Metric	USZ East	USZ West	LSZ
Composite	Count	702	644	213
Block	Count	330672	566854	108884
Composite	Mean	3.96	3.60	3.62
Block	Mean	4.04	3.77	3.65
Composite	Minimum	2.68	2.27	2.64
Block	Minimum	3.05	2.59	2.94
Composite	Maximum	4.71	4.55	4.43
Block	Maximum	4.39	4.36	4.10
Composite	Variance	0.15	0.20	0.17
Block	Variance	0.07	0.08	0.05
Composite	Std Deviation	0.39	0.45	0.41
Block	Std Deviation	0.27	0.28	0.22
Composite	CV	0.10	0.12	0.11
Block	CV	0.07	0.07	0.06

Table 14.28 Summary Statistics for Johnny Lee Cu Composite Grade (Capped) and Block Grade by Domain

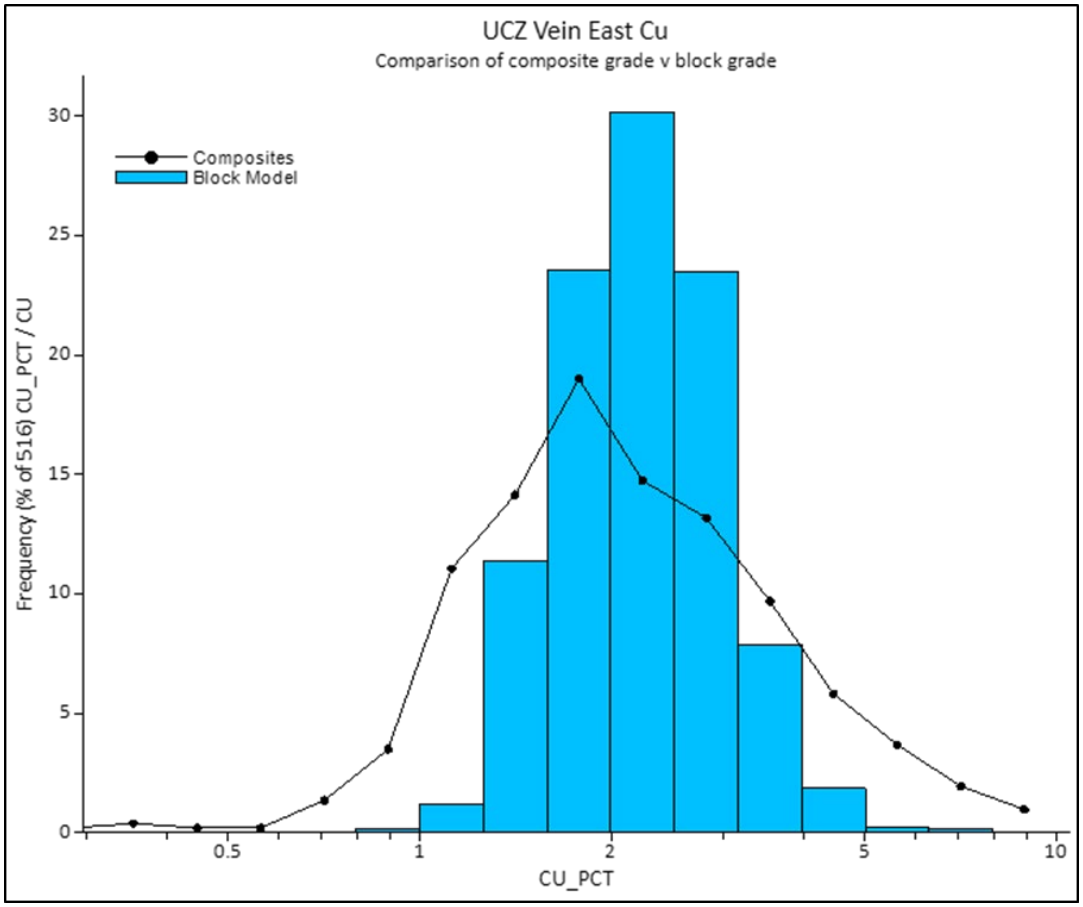


Figure 14.64 Comparative Log Histogram of Cu Composite and Block Grade Distributions for the UCZ Vein East

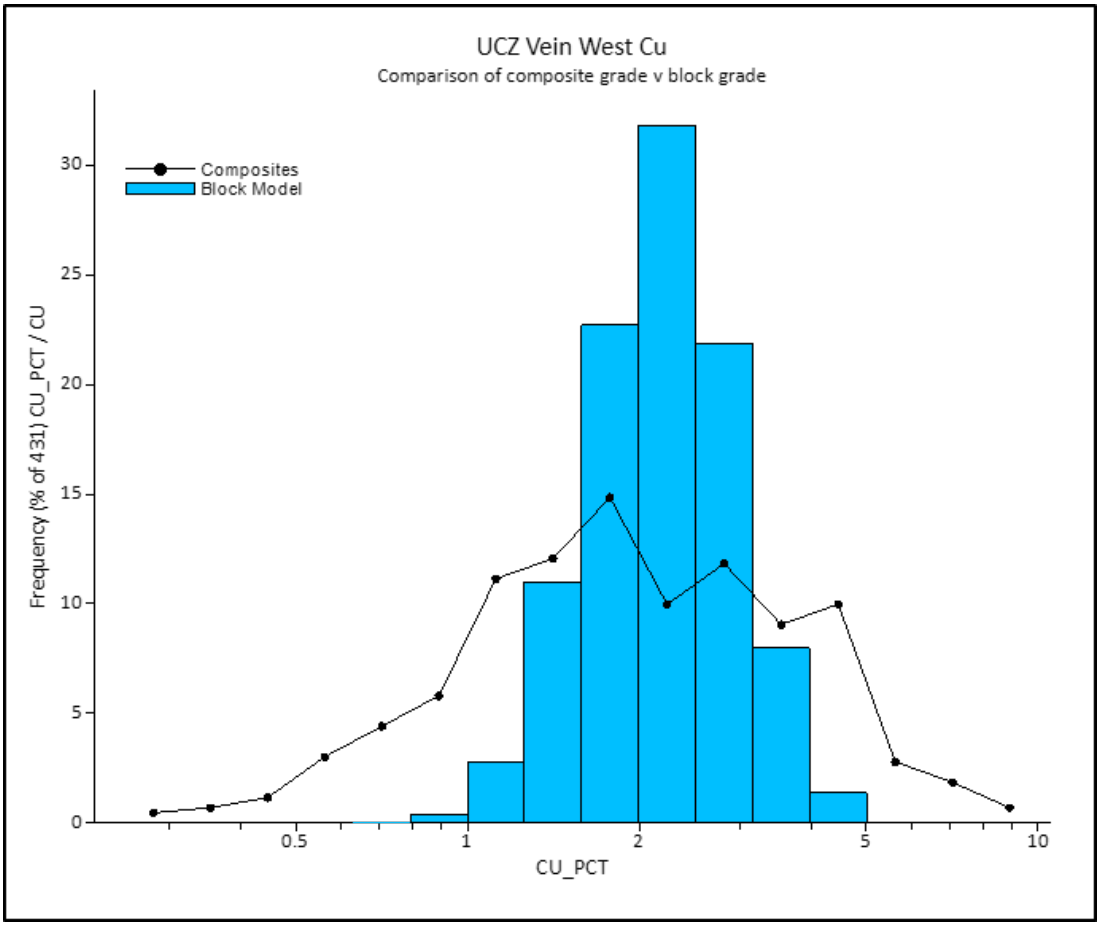


Figure 14.65 Comparative Log Histogram of Cu Composite and Block Grade Distributions for the UCZ Vein West

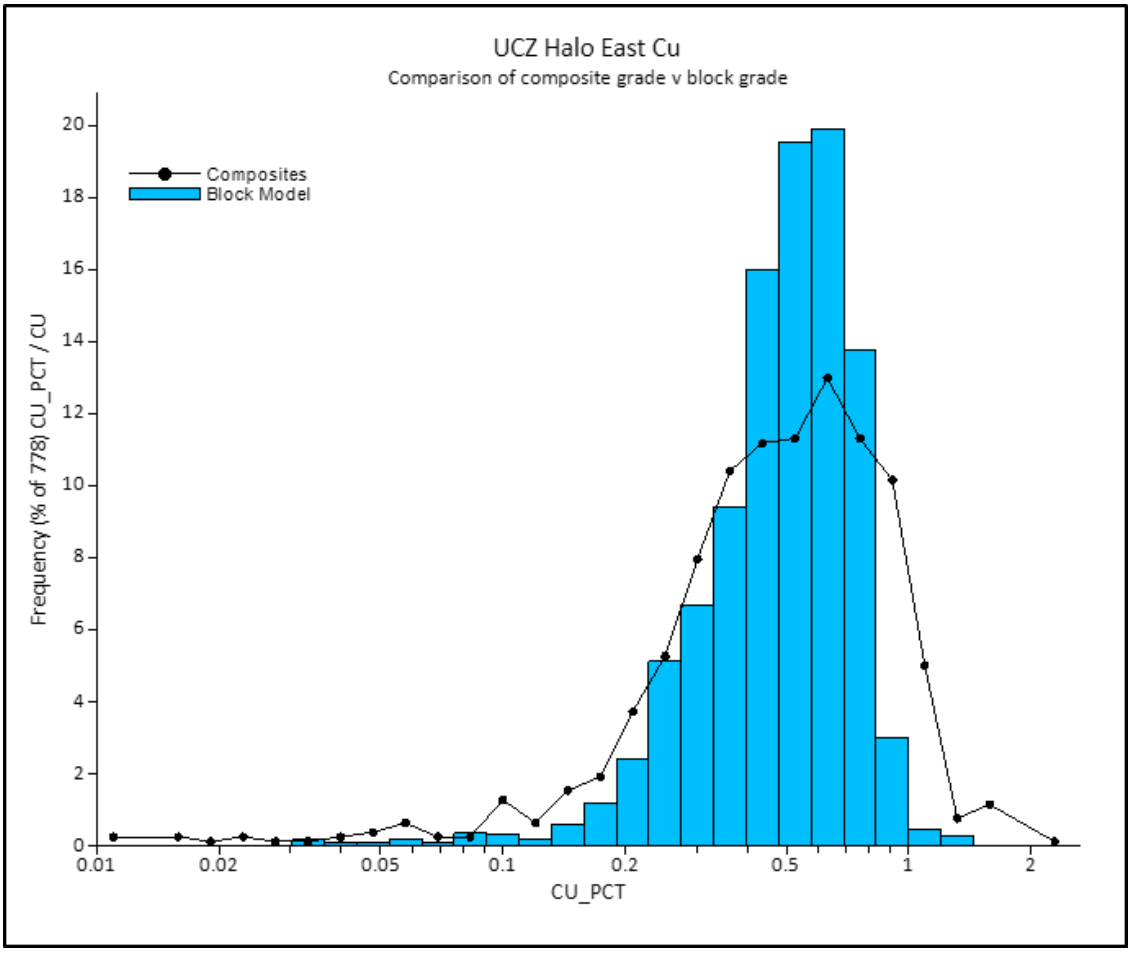


Figure 14.66 Comparative Log Histogram of Cu Composite and Block Grade Distributions for the UCZ Halo East

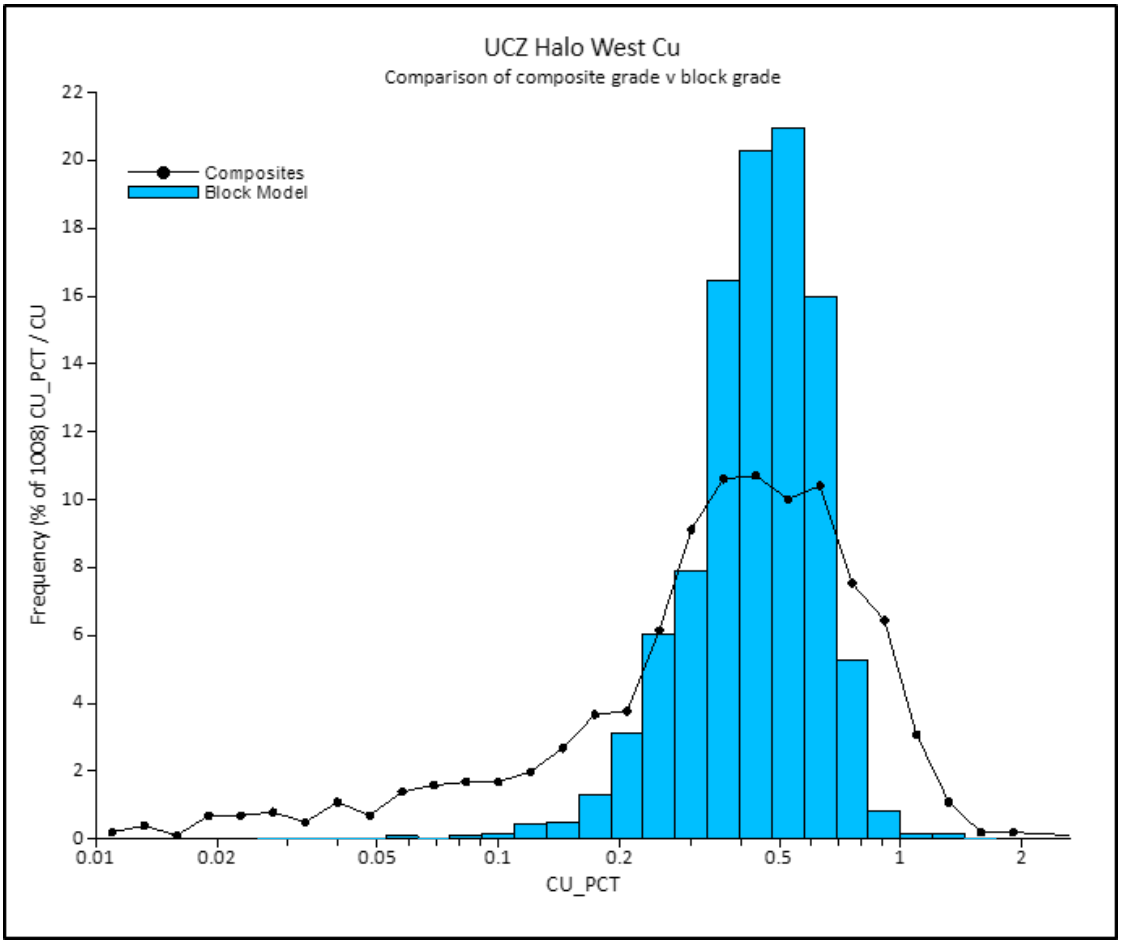


Figure 14.67 Comparative Log Histogram of Cu Composite and Block Grade Distributions for the UCZ Vein West

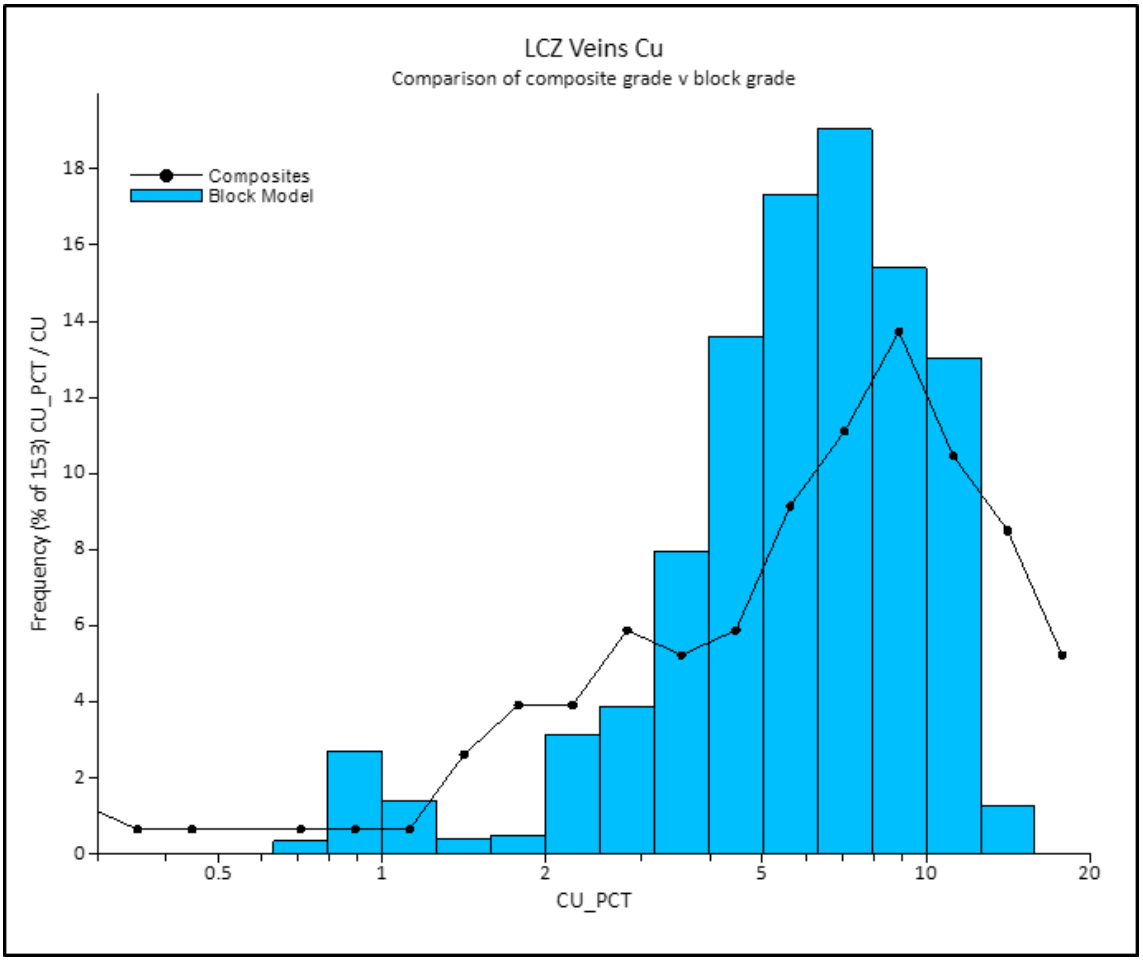


Figure 14.68 Comparative Log Histogram of Cu Composite and Block Grade Distributions for the LCZ Veins (West, Central, and East)

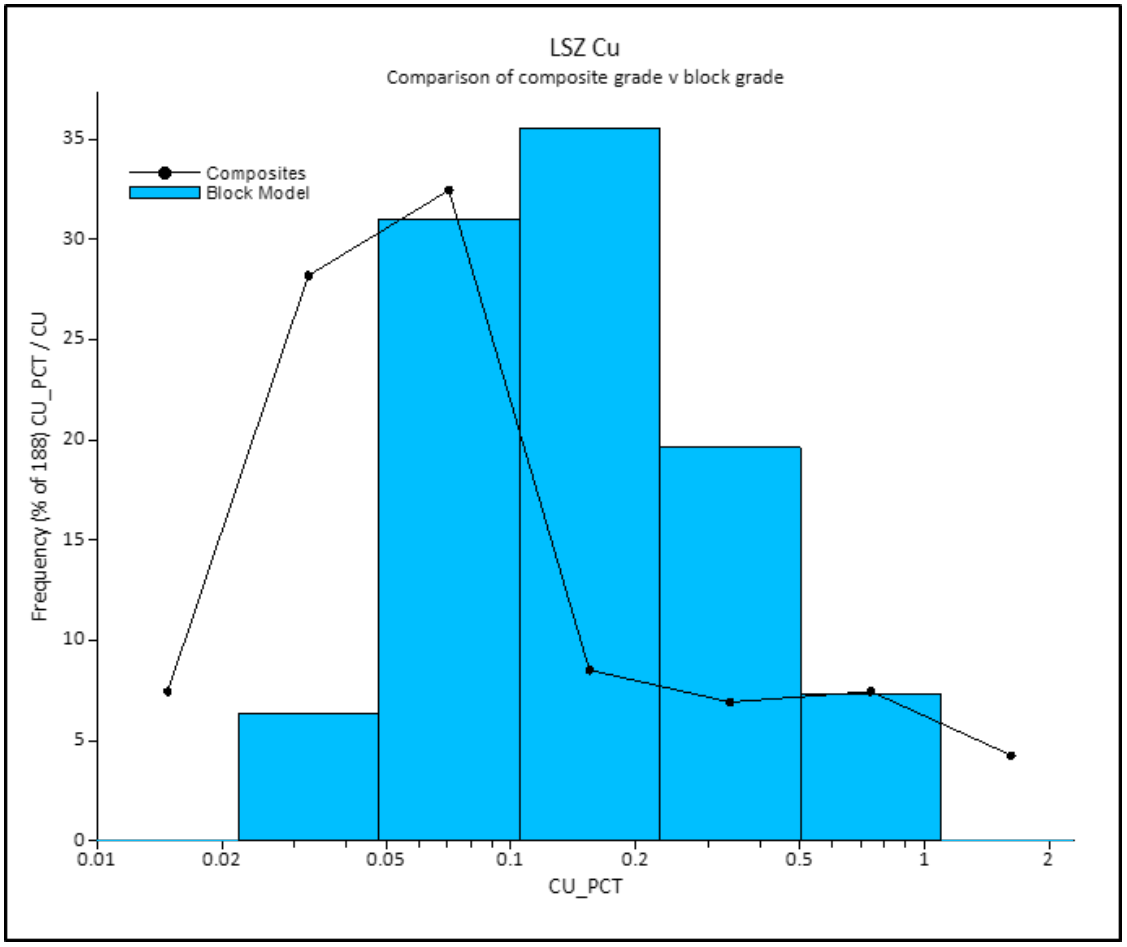


Figure 14.69 Comparative Log Histogram of Cu Composite and Block Grade Distributions for the LSZ

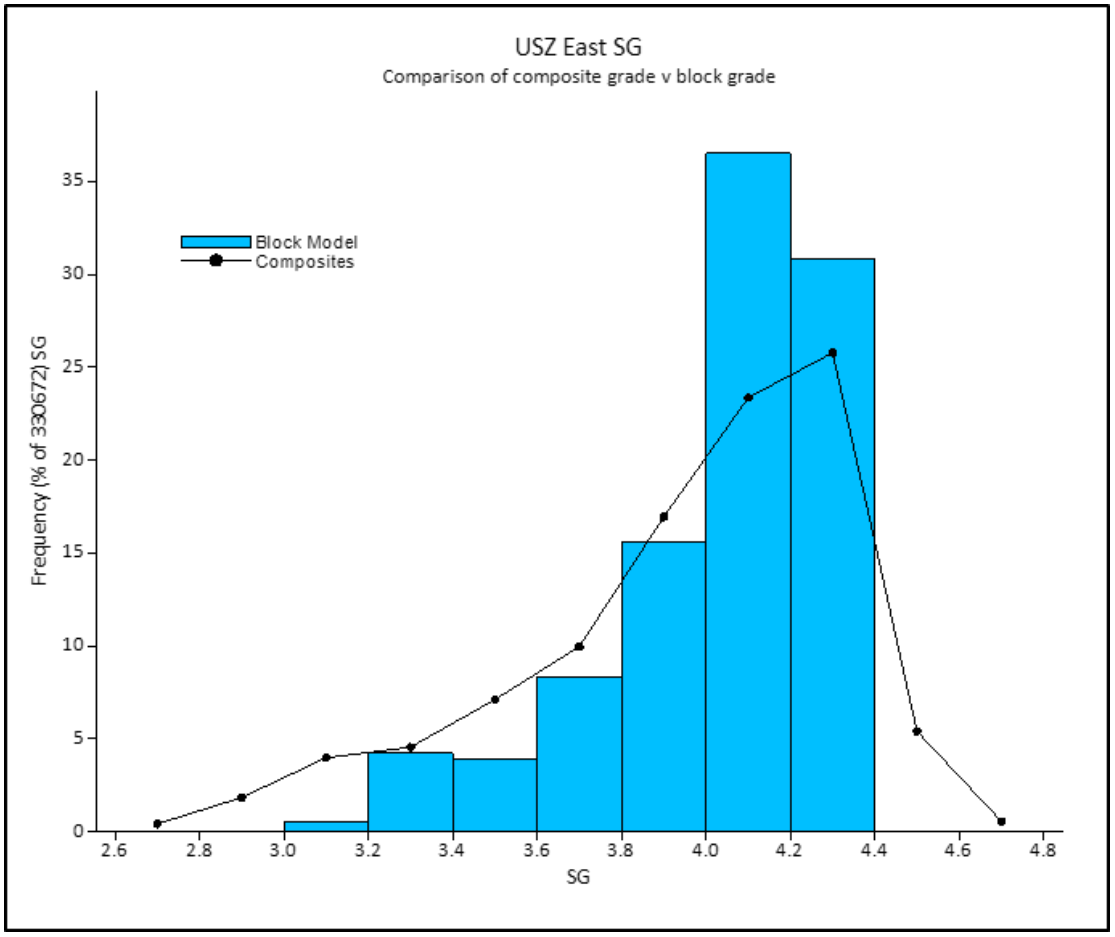


Figure 14.70 Comparative Log Histogram of SG Composite and Block SG Distributions for the USZ East

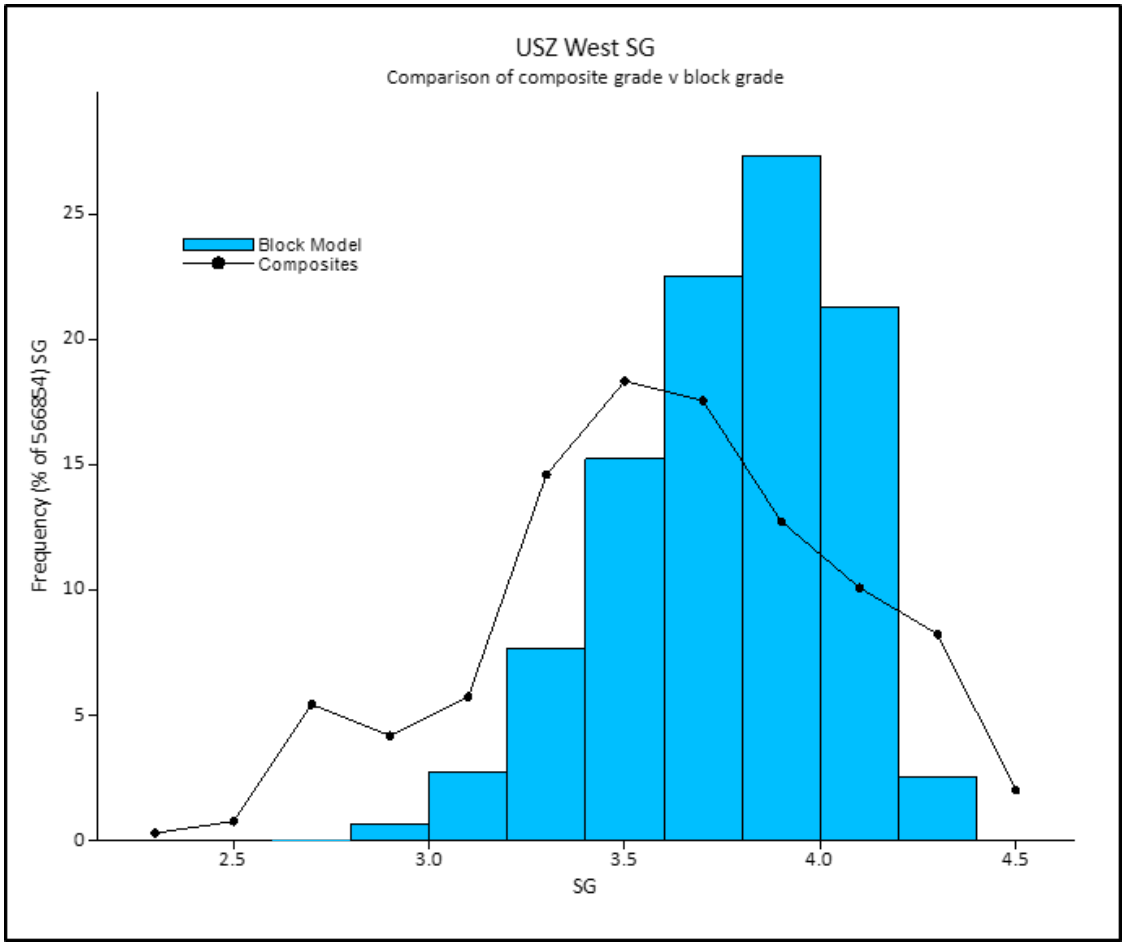


Figure 14.71 Comparative Log Histogram of SG Composite and Block SG Distributions for the USZ West

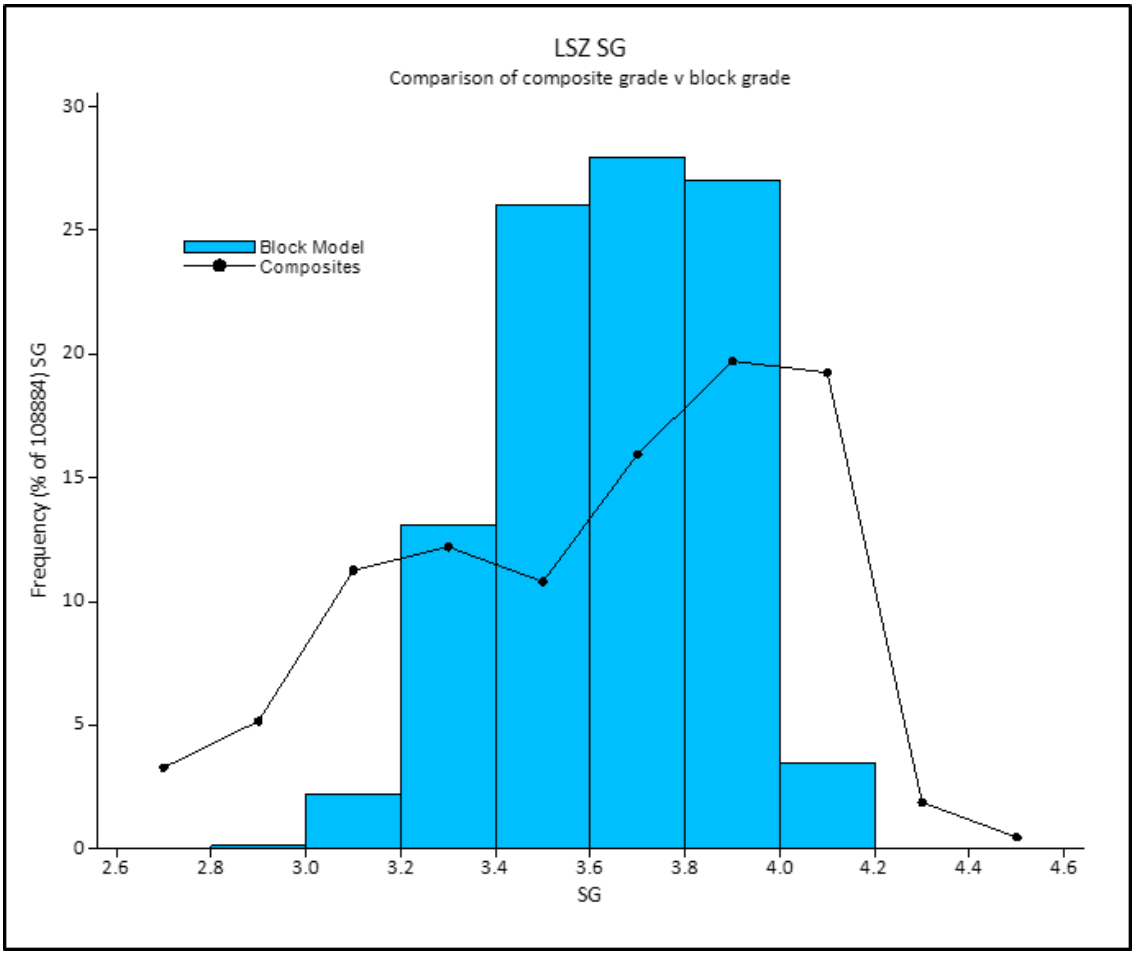


Figure 14.72 Comparative Log Histogram of SG Composite and Block SG Distributions for the LSZ

Swath Plots

Swath plots for Cu and SG were generated to validate each mineralized domain. The profiles compare mean block grade and mean composite values by northing for the UCZ and easting for the LCZ (Figure 14.73 to Figure 14.81). The swath plots illustrate an acceptable correlation between composite grades and block model grades. There are no material biases between composited and block grades for Cu and SG.

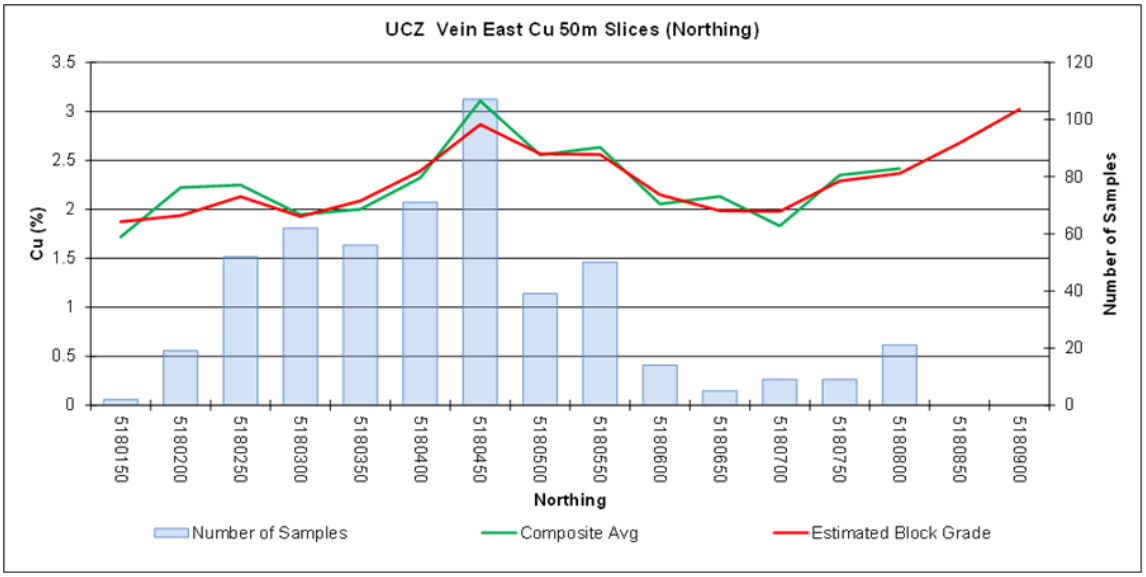


Figure 14.73 UCZ Vein East Cu Sectional Profile (Northing) – Composite Grade versus Block Model Grade

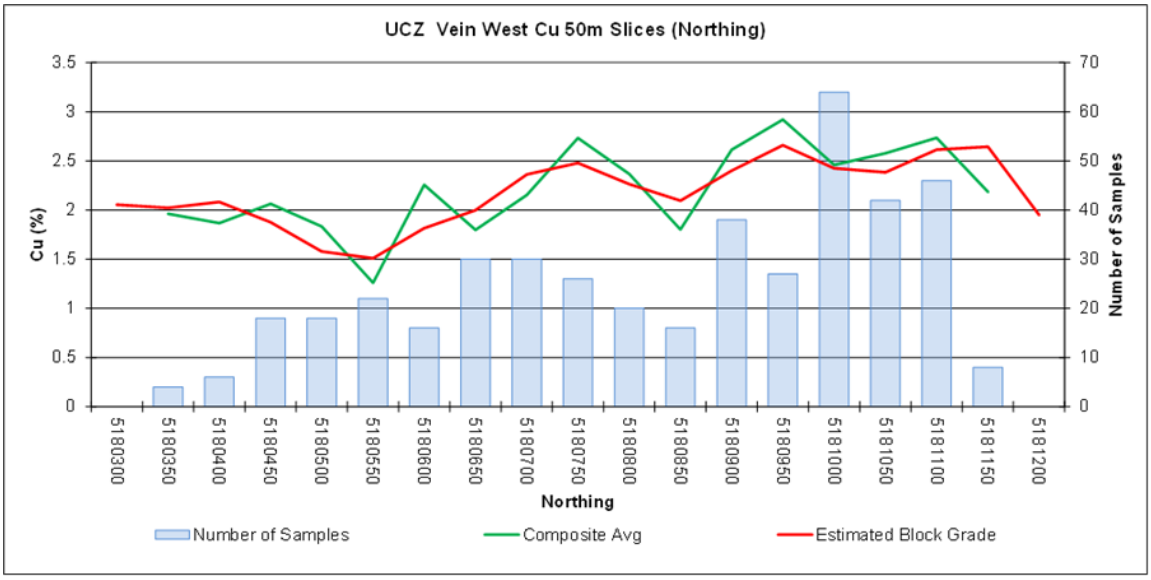


Figure 14.74 UCZ Vein West Cu Sectional Profile (Northing) – Composite Grade versus Model Grade

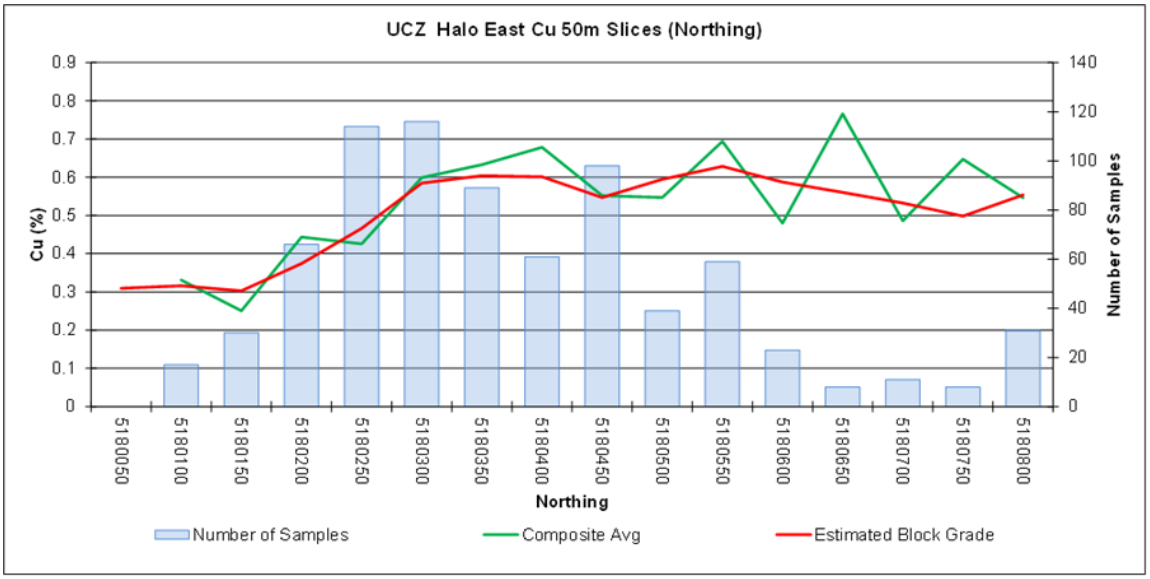


Figure 14.75 UCZ Halo East Cu Sectional Profile (Northing) – Composite Grade versus Model Grade

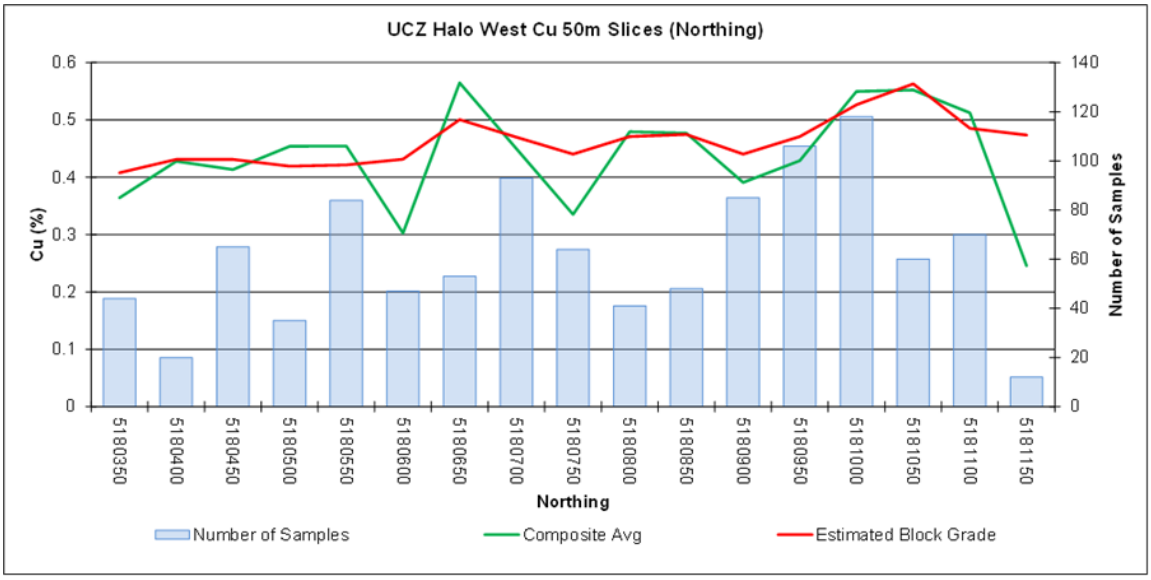


Figure 14.76 UCZ Halo West Cu Sectional Profile (Northing) – Composite Grade versus Model Grade

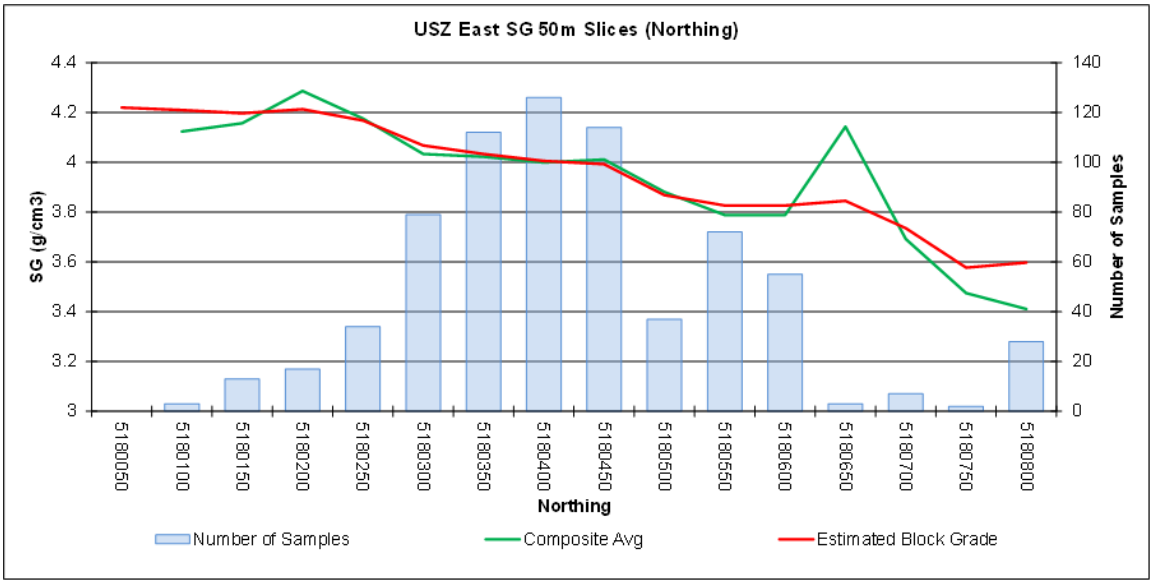


Figure 14.77 USZ East SG Sectional Profile (Northing) – Composite Grade versus Model Grade

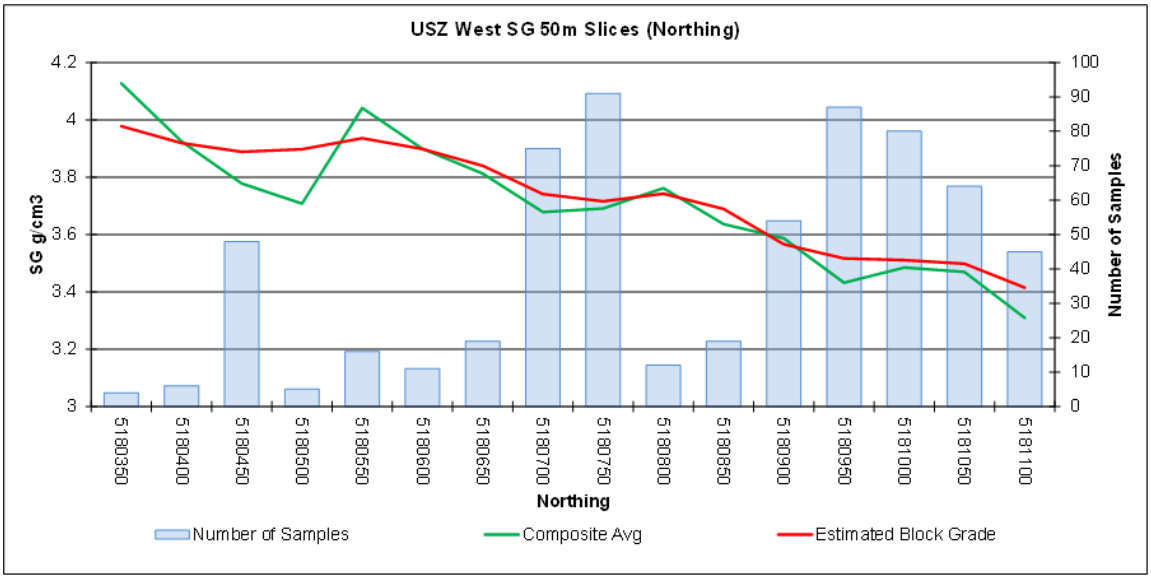


Figure 14.78 USZ West SG Sectional Profile (Northing) – Composite Grade versus Model SG

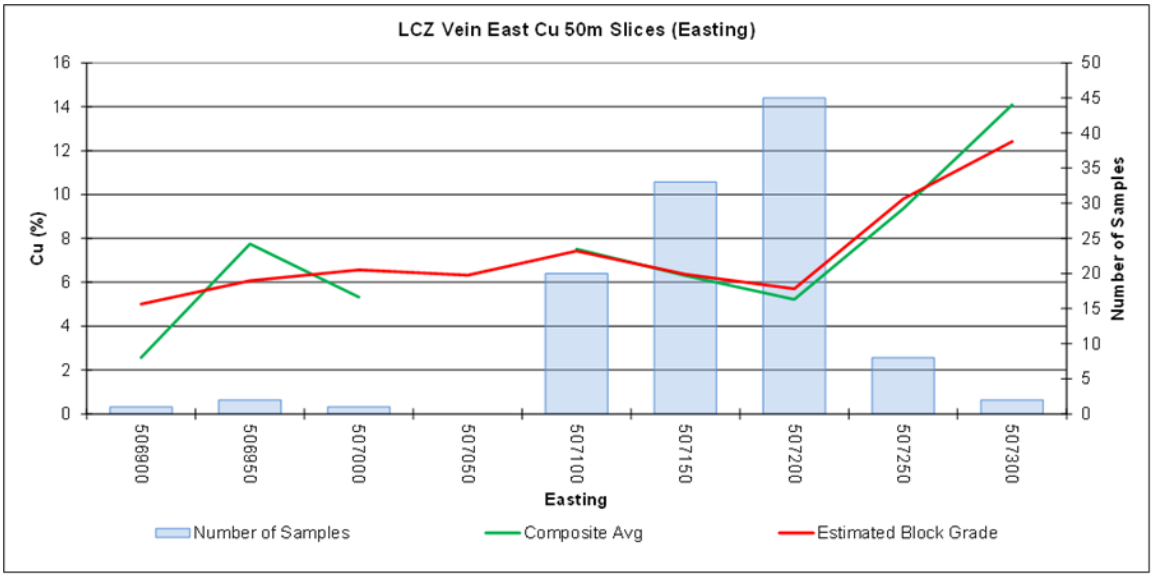


Figure 14.79 LCZ Vein East Cu Sectional Profile (Easting) – Composite Grade versus Model Grade

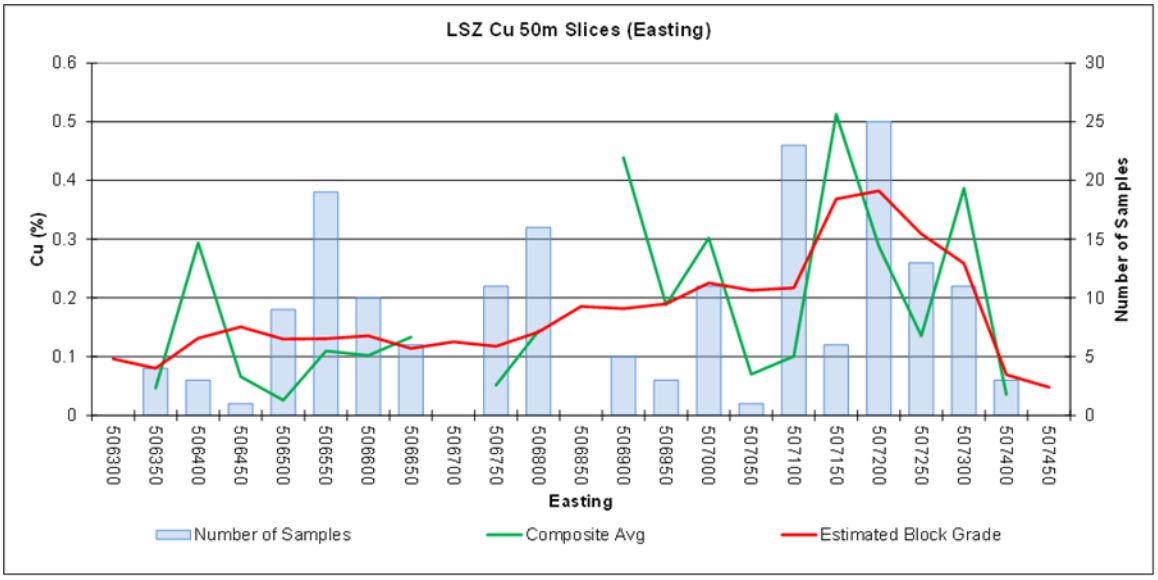


Figure 14.80 LSZ Cu Sectional Profile (Easting) – Composite Grade versus Model Grade

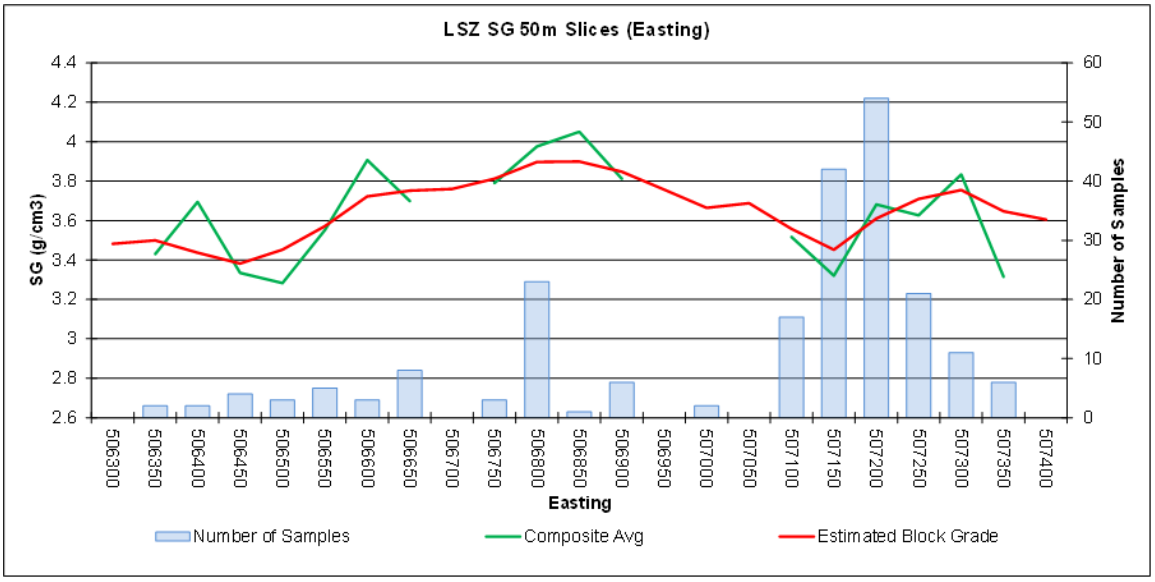


Figure 14.81 LSZ SG Sectional Profile (Easting) – Composite SG versus Model SG

14.8.2 Lowry deposit

The 2020 Lowry block model was validated using multiple techniques including: visual validation of block grades with raw drilling data and a statistical comparison of composites and block grades through swath plots and mean comparison by domain. In summary, it is the opinion of the QP that the model validates well against raw and composited data and is suitable for use in the determination of Mineral Resources as per CIM guidelines and standards.

Visual Validation

Cross-sectional visual comparisons were made between estimated block grades and raw drilling grades. Visual validation was performed using Vulcan™ software, version 11.0.4. Estimated quality variables of Cu, Ag, Co, As, and SG were compared with blocks showing an acceptable representation of drilling grades. An example of a visual validation cross-section is provided in Figure 14.82.

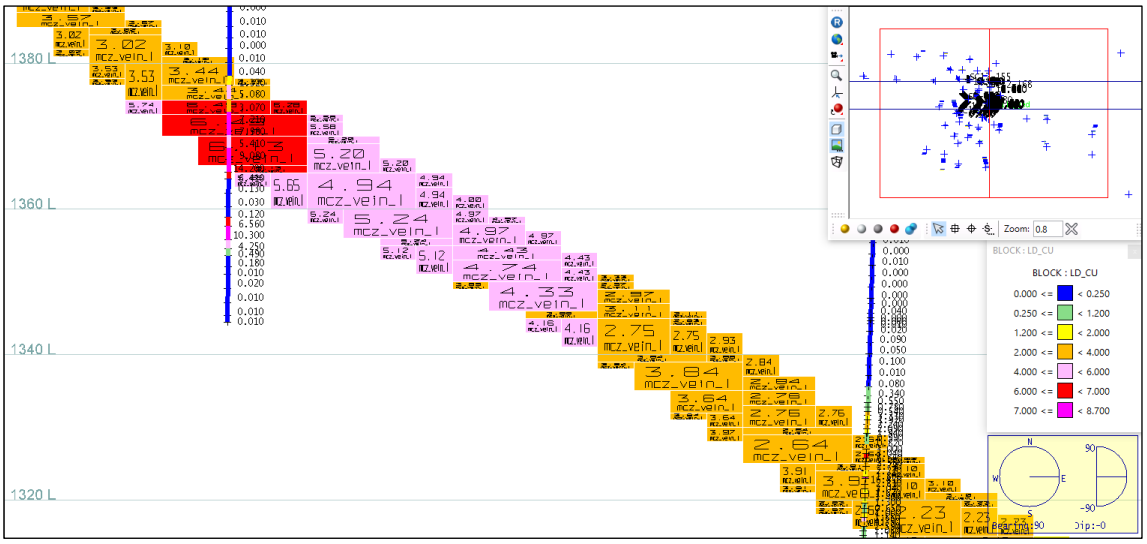


Figure 14.82 Cross-section showing Drill hole to Block Grade Comparisons

Statistics

Composites and block grades were compared to determine differences in the descriptive statistics of the data by domain through the estimation process (Figure 14.83). Statistical comparisons are performed to identify potential biases or significant changes in mean values due to the volume-variance effect where a data population's variance is reduced when using point support data (composites) to estimate volumes (blocks). In non-Gaussian distributed populations such as the Lowry high-grade (> 1.2% Cu) domains, this typically results in reduction in the mean as the tail portion of composite data distribution is reduced through the volume-variance effect during block estimation.

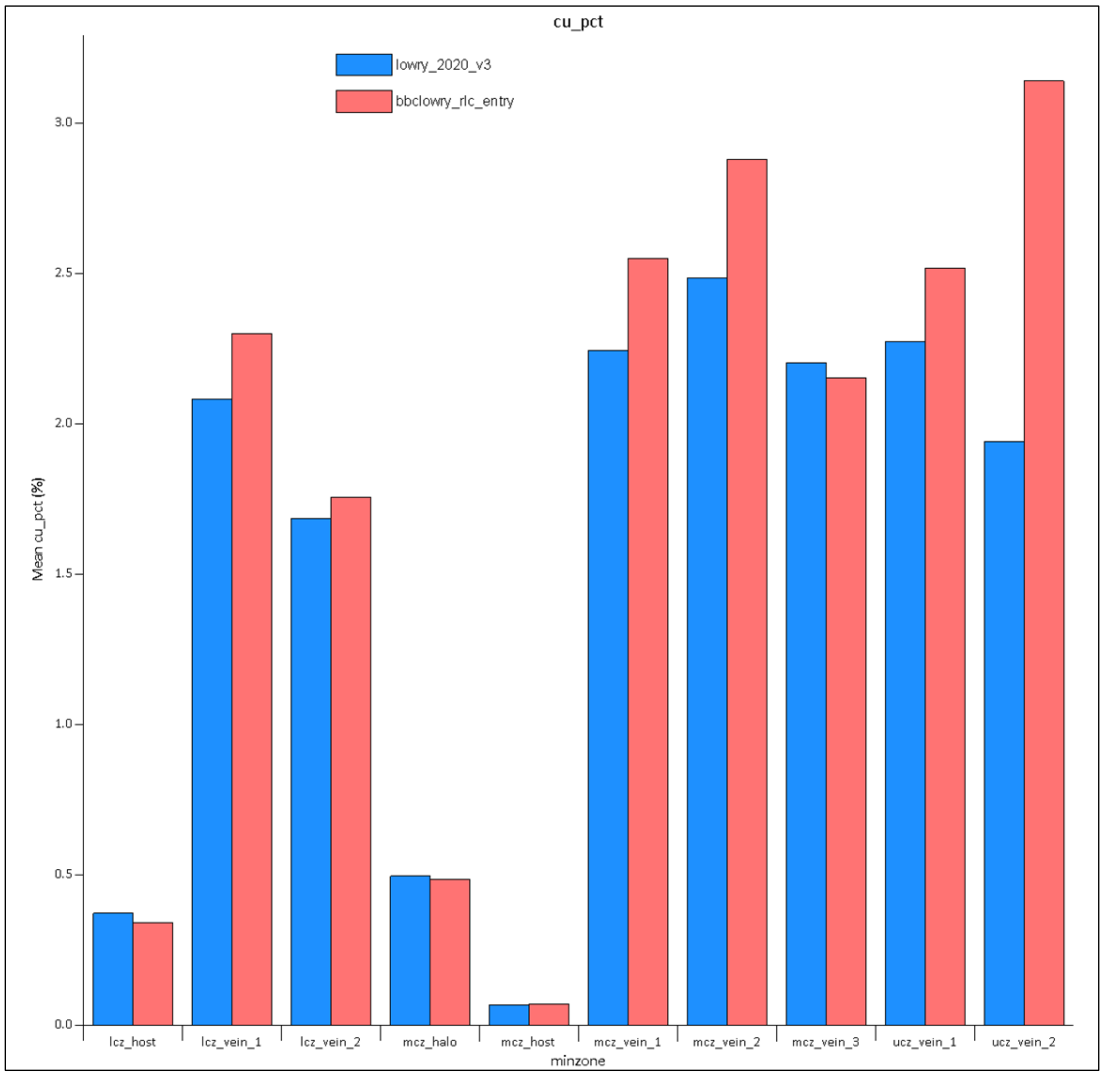


Figure 14.83 Comparison of Mean Cu Grades by Domain for Composites (red) and Blocks (blue)

SG mean grade comparison shows acceptable correlations with nearly identical mean values (Figure 14.84). This is primarily due to a near-normal distribution of SG data per domain compared with Cu and other quality variables that exhibit a near log-normal distribution. The domain “rock” and UCZ_Vein_1 did not contain any composite values due to lack of samples resulting in the lack of a composite column in Figure 14.84. The rock domain represents all in situ non-mineralized material. For these two domains, mean SG values were assigned.

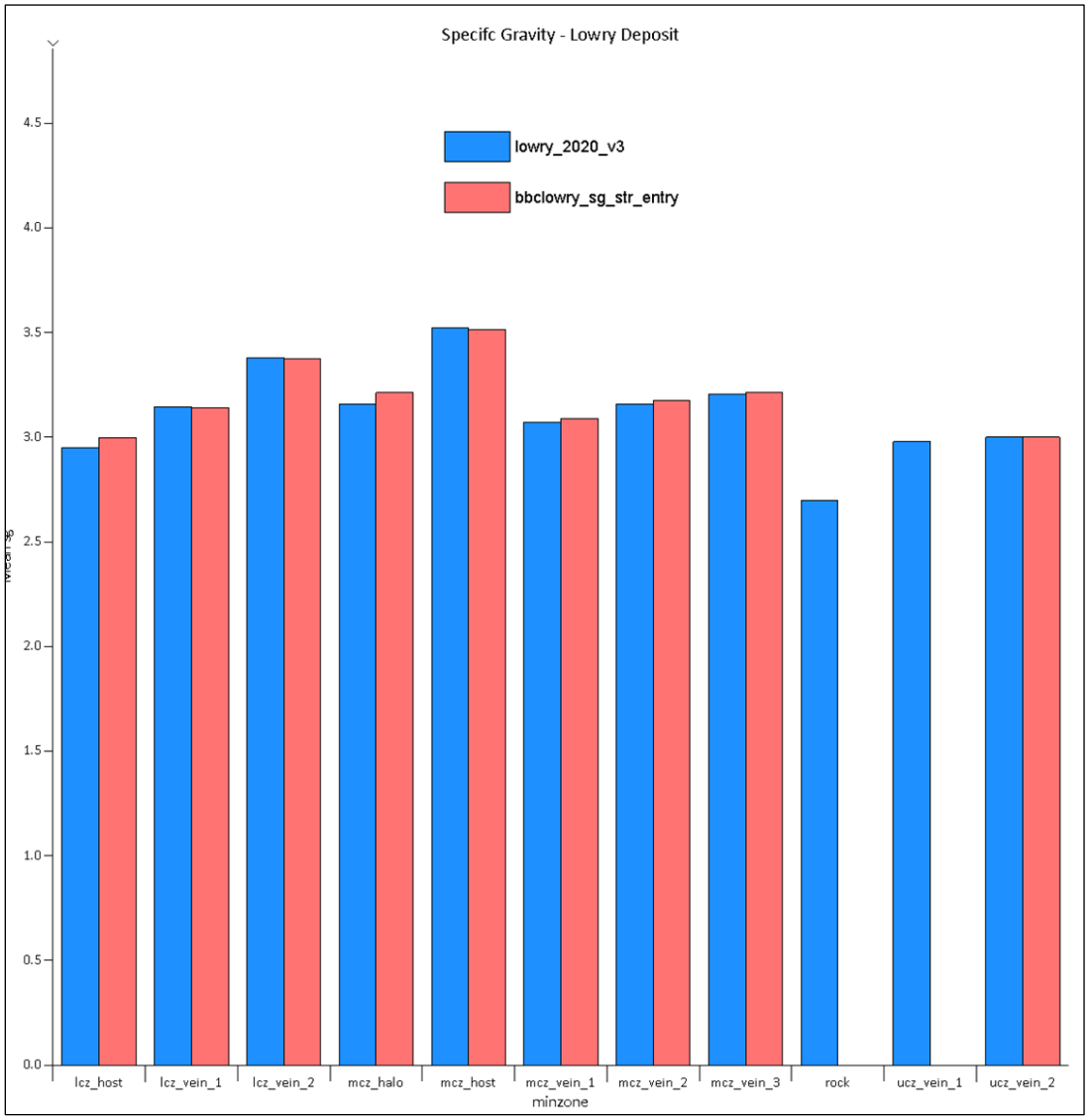


Figure 14.84 Comparison of Mean SG between Composites (red) and Blocks (blue)

Box and whisker plots were calculated for Cu and SG to compare composite grade distributions with estimated block grade distribution. Figure 14.85 shows Cu while Figure 14.86 shows SG values. In both cases, block variance is reduced due to volume increases while the means are acceptably close.

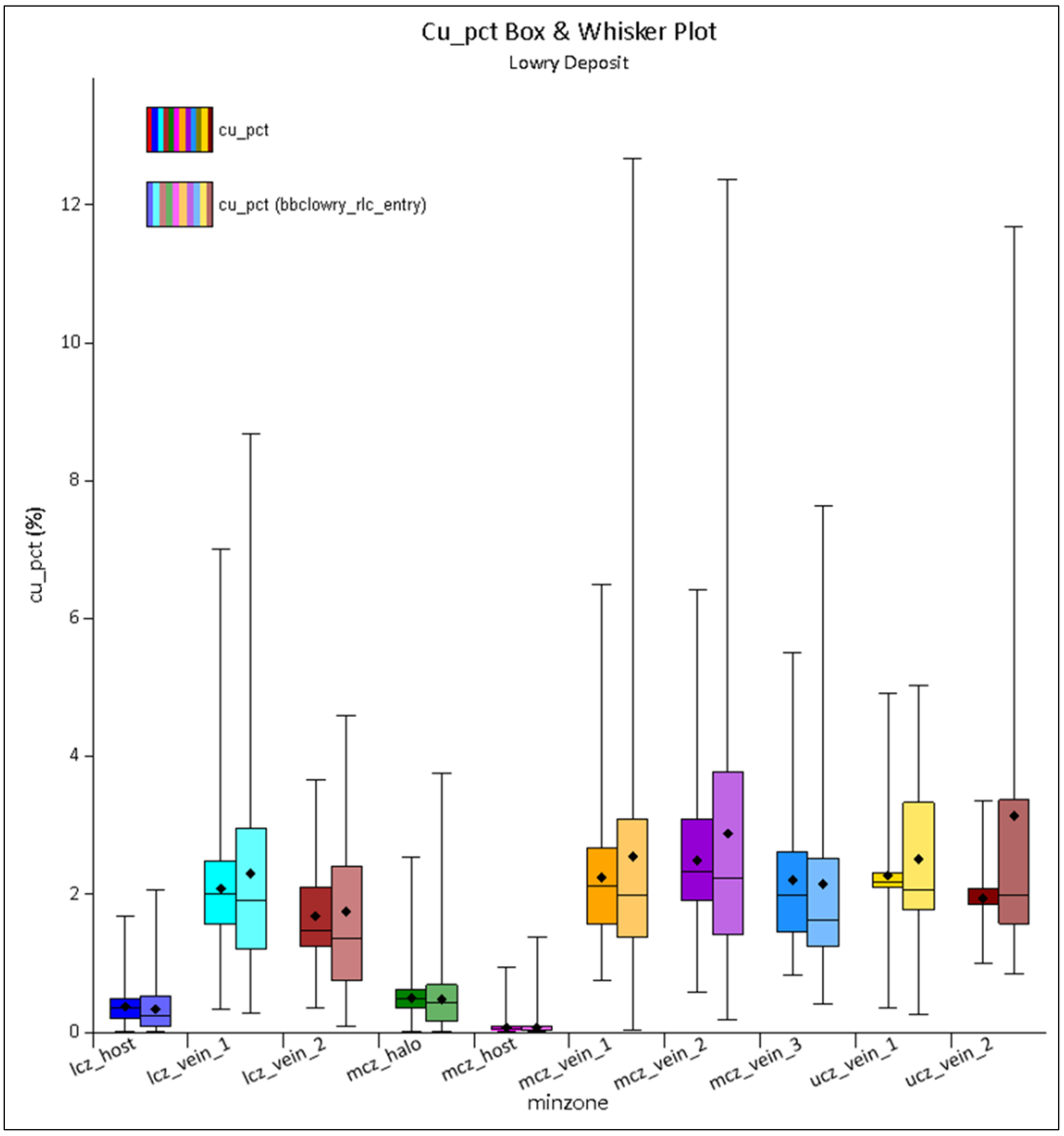


Figure 14.85 Box and Whisker Plot Comparing Cu in Composites (right side) and Blocks (left side)

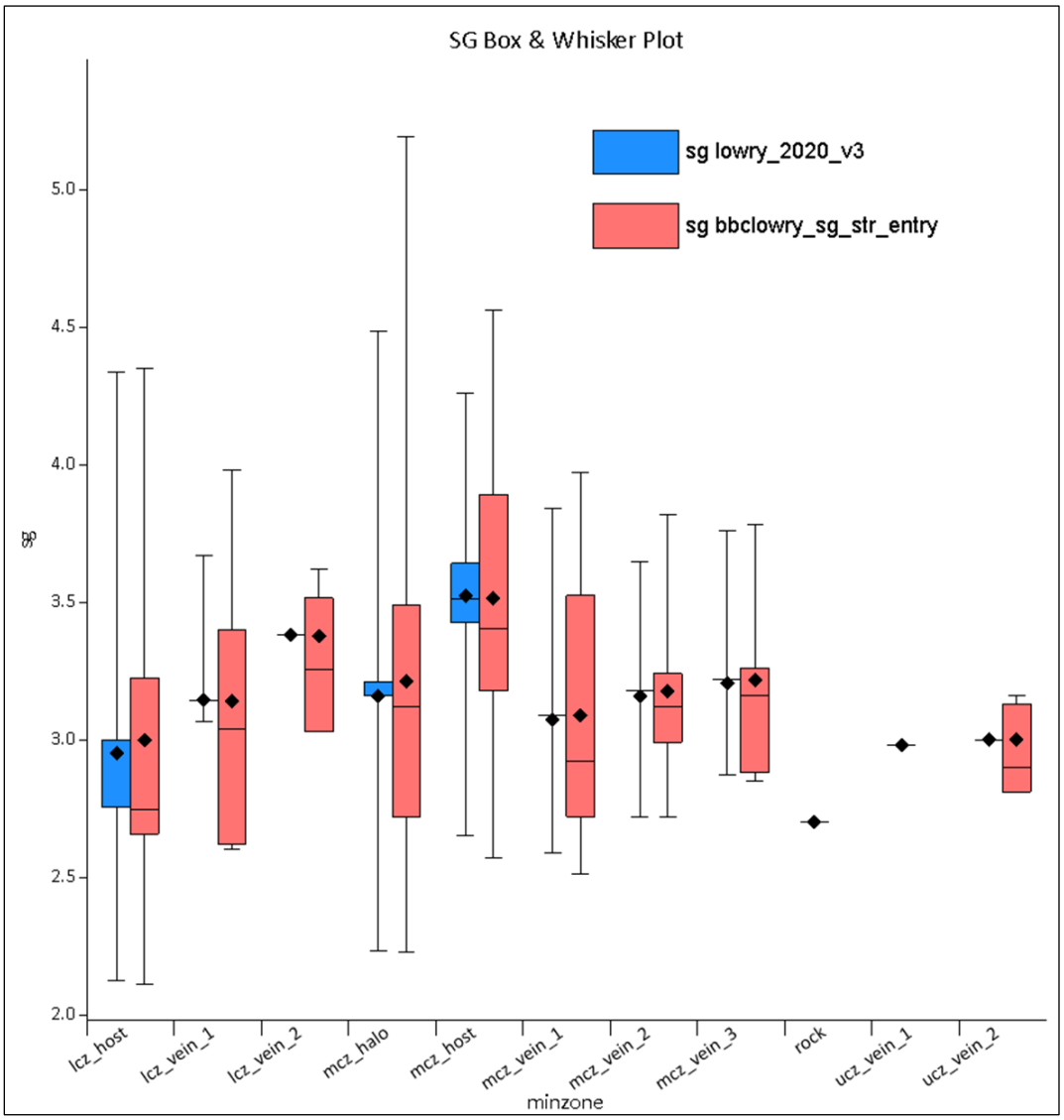


Figure 14.86 Box and Whisker Plot Comparing SG in Composites (red) and Blocks (blue)

Swath Plots

Swath plots were calculated in the X, Y, and Z directions for the major elements to compare mean composite values with mean block values along sections or swaths across the block model. This provides insight into differences and potential biases between composited and estimated block data. Commonly, mean composite data is compared with the estimated mean block data and a Nearest Neighbor (NN) estimate. The NN data provides a de-clustered local mean of composite data while composites often show extreme ranges in values due to the nature of point support. Ideally, block estimates should provide a smoother estimate of grade that lies near the NN estimate and reflects local highs and lows represented in the composites.

It is the opinion of the QP that block estimates of quality variables and SG provide an acceptable representation of composite data with no observed material biases. In the case of SG, due to low data density, the estimated values represent a near-mean value with minor local changes due to variations in the SG composites. Example swath plots are presented in Figure 14.87 through Figure 14.92.

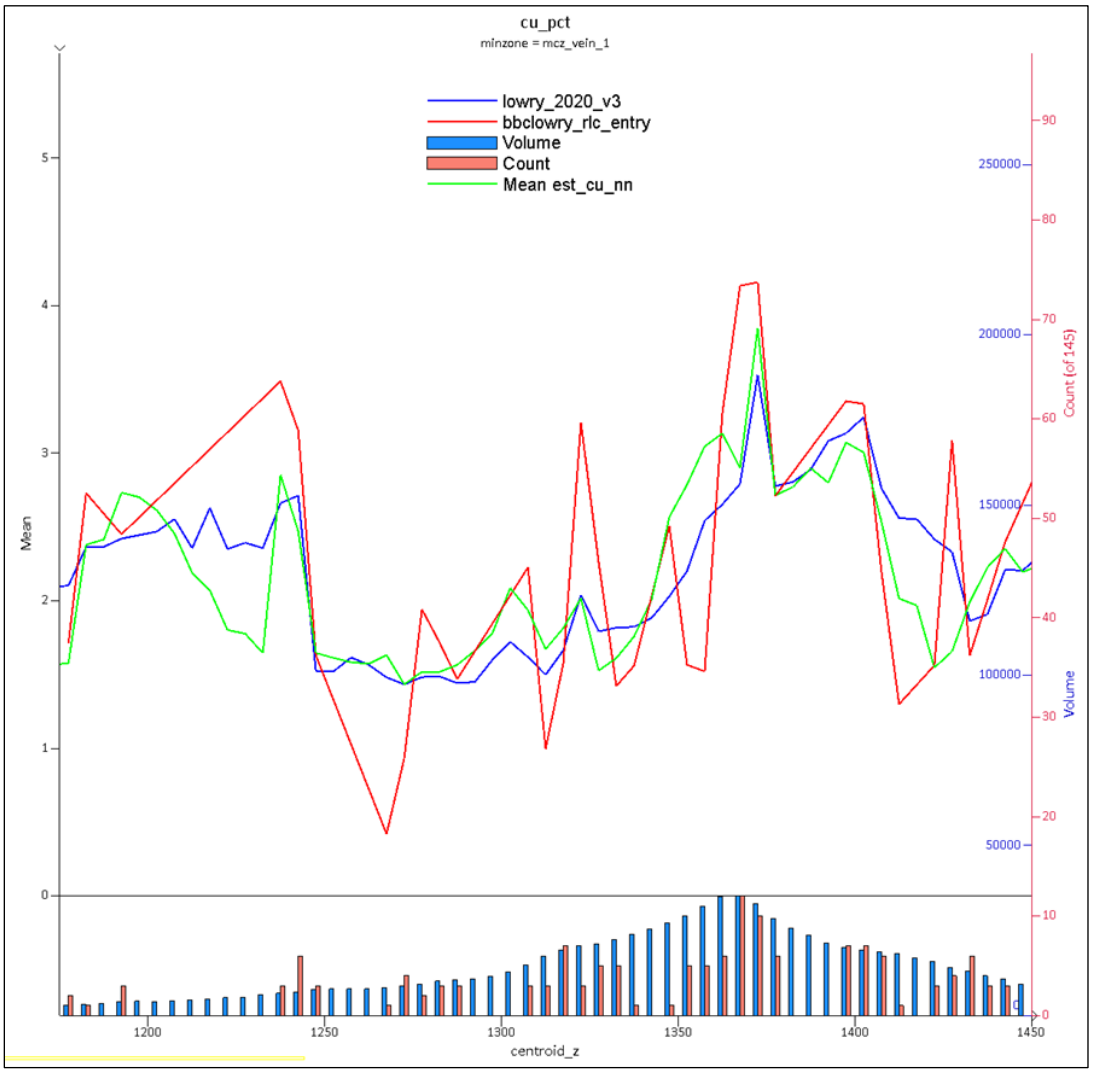


Figure 14.87 Swath Plot along Z for Cu in the LMCZ_Vein_1 Domain

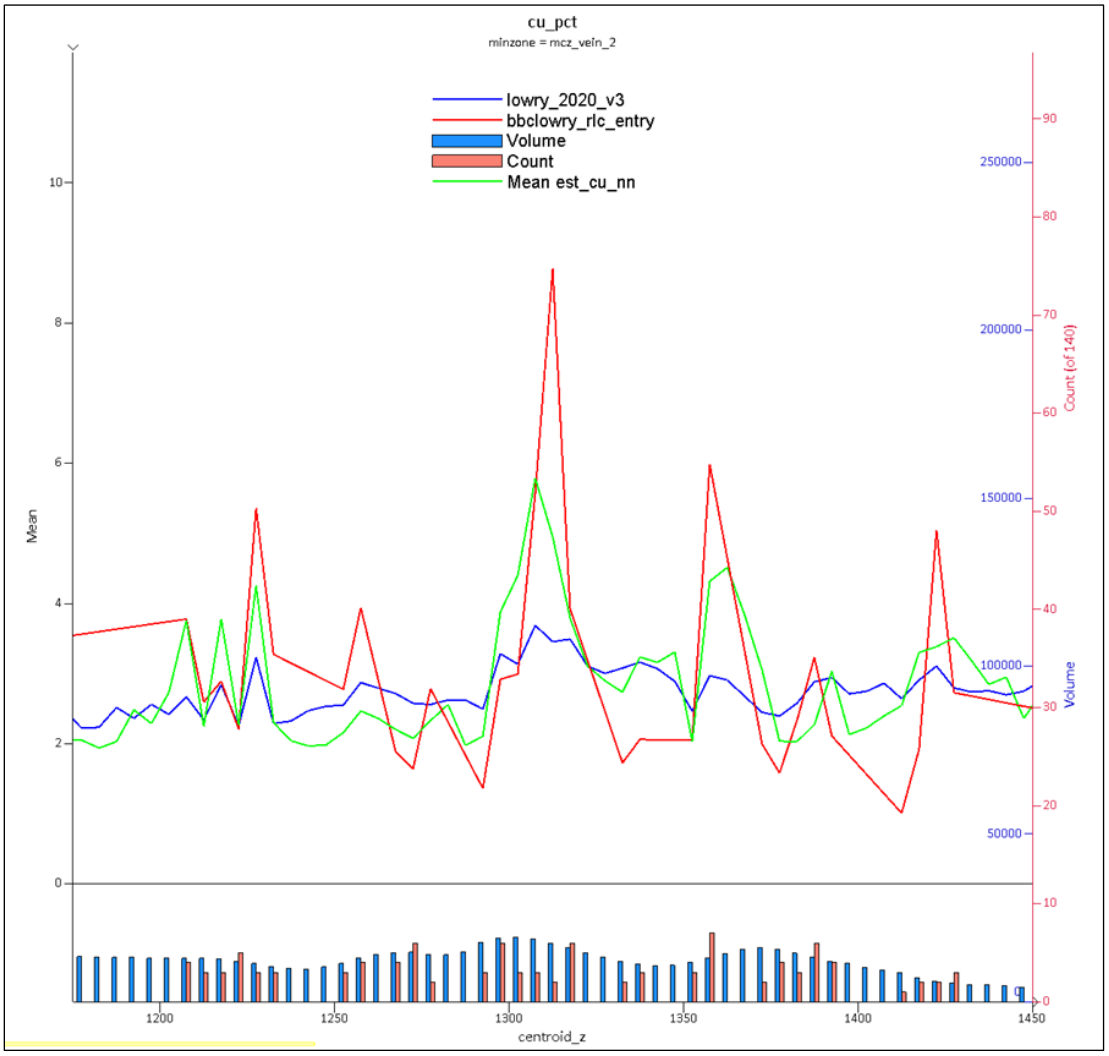


Figure 14.88 Swath Plot along Z for Cu in the LMCZ_Vein_2 Domain

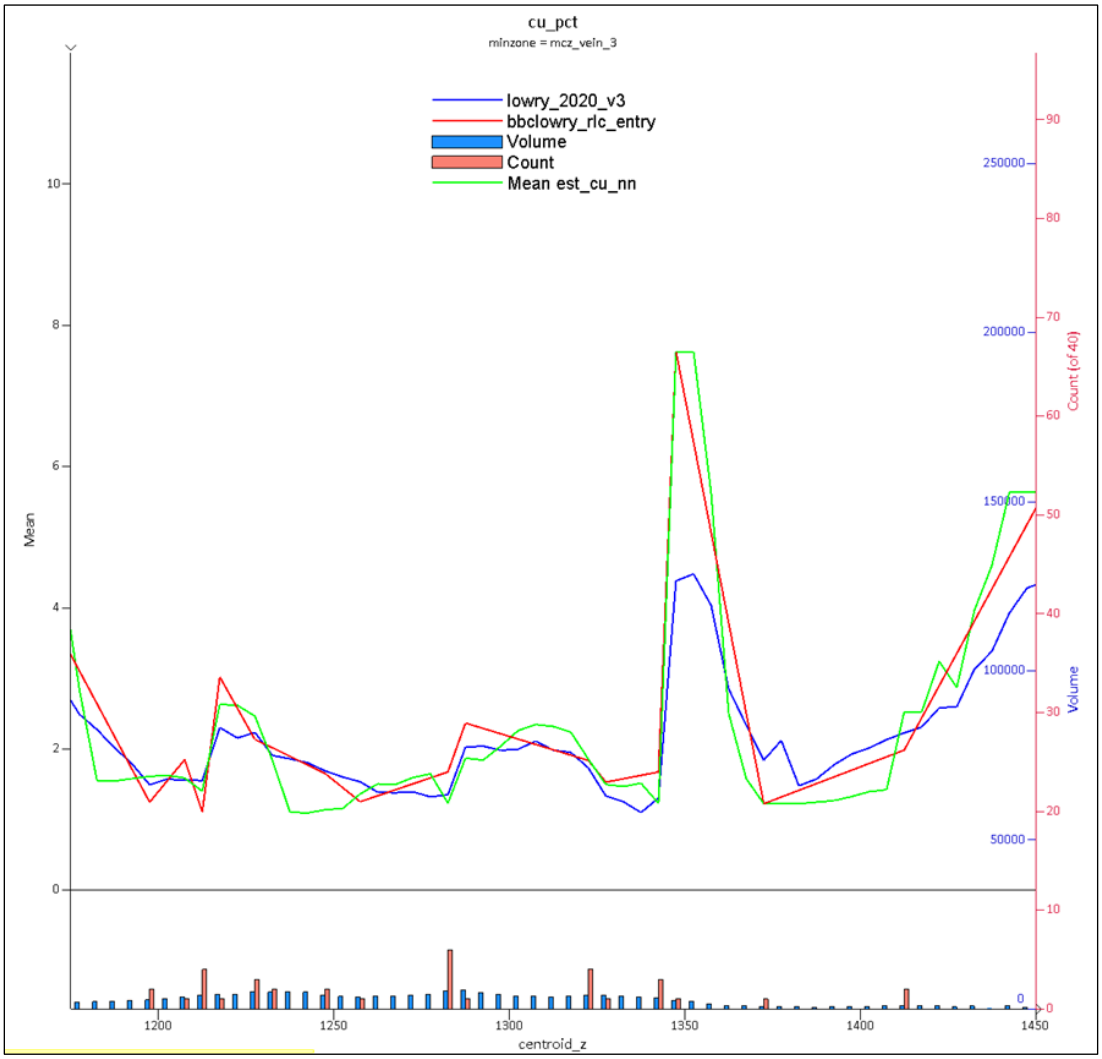


Figure 14.89 Swath Plot along Z for Cu in the LMCZ_Vein_3 Domain

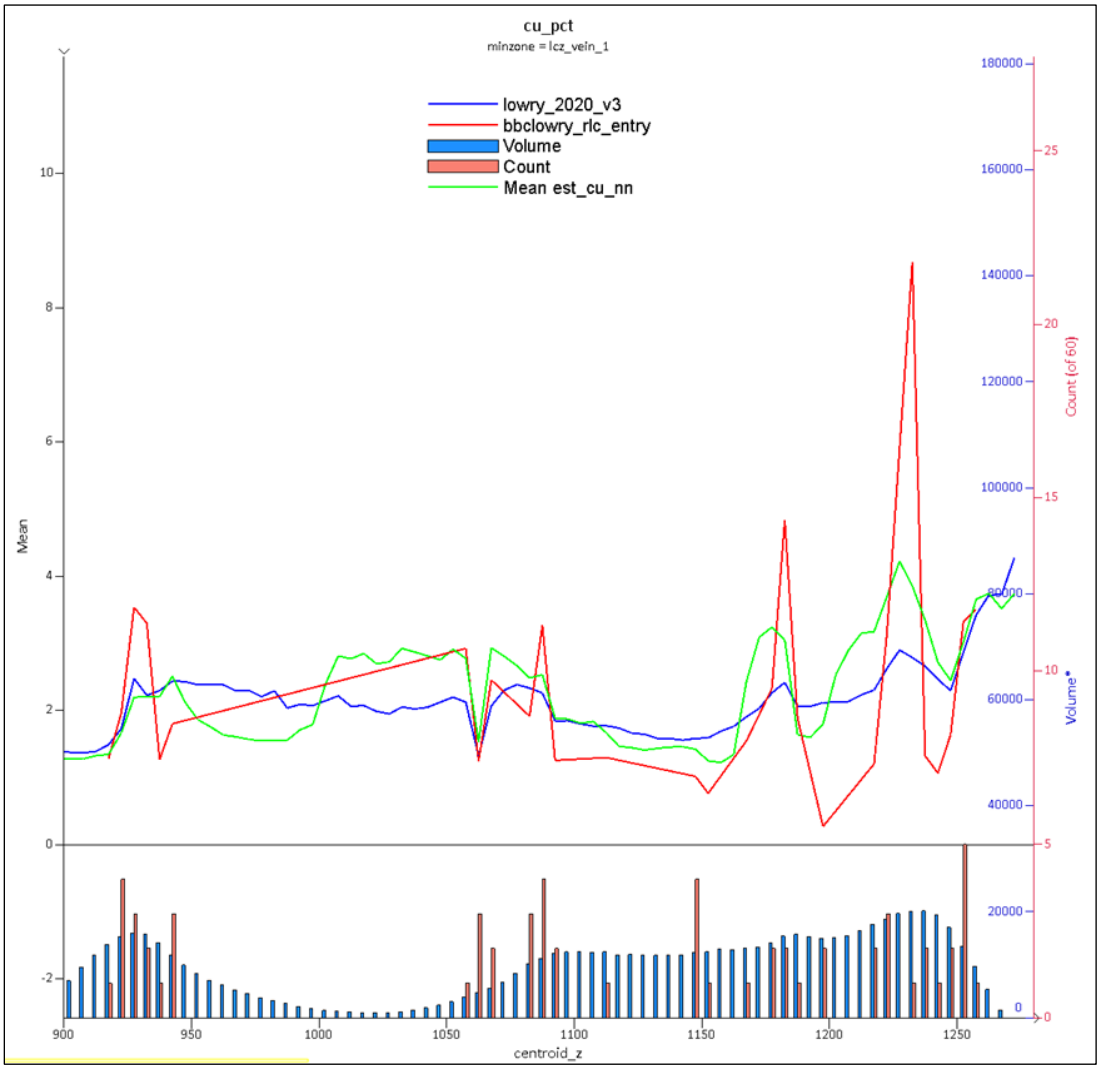


Figure 14.90 Swath Plot along Z for Cu in the LLCZ_Vein_1 Domain

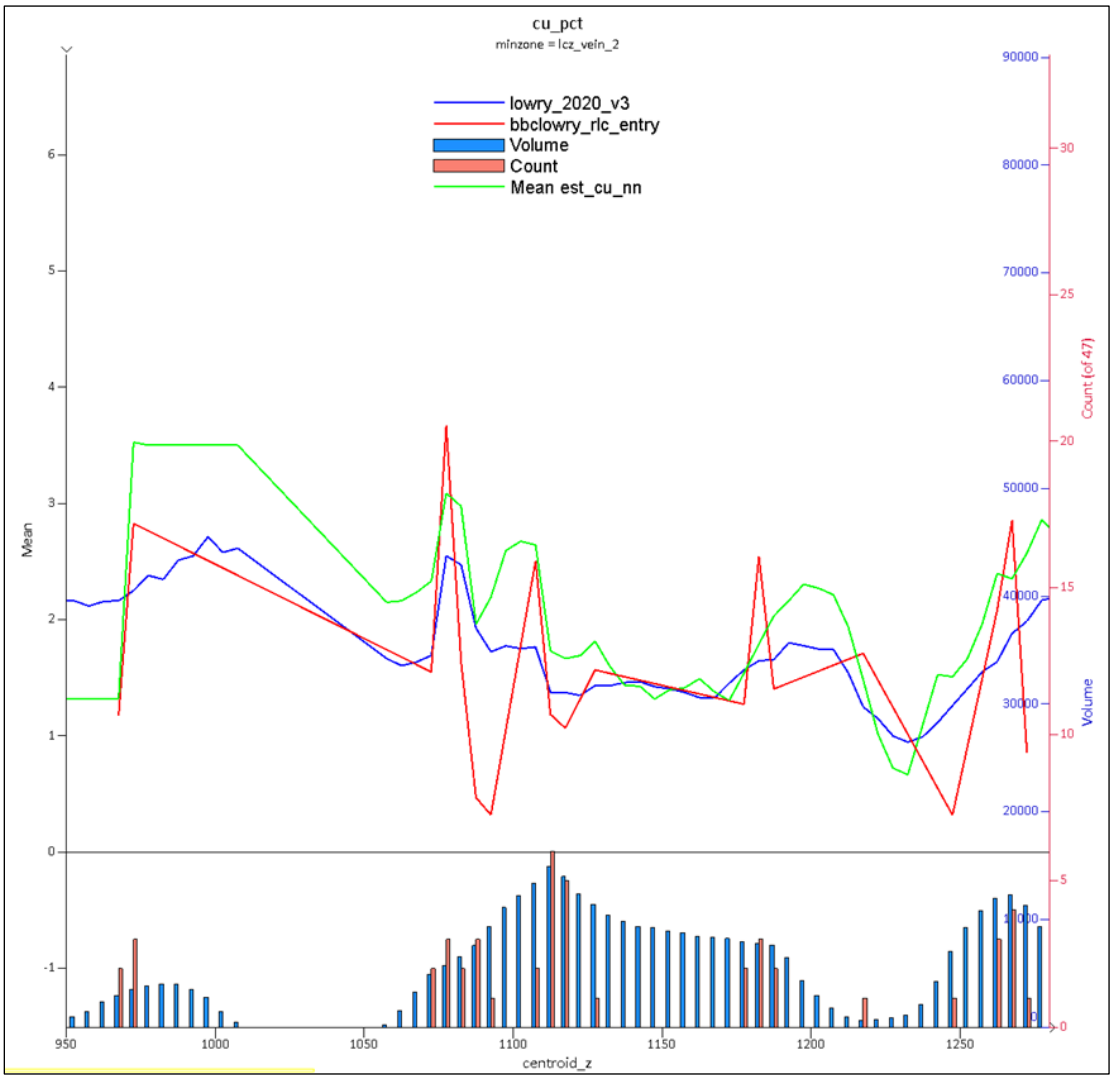


Figure 14.91 Swath Plot along Z for Cu in the LLCZ_Vein_2 Domain

Figure 14.92 Swath Plot along Z for SG in the LMCZ_Vein_2 Domain

Lowry Validation Summary

The Lowry block model validation was performed using a variety of techniques including visual and statistical validation. Summary validation shows acceptable smoothing of grades into block volumes from composite data. In domains with higher variability and highly skewed composite distributions, mean grades of blocks are commonly less than composite grades. It is the opinion of the QP that block quality estimates are acceptable and appropriate for the determination of Mineral Resources as per CIM guidelines. No material biases are observed through the estimation process.

14.9 Mineral Resource Classification

14.9.1 Johnny Lee deposit Classification

Mineral Resource classification was assigned to the Johnny Lee block models by the QP based upon: geological knowledge, continuity of Cu grade within mineralized zones, confidence in the underlying data (logging, assay, and physical testing), spatial continuity as determined through variography for Cu and SG, spatial continuity of recovery data, Kriging quality variables (kriging efficiency), and drill sample spacing on a domain basis. Blocks within the UCZ and LCZ have been categorized as Measured, Indicated, and Inferred classification consistent with NI 43-101 and the CIM definitions and guidelines (CIM, 2014). A combination of wireframe volumes and scripting of specific blocks was used to apply the appropriate block classification of Mineral Resource categories.

Measured Mineral Resource classification is assigned to blocks based upon high confidence in geology, Cu grade, and SG continuity, acceptable QA/QC of fundamental data, increased confidence in estimation quality based on Kriging efficiency values greater than 0.7, drill spacing less than 50 m, and reasonable recovery data. Measured blocks in the UCZ and LCZ are concentrated in zones of tight drill spacing as illustrated in Figure 14.93 and Figure 14.94.

Indicated Mineral Resource classification is assigned to blocks based on moderate to high confidence in geology, Cu grade, and SG continuity, acceptable QA/QC of fundamental data, moderate confidence in estimation quality based on Kriging efficiency values typically less than 0.7, drill spacing of approximately 50 m, and reasonable recovery data. Indicated blocks constitute the majority of Mineral Resources.

Inferred Mineral Resource classification is assigned to blocks based on moderate confidence in geology, Cu grade, and SG continuity, acceptable QA/QC, low confidence in estimation quality based on Kriging efficiency, and drill spacing greater than 50 m with a low degree of confidence in recovery data. Inferred blocks represent mineralized material within a modelled volume with limited or wide-spaced data supporting the continuity. The modelled volume has been bounded by drilling in most or all directions. The majority of the LCZ has been classified as Inferred Mineral Resources due to wide-spaced drilling in the western portions of the zone.

It is the opinion of the QP that the classification for the Johnny Lee deposit Mineral Resource is reasonable for the type of mineralization and deposit.

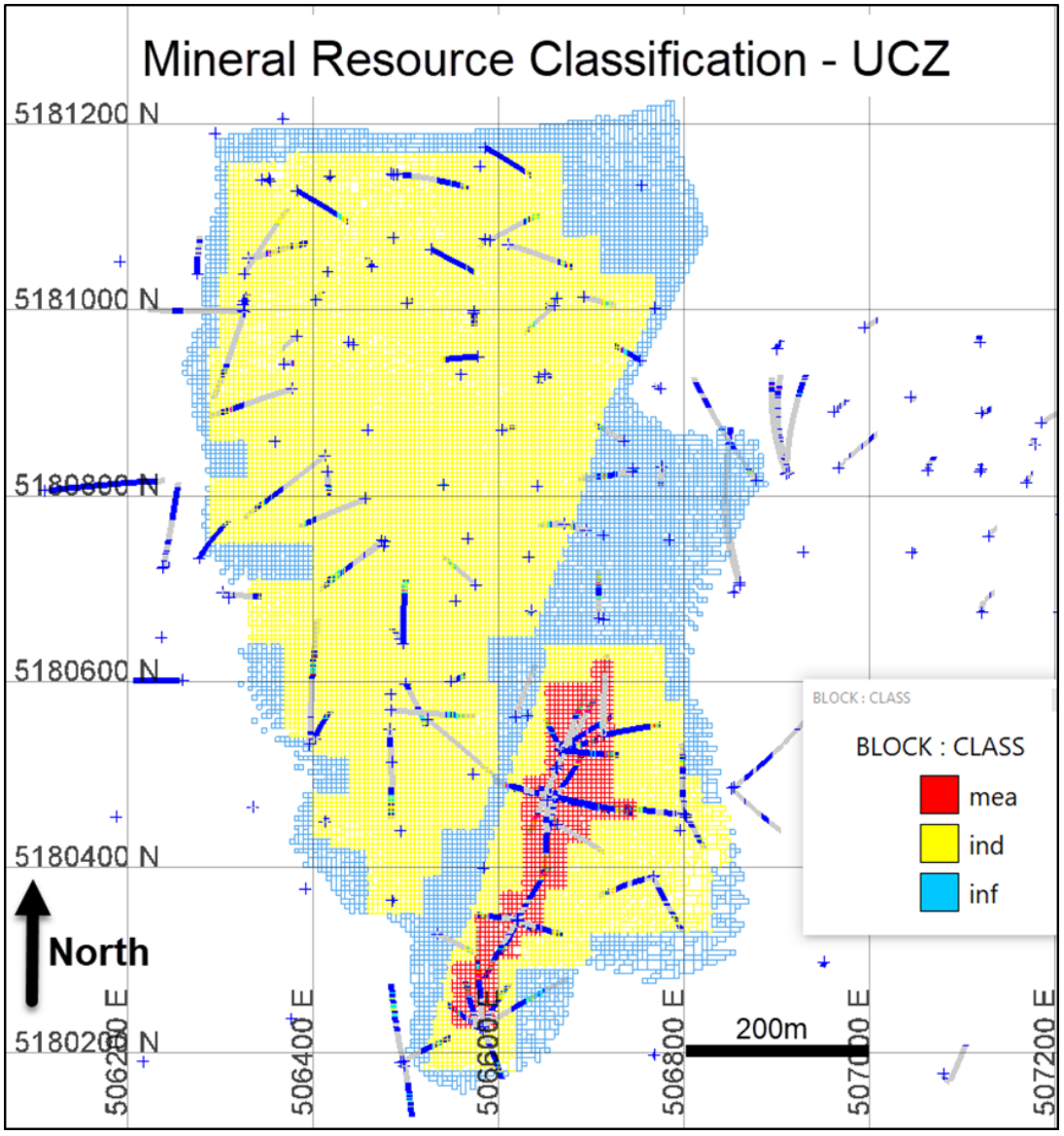


Figure 14.93 Plan View of the UCZ showing Mineral Resource Classification

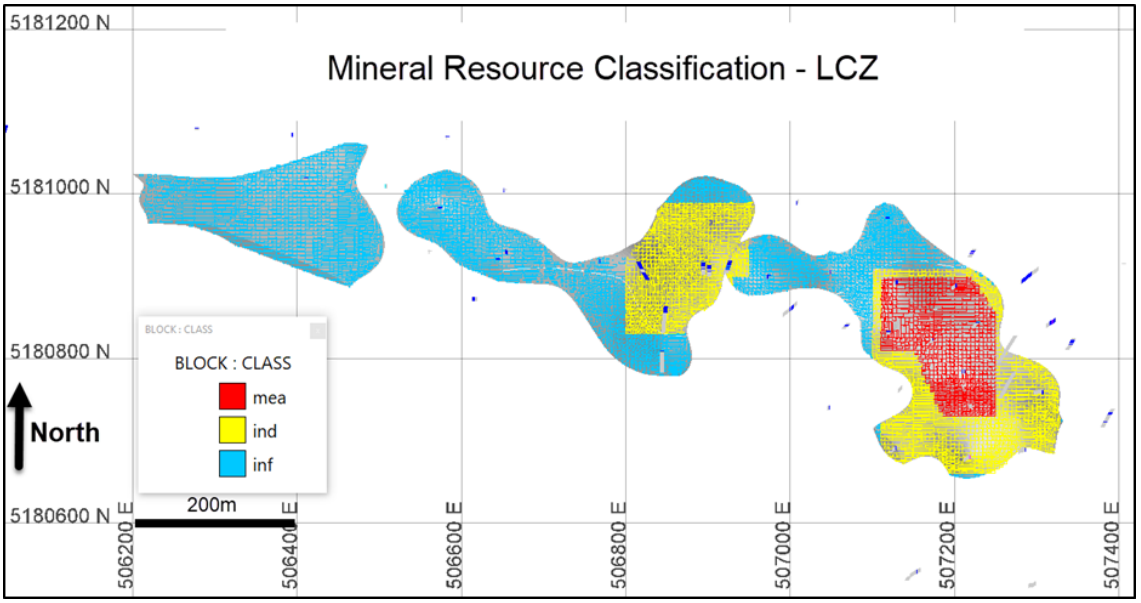


Figure 14.94 Plan View of the LCZ showing Mineral Resource Classification

14.9.2 Lowry Deposit Classification

Mineral Resource classification was assigned to the Lowry deposit block model by the QP based upon: geological knowledge, continuity of Cu grade within mineralized zones, thickness of the mineralized zones, confidence in the underlying data (logging, assay, and physical testing), spatial continuity as determined through variography for Cu, recovery data, Kriging quality variables (kriging efficiency, average distance to samples, and estimation run pass), and drill sample spacing on a domain basis. Blocks within the LMCZ and LLCZ have been categorized as Inferred classification consistent with NI 43-101 and the CIM definitions and guidelines (CIM, 2014). Mineralized material in the LUCZ and the LMCZ Halo zone was not deemed acceptable for classification at this time. A combination of wireframe volumes and scripting of specific blocks was used to apply the appropriate block classification of Mineral Resource categories.

14.10 Demonstration of Potential for Economic Extraction

As per CIM definitions of Mineral Resources, material must demonstrate reasonable prospects for eventual economic extraction. For the Black Butte Copper Project, use of cut-off grades has been applied to each deposit to satisfy this implication.

14.10.1 Johnny Lee Cut-Off Grade

SRK has applied a Cut-Off Grade (CoG) that accounts for operational costs based on the assumed mining method proposed, assumed processing costs, assumed general & administrative costs, metallurgical recovery of Cu, and market-driven Cu pricing. Updated operational costs were used along with recent

metallurgical recovery work and updated Cu price projections. The following technical and economic parameters are assumed and accounted for in the determination of CoG:

- Metallurgical recoveries: Estimated block model variable Cu recovery for the UCZ. Assigned mean 94% Cu recovery for the LCZ.
- Operational Costs: $71/t. This includes mining ($50/t), processing ($15/t) and general and administrative costs ($6/t).
- Cu Price: A long term Cu price of $3.20/lb ($7,055/t) is based on market consensus forecast for upside Cu pricing.

Using these metrics, a cut-off grade of 1.0% Cu based on recovery assumptions was used for the entire Johnny Lee deposit.

14.10.2 Lowry deposit Cut-Off Grade

SRK has applied a CoG that accounts for operational costs based on the assumed mining method proposed, assumed processing costs, assumed general and administrative costs, metallurgical recovery of Cu, and market-driven Cu pricing. The following technical and economic parameters are assumed and accounted for in the determination of CoG:

- Metallurgical recoveries: Assigned mean 86% Cu recovery for the Lowry deposit based upon multiple metallurgical tests conducted on samples across the deposit;
- Operational Costs: $71/t. This includes mining ($50/t), processing ($15/t) and general and administrative costs ($6/t). Operational cost assumptions are in-line with Johnny Lee costs as processing assumes use of the same plant and flotation facility; and
- Cu Price: A long term Cu price of $3.20 per pound ($7,055/t) is based on market consensus forecast for upside Cu pricing.

Using these metrics, a cut-off grade of 1.2% Cu based on recovery assumptions was used for the Lowry deposit.

14.11 Mineral Resource Statement

Summary Mineral Resources for the Johnny Lee (Table 14.29) and Lowry Deposits (Table 14.30) have been calculated and reported using an economic CoG applied to Cu as estimated in the resource block model. This Mineral Resource statement is supported by drilling, analyses, geological modelling, and extensive metallurgical studies to provide updated recoveries.

Category	Quantity (Mt)	Cu (%)	Total Metal (kt)
UCZ			
Measured	1.4	2.6	36.2
Indicated	8.3	2.3	191.3
Measured and Indicated	9.7	2.4	227.5
Inferred	2.2	2.2	49.5
LCZ			
Measured	0.6	5.7	32.9
Indicated	0.6	7.9	50.5
Measured and Indicated	1.2	6.8	83.4
Inferred	0.5	6.3	30.3
Combined UCZ + LCZ			
Measured	2.0	3.5	69.1
Indicated	8.9	2.7	241.8
Measured and Indicated	10.9	2.9	310.9
Inferred	2.7	3.0	79.7

Source: SRK (2019)

- The effective date for this Mineral Resource is October 15, 2019. All significant figures are rounded to reflect the relative accuracy of the estimates. Cu assay values were capped where appropriate;
- *Mineral Resources are inclusive of Mineral Reserves and have reasonable prospects for eventual economic extraction. Inferred Mineral Resources have a high degree of uncertainty as to their economic and technical feasibility. It cannot be assumed that all or any part of an Inferred Mineral Resources can be upgraded to Measured or Indicated Mineral Resources;*
- *Metallurgical recovery of Cu has been estimated on a block basis in the UCZ, averaging 77.4%, with a consistent 94.0% Cu recovery applied to the LCZ;*
- *To demonstrate reasonable prospects for eventual economic extraction of Mineral Resources, a cut-off grade of 1.00% Cu based on metal recoverability assumptions, long-term Cu price assumptions of $3.20/lb, mining costs, processing costs, G&A costs totalling $71/t;*
- *There are no known risks to the Johnny Lee Mineral Resources that could materially affect the potential development of the Mineral Resources other than that the Project is subject to legal challenges that are outlined in the Management Discussion and Analyses of the June 2020 Company Quarterly Report (also see Section 14.14 of this Technical Report). All Mineral Resources are located within land currently under control or lease to Sandfire Resources America, Inc;*
- *All Mineral Resources are located within land currently under control or lease to Sandfire Resources America Inc.*

Table 14.29 Black Butte Copper Project Mineral Resource Estimate for the Johnny Lee deposit as of October 15, 2019 – SRK Consulting (U.S.), Inc.

Category	Quantity (Mt)	Cu (%)	Total Metal (kt)
LMCZ			
Inferred	5.7	2.53	318.7
LLCZ			
Inferred	2.6	2.13	121.3
Combined LMCZ + LLCZ			
Inferred	8.3	2.41	440.0

Source: SRK (2020)

- *The effective date for this Mineral Resource is October 15, 2020. All significant figures are rounded to reflect the relative accuracy of the estimates. Cu assay values were capped where appropriate;*
- *Mineral Resources have reasonable prospects for eventual economic extraction. Inferred Mineral Resources have a high degree of uncertainty as to their economic and technical feasibility. It cannot be assumed that all or any part of an Inferred Mineral Resources can be upgraded to Measured or Indicated Mineral Resources*;
- *Metallurgical recovery of Cu has been assigned to the Lowry deposit using the mean recovery of 86% Cu;*
- *To demonstrate reasonable prospects for eventual economic extraction of Mineral Resources, a cut-off grade of 1.20% Cu based on metal recoverability assumptions, long-term Cu price assumptions of $3.20/lb, mining costs, processing costs, G&A costs totaling $71/t;*
- *All Mineral Resources are located within land currently under control or lease to Sandfire Resources America Inc., and*
- *There are potential permitting and other risks to the Lowry deposit that could materially affect the potential development of the Mineral Resources including those outlined in the Management Discussion and Analyses of the June 2020 Company Quarterly Report (see Section 14.14 of this Technical Report).*

Table 14.30 Black Butte Copper Project Mineral Resource Estimate for the Lowry deposit as of October 15, 2020 – SRK Consulting (U.S.), Inc.

14.12 Mineral Resource Sensitivity

14.12.1 Johnny Lee deposit

A series of grade-tonnage curves have been calculated to assess the sensitivity of Mineral Resources to changes in CoG. Figure 14.95 and Figure 14.96 show grade-tonnage curves for the UCZ and LCZ respectively. Summary tabulated Mineral Resource sensitivity for Measured and Indicated is presented in Table 14.31.

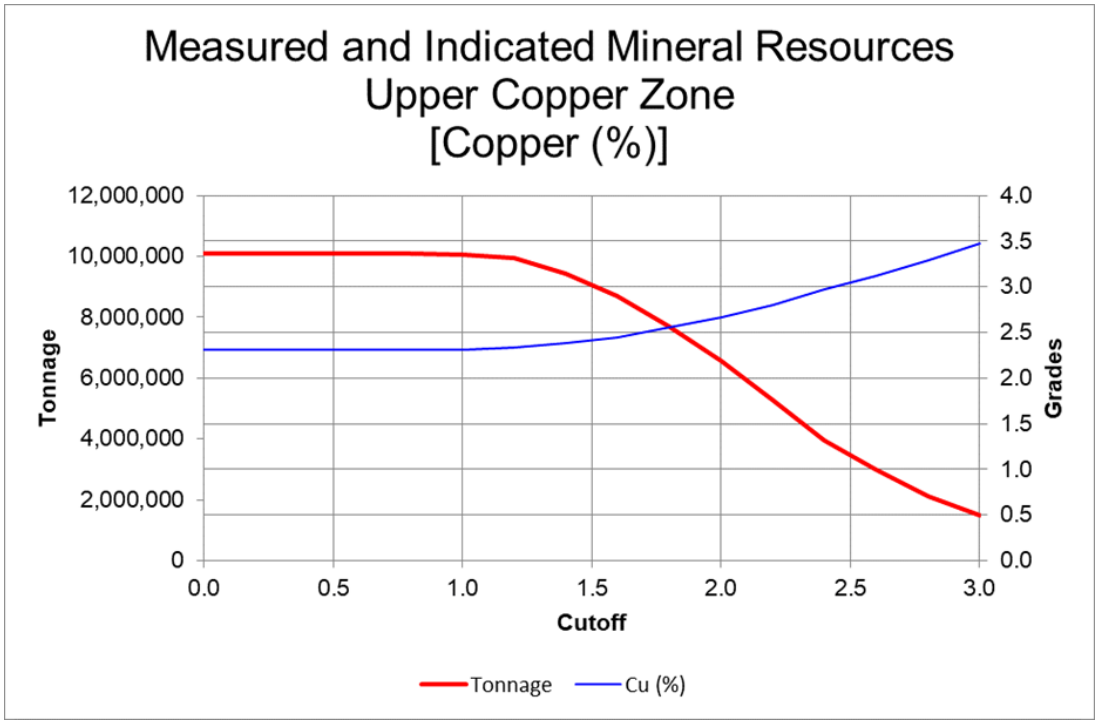


Figure 14.95 Grade-Tonnage Curve for the Johnny Lee UCZ showing Measured and Indicated Mineral Resources

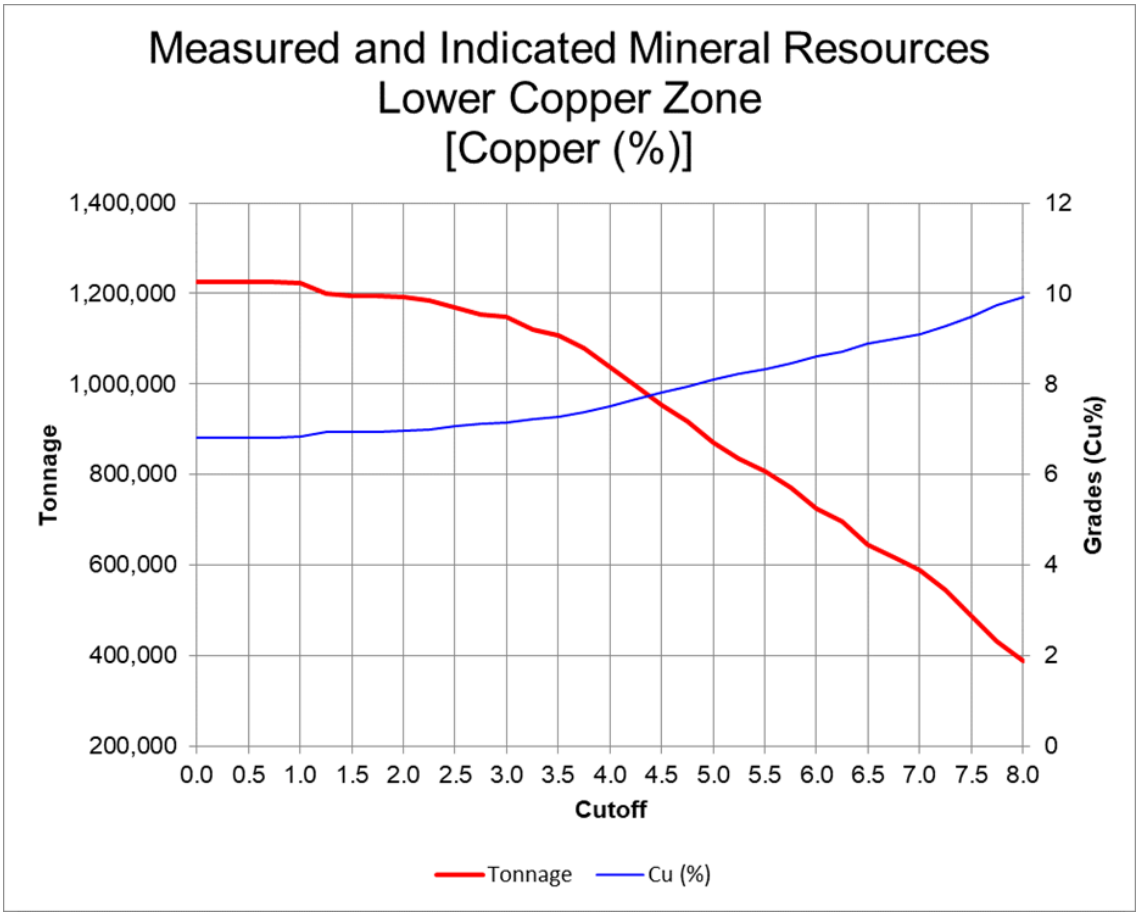


Figure 14.96 Grade-Tonnage Curve for the Johnny Lee LCZ showing Measured and Indicated Mineral Resources

Upper Copper Zone		
Cutoff	Cu (%)	Tonnage
0.4	2.31	10,091,034
0.6	2.31	10,091,034
0.8	2.31	10,086,602
1.0	2.31	10,053,764
1.2	2.33	9,927,710
1.4	2.38	9,442,759
1.6	2.45	8,693,067
1.8	2.55	7,703,103
2.0	2.66	6,553,338
2.2	2.80	5,266,876
2.4	2.97	3,957,391
2.6	3.12	2,998,815
2.8	3.29	2,122,509
3.0	3.47	1,480,408

Lower Copper Zone		
Cutoff	Cu (%)	Tonnage
0.5	6.82	1,224,553
1.0	6.83	1,221,925
1.5	6.95	1,195,444
2.0	6.97	1,190,870
2.5	7.06	1,168,614
3.0	7.14	1,147,675
3.5	7.28	1,106,614
4.0	7.51	1,038,634
4.5	7.81	953,463
5.0	8.10	870,197
5.5	8.32	807,146
6.0	8.61	723,935
6.5	8.90	645,146
7.0	9.10	589,415
7.5	9.49	488,389
8.0	9.93	388,473

Table 14.31 Johnny Lee Mineral Resource Sensitivity (Measured and Indicated Only)

14.12.2 Lowry deposit

A grade-tonnage curve has been calculated to assess the sensitivity of Mineral Resources to changes in CoG. Figure 14.97 shows the grade-tonnage curve for the combined Inferred Mineral Resources of the LMCZ and LLCZ of the Lowry deposit. Summary tabulated Inferred Mineral Resource sensitivity is presented in Table 14.32.

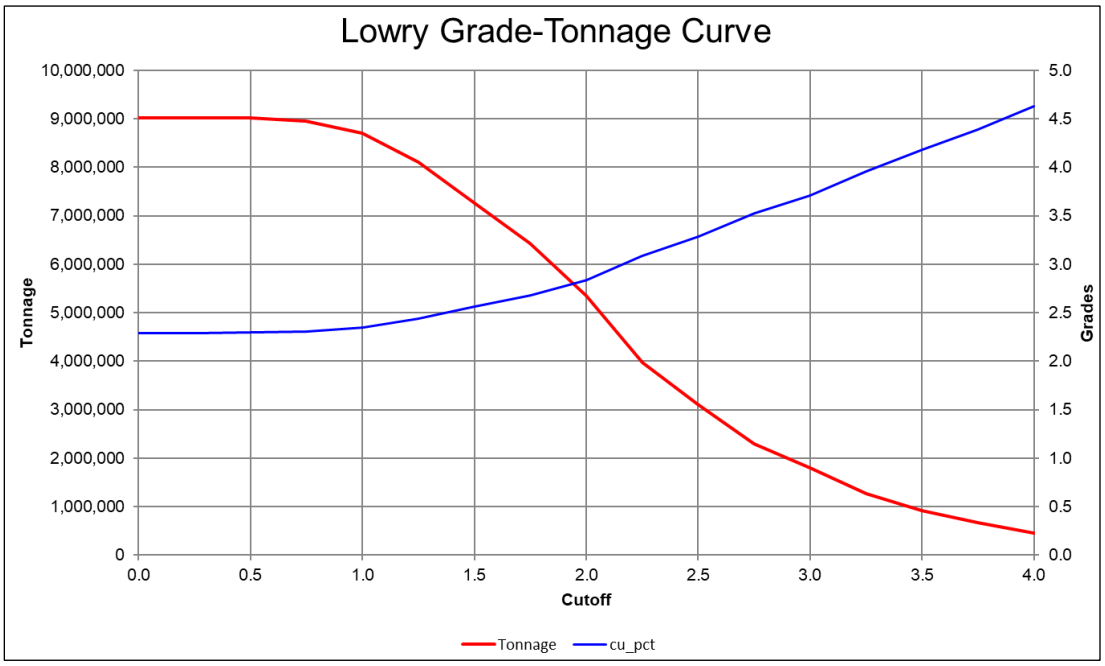


Figure 14.97 Grade-Tonnage Curve for Inferred Mineral Resources at the Lowry deposit

Cut-off	cu_pct	Tonnage	Increment	Increment	Inc Grade	Inc Tonnage
0.5	2.30	9,017,614	0.5	0.75	0.67	59156
0.75	2.31	8,958,458	0.75	1	0.90	249515
1	2.35	8,708,943	1	1.25	1.15	604577
1.25	2.44	8,104,366	1.25	1.5	1.37	837993
1.5	2.56	7,266,373	1.5	1.75	1.63	827930
1.75	2.68	6,438,443	1.75	2	1.88	1079210
2	2.84	5,359,232	2	2.25	2.12	1380083
2.25	3.09	3,979,150	2.25	2.5	2.36	865160
2.5	3.29	3,113,990	2.5	2.75	2.62	816074
2.75	3.53	2,297,915	2.75	3	2.87	505766
3	3.71	1,792,150	3	3.25	3.12	526143
3.25	3.96	1,266,006	3.25	3.5	3.37	349693
3.5	4.18	916,314	3.5	3.75	3.62	250177
3.75	4.39	666,137	3.75	4	3.86	210985
4	4.63	455,152	4	-	4.63	455152

Table 14.32 Lowry deposit Mineral Resource Tabulated Grade-Tonnage Data by Cut-off Grade

14.13 Comparison to Previous Mineral Resource Estimates

The previous Mineral Resource estimate for the Johnny Lee deposit was completed in 2019 as reported in Ronald and Malhotra (2019). That Mineral Resource estimate remains unchanged with an effective date of October 15, 2019.

The previous Mineral Resource estimate for the Lowry deposit was completed in 2013 (effective date July 12, 2013) as part of the SRA PEA (Winckers et al., 2013). The 2013 Mineral Resource estimate is provided in Table 14.33 and included preliminary assumptions on recovery along with assumed economic viability of multiple elements including Co, Ag, and Au. Continued work since the release of the PEA Report has shown the current economic viability of polymetallic products other than Cu has shown poor results in current flotation testing. Therefore, for the 2020 Mineral Resources at the Lowry deposit, Co, Ag, and Au have been excluded.

The 2020 updated classification for Lowry deposit Mineral Resources is aligned with the 2019 Johnny Lee deposit classification. This has resulted in a change from the 2013 Lowry deposit Mineral Resources which reported a combination of Indicated and Inferred Mineral Resources at the time. Updated 3-D wireframing of the major mineralized zones, spatial continuity analyses, and a review of estimation criteria has resulted in the updating of Lowry deposit resources to be entirely classified at Inferred Mineral Resources.

The total quantity of Mineral Resources has increased in 2020 from the 2013 statement. This is due to updated mineralized 3-D wireframes and the estimation of SG values compared to assignment of mean SG data in 2013. The increase in volume and change in local SG values result in the increase in tonnages at the Lowry deposit.

The average Cu grade has decreased in the 2020 Mineral Resources compared to the 2013 statement. This is due to changes in the composite size from 1 m in 2013 to 1.5 m in 2020, use of OK estimation method in 2020 compared to IDW3, improved search neighborhood incorporating multiple samples and search ellipsoid aligned with the dominant directions of mineralization, reduced CoG of 1.2% Cu from 1.6% Cu in 2013, and modified domains constraining estimation to zones of approximately greater than 1.2% Cu.

LMCZ Category	2013		2020		Percent Difference	
	Quantity (Mt)	Cu (%)	Quantity (Mt)	Cu (%)	Quantity (Mt)	Cu (%)
Indicated	4.1	2.94	n/a	n/a	-100%	-100%
Inferred	0.8	2.58	5.7	2.53	614%	-2%
LLCZ Category	**2013**		**2020**		**Percent Difference**	
	Quantity (Mt)	Cu (%)	Quantity (Mt)	Cu (%)	Quantity (Mt)	Cu (%)
Inferred	n/a	n/a	2.6	2.13	100%	100%

Note: *From the 2013 to the 2020 Mineral Resources, changes include:*

1) change in classification, and
2) addition of Mineral Resources from the LLCZ.

Table 14.33 Comparison of Historic 2013 and 2020 Mineral Resource Summary for the Lowry deposit

14.14 Relevant Factors

For the Johnny Lee deposit Mineral Resource the QP is not aware of any environmental, title, permitting, marketing, political, or other factors that could affect the Mineral Resources for the Black Butte Copper Project. The Black Butte Copper Project is subject to legal challenges including the mine operating permit and objections to the Company leasing of mitigation water rights as discussed in the June 2020 Management Discussion and Analyses (MD&A) Company Quarterly Report that will be addressed in State court in 2021. Mineralized material within the UCZ located beneath the McGuire Parcel (Figure 4.3) has been excluded from the calculation of Johnny Lee Mineral Resources as this land is not currently under control of SRA. The QP recognizes that unforeseen factors resulting in changes to the current assumptions on price, costs or other factors effecting the cut-off grade, may materially affect the Mineral Resources.

For the Lowry deposit Mineral Resource the QP is aware that there may be potential environmental, permitting and other risks outlined in the June 2020 MD&A Quarterly Report that could materially affect the potential development of the Mineral Resources. In addition, the Lowry deposit Mineral Resource are not included in the 2020 Mine Operating Permit.

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15. MINERAL RESERVES ESTIMATES

15.1 Introduction

A Mineral Reserve is the economically mineable part of a Measured and/or Indicated Mineral Resource demonstrated by at least a PFS. This FS includes adequate information on mining, processing, metallurgical, economic, and other relevant factors that demonstrate, at the time of reporting, that economic extraction is justified.

Mineral Reserves are those parts of Mineral Resources, which, after the application of all mining factors, result in an estimated tonnage and grade that is the basis of an economically viable project. Mineral Reserves are inclusive of diluting material that will be mined in conjunction with the economic mineralized rock and delivered to the treatment plant or equivalent facility. The term “Mineral Reserve” need not necessarily signify that extraction facilities are in place or operative or that all governmental approvals have been received. It does signify that there are reasonable expectations of such approvals.

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

The reserve classifications used in this Study conform to the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) classification of NI 43-101 resource and reserve definitions and Companion Policy 43- 101CP. These are listed below.

A “Proven Mineral Reserve” is the economically mineable part of a Measured Mineral Resource demonstrated by at least a PFS. This Study must include adequate information on mining, processing, metallurgical, economic, and other relevant factors that demonstrate, at the time of reporting, that economic extraction is justified. Application of the Proven Mineral Reserve category implies that the QP has the highest degree of confidence in the estimate with the consequent expectation in the minds of the readers of the Study. The term should be restricted to that part of the deposit where production planning is taking place and for which any variation in the estimate would not significantly affect potential economic viability.

A “Probable Mineral Reserve” is the economically mineable part of an Indicated Mineral Resource, and in some circumstances a Measured Mineral Resource, demonstrated by at least a PFS. The Study must include adequate information on mining, processing, metallurgical, economic, and other relevant factors that demonstrate, at the time of reporting, that economic extraction can be justified.

15.2 Preparation of Block Model for Mine Design

A block model was provided by SRA in Datamine format. The block model contained the following fields of note.

- CU-Estimated Cu Grade in %;
- T_REC-Estimated Metallurgical Recovery of Cu to the concentrate;
- CU_R-Recoverable Cu Grade (CU*T_REC/100);
- TENNAN-Estimated As grade in concentrate (in ppm).

The NSR was determined using the inputs in Table 15.1 below. All blocks that had been classified either Measured or Indicated were coded with an NSR value while all blocks classified as Inferred were given a value of 0. The field that was created and coded for the NSR values was labelled NSR_MI. The CU and CU_R values for blocks classified as Inferred were also coded to 0.

Item	Units	Value
Cu Price	$/lb	3.10
Concentrate Grade	% Cu	24
Moisture Content	% Moisture	9
Cu Min Deduction	% Cu	1
Treatment Charge	$/dmt Concentrate	90
Refining Charge	$/lb Cu Payable	0.09
Freight	$/wmt Concentrate	177
As Penalty Threshold	% As	0.5
As Penalty Above Threshold	$/dmt per 0.1%> Threshold	3
Royalties	% NSR	2

Table 15.1 NSR Inputs for Reserve Calculations

The above parameters were used to calculate the NSR per tonne for each block within the block model. The calculation can be simplified to the following formula:

$$NSR = 0.98 * \left(51.805 * (CU_R) - \max(0, \frac{3 * TENNAN}{1000} - 15) * \frac{CU_R}{24}\right)$$

15.3 Cut-off Value

Mineable stopes were defined based on Cut-Off Values (COV) greater than $70/t after dilution and mining recoveries are applied. The parameters used for the calculation of the COV are shown in Table 15.2 and the parameters used in the calculation of the NSR are shown in Table 15.1. The NSR parameters, as well as operating cost parameters used in this section, are not the final NSR or the operating cost estimates used in the economic analysis, but rather preliminary numbers to identify the mineable reserves. For further details on the actual NSR calculation and the operating cost estimate see Sections 19, 21 and 22.

These initial parameters were used to define potentially mineable shapes. Following this, all shapes were reviewed with all incremental costs applied to them to determine if they were economic.

A marginal COV was used for material that was mined while accessing economically mineable stopes. This material would be hauled to the surface and otherwise be treated as waste. This lower COV of $38/t is the cost of processing ore as well as the general and administration costs for material that would otherwise be sent as waste.

Operating Cost Area	Cost ($/t)
Mining	32
Processing	31
General and Administrative	7
Total	**70**

Note: *Assumptions in this table are for Mineral Reserve estimate only*

Table 15.2 Inputs for Cut-off Value Calculation

15.4 Mining Shapes

Preliminary mining shapes were created using Datamine Mineable Shape Optimiser Package with a COV of $70/t. These shapes were used as an initial guide to identify areas above a minimum thickness of 4 m and the COV of $70/t. With these guides, shapes were laid out according to the mining method selected. These shapes were manually adjusted to ensure maximizing extraction of above $70/t material while at the same time minimizing the amount of below $70/t material in the shapes as well as ensuring mineability. The shales were also assessed for mineability.

A final check was undertaken on each drift and area to ensure that it pays for itself as well as any associated access development infrastructure.

15.5 Dilution and Recovery Estimates

Two types of dilution were applied to the stope and development designs:

- External dilution – Additional material (overbreak) that is mined outside of the mineralized zone; this includes backfill dilution from adjacent stopes;
- Internal dilution – Additional lower than COV material that is within the planned stope or development design shape and cannot be mined separately.

15.5.1 External Dilution

Dilution for the stopes will primarily come from overbreak in the walls and backs. The dilution estimate is applied to the walls and the backs of the stopes individually and summed for each stope. For the walls or back of the stope that have a future stope immediately adjacent, no dilution has been applied to these faces. This is because the subsequent drifts will be "field-fit" in place, to not create metal and tonnes by double-counting material.

For faces that do not have a future stope immediately adjacent, a dilution skin of 0.3 m has been applied. For faces that have a previously mined drift that has been paste-filled immediately adjacent, a 0.25 m dilution skin has been incorporated. Both of these values are consistent with similar operations. The paste backfill dilution has a grade of 0% Cu applied to it. The rock dilution in the UCZ has a grade of 0.25% Cu and a metallurgical recovery of 76% applied to it. The rock dilution in the LCZ has a grade of 0.25% Cu and a metallurgical recovery of 94% applied to it. These numbers are based on the grades and recoveries immediately adjacent to the mined areas in the block model.

15.5.2 Internal Dilution

Additional sources of dilution include sub-economic material that is within the minimum mineable shape. This also includes material classified as Inferred Resources for which a 0 value was applied.

15.5.3 Mining Recovery

Mining recovery is a function of any mineralized material left behind in the mining process. This will primarily be material on the floor of the stopes that is not mucked. A 97.5% mining recovery has been applied to all stopes in this Study which is consistent with operations that employ similar development mining methods.

15.6 Risk to Reserve

The permits for the Black Butte Copper Project are substantially in place having received the Final Mine Operating Permit from the MDEQ. This decision is currently subject to litigation.

The FS supporting this Mineral Reserve complies with the Mine Operating Permit and only mines the areas that are currently permitted (the Johnny Lee deposit).

The largest risk to the Mineral Reserve is the metallurgical recovery in the UCZ which is quite variable. A significant amount of test work has been undertaken and that is accounted for in the Mineral Reserve estimate.

15.7 Mineral Reserves

The mining stope and level designs with dilution and mining recovery factors applied determined the Mineral Reserve estimate shown in Table 15.3.

Class	Diluted Tonnes	Cu Grade	Contained Cu Metal (t)
Proven	1,998,000	3.0%	60,700
Probable	6,804,000	2.4%	165,400
Total	**8,802,000**	**2.6%**	**226,100**

Notes:

1. *The Qualified Person for the Mineral Reserve estimate is Brad Evans MAusIMM CP(Mining).*
2. *Effective date: October 19, 2020. All Mineral Reserves have been estimated in accordance with Canadian Institute of Mining and Metallurgy and Petroleum (CIM) definitions, as required under NI 43-101.*
3. *Mineral Reserves were estimated using a $3.10 /lb Cu price and a NSR cut-off value of $70/t.*
4. *Tonnages are rounded to the nearest 1,000 t, metal grades are rounded to one decimal place. All units are metric.*
5. *Rounding as required by reporting guidelines may result in summation differences.*
6. *Average Metallurgical Recovery is 84%*

Table 15.3 Mineral Reserve Estimate Johnny Lee deposit

The Mineral Reserves identified in Table 15.3 comply with CIM definitions and standards for a NI 43-101 Technical Report. Detailed information on mining, processing, metallurgical, and other relevant factors are contained in the following sections of this Study and demonstrate, at the time of this Study, that economic extraction is justified. This Study did not identify any mining, metallurgical, infrastructure or other relevant factors that may materially affect the estimates of the Mineral Reserves or potential production.

The conversion of Mineral Resource to Mineral Reserves is high in the LCZ due to the narrow, discrete, high-grade, and high-recovery nature of the orebody. Due to these factors, in the LCZ, a higher amount of internal dilution was mined which included some amount of Cu that was in blocks below the resource cut-off grade. Each mining area was broken up into 4 m long segments, if greater than 50% of the ore tonnes were classified as Measured than the segment was classified as Proven, if greater than 50% of the ore tonnes were classified as Indicated, the segment was classified as Probable.

Table 15.4 below shows the Mineral Reserves broken out by zone.

Zone	Class	Diluted Tonnes	Cu Grade	Contained Cu Metal (t)
UCZ	Proven	1,159,000	2.2%	25,900
	Probable	5,693,000	2.1%	116,900
	Total	6,852,000	2.1%	142,800
LCZ	Proven	839,000	4.1%	34,800
	Probable	1,111,000	4.4%	48,500
	Total	1,950,000	4.3%	83,300
Grand Total	**Total**	**8,802,000**	**2.6%**	**226,100**

Table 15.4 Mineral Reserves by Zone

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16. MINING METHODS

16.1 Introduction

The Black Butte Copper Project contains four zones, two of which, the Johnny Lee Upper Copper Zone (UCZ) and the Johnny Lee Lower Copper Zone (LCZ) are scheduled to be mined as part of this FS. Both zones are characterized as being high-grade, flat-lying and with relatively narrow widths. All deposits have anomalous Ag and Co mineralization, however Cu is the only economic product considered in this Study.

16.2 Geotechnical Analysis and Recommendations

16.2.1 Geotechnical Data Gathering and Previous Work

Geotechnical data was gathered from logging of the diamond drill core performed by SRA geologists as well as part of previous work by MDEng (Kalenchuk et al, 2015). Specific geotechnical holes were drilled along the projected Main Decline and one of the ventilation raises and logged by Mining Plus to support the FS. Mining Plus in collaboration with SRA geologists undertook a QA/QC audit of the data gathered that included training SRA geologists. Acoustic Televiewer and oriented core data were used to determine structural information. In addition to the data logging, multiple rock property tests were performed on different rock types including, Uniaxial Compressive Strength (UCS) testing, Triaxial Strength testing, Cerchar Abrasiveness testing and Slake Durability testing (Agapito Associates Inc., 2019; Nazeri, H., 2019).

16.2.2 Rock Types

For UG development, there are seven rock types that are to be expected within the mining horizons. Detailed description of the rock units was provided by MDEng (Kalenchuk et al, 2015) and summarized below. Figure 16.1 shows the cross-section of the rock units and outlines the mineralized zone.

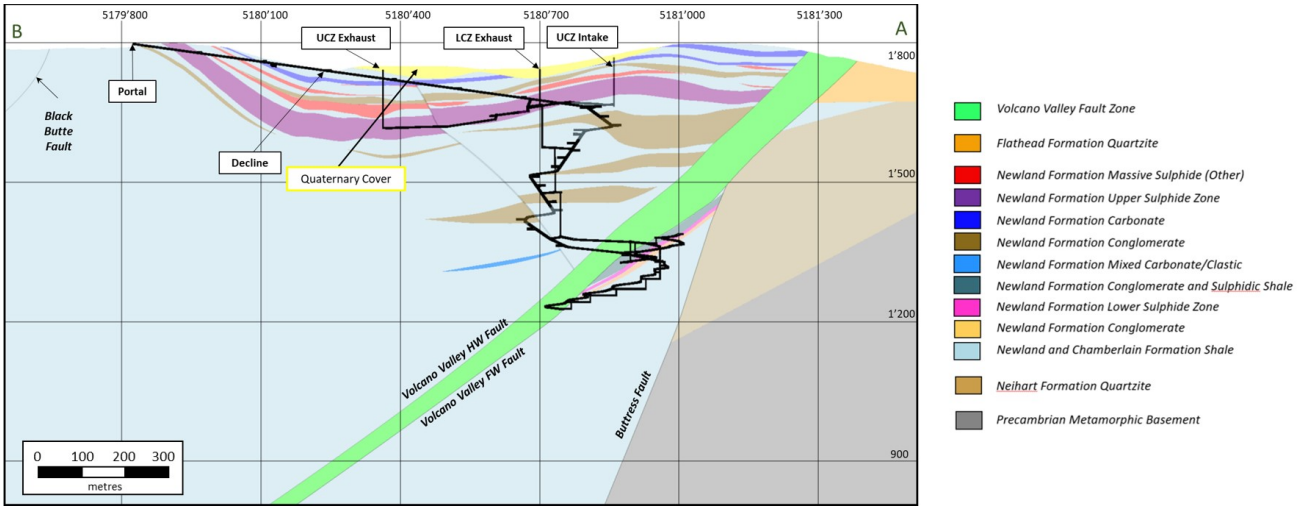

Figure 16.1 Geological Section along 506600 mE (looking west)

- Chamberlain Shale (Yc): Poor quality rock mass, overlies the Newland Unit. Silty black shale with planar, smooth, closely spaced graphitic coated discontinuity surfaces in proximity to Buttress Fault; away from fault discontinuity condition is smooth and altered with low friction materials. Occasional andesite intrusions within the unit. Rock type will form the majority of the decline;
- Newland Formation (Ynl): Poor to fair quality rock mass upper unit lies above the VVF. Consists of thinly bedded silty dolomites and shales with occasional clayey seams and conglomerate zones. Closely spaced bedding planes dip gently to the SW. Steep fractures dip to the west and north. Highly damaged ground in proximity to the VVFZ, (folding shears and calcite veins);
- Lower Newland Unit (Ynl L): Poor to fair quality rock mass. Located below the VVF is a massive silicified conglomerate with shale. The unit contains quartz veins, occasional gravel and clay seam, cross-cut by mafic dykes. Weathered and damaged zones are present;
- USZ: Poor to fair quality rock mass, consisting of thickly to thinly bedded massive sulphides and barite, mixed with siliceous black shale and dolomite. Shearing parallel to bedding common in shale zones. Increasing strength within shale zones with depth;
- LSZ: Fair to good quality rock mass consisting of shale, siltstone, dolostone and conglomerate – sulphides present as clasts, within matrix or in veining. Bound by silicified conglomerates commonly. Sheared/damaged in upper part of unit and has a transitional contact with the VVF;
- VVFZ: Very poor quality rock mass sheared black shale and gouge with blocks of heavily veined dolostone and shale. Graphite and pyrite coated fracture surfaces common, brecciated zone often cohesive or re-cemented. Typically discrete HW contact. FW is transitional with smaller gouge-filled splays extending below the main fault zones. A wedge of poor ground between the fault and the HW of the LSZ will potentially cause ground control issues;
- Neihart Quartzite (Yne): Mixed quartzite and mudstone unit. Clay like minerals observed coating on joint surfaces. Poor geotechnical drill coverage, however, geology drilling within the unit reports strong artesian pressures. Preliminary analysis found that the modified Rock Quality Index (Q') is expected in the order of 10 (Kalenchuk et al, 2015). The mine plan avoids exposure of the Neihart Quartzite;
- As part of the 2015 rock mass characterization program, the Q' classification system (Barton et al, 1974) was used. Table 16.1 below represents the results for the primary mining zones. Mining Plus' further work has confirmed these values to be reasonable.

Rock Unit	Mean Q'	30th Percentile Q'
LSZ	3.4-6.8	2.0-4.0
USZ	3.9-5.2	3.2-4.3
VVFZ	0.7-0.8	0.5-0.6
Yc	2.7	1.6
Ynl Upper	5.7-7.6	3.6-4.8
Ynl Lower	3.7-4.9	2.2-2.9

Table 16.1 NGI Q' Rock Mass Ratings for Mining Zones (Kalenchuk et al, 2015)

16.2.3 Portal Support

It is expected that a weathered horizon will be encountered during the excavation of the portal. The portal will require 10 m of sound supported rock above the excavation.

Minimal ground support guidelines for rock slopes cut in excess of 5 m are to be installed as follows:

- 3.0 m long, #8 (25 mm) diameter Williams style (threaded) bolts, fully encapsulated resin (600 mm fast set resin, 2,400 mm slow set) installed on a 1.8 m x 1.8 m pattern at a plunge of 18 degrees from horizontal;
- 6.5 m long twin strand bulbed cable bolts around the perimeter of the portal decline drilled perpendicular to the face to acts as spilling cables;
- Welded wire mesh should be nominally 4" x 4" squares (S) of #6 gauge or equivalent. Overlap should be 300 mm (12" or 3 rows) either and the overlap should be secured through ground support (no pinning with shorter support);
- Where rock mass with a NGI Q' Rating is determined to be 1.0 or less, it is to be sprayed with 60 mm of fibrecrete. Adequate drainage through the shotcrete is to be provided to avoid ice-jacking or high-water pressure developing behind the shotcrete.

16.2.4 Decline Support

The ground support recommendations are made based on a rock mass assessment of geotechnical drilling, laboratory testing and empirical and analytical design. The decline support as presented is based on the upper sedimentary unit that was drilled as part of the 2018/19 geotechnical campaign and addresses support through the decline up to the intersection with the VVFZ.

The bedded sedimentary units with low estimated shear strength will be problematic for development and require immediate support to ensure that the roof beam remains intact and stable. Loose rock is to be expected in the immediate back and welded wire mesh is required throughout the decline. Two metre advances are recommended with low density smooth wall blasting techniques employed for all perimeter holes.

Ground support for the initial decline support is as follows:

- Spraying of 75 mm of fibercrete is to be applied immediately after mucking and scaling. Provide 45 mm of shotcrete cover of previous round shotcrete (total 120 mm of shotcrete) tied into fibercrete arches;
- Installation of welded wire mesh and bolts consisting of 3.0 m long combination rock bolts (DSI CT or equivalent) on 1.2 m ring spacing are to be installed – all CT bolts need to be grouted prior to the blasting of the next cut;

- Installation and spraying of fibercrete arches at 3.0 m centers from previous arch;
- Installation of 6.0 m long single-strand cablebolts, midway between shotcrete arches.

Due to the critical nature of the Main Decline being the primary source of intake, paste fill as well as access into the mine, a higher factor of safety is used in the design to control the higher consequences of requiring rehabilitation in the future. Table 16.2 below outlines the minimum ground support applications that will be used for the Main Decline.

Domain	Q' Range	Bolt Spacing (m)	Bolt Length (m)	Surface Support
1	0.04 - 0.2	1.2	2.1	Fibercrete 9 - 12 cm
2	0.2 - 1.2	1.5	2.1	Fibercrete 6 - 9 cm
3	1.2 - 5.6	1.5	2.1	Fibercrete 5 cm
4	5.6 - 2.8	1.2 x 1.2	2.1	Welded Wire Mesh

Table 16.2 Minimum Recommended Ground Support for Decline

Approximately 70% of the development will be within Domain 4 with the remainder in Domain 2 and 3. Domain 1 is expected to only be required when crossing major structures such as the VVFZ.

16.2.5 Ground Support in Upper Copper Zone

The UCZ will be mined using Drift and Fill methods (DF) with paste fill. The sequence will be a primary-secondary-tertiary method with primary drifts mined and tight filled followed by the mining of a secondary drift and then the unsupported slashing of a tertiary drift. See Figure 16.2 and Section 16.3.1 below for further information on the UCZ mining sequence.

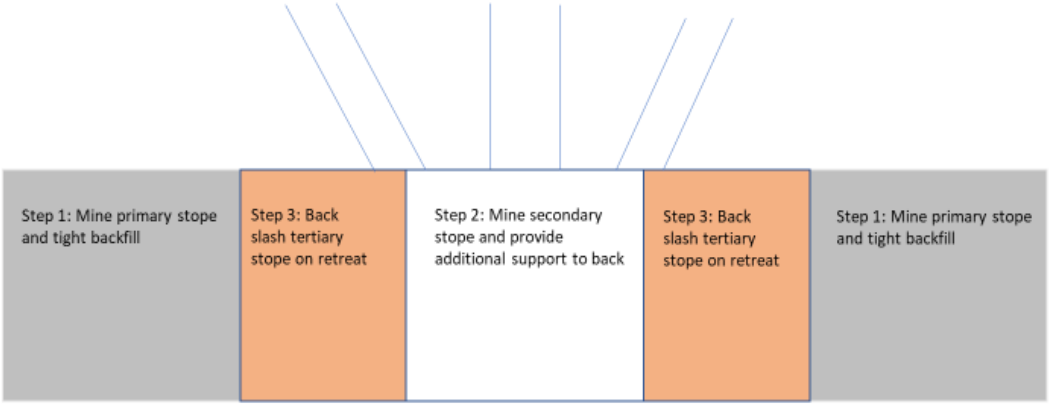

Figure 16.2 UCZ Mining Sequence with Additional Long Support in Secondary Drives

A modified stope stability method (Potvin et al, 1988) approach was taken in analyzing the maximum width of the combined secondary and tertiary supports. A maximum combined secondary and tertiary width of 12.2 m is possible in good quality rock (Q'>2.0) based on the modified stope stability method. For the purposes of this Study a maximum span of 10 m was used. Where weak rock is experienced in the back (Q'<2.0), the tertiary stopes should be eliminated.

Minimum ground support recommendations for the UCZ can be found in Table 16.3 below.

Domain	Q' Rating	Description	Comments	Back Support	Wall Support	Surface Support
U1	0.6 - 20	Permanent	3-2-3 Pattern	1.8 m Resin Rebar 1.2 m x 1.2 m Pattern	1.8 m SS39 Friction Bolt 1.2 m x 1.2 m Pattern	Welded Wire Mesh #6 AWG
U2	<0.6	Permanent	3-2-3 Pattern	1.8 m Resin Rebar 1.2 m x 1.2 m Pattern	1.8 m SS39 Friction Bolt 1.2 m x 1.2 m Pattern	75 mm Shotcrete
U3	All	Up to 9 m Intersection	Apply U1-U2 and Augment	3.6 mL Extendable Swellex 2.0 m x 2.0 m Pattern	-	-
U4	0.6 - 20	Temporary up to 5 m W	3-2-3 Pattern	1.8 m L SS39 Friction Bolt 1.2 m x 1.2 m Pattern	1.8 m SS39 Friction Bolt 1.2 m x 1.2 m Pattern	Welded Wire Mesh #6 AWG
U5	<0.6	Temporary up to 5 m W	Square Pattern	1.8 m L SS39 Friction Bolt 1.2 m x 1.2 m Pattern	1.8 m SS39 Friction Bolt 1.2 m x 1.2 m Pattern	75 mm Shotcrete
U6	0.6 - 20	Temporary up to 8 m W	Apply U4 and Augment	3.6 mL Extendable Swellex 2.0 m x 2.0 m Pattern	1.8 m SS39 Friction Bolt 1.2 m x 1.2 m Pattern	-
U7	<0.6	Temporary up to 8 m W	Apply U5 and Augment	3.6 mL Extendable Swellex 2.0 m x 2.0 m Pattern	1.8 m SS39 Friction Bolt 1.2 m x 1.2 m Pattern	-

Table 16.3 Ground Support Recommendations UCZ

16.2.6 *Ground Support in Lower Copper Zone*

Development within the LCZ is comparable to the UCZ in terms of rock mass quality. However, the ground stresses within the LCZ will be problematic as they will be comparable to the expected rock mass strength. In these areas, ground squeezing will be expected and given the weak shear strength of the discontinuity surfaces, excessive bulking of the ground is likely.

A support that can accommodate the squeezing of the ground will be required in the LCZ. A Jenmar 'Nevada Bolt', or DSI 'D' Bolt is recommended to replace the stiff rebar support. Fibrecrete is to replace shotcrete in squeezing ground such that a yielding surface support is possible. Table 16.4 provides the recommended ground support for squeezing ground.

Domain	Q' Rating	Description	Comments	Back Support	Wall Support	Surface Support
L1	0.6 - 20	Permanent	3-2-3 Pattern	1.8 m D-Bolt (or similar) 1.2 m x 1.2m Pattern	1.8 m SS39 Friction Bolt 1.2 m x 1.2 m Pattern	Welded Wire Mesh #6 AWG
L2	<0.6	Permanent	3-2-3 Pattern	1.8 m D-Bolt (or similar) 1.2 m x 1.2 m Pattern	1.8 m SS39 Friction Bolt 1.2 m x 1.2 m Pattern	75 mm Fibercrete
L3	All	Up to 9 m Intersection	Apply U1-U2 and Augment	3.6 mL Extendable Swellex 2.0 m x 2.0 m Pattern	-	-
L4	0.6 - 20	Temporary up to 5 m W	3-2-3 Pattern	1.8 m L D-Bolt (or similar)1.2 m x 1.2 m Pattern	1.8 m SS39 Friction Bolt 1.2 m x 1.2 m Pattern	Welded Wire Mesh #6 AWG
L5	<0.6	Temporary up to 5 m W	Square Pattern	1.8 m L D-Bolt (or similar)1.2 m x 1.2 m Pattern	1.8 m SS39 Friction Bolt 1.2 m x 1.2 m Pattern	75 mm Fibercrete
L6	0.6 - 20	Temporary up to 8 m W	Apply U4 and Augment	3.6 mL Extendable Swellex 2.0m x 2.0m Pattern	1.8m SS39 Friction Bolt 1.2 m x 1.2 m Pattern	-
L7	<0.6	Temporary up to 8 m W	Apply U5 and Augment	3.6 mL Extendable Swellex 2.0 m x 2.0 m Pattern	1.8 m SS39 Friction Bolt 1.2 m x 1.2 m Pattern	-

Table 16.4 Minimum Ground Support for LCZ

16.2.7 Ground Support for Volcano Valley Fault Zone

The VVFZ is a zone of extremely poor to very poor ground conditions. Development within the fault should be kept to a minimum and traverse through the zone should be performed through as direct a path as possible.

Difficult development conditions requiring a delay in the mine schedule are to be expected. Short round development and fore-poling (3.6 m long, self-drilling bolts on a 1.0 m spacing) are recommended for advance with low density smooth wall blasting techniques employed for all perimeter holes. Support needs to be installed immediately after mucking, commencing with 75 mm of shotcrete. The use of wide spans and intersections are to be avoided within the VVFZ and the areas below the HW.

Ground support in line with Domain 1 in Table 16.2 is to be employed within the fault and within two rounds prior and past the known boundaries of the fault. Ore development should not be performed within 6 m of the VVFZ without detailed study of the ground based on operational understanding.

16.2.8 Ground Support for Vertical Development

Three ventilation raises will be excavated from surface as well as raises for second means of egress. The first leg of all three ventilation raises will be 90 m. One raise will then be extended further UG. The maximum length of any second means of egress will be 90 m.

All the ventilation raises will be 3.5 m diameter. Based on the current geotechnical investigation they will require full support and conventional sinking or a combination of conventional sinking and small diameter raiseboring. Ground support will primarily consist of 2.4 m rebar on a 1.8 m x 1.8 m pattern with shotcrete and mesh as required.

For ventilation legs below the first 90 m leg the ground conditions improve and are able to be excavated using raisebore with 75 mm shotcrete. Further detailed design and geotechnical drilling will be required prior to excavating these raises.

All the second means of egress raises will be excavated using a small diameter raisebore and then lined with a raisebore tube.

16.3 Drift and Fill Stoping with Paste Fill

The mining methods used will be a mix of DF and Cut and Fill (CF) mining methods. Both DF and CF are mechanized mining methods where a development drift is driven and paste filled and then another drift is extracted adjacent to the filled drift. In DF the drifts are all on the same elevation, while in CF the drifts are extracted in “lifts” with the lift being extracted and paste filled then another lift being extracted on top (overhand) of the previously mined paste fill.

16.3.1 *Upper Copper Zone Mining Methods*

The UCZ is a flat dipping deposit with multiple parallel lenses. It will be entirely extracted using DF mining methods.

The DF panels will be setup with two parallel access drifts spaced 60 - 90 m apart where possible. Primary ore drifts spaced 15 m apart will be developed from one access drift to within 5 m of the second access drift. Once the drift has completed mining, all the services will be stripped from the drift and a barricade will be built at the access. Paste and breather holes will then be drilled from the second access drift through the pillar and paste fill will be placed in the drift.

Once two adjacent primary drifts have cured, a secondary ore drift will be driven to the end of the panel in the middle of the two cured drifts. This drift will be 5 m wide with two 2.5 m wide pillars on either side. These pillars will then be extracted by drilling unsupported slashes up against the paste and mucking out on remote control. Once the secondary drift has been mined out, a barricade will be built, and the secondary drift will be filled using the same process as the primary drifts.

Once an entire panel has been mined out and both access drifts are paste filled, the pillar is mined out and then paste filled. Figure 16.3 below demonstrates the layout of an UCZ DF panel.

In areas where there is no second access to fill from, the same general sequence will be followed. However, the primary drifts will be located 10 m apart and the secondary drifts will be driven at a 5 m width, fully extracting the pillars between the primary drifts with no unsupported slashing. This is to allow the proper tight filling of the secondaries.

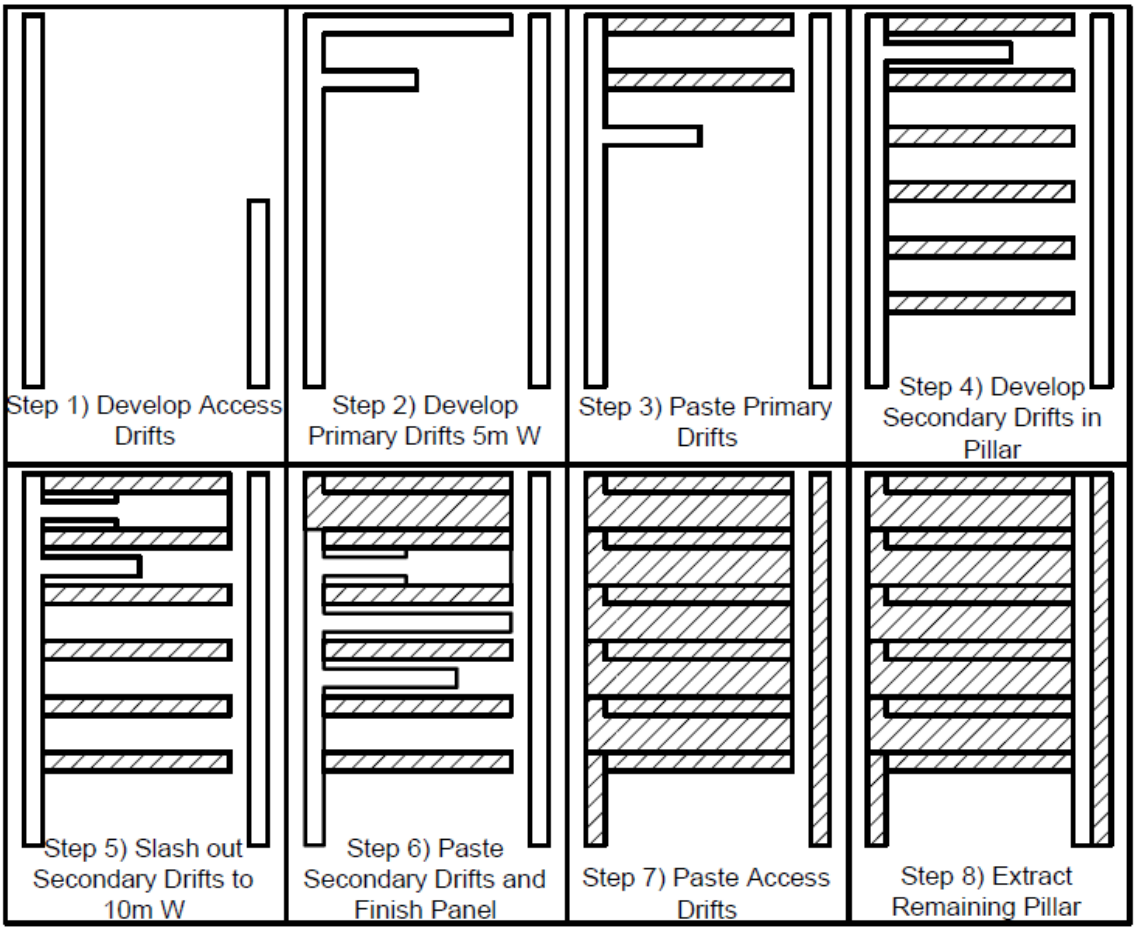

Figure 16.3 UCZ Mining Sequence

16.3.2 Lower Copper Zone Mining Method

The CF mining method has been selected for the LCZ mining area. Due to orebody geometry dipping slightly steeper than the UCZ and no parallel lenses, multiple lifts will be taken from a single attack access. The attack access will be driven from the LCZ access. In this method, extraction will commence from the lowest lift and progress up vertically. In each lift, primary CF production drifts will be extracted and then filled with paste. After the primary CF production drifts have been backfilled, mining of secondary drifts will commence. The secondary production drifts and stopes will also be backfilled using paste fill.

Once all the secondary drifts on a lift have been extracted the access will be backslashed and the lift above will be extracted. The sequence will be strictly bottom-up except for a few locations where sill pillars have been selected to increase production from the LCZ. Figure 16.4 and Figure 16.5 below show a typical plan and section view respectively of the LCZ mining.

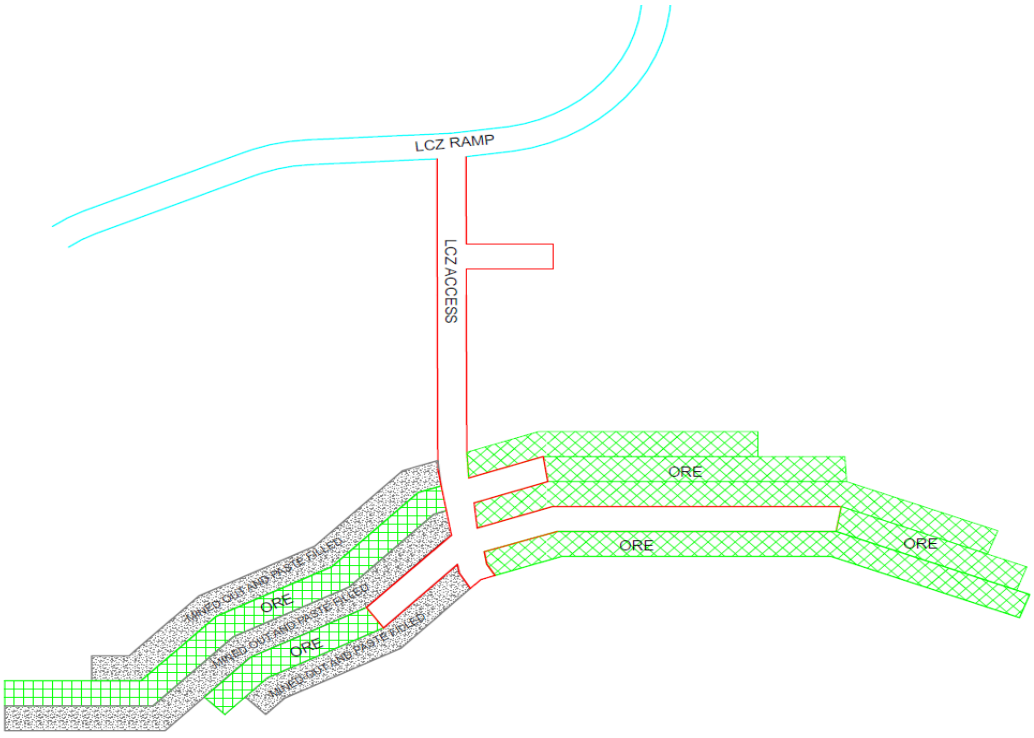

Figure 16.4 Plan View LCZ Ore Extraction

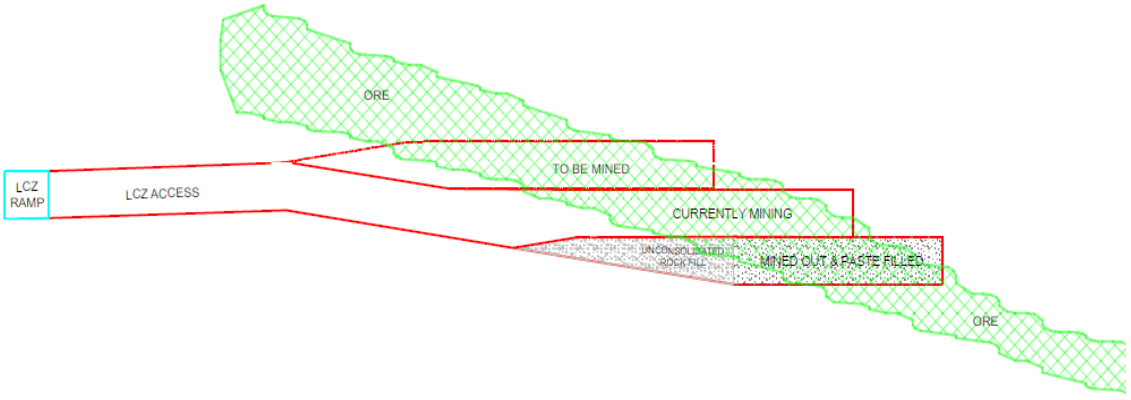

Figure 16.5 Section View LCZ Mining Method

16.4 Mine Design

The Johnny Lee deposit will be accessed by a single Main Decline driven from surface. The decline dimensions will be 5 m wide by 5 m high and excavated with a flat back to maximize the stability of the flat dipping joint sets that are prevalent throughout the Project.

The decline will be excavated at a maximum gradient of -15% from the surface and pass to the east of the UCZ. A single access drift will be driven from the decline into the UCZ. From this access drift an initial ventilation connection and second means of egress to surface will be established. This access drift will serve as a take-off point for access drifts for mining the UCZ panels.

The decline will continue to spiral below the UCZ connecting to the return air and secondary egress system which will be advanced in raises until the access is in the HW of the VVFZ. Due to poor expected ground conditions in this area, rather than driving raises, two parallel drifts, one acting as the Main Decline and the second acting as the return airway and secondary egress, will be driven through the VVFZ to access the LCZ.

The LCZ zone is made up of two pods both of which will be accessed from a decline driven in the FW of the pods. Attack drifts will be driven off the decline at regular intervals to access the ore.

All vertical development will be performed by a contractor. Development of the Main Decline as well as the UG infrastructure will also be performed by a contractor while the ore production will primarily be performed by the owner. Figure 16.6 is an isometric view of the UG design looking from the SE.

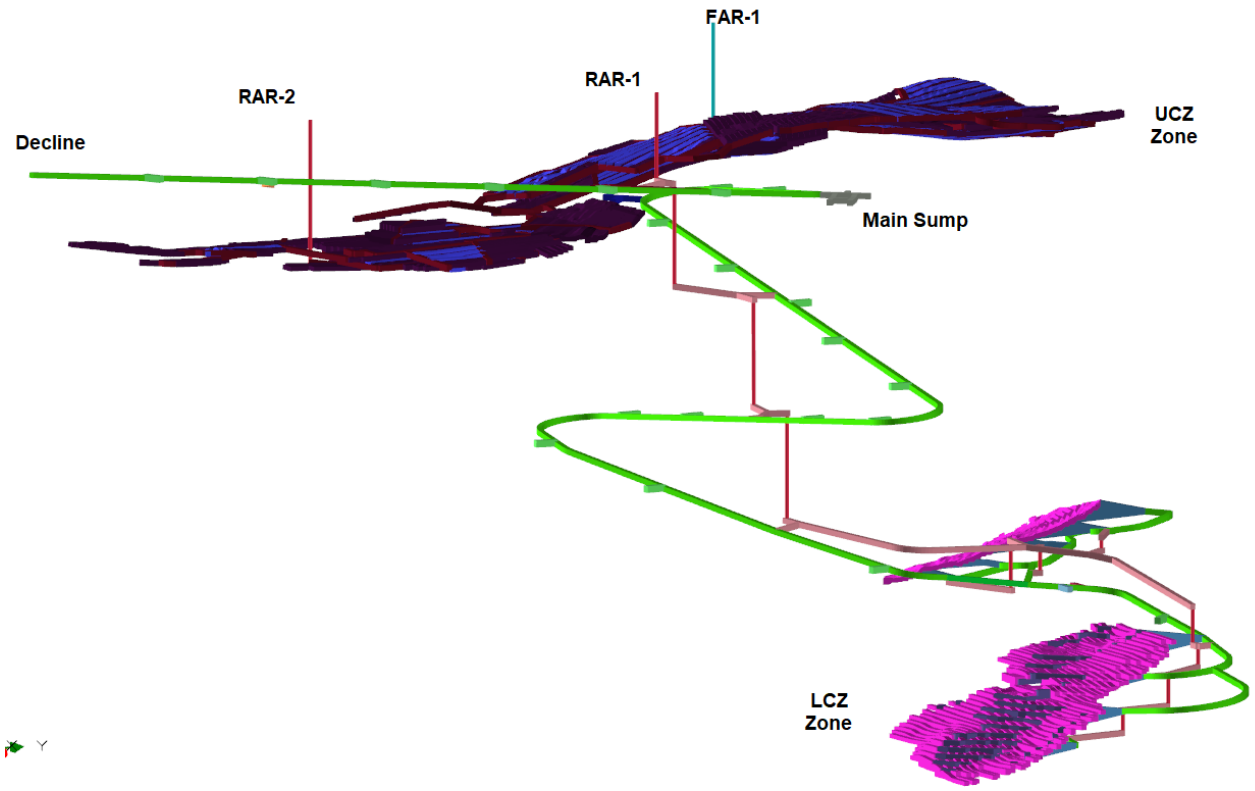

Figure 16.6 Isometric View UG Design from the SE

16.5 Paste Backfill

All UG production openings will be backfilled with cemented paste backfill. Paste backfill will be required to provide a working floor for some areas as well as to support the back and allow for complete extraction of the ore with no pillars remaining in-situ.

16.5.1 Required Strength

The modified Mitchell equation was used to determine the required fill strength of the paste backfill. The required loads are a function of stope height and range from 400 - 650 kPa with a factor of safety of 1.5. As per the MOP the binder content for all paste UG will be a constant 4% binder. Testing by Paterson & Cooke (Bachman et al, 2020, Appendix H) demonstrates that this strength is achievable at this binder percentage with the tailings from the mill circuit within seven days.

16.5.2 Paste Fill Reticulation

Paste fill will be mixed in the processing plant and distributed to the portal and throughout the UG by pumping through a piped distribution system. Distribution to the UG will be via a NB150 SCH80 Steel Pipe mounted in the back of the Main Decline to the UCZ access. From there a borehole will distribute the paste to the LCZ FW decline where it will be piped to the final distribution. A second line will be installed into the UCZ. As the paste line is located in the primary access and any blockage of it will cause a significant production delay, automated flow diverter valves will be installed along with extensive monitoring to ensure that any blockages are addressed as quickly as possible to reduce the risk of a complete blockage.

16.5.3 Fill Wall

The fill walls will be constructed using prefabricated arch wall kits. A prefabricated adjustable steel structure will be installed and then steel mesh and hessian fabric hung from it. This will be sprayed with shotcrete and allowed to cure to strengthen the wall. Figure 16.7 below shows a similar style paste wall under construction prior to the application of shotcrete.

Figure 16.7 Paste Wall prior to Shotcrete Application

16.6 Materials Handling

All material will be mucked using 17 tonne Load Haul Dumps (LHD's) and hauled to surface using 51 tonne trucks. Waste will be stockpiled on the waste pad and used to construct the CTF. Ore will be stockpiled on the surface ROM pad and fed to the surface crusher using a Front-End Loader (FEL) or direct tipped into the surface crusher as required.

16.7 Ventilation

16.7.1 Primary Ventilation

Two negative pressure ventilation circuits will ventilate the mine: one for the UCZ and one for the LCZ. Primary fans will be located UG at both dedicated exhaust raises which will cause air to be drawn into the main portal and intake raise, then along the respective access declines to working levels. For the UCZ, a series of airdoors and temporary regulators will redirect the air between the intake and exhaust to ensure all work areas are connected to a primary source of airflow. The LCZ is divided into a series of two smaller ventilation loops regulated by louvres at return raises. All main circuit fans will be equipped with variable speed drives to allow for the proper balancing of airflows between areas as required.

The ventilation quantity was determined using 0.06 m³/s per kW of engine horsepower. Figure 16.8 shows an isometric view of the ventilation circuit. Peak ventilation flows will be 270 m³/s and will occur two years following the beginning of UG development. All mine air will be heated by propane mine air heaters located at the intakes during the winter months to ensure the safety and comfort of all workers.

Initial ventilation to the decline prior to breakthrough of RAR-1 will be via two 110 kW fans and ventilation tubing. Following the breakthrough of RAR-1 and installation of the primary ventilation fans, the tubing and fans located at the portal will be removed and the air will be drawn down the portal by the UG primary fans. This ventilation circuit will be extended as the decline and RAR-1 deepen during mine development, and ventilate the LCZ upon completion of capital development. Prior to production mining in the UCZ ramping up, RAR-2 and FAR-1 will be developed creating a second independent ventilation circuit to feed the UCZ.

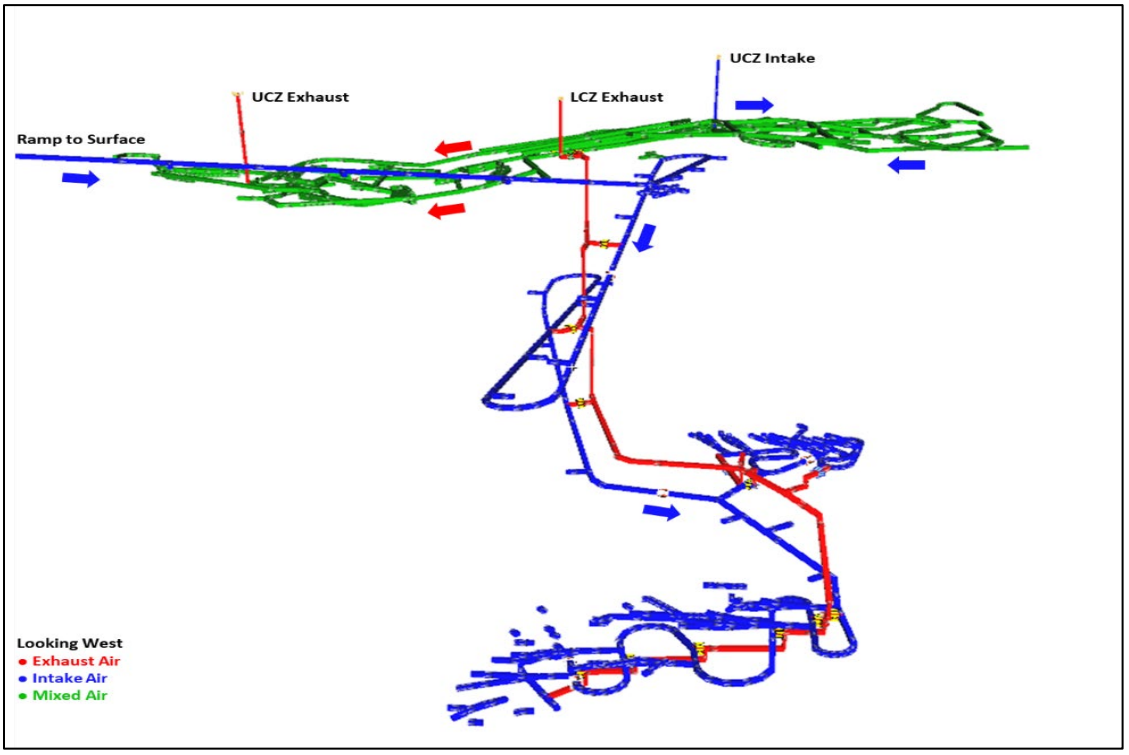

Figure 16.8 Ventilation Schematic

16.7.2 Secondary Ventilation

Each production area will be provided with a split of air from the primary ventilation circuit, sufficient for the operation of an LHD. Air returning from the face will re-join the primary air circuit.

Airflow to ore/production drive ends will be provided via auxiliary ventilation systems, typically employing 1.067 m diameter axial fans and parallel 1.067 m diameter duct. Due to the large number of headings on each level, care will be required to ensure proper ventilation control.

16.8 Sequence and Schedule

Initial mine development will be carried out by a mining contractor. The mining contractor will excavate and support the portal and commence development of the Main Decline. Simultaneously they will commence sinking RAR-1 so that the first leg of RAR-1 is complete upon the decline reaching the breakthrough point.

Once the initial ventilation circuit is complete, the Main Decline will remain the focus of the lateral development until 18 months after decline development begins, at which point lateral development will resume in the UCZ, to establish FAR-1 and RAR-2 as well as to begin stockpiling ore for the mill. This strategy will allow the higher grade LCZ to contribute ore feed as early as possible to the mill.

Once the LCZ has been accessed the owner will shift a portion of their fleet to the LCZ and commence stoping in the LCZ. Stoping will continue in both areas until they are depleted. The overall mine life from commencing development of the decline is just over nine years. Production mining is currently at eight years.

The production mix is initially solely UCZ ore while the decline is being developed and the LCZ is being established. Thereafter it moves to 25-30% LCZ ore with the remainder made up by UCZ ore until the end of mine life at which point the LCZ has been depleted.

16.9 Mine Physicals Summary

The year by year lateral and vertical development, ore and waste tonnage, mined head grade and paste backfill volume is given below in Table 16.5.

	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10
Lateral Development (km)	2.3	4.9	17.1	14.6	15.5	15.5	12.5	11.1	12.0	2.2
Vertical Development (m)	80	795	309	0	0	0	0	0	0	0
Waste Tonnes (kt)	230	272	288	9	24	33	5	11	10	3
Ore Tonnes (kt)	0	197	1,203	1,199	1,201	1,198	1,203	1,200	1,181	220
Ore Mined Head Grade (% Cu)	1.2	1.9	2.8	3.3	2.9	2.7	2.3	2.1	2.1	1.8
Paste Backfill ('000 m^3)	0	23	219	335	321	290	297	358	314	81

Table 16.5 Mine Physicals Summary

16.10 Mine Equipment

Production equipment was selected based on its suitability to the mine design criteria, cost, fleet standardization, support, autonomy, experience, and lead time. All equipment will be diesel powered conventional rubber-tired equipment. The owner's equipment fleet by year over the life of the Project is illustrated in Table 16.6.

Equipment	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10
Haul Truck	0	2	4	4	4	4	3	2	2	2
Loader	0	3	3	2	2	2	2	2	2	2
Jumbo	0	3	3	3	3	3	3	2	2	2
Bolter	0	2	2	2	2	2	2	2	2	2
ANFO Loader	0	2	2	2	2	2	2	2	2	2
Scissor Lift	0	2	2	2	2	2	2	2	2	1
Shotcrete Truck	0	2	2	2	2	2	2	2	2	2
Shotcrete Sprayer	0	2	2	2	2	2	2	2	2	1
Fuel/Lube Truck	0	1	1	1	1	1	1	1	1	1
Boom Truck	0	1	1	1	1	1	1	1	1	1
Motor Grade	0	1	1	1	1	1	1	1	1	0
Personnel Carrier	2	9	9	9	9	9	9	9	9	3
Total	2	30	32	31	31	31	30	28	28	19

Table 16.6 Mine Equipment

16.11 Organization and Structure

The mine organization and structure is summarized in Table 16.7 below.

All Mine Department personnel will either be on a 9 - 5 rotation schedule or a 7 - 7 rotation schedule.

Production personnel will be on a 7 days on, 7 days off rotation. The rotation schedule will be 7 x 12 hour night-shift rotations followed by 7 days off then 7 x 12 hour day-shift rotations. At the conclusion of their day shift rotations they will work 1 x 8 hour day shift where there will be a safety meeting and training will occur.

All other personnel in the mine department will work a 9 days on, 5 days off rotation comprised of 9 x 9 hour day shifts followed by 5 days off.

Position	Rotation	Pre-Production	Average	Peak
Mine Management				
Mine Manager	9 - 5	1	1	1
Technical Services Superintendent	9 - 5	1	1	1
Underground Superintendent	9 - 5	0	1	1
Underground Services				
Dispatcher	7 - 7	0	4	4
Production Supervisor	7 - 7	0	4	4

Position	Rotation	Pre-Production	Average	Peak
Technical Services				
Senior Mine Planner	9 - 5	2	3	3
Mine Planning Engineer	9 - 5	2	4	4
Surveyor	7 - 7	1	4	4
Rock Mechanic	9 - 5	1	2	2
Projects/Ventilation Engineer	9 - 5	0	1	1
Senior Geologist	9 - 5	1	2	2
Mine Geologists/Technicians	7 - 7	2	4	4
Mine Operations				
Miner 1	7 - 7	0	12	17
Miner 2	7 - 7	0	17	24
Miner 3	7 - 7	0	8	13
Mine Operations				
Lead Hand Mechanic	7 - 7	0	4	4
Mechanics	7 - 7	0	8	8
Electrical Lead Hand	7 - 7	0	4	4
Electrician	7 - 7	0	6	6

Table 16.7 Organization and Structure Summary

16.12 Underground Infrastructure and Service

16.12.1 Controls and Communication System

Radio communications for the mine will be provided over a leaky feeder system. The system will be distributed throughout the mine for communication purposes. A combination of handheld and fixed radios will be used to communicate over the leaky feeder network.

A fibre optic cable will be installed down the Main Decline to provide enhanced communications and control. Features to be added include equipment and personnel tracking at key nodes, ventilation on demand, monitoring and control of the paste system and Wi-Fi communications to the surface at key locations.

16.12.2 Underground Power

Power will be distributed to the UG mine at 13.8 kV via two primary feeders from the surface. One feeder will be fed through the Main Decline while the second will be fed through RAR-1.

Each feeder will be connected to substations that will step down the voltage to 460 V for finalized distribution and usage.

	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10
Power Consumption (MWh)	1,909	4,537	10,029	7,416	7,323	7,333	6,410	5,474	4,349	1,668

Table 16.8 Mine Electrical Consumption

16.12.3 Compressed Air

There will be no central compressed air line. All mobile equipment that will require compressed air will make use of onboard compressors.

16.12.4 Potable Water

Potable water will be supplied by five gallon water jugs placed in refuge chambers and other strategic locations.

16.12.5 Drainage and Underground Sumps

The UG dewatering system has been sized and designed to handle all ground water inflows as well as any water produced by mining activities with a significant buffer capacity. Peak water inflow is estimated to be 3,000 m^3 per day (550 gpm) when all capital development has been completed and full production is achieved.

The main dewatering station will be located off the decline below the UCZ. All water in the mine will report to the main sump where it will be decanted and filtered. The water will be pumped using multistage centrifugal pumps up the Main Decline. A portion of the water will be stored at an UG water recycle dam and used as service water while the remainder will be pumped to the surface.

A fully integrated mobile slurry pump will be advanced with the Main Decline development and pump to the main sump. When decline distance exceeds the effective pumping distance of this pump, a booster pump will be integrated into the mobile pump system to maintain effective water management. This mobile pump will eventually be installed in the permanent pumping station for the LCZ.

Water in the UCZ and LCZ production areas will drain to temporary sumps where it is collected and pumped by electric submersible pumps to the nearest main pump station.

16.12.6 Service Water

A water reservoir will be located off the decline approximately 100 m down the Main Decline from the portal. The reservoir will have a capacity of 190 m^3 and will be fed by filtered clean water from the main sump. Service water will be distributed through the Main Decline using NPS4 DR9 HDPE pipe. Pressure management throughout the Main Decline is facilitated by installation of pressure reduction valves.

16.12.7 Refuelling and Service Bay

Major equipment maintenance as well as maintenance work on vehicles that regularly travel to surface will be performed in the surface workshop. This workshop is discussed further in Section 18.7.

Two service bays will be constructed, one in the UCZ and one off the Main Decline above the LCZ. Each service bay will be equipped with a five tonne jib crane. All service bays will be equipped with fire doors and have secondary egress and will perform minor repairs as well as regular preventative maintenance on equipment that does not regularly travel to the surface.

16.12.8 Explosives Magazine and Use

The main UG storage facility is located off the Main Decline and is protected from vehicle traffic. The main storage facility is equipped with a five tonne jib crane that will facilitate the delivery of explosive products from raised platforms to the loaders. Packaged emulsion product will be used in decline headings and other areas where there is significant water present.

16.13 Grade Control

16.13.1 Introduction

Geostatistical analysis of existing drill data, within the framework of the Mineral Resource block model, has been completed in order to estimate the required drillhole spacing to convert the areas of the Johnny Lee deposit that are currently in the Indicated Resource category to the Measured Resource category. For the UCZ, a 35 m drill spacing is required for a Measured Resource. A 50 m drill spacing is required for the LCZ. These drillhole spacings have been utilized to develop a grade control drill program for the deposit.

16.13.2 Upper Copper Zone

Figure 16.9 shows existing UCZ drilling and the orebody intersections that will be required to convert the existing Indicated Resource to Measured Resource. Given the shallow depth of the deposit and lack of UG platforms suitable for drilling it is assumed that all drillholes shall be completed from surface. Drillholes are planned to be collared using a tri-cone bit and then diamond drilled once competent rock is encountered.

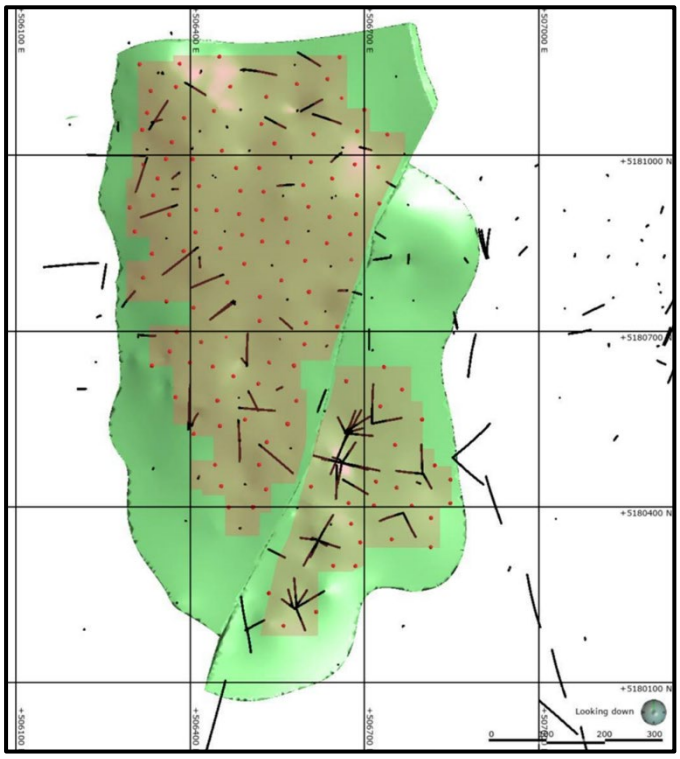

Figure 16.9 *Plan View of UCZ showing current Indicated Resource (shaded light red) and drillhole intersections required to convert to Measured (Red Symbols)*

Using surface topography and satellite imagery, suitable surface collar positions were identified from which the grade control target intersections could be completed. To minimize environmental impact, planning was undertaken such that as many drillholes as possible could be completed from each surface drill pad (Figure 16.10).

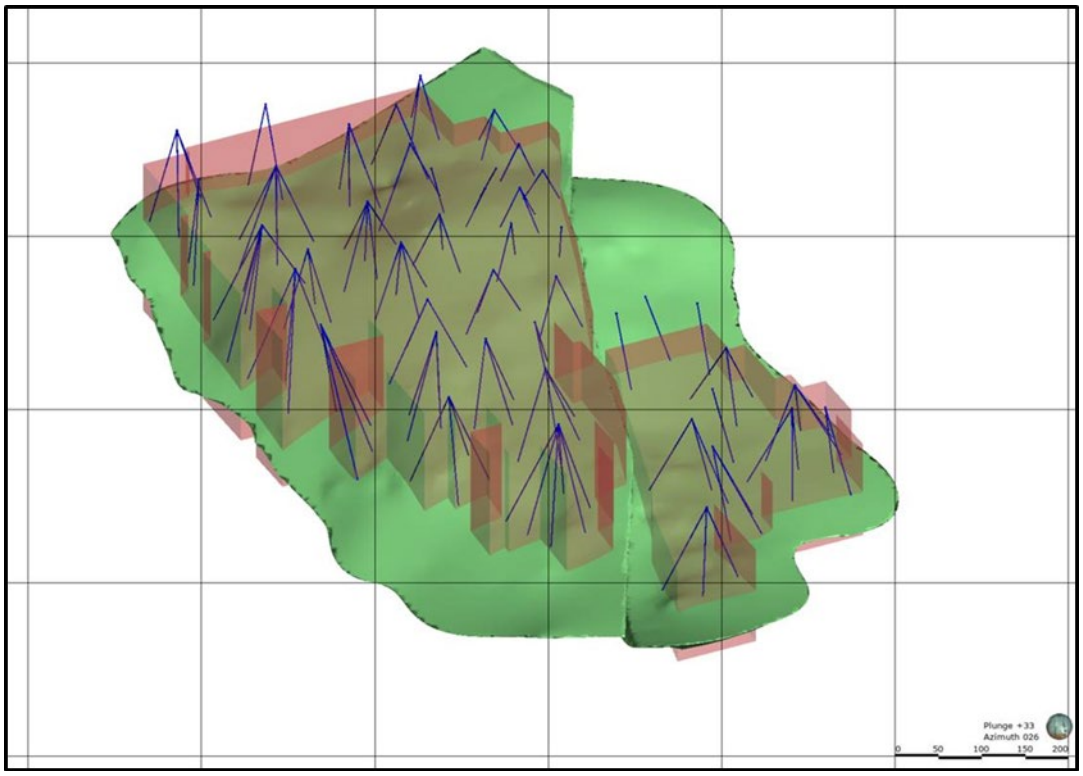

Figure 16.10 Oblique view of the UCZ showing planned grade-control drillholes

The planned grade control drill program includes 19,085 m of drilling. Costing of the drill program has been undertaken using actual costs for drilling ($261/m), site preparation and rehabilitation ($1700/drill site) during the 2019 Resource drill program. Assay and QA/QC costs for each drillhole are estimated at $3200. Mineragraphic analysis of three polished mounts from each drillhole has been planned to provide recovery estimate data, for incorporation within the grade control model, at a cost of $500/mount.

16.13.3 Lower Copper Zone

Figure 16.11 shows drillhole pierce points required to convert Indicated Resource to Measured Resource for the LCZ. The drilling for the LCZ will be accomplished from two drill cut-outs from the Main Decline. 21 drillholes for a total of 6100 m will be required.

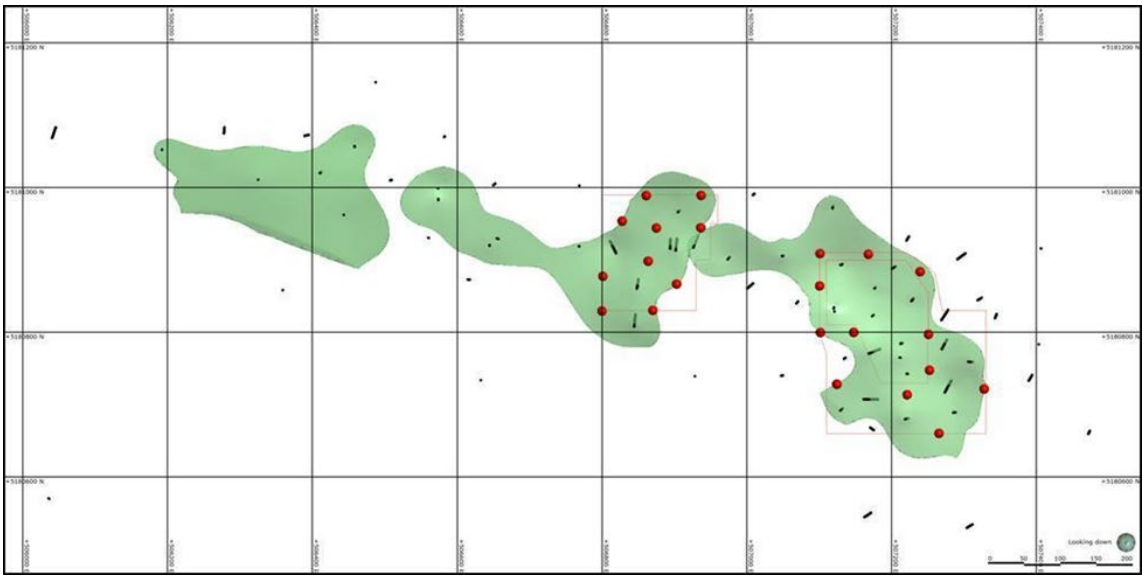

Figure 16.11 Plan View of LCZ showing intersections required to convert the current Indicated Resource to Measured Resource

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17. RECOVERY METHODS

17.1 Introduction

The selected process recovery method will utilize industry standard comminution and flotation process technologies to produce a copper sulphide concentrate for market export. The design processing rate of 150 dry tph will provide for the annualized 1.2 Mtpa treatment of UG sulphide mineralization, producing 122 ktpa of copper sulphide concentrate. The process plant will operate on a continuous 24 hour-a-day basis, via two 12 hour processing shifts. The design availability of the crushing and grinding circuits will be 70% and 92%, respectively.

UG ore will be crushed using a primary jaw crusher to produce an 80% passing 99 mm feed size for the grinding circuit. The grinding circuit will consist of a conventional two stage SAG and Ball mill configuration to produce a cyclone overflow product at the target 80% passing size of 38 microns for feed to the flotation circuit. The cyclone overflow slurry will be processed through the Cu rougher and scavenger flotation circuit, followed by a concentrate regrind circuit. The reground rougher/scavenger concentrate will be upgraded in four stages of Cu cleaning, consisting of a pre-cleaner Jameson flotation cell, followed by three stages of cleaning in conventional flotation cells. Cu scavenger tailings, together with the first Cu cleaner scavenger tailings, will be dewatered using a conventional tailings thickener. Thickened tailings will be pumped to a paste plant for the generation of cemented paste for mine backfill or a cemented paste for storage in the Cemented Tailings Facility (CTF). Concentrate from the third Cu cleaner combined with the pre-cleaner concentrate, will form the final Cu concentrate at an average grade of 23.5% Cu. The Cu concentrate will be dewatered using a conventional thickener with subsequent pressure filtration to produce a filter cake for dispatch to port via road using sealed containers.

A simplified process flow diagram is shown in Figure 17.1

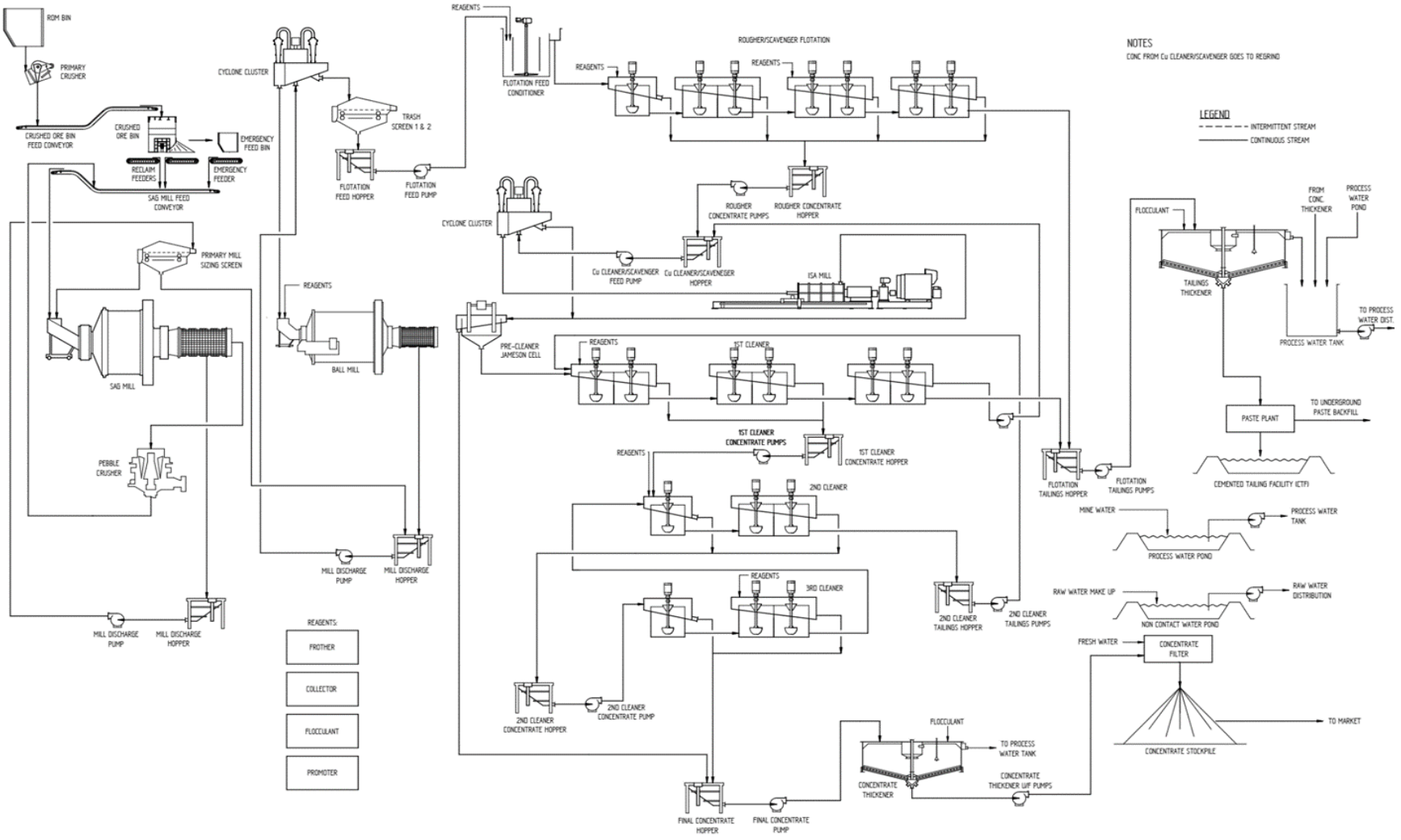

Figure 17.1 Plant Flowsheet

17.2 Process Design Basis

The process plant has used the key criteria listed in Table 17.1 as the basis of design development.

Key Criteria	Units	Process Values
Annual Throughput	tpa	1,204,500
Feed Grades ▪ Cu	 %	 2.9
Copper Concentrate ▪ Cu Grade ▪ Cu Recovery ▪ Concentrate produced	 % % tpa	 23.5 80.8 119,948
Operating Hours ▪ Crushing Circuit – UG Ore ▪ Milling Circuit – UG Ore	 hours hours	 6,132 8059
Treatment Rate ▪ Crushing-UG Ore ▪ Milling Circuit – UG Ore	 t/h t/h	 254 150
Primary Grind Size (P_{80}) Concentrate Regrind Grind Size (P_{80})	µm µm	38 10

Table 17.1 Key Processing Criteria

17.3 Process Plant Description

17.3.1 Overview

Feed to the new treatment plant will consist of fresh massive sulphide UG ore. The processing plant will be a conventional flotation-based concentrator consisting of crushing, grinding, sulphide flotation, thickening and filtering techniques. Low Cu plant tails will be converted to paste for use as UG backfill or stored in a new surface CTF. Final concentrate will be shipped off site for export.

17.3.2 Crushing

The crushing circuit was designed by Orway Mineral Consultants (OMC)(Borger and Kock, 2019), based on an Impact Crush Work index (CWi) of 10.7 kWh/t. A single stage crushing circuit has been selected to complement the selection of a SAG mill as the primary grinding method.

UG ore will be delivered to the Run Of Mine (ROM) pad and stored in a number of separate stockpile fingers according to ore type and grade to facilitate blending. Stockpiled material will be reclaimed by a FEL for direct feed to the crushing plant ROM bin.

Ore will be withdrawn from the 90 t capacity ROM bin using a variable speed apron feeder which will discharge onto a 1.12 m wide by 6.16 m vibrating grizzly with 64 mm bar spacing. Based on the selected ROM size distribution, 67% of the ROM feed will report to the jaw crusher, while the grizzly undersize will bypass the crusher via a chute directly onto the crushed ore conveyor. A single toggle jaw crusher (CJ411) has been selected for the crushing duty. The jaw crusher Closed Side Setting (CSS) will be maintained at 100 mm to achieve a crushed product size P_{80} of 99 mm.

Crushed rock will be conveyed to a Coarse Ore Bin (COB) of 2,500 t capacity that will provide 16.7 hours of surge.

17.3.3 Grinding

A two stage, SAG and ball mill grinding circuit has been selected to achieve the target grind size P_{80} of 38 µm to the flotation circuit. The mill selection has been based on the OMC simulation and an SMC specific energy calculation to determine the grinding power requirements and mill sizes.

Fresh ore will be reclaimed from the COB by two variable speed feeders and discharged onto the fixed speed mill feed conveyor. The primary SAG mill, with dimensions 6.10 m Ø x 3.05 m Effective Grinding Length (EGL), will be equipped with a 2,000 kW variable speed mill motor. A grate discharge system using 25 mm grates has been selected in combination with a discharge trommel at 12 mm apertures. Oversize from the SAG mill trommel will be conveyed to a pebble crusher for size reduction of the pebbles prior to returning to the mill feed conveyor. Trommel undersize pulp will be pumped to a primary sizing screen fitted with 2 mm polyurethane screen panels; screen oversize will be treated in closed circuit with the SAG mill while undersize material will gravitate to the ball mill discharge sump. The nominal SAG mill transfer size (T80) to the secondary ball mill will be 600 µm and controlled by the primary sizing screen.

The secondary ball mill will be fixed speed grate discharge mill (5.50 m Ø x 8.10 m EGL), operating at 75% of critical speed with a design ball charge of 27%. A grate discharge system using 15 mm grates will provide slurry flow to the mill trommel screen, fitted with 10 mm aperture panels. The trommel screen is required to remove ball scats only.

A summary of the grinding circuit design criteria is given in Table 17.2.

Method	Units	Value (80th Percentile)	Design Value
Treatment Rate	t/h	150	150
Feed Size F_{80}			99
Product Size P_{80}			38
SMC Rock Breakage Parameters			
- A*b		48.4	48.4
- DWi	kWh/m³	6.38	6.38
- Specific Comminution Energy	kWh/t		35.4
SAG Mill			
- Size (diameter x EGL)	M		6.1 x 3.05
- Specific Comminution Energy	kWh/t		9.6
- Installed Motor Power	kW		2,000
- Power Drawn	kW		1,440
- Discharge Configuration	type		Grate
Ball Mill			
- Size (diameter x EGL)	M		5.5 x 8.10
- Installed Motor Power	kW	5,500	4,600
- Power Drawn	kW	4790	3,860
- Specific Comminution Energy	kWh/t		25.7

Table 17.2 Grinding Process Criteria

The OMC simulations highlighted the importance of minimizing the SAG mill transfer top size to the secondary ball mill. Consequently, to optimize grinding efficiency, a vibrating screen has been installed to classify the SAG mill discharge product with a sizing screen fitted with 2 mm screen decks, and to control the transfer size to the ball mill circuit to 600 µm.

The secondary ball mill will operate in closed circuit with a cluster of 250 mm diameter cyclones. The cyclones have been designed with a circulating load of 250% and will have an overflow pulp density of 35% solids to achieve the flotation feed size P_{80} of 38 µm. The cyclone overflow will report to a horizontal, vibrating trash screen 1.5 m wide by 4.8 m long. The trash screen will be fitted with polyurethane screen panels having an aperture of 1.0 mm. Cyclone underflow will be directed to the ball mill feed chute.

Trash screen oversize will gravitate directly to a trash bin. Undersize from the trash screen will gravitate to the Cu flotation feed conditioning tank. A launder sampler will be located on the trash screen underflow and the sample will be pumped to the On-Stream Analyzer (OSA) for elemental and density analysis.

17.3.4 *Cu Flotation Circuit*

The Cu minerals, chalcopyrite and tennantite, will be recovered from the pyrite and non-sulphide gangue throughout the Cu flotation circuit at an elevated pulp pH of 9.5. Lime will be used to maintain the flotation circuit pH conditions. Aero 3477 will be the main copper sulphide collector and will be added in a stage-wise manner to avoid overdosing, which may result in the inadvertent recovery of the pyrite (FeS_2) to the Cu concentrate.

Depressants, sodium monophosphate and dextrin, will be used to minimize the flotation of the pyrite and carbonaceous gangue material. Depressants for the rougher/scavenger flotation circuit will be added in the grinding circuit. Depressants will also be added into the concentrate regrind circuit to maximise the rejection of pyrite in the cleaner flotation circuit.

The Cu rougher/scavenger circuit will consist of three rougher 30 m³ tank cells and four 30 m³ scavenger tanks cells.

Rougher/scavenger concentrate at 9% to 11% Cu, will be pumped to a cluster of 150 mm diameter dewatering cyclones to enable control of the feed density from the cyclone underflow to the regrind mill. Cyclone underflow will gravitate to a regrind mill feed hopper for further size reduction by a self-classifying M5000 IsaMillTM. The regrind feed cyclone overflow will be combined with the regrind mill discharge and pumped to the 10 m³ pre-cleaner conditioning tank.

Cu cleaning will be conducted in four stages and consist of:

- Cleaner scalping in a Jameson cell;
- An open circuit first cleaner/cleaner scavenger; and
- Closed circuit second and third cleaners.

The success of Jameson Cells in the roll of a cleaner scalper duty at the PanAust - Phu Kham and Oz Minerals - Prominent Hill operations, which increases both concentrate recovery capacity and operating flexibility for a low capital cost, has led to the inclusion of a Jameson Cell as a cleaner scalping duty in the SRA flowsheet. An E1723/4 model Jameson cell has been selected for the proposed duty. Tailings from the Jameson cell will gravitate to the first cleaner, whist the Jameson cell concentrate will be combined with the concentrate from final Cu cleaning stage.

The first cleaner flotation stage will consist of three conventional 8 m³ cleaner cells followed by three 8 m³ cleaner scavenger cells. The concentrate from the first Cu cleaner cells will be pumped to the second Cu cleaner cells for further upgrading, while the concentrate from the cleaner scavenger cells will be recirculated to the rougher/scavenger regrind circuit. The tailings from the cleaner scavenger cells will be pumped via the OSA to the final tailings feed box.

The second cleaning stage will consist of three 8 m³ conventional cells. The tailings from the second cleaner stage will discharge directly into the first cell of the first cleaner flotation stage. The second cleaner concentrate will be pumped to the third cleaning stage which will consist of three 4.3 m³ flotation cells. Tailings from the third cleaner stage will be directed to the second cleaner flotation stage. Concentrate from the final Cu cleaning stage (3rd cleaners) will be combined with the Jameson cell concentrate to produce the final concentrate of 23.5% Cu. The final Cu concentrate will be pumped via the OSA to the Cu concentrate thickener feed hopper.

17.3.5 On Stream Analysis System

A six-stream Courier analyser system will be installed in the flotation circuit to provide for continuous online analysis. Slurry samples from nominated streams will be directed to the Courier for multi-element analysis. Analytical results from the Courier will be displayed and recorded on a monitor in the plant control room and shift composite sub samples will be collected for metallurgical accounting purposes.

The following streams will be measured online by the Courier:

- Flotation feed;
- Cu rougher/scavenger concentrate;
- Cu scavenger tail;
- Cu cleaner scavenger tail;
- Cu final concentrate;
- Jameson cell concentrate.

A particle size analyser Outotec PSI 500 will also be installed to measure the particle size distribution of the flotation feed and Cu regrind product.

17.3.6 Concentrate Production

Concentrate Thickening

Frothbuster® technology has been included in the design to reduce the build-up of froth on the surface of the Cu concentrate thickener, and to minimize the potential loss of Cu concentrate to the thickener overflow. Final Cu concentrate slurry, filtrate from the Cu filter and spillage from the Cu concentrate area, will be pumped to the Cu concentrate Frothbuster® which will operate at 100 kPa to deaerate the pulp prior to discharging into the feedwell of the Cu concentrate thickener. A 12 m diameter high rate Cu concentrate thickener fitted with an auto-dilution feed system has been selected. Flocculant will be mixed into the feed slurry to increase the solids settling rate and control the clarity of the thickener overflow. The Cu concentrate slurry will be thickened to 57% solids and will then be pumped to a 300 m³ agitated concentrate storage tank by one of two peristaltic type pumps in a duty/standby configuration. The Cu concentrate thickener overflow will be pumped to the tailings thickener overflow tank.

Concentrate Filtering

The Cu concentrate storage tank will have capacity to store up to 18 hours of Cu concentrate production. A single duty filter feed pump will pump the Cu concentrate to a plate and frame type pressure filter which has been selected due to the fine particle size distribution of the concentrate. A narrow chamber depth has been used for sizing based on the test work results to realize the target concentrate moisture content of 10%. A filter, fitted with 1.5 m by 1.5 m plates and 42 chambers, has been selected. The filter will operate automatically via a dedicated vendor supplied PLC and operator interface system linked to the plant control system. The filter filtrate will be pumped to the Frothbuster® feed hopper via a blowdown vessel.

The filter cake will discharge onto the floor of the concentrate storage shed from where it will be loaded into sealable containers by FEL on loadout trucks while parked on a weighbridge. The trucks will then transport the concentrate either directly to port or to a rail header for transfer to rail/port transport options. Containers will be unloaded or stored on a hard stand at the port awaiting shipment in bulk carriers.

17.3.7 Tailings Thickening

The final tailings, consisting of the Cu scavenger tails and the Cu cleaner scavenger tails, will be dewatered in an 18 m diameter high rate thickener. Additional streams from the water treatment plant, the paste plant and decant water from the CTF, will also feed the thickener. Flocculant will be added to increase the settling rate and underflow density to between 55% and 60% solids. Thickened tailings will then be pumped to the paste plant filter feed tank. Thickener overflow will gravitate to a tailings thickener overflow tank, which also collects the overflow from the Cu thickener. Water from the tank will be used as process water within the circuit and will be supplemented by process water from the process water dam.

17.3.8 Paste Backfill Plant

Thickened plant tailings from the tailings thickener will be pumped to a 300 m^3 capacity paste filter feed tank. Variable speed paste filter feed pumps will transfer the material at a feed density of 55% to 60% solids to two conventional vacuum disc filters, with a third filter installed as a standby unit. The design has allowed for 85% of the thickened tailings to be feed to the filters, with the remaining 15% of the underflow being recombined at the paste mixer to generate the paste and avoid the requirements of additional water dilution.

17.4 Reagent Mixing, Storage and Distribution

Treatment of the UG sulphide ore by the flotation process will require a specific suite of flotation reagents described below to ensure the activation of selected minerals, the depression of gangue mineral, frother for froth stability and collectors for specific mineral recovery.

Copper Collector- Aero 3477

The Cu collector will be a dithiophosphate with the trade name Aero® 3477. The Cu collector will be supplied as a 100% concentrated solution in 1,000 L bulk boxes.

Frother - MIBC

The frother will be a Methyl Isobutyl Carbinol (MIBC). The frother will be supplied as a 100% concentrated solution in 1,000 L bulk boxes.

Frother – POLYFROTH W

Huntsman POLYFROTH® W31, polyglycol ether frother, will be used as an additional frother in the Cu cleaner flotation circuit. POLYFROTH® W31 produces relatively mobile and freely draining froth of intermediate strength. It offers a combination of selectivity, froth stability and moderate kinetics. The frother provides enhanced stability for coarse particle recovery. The frother will be supplied as a 100% concentrated solution in 1,000 L bulk boxes.

Sodium Monophoshate

Sodium Monophosphate (SMP) is a modifying/depressant agent to stop the activation of iron sulphide minerals from free Pb and Cu ions. Depressants change the surface chemistry of selected minerals to depress the action of collectors in flotation. SMP will be supplied in powder form in 1,000 kg bulk bags.

SMP mixing will be completed manually by the operator. The bulk bags will be lifted by the reagents hoist, split, and then mixed with raw water to generate a 20% (w/v) solution. The hood over the mixing tank will be fitted with an extraction fan.

Hydrated Lime

Lime will be used to increase the pH of process streams and neutralize the acidic condition of the stream, and to form hydroxides of particular metals. The lime will be supplied as hydrated lime ($Ca(OH)_2$) in bulk as a powder. Upon receipt at site, it will be pneumatically transferred into an 80 t capacity silo. The hydrated lime will be mixed with raw water in a lime mixing tank to produce hydrated lime slurry. Lime will be automatically mixed with raw water on a batch basis to generate a 20% slurry of hydrated lime. The hydrated lime slurry will be automatically transferred from the mixing tank to the storage tank. Both the mixing and storage tanks will be agitated. Hydrated lime slurry will be pumped via a ring main for distribution throughout the plant. Individual addition points will be controlled by an automatic valve and timer.

Dextrin

Dextrins are water soluble polysaccharides and are used as depressants for carbonaceous pyrite.

Dextrin mixing will be completed automatically in a mixing system similar to that of flocculant (refer to the flocculant mixing description below). The bulk bags will be lifted by the reagents hoist, split, and then mixed in batch operation with raw water to generate a 10% (w/v) solution. This solution will be transferred to a storage tank for distribution to the primary SAG mill and concentrate regrind mill circuits.

AntiScalant

Antiscalant will be added to the suction side of both the process water pumps to inhibit the formation of scale in the process water system. The antiscalant will be supplied as a solution in 1,000 L bulk boxes.

Flocculant

The flocculant will be an anionic flocculant with a trade name Magnafloc 10. Flocculant will be supplied as a powder in 800 kg bulk bags.

The flocculant mixing system will be a proprietary packaged plant consisting of a dry flocculant hopper, powder feed and wetting system, mixing tank, transfer pump and storage tank. Flocculant will be mixed automatically with raw water on a batch basis to generate a 0.25% (w/v) solution. The flocculant solution will be automatically transferred from the mixing tank to the storage tank.

Granulated Blast Furnace Slag

Ground blast furnace slag – Semi Cementitious Material (SCM) will be used in the paste plant as the binder to the UG stopes and the CTF. The binder is added on a ratio to dry tonnes of mine backfill and is mixed in a paste mixer prior to being pumped back down to the required backfill stope.

Cement

Portland type (I/II) type cement will be used in the paste plant as the cement for the UG stopes and the CTF paste requirements. The cement will be added on a ratio to dry tonnes of mine backfill or CTF paste and is mixed in a paste mixer prior to being pumped to the desired location.

17.5 Services

17.5.1 Control Systems

Operators will monitor and run the plant from PC-based Human-Machine Interface (HMI) systems located in the process plant control room. Operations will be monitored and controlled from the process plant control room via PC-based Supervisory Control and Data Acquisitioning (SCADA) screens using Citect software. The ability to remotely start and stop plant equipment from the operator screens via start and stop sequences or by "manually" starting or stopping individual equipment will be provided. Recording of plant data by operations personnel will be required on a shift by shift basis, with the Courier system providing online assay results.

Motor starters, main isolators and distribution board feeders will be located in the Motor Control Centres (MCC's). Each MCC will contain a PLC which with the SCADA system, will make up the Process Control System (PCS). The PCS will provide the interface between drives and instrumentation and the operators. Hardwired outputs and inputs for plant equipment and field devices will be interfaced through these PLC's in which plant start-up/shutdown sequences and interlocks will be programmed. The plant functions will be controlled automatically by the PLC and the operators will be required to monitor the system performance and perform manual checks.

All equipment will be equipped with local start and stop buttons. Start and stop will also be possible via HMI. It is intended that there will be two modes of operation for each piece of equipment: local and remote. The status of each piece of equipment (i.e. whether in local or remote mode) will be displayed at the HMI. Local or remote mode will be selected using a switch mounted on the local control panel. The stopped/started status of each piece of equipment will be displayed at the HMI.

17.5.2 Water Services

Water Treatment

UG dewatering water at a nominal rate of 114 m^3/h (500 gpm) will be treated in a proprietary RO Water Treatment Plant (WTP) located at the processing plant site. Treated water will be to RO quality and will be used in part as water for the process plant with the bulk of the water returned to an UG infiltration gallery system. The water treatment plant will be a sub-contract plant operated by others.

Raw Water

Raw water will be a combination of returned water from the process water pond and the water treatment plant, following the clarification and filter stage. It will then be stored in the new raw water tank at the plant site. It will be further cleaned by a sand filter prior to return to the plant.

Raw water will be used for the following purposes:

- Process water make up;
- Reagent mixing;
- Flocculant dilution;
- Cloth washing in filtration;
- Gland water;
- Regrind milling cooling water; and
- Fire water.

Two raw water distribution pumps arranged in a duty-standby configuration will be located at the raw water tank. These pumps will distribute the raw water throughout the processing facility. In addition, there will be two dedicated pumps, arranged in a duty-standby configuration, for gland water.

The lower portion of the raw water tank will provide a dedicated fire water reservoir for the fire water system. The fire water system will include two pumps, an electric fire water pump and a diesel driven fire water pump to ensure a continuous supply of water to the fire system in the event of a power failure.

Process Water

Process water will be primarily recovered from the process plant as thickener overflow streams to the tailings thickener overflow tank. Two dedicated process water pumps will then supply the water around the circuit. Inflow from the water treatment plant and makeup water from the Process Water Pond (PWP) will provide the additional makeup for operations. The PWP has a total fill capacity of 200,000 m^3 and will be used to store process water as the main storage facility.

Potable Water

Potable water will be provided from the WTP after the RO skid treatment system discharge and pumped to a potable water tank. A UV sterilization system will be included for the potable water discharge from the tank.

17.5.3 Air Services

Compressed air will be supplied by two dedicated plant air compressors along with a standby unit for any high demands and redundancy in the system. Instrument air will be provided from the plant air system via an air dryer and filter. Each will have their own respective air receiver with additional air receivers located at positions within the plant to ensure adequate storage capacity.

An independent air system will be used for the paste plant filter snap air system, it will be located adjacent to the paste plant.

Flotation air to the flotation circuits will be supplied by two dedicated blowers arranged in a duty-standby configuration.

17.6 Processing Labor Force

The operating philosophy of the processing plant has been based on a continuous 24-hour operation, with two 12-hour shift rotations. The shift rotations will use a 7 days on / 7 days off, 4 panel shift roster to achieve the production targets. Technical based personnel will be on a 9 days on / 5 days off roster with a back to back arrangement for supervisors and superintendent roles. Senior management and administration will be based on a 5 days on / 2 days off arrangement. All roles are locally based with no fly-in or out option.

The Project is to be operated with a departmental structure as shown in Figure 17.2. The total direct workforce requirement for process plant operation is 124 employees. The processing structure provided is considered adequate to maintain and operate a plant of this size and complexity.

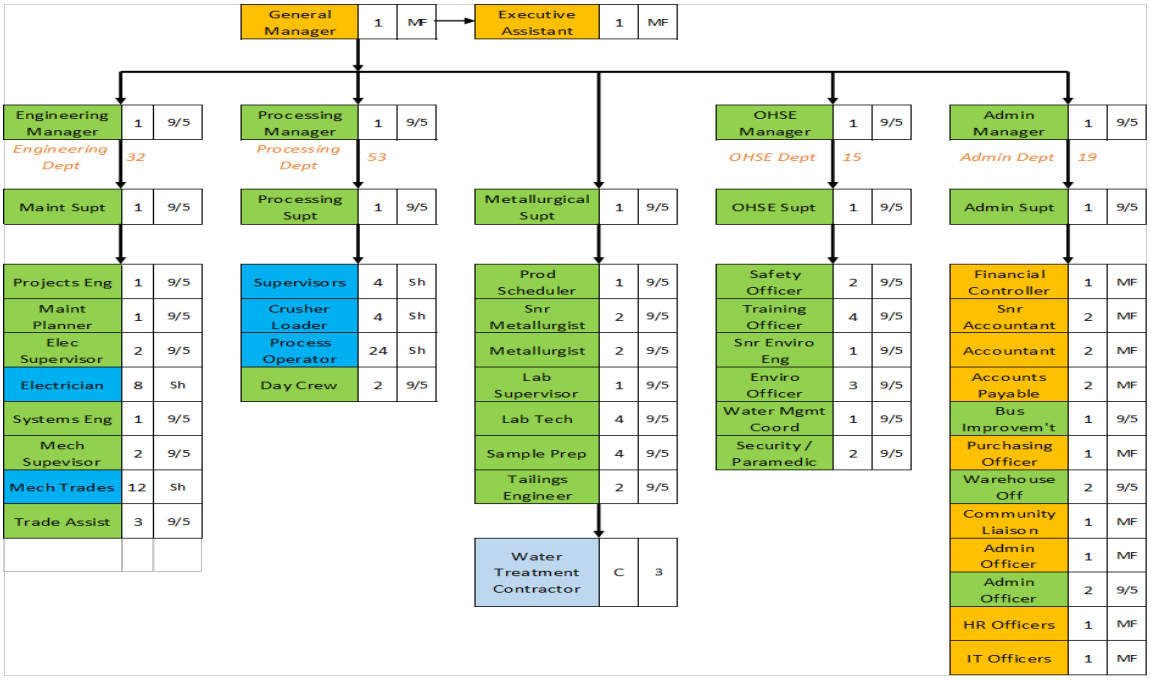

Figure 17.2 Processing Organizational Structure

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18. PROJECT INFRASTRUCTURE

18.1 Introduction

The Project will require the development of various ancillary facilities and related infrastructure. Their location has been selected to take advantage of local topography; to accommodate environmental considerations; and to reduce capital and operating costs.

Project facilities and infrastructure that support the operations will include:

- Access:
  - Roads;
  - Rail; and
  - Airport.
- Water supply:
  - Water requirements;
  - Potable water supply;
  - Mine dewatering fresh water supply;
  - Water treatment; and
  - Water management.
- Power supply and distribution:
  - Utility connection;
  - Construction and emergency power;
  - Mine site substation; and
  - 13.8 kV power distribution from the mine site substation to all the facilities.
- On-site ancillary facilities:
  - Buildings and structures;
  - Fuel and explosives;
  - Sewage treatment;
  - Fire protection system;
  - Communications;
  - Water treatment plant (WTP); and
  - Paste plant.
- Other surface infrastructure:
  - Portal pad;
  - Waste Rock Storage (WRS),
  - Cu-enriched rock stockpile;
  - Process Water Pond (PWP);

- Brine/Contact Water Pond (BCWP);
- Non-Contact Water Reservoir (NCWR);
- Treated Water Storage Pond (TWSP);
- Cemented Tailings Facility (CTF); and
- Site supported equipment.

The locations of the main Project facilities are shown in Figure 18.1. Additional Plant layout drawings are presented in Appendix G. Appendices C and D include process flow diagrams and piping and instrumentation diagrams, respectively.

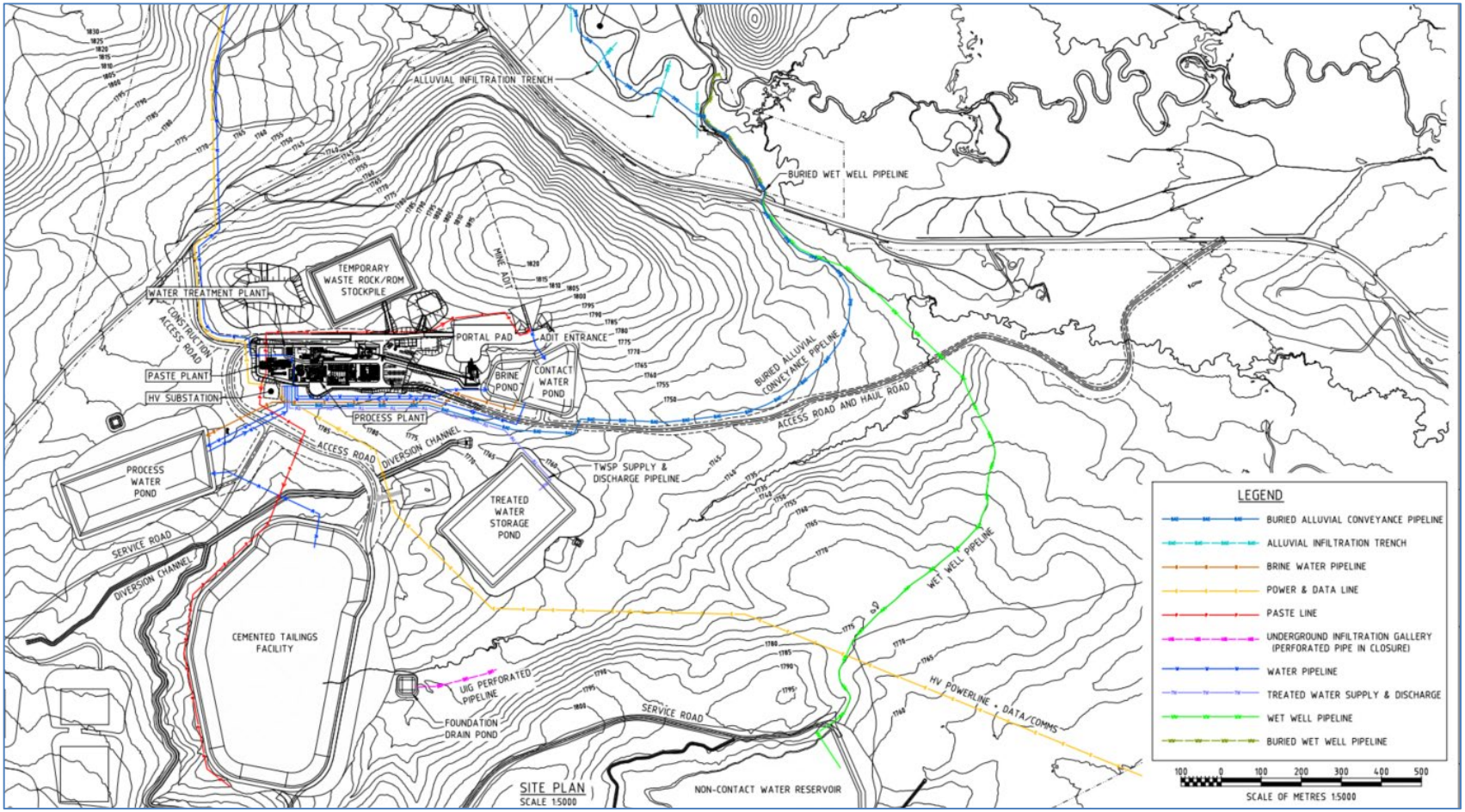

Figure 18.1 Overall Site Plan

18.2 Infrastructure Coordinate System

Infrastructure design plans and drawings used in the MOP Application (Tintina Montana, 2017), Tetra Tech Phase 1 construction drawings (Cawlfield, 2020), and the FS design drawings are presented on topographic maps based on aerial Lidar deliverables collected by MT Lidar in October 2012 and utilize a grid coordinate system. This grid coordinate system used herein is the Universal Transverse Mercator (UTM) Zone 12 North (N), and the datum is the 1983 North American Datum (NAD83).

Note that all exploration and resource infill drillholes and installed wells have been surveyed in ground coordinates to date (see Section 10 of the Report). WWC Engineering Inc. has conducted the collar drillhole and well surveys at the Project since SRA started work at the Project and recently documented the slight differences between the grid and ground coordinate systems in a memorandum (WWC Engineering, 2020). SRA plans to consolidate the Project drillhole and well data and the recent Phase 1 Development surface construction into one likely ground-scaled coordinate system prior to the start of underground mining.

18.3 Surface Disturbance

The total surface disturbance for the major mine facilities from Zieg (2018) is shown in Table 18.1 and was approved by the MDEQ (2020a; 2020c, 2018).

Facility or Activity	Surface Disturbance (ha)	Surface Disturbance (Acres)
New Access Roads	23.4	57.7
Direct Underground Mine Support	3.2	7.9
Waste Rock Storage and Cu-enriched rock stockpile	4.9	12.1
Brine/Contact Water Pond	3.6	9.0
Mill/Plant Site	4.0	9.8
Process Water Pond	11.6	28.7
Cemented Tailings Facility	33.4	82.5
Non-Contact Water Reservoir	3.1	7.6
Water Supply	2.5	6.3
Underground Infiltration Galleries	2.2	5.4
Material Stockpiles	13.1	32.4
Treated Water Storage Pond	8.2	20.2
Wet Well Diversion and Pipeline	1.0	2.4
Other/Miscellaneous	0.2	0.6
Subtotal	***114.4***	***282.6***
Construction Buffer Zone/Misc. (10%)	11.4	28.3
Total Disturbance	**125.8**	**310.9**

Table 18.1 Surface Disturbance Consolidated by Major Facility (from Zieg, 2018)

Total surface disturbance during early Phase 1 Development preconstruction that started on August 17, 2020 is focusing on the BCWP, portal pad, WRS pad, associated access roads, equipment laydown areas, and borrow pit located in the northern part of the CTF excavation footprint. This Phase 1 surface disturbance will cover a maximum area of 49.7 ha (122.8 acres). The borrow pit is required to source granodiorite required for construction of these early site facilities in addition to the other larger facilities included in Phase 2.

18.4 Engineering Evaluations and Design Standards

As part of the Project advancement, SRA initiated and completed geotechnical (Collison et al, 2017) and hydrogeological site investigations (Hydrometrics, 2017) in the spring of 2015. These investigations included the collection of pertinent geotechnical and hydrogeological information necessary to characterize the conditions at the proposed plant site and portal pad, as well as, to support feasibility level design for the BCWP, CTF, PWP, and NCWR. Additionally, it provided confirmation of the suitability of potential borrow sources and other on-site materials from the facility excavations for construction material. The results of these engineering evaluations developed the design criteria for the waste rock and water management facilities to satisfy State, Federal and International standards.

18.4.1 Plant Site

The geotechnical investigation of the proposed surface facility sites (Collison et al, 2017) showed that it consists of an overburden overlaying a bedrock. The overburden consists of topsoil and local subsoil. These soils are mainly a firm to compact, moist, sandy silt and clayey sand with some gravel. The gravel is sub-angular to sub-rounded and poorly graded. The overburden contains some roots and organics. Thickness varies from 0.3 to 1.4 m.

The bedrock is predominantly shale with associated sedimentary rocks. The average field estimated UCS of the shale is approximately 35 MPa and the average Bieniawski Rock Mass Rating (RMR) is 42, indicating a rock mass designation of FAIR.

18.4.2 Cut and Fill Construction Material Balance

The MOP Application (Tintina Montana, 2017) provided evidence that there is sufficient non-acid generating construction materials sourced from the on-site facility excavations to: (1) construct all of the planned mine facilities and (2) reclaim and close all of the facilities after production mining and milling has ceased to pre-mining topographic.

The TWSP was added to the water management facilities after the MOP Application. In addition, detailed design of the BCWP and portal pad by Cawlfield (2020) was also completed after the MOP Application.These facility design changes post MOP Application result in an updated total project bulked cut bedrock volume of approximately 3,214,996 m^3 and a bulked construction fill volume required for construction of approximately 2,867,594 m^3. This results in an excess of 347,402 m^3 bedrock which is still sufficient to successfully close and reclaim the mine facilities at the end of mine life to approximate pre-mining topography (except the CTF). As described in the MOP Application (Tintina Montana, 2017), the facility design allows flexibility during construction and reclamation to utilize all the excess bedrock and soils and minimizes the need for imported materials from off-site.

18.4.3 Water Management Facilities

The Project's water management facilities include the CTF, BCWP, PWP, NCWR and the TWSP. It should be noted that while not considered a tailings storage facility by the MDEQ (2017), the Project has elected to commit to the substantive provisions of the Metal Mine Reclamation Act as applicable to a "tailing storage facility" and instilled the stricter design criteria to the CTF structure. These facilities were first classified per the Federal Emergency Management Agency (FEMA) guidelines and the Administrative Rules of Montana (ARM) requirements for dam safety, then designed to standards outlined by the International Commission On Large Dams (ICOLD) which incorporated the Montana Code Annotated, Title 82 (MCA 82) requirements.

The first of the drivers of such facility designs is the ARM evaluation for dams which is based on the consequences of a dam failure and not the condition, probability or risk of failure. Using this information, together with the FEMA guidelines for dam safety, a Hazard Potential Classification was assigned to each facility. ICOLD addresses Probable Maximum Flood (PMF) conditions, as well as the seismic design requirements. MCA 82 states that a dam operating in Montana must withstand either the Maximum Credible Earthquake event (MCE) or the 1 in 10,000-year earthquake event, whichever is greater.

In the instance of this Project, the classifications for the CTF, PWP, NCWR and the BCWP has been defined as high, high, low and low, respectively. As such, the PWP and the CTF were designed to store the PMF volume in addition to their normal operations volumes. Since the BCWP and NCWR are "low" risk facilities, they have been designed to safely pass a peak instantaneous discharge associated with the 1 in 200-year, 24-hour storm event. For these facilities, the seismic design criteria requirements require the use of the 1 in 10,000-year earthquake event requirement, as it proved to be the more stringent criteria.

18.4.4 *Off-site Roads*

The Project is easily accessed from U.S. Highway 89 (US-89). US-89 is an all-weather, two-lane paved highway that is maintained by Montana Department of Transportation (MDOT) and lies east of the Project area. Either travel north on US-89 for approximately 24 km (15 miles) from White Sulphur Springs, MT or south about 134 km (83 miles) from Great Falls, MT to Sheep Creek Road. Then travel west for 2.4 km (1.5 miles) on Sheep Creek Road to the site access road. Sheep Creek Road is a well maintained gravel county road (Figure 18.2).

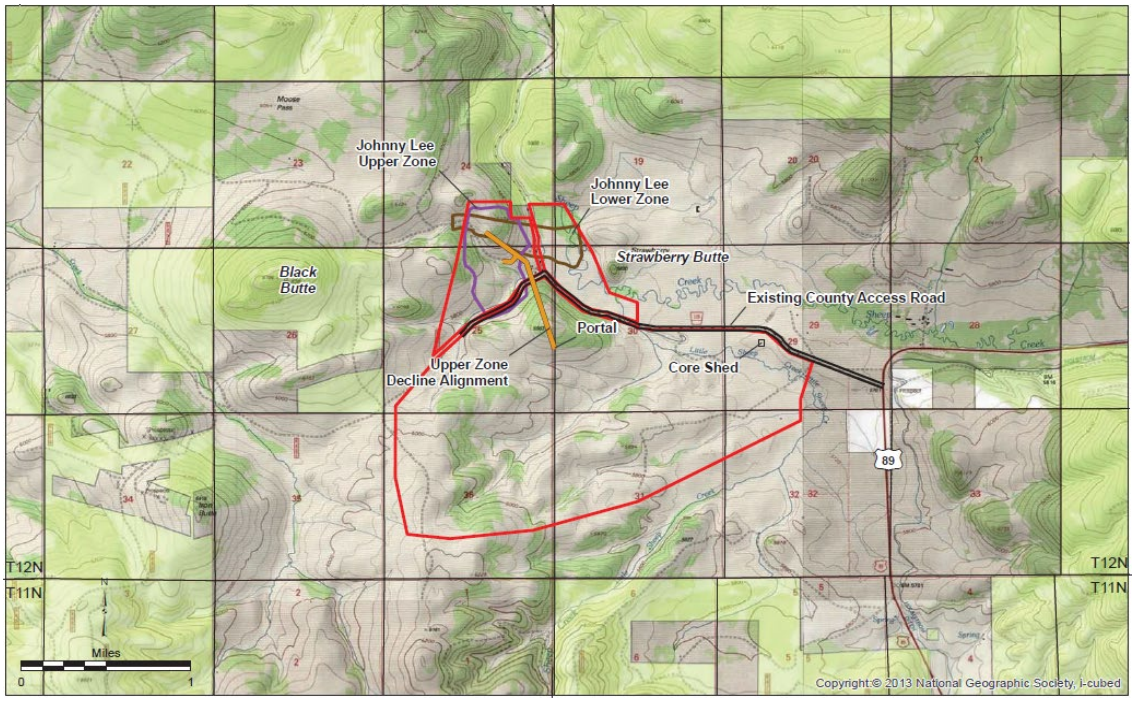

Figure 18.2 Site Vicinity Map with Mine Permit Boundary

A traffic impact study will be completed to assess any traffic control and intersection improvements where the main Project access road ties into the county roads and where the county roads tie into US 89.

18.4.5 Main Access Road

Within the Project area there is approximately 35 km (22 miles) of unpaved roads that traverse the property. The roads are a mix of established gravel roads and less frequented dirt or grass two-track roads. The established county gravel roads include Sheep Creek Road and Butte Creek Road. There are two unnamed gravel roads that provide access to residential buildings in the northern portion of the Project site. The dirt and grass two-track roads are fairly well distributed across the Project property; the central portion of the Project area has fewer two-tracks than other areas.

The access road route alignment was designed to minimize impacts to local ranchers, recreationalists, property owners, creek crossing widths, impacts to wetlands, and overall travel distances. The permanent access road will be a private gravel road designed and constructed to Meagher County and MDOT specifications and standards. The access road will support personnel access, delivery of supplies, and the transport of concentrate. Vendor use and deliveries along the road will be restricted to daylight hours and weekdays, whenever possible. There will be a gate and guard house at the western end of the main access road to restrict public access onto the mine site, along with a set of truck scales in the vicinity of the gate/guard house area.

The access road will be gravel surfaced over a compacted sub-grade and crushed gravel upper surface. The road will be designed for a maximum speed of 55 kph (35 mph), crowned, and have maximum grade of 7%. The roadway will be two lanes, each 3.7 m (12 ft) wide, with minimum 0.3 m (1 ft) wide shoulders developed with side slopes of 2.5H:1V (horizontal to vertical). The road will require berms of one-half axle height or greater for the largest vehicle using the road as per MSHA safety requirements. The road will be routinely watered to reduce airborne dust. Other controls for dust may include, speed and traffic controls and treatment with magnesium/calcium chloride. Periodic grading will be conducted as necessary.

The road will have drainage control, culverts, and sediment control basins, as necessary. Water diversions off of the main access road, through berms where required and into storm-water diversion ditches, will be laid-out on a site-specific basis maximizing the use of natural topography and avoiding diverting storm water directly into natural ephemeral or perennial drainage channels. The berms can have openings at any number of pre-planned locations to prevent excessive water flows and to mitigate the erosion resulting from storm-water diversion. Drainage from the road is deemed to be non-contact water and will be collected in roadside ditches designed to accommodate the 1 in 10-year return storm event. Water will be diverted to natural drainage areas.

Topsoil and subsoil from access road construction will be temporarily stored and reused in concurrent reclamation along the roadways. Excess soils will be salvaged separately and transported to stockpiles.

The MOP Application describes two planned creek crossings on the private mine access road. The access road crossing location on Brush Creek, the nearby buried alluvial conveyance pipeline, and the nearby proposed powerline, are located to avoid a cultural site. This crossing location also slightly decreased the amount of fill within wetlands (<0.004 ha; <0.01 acres) at the Brush Creek crossing. The sites for stream crossings were selected specifically to minimize impact to wetlands.

At each creek crossing, a 3 m (9.8 ft) diameter bottomless pipe arch, and two 1.8 m (6 ft) diameter round culverts will be installed, one on each side of the half pipe arch. Any storm flow not accommodated is expected to potentially overtop or damage the road requiring occasional repairs. The two creek crossings will permanently impact two wetlands and two streams.

18.4.6 Construction Access Road

A temporary access road will be required during Phase 1 Development construction. SRA plans on using the existing road entering the site from the west via the Butte Creek County road. A gate will be installed at the west end of this construction access road to control the public from entering onto the site.

18.4.7 Cemented Tailings Facility Road

The proposed CTF road runs from the portal pad to the north of the mill pad, and then SE to the CTF. There will be a short branch off the CTF road to access the WRS/ROM rock stockpile pads. The proposed CTF road will be constructed for use by 40 t (44 ton) trucks hauling waste rock from UG to the WRS and ROM rock pad.

Roadway design criteria for the CTF surface facility road are similar to access roads. Maximum grades are set to 8% and maximum speed limits on all roads are expected to be approximately 50 kph (30 mph). The CTF haul road will require berms of one-half axle height or greater for the largest truck using the road as per MSHA safety requirements. Crossing culverts will be installed as needed along the route.

Topsoil and subsoil from the CTF road construction will be temporarily stored and then used in concurrent reclamation along the roadways. Excess soils will be brought to the topsoil and subsoil storage areas west of the CTF.

Drainage from the CTF road will be deemed contact water as the road is used to haul waste rock from the mine to the CTF. This water will be collected in High Density Polyethylene (HDPE) or half culvert lined roadside ditches that transfer it to the CWP for subsequent treatment. Some of the water from the CTF haul road along the east side of the CTF will need to be diverted to the CTF foundation drain pond, prior to being pumped to the PWP or directly to the water treatment plant.

18.4.8 Service Roads

Service roads will access the PWP, NCWR, CWP, and topsoil and subsoil storage areas. These service roads will be designed using applicable low volume road design criteria that include a 5 m (16 ft) wide roadbed, maximum 8% grade, and 2.5H:1V side slopes and design speeds of approximately 40 kph (25 mph).

There will be other minor one lane tracks as required on the site for maintenance access purposes. Road-side ditches for the service roads will discharge to their existing drainage catchment outlets. The ditches will be hydroseeded with native species or riprapped for erosion protection depending upon the expected velocities in the ditch section. Crossing culverts will be installed as needed along the route. Service roads are intended to only be used by operations personnel using single haul units and light vehicles.

18.4.9 Railroads

SRA proposes to transport concentrate to a west coast port facility for shipment to probable smelters. The Burlington Northern Santa Fe (BNSF) and the Montana Rail Link (MRL) operate 59% and 25% of the rail lines in the State of Montana, respectively. Upon early inspection, the closest rail facilities to the Project area appear to be MRL's line that is about 64 km (40 miles) W/SW of White Sulphur Springs in the town of Townsend, MT and reachable via US-12. This segment of MRL line is part of a longer SE-NW line that connects Huntley, MT and Sandpoint, Idaho (MRL, 2015) and also passes through the town of Livingston approximately 140 km (87 miles) from the Project, see Figure 18.3. MRL is a Class II regional railroad that operates 1,408 km (875 miles) of track within Montana (Cambridge Systematics, 2010).

Alternatively, BNSF operates 92 km (57 miles) east of White Sulphur Springs near Harlowton, MT. This BNSF line runs N/NW from east of Laurel to Great Falls, MT (BNSF, 2014). BNSF is a Class I railroad, that as of 2006, operated 3,125 km (1,942 miles) of track in Montana.

To clarify how the use of rail will occur on the Project, SRA engaged a materials management and logistics group that works in the minerals industry. This contractor conducted a Supply Chain Study for moving 160,000 t (≈176,400 tons) per year of Cu concentrate using a combination of truck and rail from the mine to a west coast port facility.

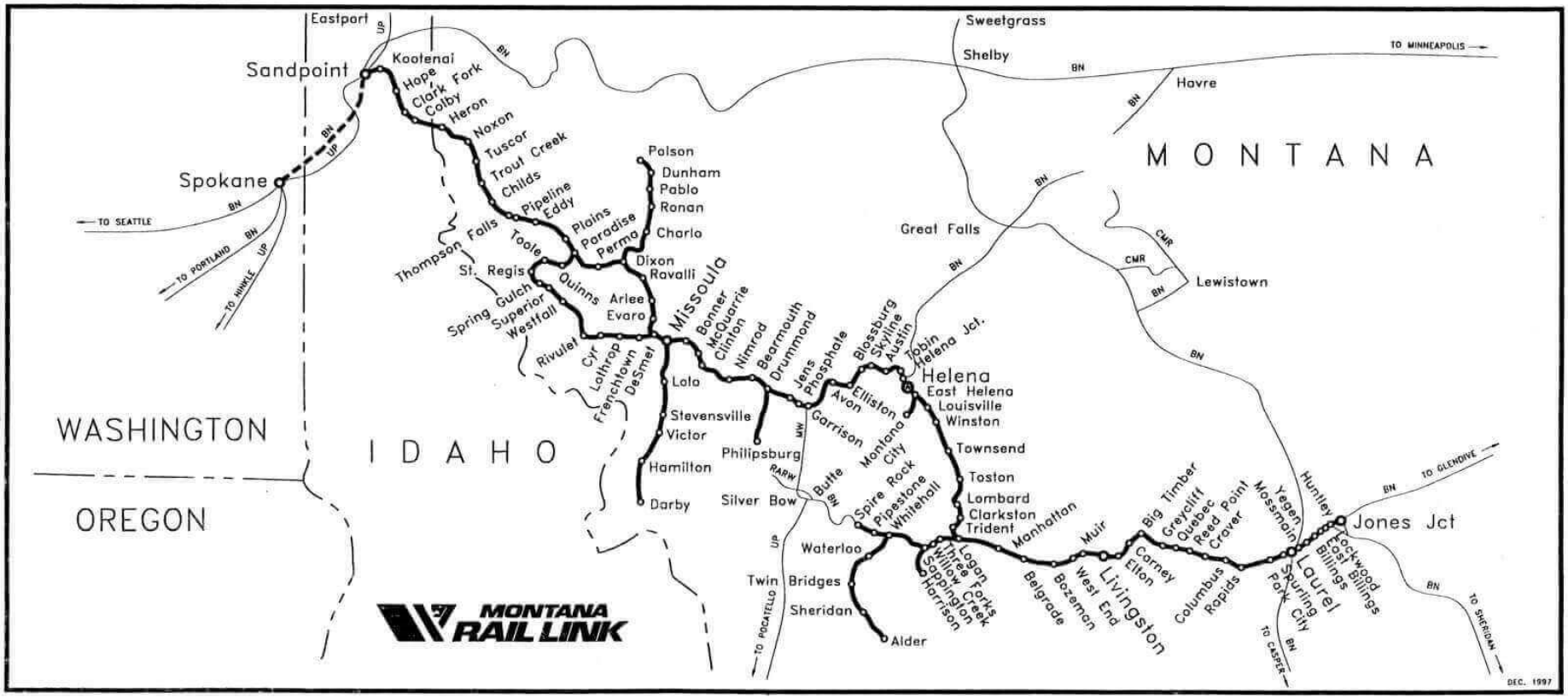

Figure 18.3 Montana Rail Link

18.4.10 Airports

The closest public airport is located 4.8 km (3 miles) south of White Sulphur Springs on the west side of MT Hwy 12 / US-89, approximately 31 km (19 miles) due south of the Project area. The City of White Sulphur Springs and Meagher County (AirNav, 2019) jointly own the airport, FAA identifier 7S6. The airport can accommodate single engine airplanes and has an average of 93 aircraft operations per week. Facilities at the airport consist of one asphalt runway and one turf runway, measuring 1,860 m (6,100 ft) and 975 m (3,200 ft), respectively, it has a single hangar but does not have a control tower. Airport operations fall under the control of Salt Lake City Air Route Traffic Control Center (ARTCC). This airport is unattended. Airport operational statistics (AirNav, September 2019) show that transient general aviation contributes 62% of operations and local general aviation contributes 38%. Great Falls, Helena, Bozeman, and Billings, MT have regional or international airports with regularly scheduled passenger services making these locations more suitable for travel to the Project area.

18.5 Water

18.5.1 Water Requirements

The Project will use both fresh water and treated water in its operation, and will require fresh water, potable water, process water and fire water. A nearby well will be the key source for potable water. As the Project becomes operational, the water required for the process plant, for first fill, make-up and fire water, will be sourced from the dewatering unit operations. Water will initially come from the dewatering of UG mine workings and subsequently recycled process water. This recovered water will be treated in the WTP and recycled within the circuit to the maximum extent possible.

At steady state, the operation requires 11.4 million Litres per day (Lpd) or 3 million gallons per day (gpd) of process water. The site requires 32,595 Lpd (8,575 gpd) or 23 Litres per minute (Lpm) (6 gpm) of potable water on average. The potable water demand requirement is based on 243 persons at the site and will cover the supply for drinking water, showers, laundry and restroom facilities. Additionally, there is a dedicated reservoir of 454,200 L (120,000 gal) of fire water in the bottom portion of the fresh water tank. This fire water reserve will meet a two-hour demand at 3,785 Lpm (1,000 gpm). These water supply tanks will be located adjacent to the WTP.

18.5.2 *Potable Water Supply*

The Project is proposing to source potable water by either converting Observation Well PW-6 into a permanent supply well, or by establishing a permanent well in the immediate vicinity of PW-6. Potable water will be pumped along a 1,800 m (5,905 ft) long buried pipeline that follows the powerline from the proposed Public Water Supply (PWS) pumping well, PW-6, to the potable water treatment system located in the WTP. This water will be chlorinated and filtered, as necessary, to meet drinking water standards. The 1.1 ha (2.7 acre) surface disturbance associated with this pipeline assumes a 6 m (20 ft) wide construction zone, which will be reclaimed immediately following construction of the pipeline. Treated water will be pumped to a covered potable water storage tank (on the west side of the mill facility) and to a potable water pump house at the truck shop complex. It will be distributed to various facilities, including the mill.

As of this report, SRA has not initiated the permitting effort with the MDEQ Public Water Supply and Subdivision Bureau for the potable water supply as the final design flow has not been confirmed to account for all potable water uses such as drinking fountains, toilets, showers, and laundry use. This water supply system will be completed in accordance with Circular DEQ-1, Standards for Water Works (MDEQ, 2014).

18.5.3 *Mine Dewatering Fresh Water Supply*

Groundwater modelling results for the Project indicate that the maximum average annual dewatering rate from the UG mine workings will be about 32 Lps (500 gpm). Excess water pumped from the workings will be sent directly to the WTP and will be discharged to the alluvial Underground Infiltration Gallery (UIG) under a Montana Pollution Discharge Elimination System (MPDES) permit. During normal operations, the Project will use about 32 Lps (500 gpm) on average for use in the plant, make-up water for the PWP, and miscellaneous freshwater needs e.g., dust suppression and truck wash. The plant will pump its water needs from the PWP.

SRA has initiated discussions with the Montana Department of Natural Resources and Conservation (MDNRC) and submitted an application in September 2018 for a groundwater beneficial water use permit (groundwater permit) for the use of groundwater at the Project. SRA is applying for a maximum flow rate of 32 Lps (500 gpm) for the beneficial use of groundwater to allow the PWP to be filled to operational levels during start-ups and periods of low water in the storage facility. The MDNRC issued a Preliminary Decision to grant the water right and change applications associated with the groundwater beneficial use permit on March 13, 2020. The final determination to grant the water right and change applications is expected in 2021 and depends on resolution of objections to the preliminary determination.

Process Water Supply

Process water will be comprised of recycled water from the PWP, the milling process, and as needed, treated water from the WTP. These water streams will be directed to the process water tank that is 12.5 m (40 ft) diameter by 14.0 m (45 ft) high with a storage capacity of 1,601,172 L (422,985 gal). Water from this tank will be distributed to the process plant and other process related service locations.

Fresh Water and Fire Water Supply

Fresh water will be derived from the WTP and piped to the combined fresh water/fire water tank located on the west side of the mill pad area. Treated fresh water will be used in the process for cooling, pump gland lubrication, reagent preparation, fire suppression and fire protection water.

The combined fresh water/fire water tank is 12.5 m (40 ft) diameter and 6 m (20 ft) high with a capacity of 711,627 L (187,992 gal). The dedicated fire water reserve is located in the lower portion of the tank and will be drawn from below the primary water supply nozzles. This reserve capacity will meet a two-hour demand at 454,200 L (120,000 gal).

18.5.4 Water Treatment

The WTP will be located on the west corner of the mill pad site and will be a Reverse Osmosis (RO) system which has been designed by Kimball (2017) and Kimball (2020) and repriced by IDE Technologies (2020) for the capital cost estimate. The design for water treatment is based upon the planned development of the Project, the water budget for the Project, the characteristics of the water that requires management, and the effluent treatment goals. Water will require management during three phases of the mine life cycle. Each phase will have different design flows and raw water quality. These phases are described below.

Construction Phase

Groundwater from mine dewatering and water collected in the CWP will require treatment at an estimated maximum flow rate of approximately 946 Lpm (250 gpm) during the second year of construction. This flow rate will not exceed the capacity of the water treatment plant, and storage will be used to provide steady state feed flow rates to the treatment system. During the latter part of the construction phase (between years 2 and 2.5), the flow rate may exceed 946 Lpm (250 gpm). If this situation occurs, the flow in excess of the 946 Lpm treatment capacity will be stored in the CWP or PWP and treated during the operational phase in the WTP. SRA will keep the construction phase RO treatment system on-site for standby purposes, providing an additional 946 Lpm (250 gpm) of treatment capacity if needed. Dewatered solids will be stored on-site for ultimate disposal in the tailings facility after construction is complete.

Operational Phase

The operational phase WTP will have an influent capacity of 2,226 Lpm (588 gpm), which is equal to the maximum annual average influent rate. However, not all of this water is treated by the RO system. Approximately 340 Lpm (90 gpm) will be delivered to the mill unit operations after the pretreatment stages to be used as gland water for pump lubrication. The remaining 1,885 Lpm (498 gpm) will be treated by the RO system, producing 1,536 Lpm (406 gpm) permeate water and 345 Lpm (91 gpm) RO brine concentrate. If higher flows continue for longer than anticipated, the excess water will be stored in the PWP, and additional treatment equipment for the front-end components will be rented or purchased to provide additional treatment capacity for long-term treatment. Treated water will meet discharge standards and will be discharged to the UG infiltration galleries, and liquid and solid treatment residuals will be disposed of on-site using the PWP and CTF, respectively.

Closure Phase

Prior to decommissioning, water from the PWP and other sources will be treated in the WTP using similar processes as the operational phase. The flow rate for water treatment during closure will be a maximum of about 1,900 Lpm (502 gpm), which is the capacity of the RO system plus water in the sludge removed in the clarifier ahead of the RO system. Treated water will be used to flush the UG workings (see Section 20.10.7 – Site Specific Closure Activities Related to Permanent Closure - for additional UG mine closure details). The brine section of the CWP will be used for brine storage prior to shipment off-site for disposal. Solid and liquid water treatment residuals will be disposed of off-site (see Section 20.10.6 – Permanent Reclamation and Closure Activities).

Water Management

The water management infrastructure will include all structures related to the diversion, conveyance, collection, and storage of surface water passing though the Project footprint.

Storm Water Control and Management

Prior to any surface disturbing activities, SRA is required to abide by the regulations included in the existing General Permit for storm water discharges associated with construction activities issued by the MDEQ's Water Protection Bureau (MDEQ, 2017b). SRA has also prepared a Storm Water Pollution Prevention Plan (SWPPP) designed to protect State waters from pollutants, which during construction are principally sediment. There are three major components to the SWPPP:

- Assessing the characteristics of the site such as nearby surface waters, topography, and storm water runoff patterns;
- Identifying potential sources of pollutants such as sediment from disturbed areas, and stored wastes or fuels; and

- Identifying Best Management Practices (BMP's) which will be used to minimize or eliminate the potential for pollutants to reach surface waters through storm water runoff.

The storm water management for the Project is using an integrated approach designed by Hydrometrics (2018) that has been applied at other mines in Montana. Early storm water discharge associated with mine construction activities will follow regulations in the General Permit as described above (including having a SWPPP). After the mine construction is complete, the storm water discharge regulations will be transferred to a General Permit for Stormwater Discharges Associated with Industrial Activities (required at active mine sites in Montana).

The majority of storm water runoff at the site will be controlled by diversion around disturbed soils. Diversion structures will consist of drainage ditches or swales, spreaders, sediment traps, rock berms, straw wattles, and slash windrows. Drainage structures will be sized to safely convey the 24-hour, 1 in 100-year storm event. All storm water controls will be constructed prior to, or in conjunction with, soil removal and stockpiling. Section 20.8.5 – Storm water describes additional details of storm water management at the Project.

Non-Contact Surface Water Diversion Structures

The primary objective of the diversion ditches is to maximize the collection of non-contact runoff from the catchments upstream of mining disturbances and convey it around these facilities for downstream discharge. The diversion ditches reduce the amount of runoff contributing to the mine facilities by diverting their respective upstream catchments. Separation of contact and non-contact waters reduces the downstream impact of mining activities, and it is more economical to minimize the quantity of water that requires treatment by physical, chemical or biological means. This also reduces the capacity required in the facilities to meet storm water Probable Maximum Precipitation (PMP) and Probable Maximum Flood (PMF) storage requirements. Diversion of non-contact water also reduces flow impacts downstream of the Project.

A number of diversion ditches will be established during the initial construction phase to divert non-contact water runoff. These channels are typically trapezoidal or V-shaped in cross-section. The majority of the diversion channel slopes are going to be excavated into weathered bedrock, and therefore should not have any issues with long term stability. Portions of the diversion channel will be constructed from fill; however, they are going to be constructed using proper fill placement, compaction, and armoring with rip-rap to ensure long term stability.

The network of diversion structures will include long term fixed and temporary diversion ditches and channels. All sections of the diversion ditch system for the CTF and PWP are designed to carry the predicted peak flow generated during a PMF event. The diversion channels for the WRS, BCWP, and NCWR are designed to carry predicted peak flow for a 1 in 200-year storm event.

Contact Water Collection Structures

Contact water will be intercepted in collection ditches down gradient from the mill pad, WRS, ROM storage, portal pad and other areas that have been disturbed by construction and mining activities. These facilities are optimally located to facilitate drainage by gravity. They consist of side-hill ditches that intercept overland sheet flow. Runoff collected by these collection ditches is considered contact water and is directed to the CWP facilities located at a topographic low point that facilitates gravity drainage. The function of this pond is described in Section 18.8.5 Brine/Contact Water Pond.

Similar to diversion ditches, both permanent and temporary collection ditches will be required.

Temporary Contact Water Collection Ditches

During construction and through expansion phases of mine operations, temporary (generally consisting of six months to a year) collection ditches will be required. Construction and maintenance of these structures would be consistent with that of permanent collection structures.

18.6 Power Supply and Distribution

18.6.1 Utility Connection

The planned source of primary electric power for the Project will be from the NorthWestern Energy (NWE) grid. NWE currently generates most of the electricity supply surrounding the vicinity of the Project. It sells wholesale power to rural cooperatives such as Fergus Electric Cooperative Inc. (FEC), or directly to customers in several cities, plus directly supplies large industrial customers. NWE's primary source of power generation is from several hydro-generation facilities. This supply is supplemented by a natural gas unit, as well as a wind generation station. NWE owns, operates and maintains the Kings Hill – Martinsdale 100 kV transmission line where the Point Of Interconnection (POI) will occur. Fergus Electric Cooperative Inc. (FEC) is the nominated co-op wholesaler for the supply of electricity to the Project.

As of this report, SRA and FEC have negotiated the terms of a Letter Of Intent (LOI) dated March 17, 2020 for the provision of electric service to the Project. The LOI outlines the agreed-to arrangement for the transmission and delivery of wholesale power and energy to the Project at the respective points of interconnection. From the power supply side, NWE will construct and install a substation from where FEC will tie-in and construct its transmission line to the mine property primary substation. The Project initially requires 16 MW of power for 10 years. FEC is responsible for establishing and obtaining; the Right-Of-Way (ROW), the engineering, design and installation of the approximately 58 km (36 mile) of 100 kV overhead single-pole structure power transmission line connection. The Project will bear the costs of the NWE POI, as well as the FEC transmission line.

The power supply line will enter onto the Property from the existing ROW adjacent to the Sheep Creek County road, into the permit boundary area just east of the present day core sheds. The powerline would then continue along a new ROW parallel to the Project's main access road, and from there to the electrical substation along the north side of the mill pad. The total length of the powerline within the permit boundary to the substation is 2.75 km (1.7 miles). With the exception of a surface distribution powerline to the PWS well measuring 1.8 km (1.12 miles), all other power distribution lines are to be buried.

The primary substation will step-down the voltage from 100 kV to 13.8 kV, which is suitable for the UG and surface uses at the site. Overhead power poles will be single pole and designed to be avian safe with insulated conductors at cross arms. These overhead power poles will be spaced approximately 150 m (492 feet) apart. The estimated total site wide power line length is 4.55 km (2.82 miles), including the line to the PWS well. The estimated number of power poles required for distribution is approximately thirty.

18.6.2 Mine Site Substation

A 100/13.8 kV switchyard will be established on the site adjacent to the process plant area. The switchyard will consist of a landing structure, surge arrestors, 100 kV outdoor switchgear, combined disconnect/earth switches, SF6 circuit breakers, potential and current transformers, and a single 25 MVA transformer. The yard will be fenced and appropriately earthed and fitted with overhead shielding to protect against lightning strikes. The transformer will feed an indoor metal enclosed 13.8 kV switchboard, installed in a switch room located close to the grinding mills. Power factor correction and voltage support capacitors will be installed to ensure that the transmission system can be operated within the North American Electrical Reliability Corporation (NERC) voltage limits.

18.6.3 13.8 kV Power Distribution

Power will be distributed throughout the site at 13.8 kV, generally via buried cables feeding pad-mounted transformers. Reticulation to grouped loads, such as buildings, will generally be via low voltage buried cables from a nearby substation. The SAG mill variable speed drive, ball mill and regrind mill will all be supplied at 13.8 kV. Substations throughout the site and the UG mine will also be supplied at 13.8 kV.

18.6.4 Construction and Emergency Power Supply

Two, diesel-fuelled, Tier 4 certified and EPA emissions compliant generator sets (a 545 kW and a 320 kW) will provide power on the portal pad prior to the primary substation coming online. Other (320 to 1,800 kW) trailer-mounted mobile generators will be used around the Project site to support specific construction projects. These generator sets are temporary and will be replaced by permanent sets for emergency power supply to support the operation in the event of an outage.

These replacement generator sets would be two 2,500 ekW (site-rated: 2,135 kW), diesel-fuelled, U.S. EPA certified for stationary emergency use only (Tier 2 nonroad equivalent emission standards), low fuel consumption certified generator sets, located adjacent to the primary substation that will provide operational back-up emergency power. The electrical system would be configured to allow the diesel generators to be synchronized to the grid, to allow for regular load testing. These generators will have weatherproof enclosures that also provide a sound attenuating design, heaters, 13.8 kV generators, 1,200 A/13.8 kV circuit breakers with lightning and surge arrestors and neutral grounding resistors. The most critical power loads will be connected to this emergency supply that include firefighting equipment/pumps, mine ventilation exhaust fans, flotation cell drives and paddles, thickener rakes, reagent agitators/pumps, paste transfer pumps, emergency lighting, communications and electrical heaters.

18.7 On-Site Ancillary Surface Facilities

18.7.1 Buildings and Structures

The on-site buildings and structures vary in use but will generally provide workspaces and change room accommodations for the employees at the Project. These buildings and structures will be in one of two different locations on the property: at the mill pad or at the portal pad areas. Table 18.2 identifies the key buildings on-site.

ID	Facility Function	Location	Area	Reference Drawing
1	Administration Building & Mine Dry	Mill Pad	1,680 m^2	300071-L-020
2	Heavy Vehicle Shop	Mill Pad	910 m^2	300071-L-017
3	Tire Changing Pad	Mill Pad	850 m^2	TBD
4	Equipment Wash Pad	Mill Pad	440 m^2	TBD
5	Warehouse	Mill Pad	TBD m^2	TBD
6	Mill Maintenance Shop	Mill Pad	TBD m^2	300071-L-018
7	Bulk Fuel Depot	Mill Pad	665 m^2	TBD
8	Laboratory	Mill Pad	585 m^2	300071-L-011
9	Reagent Storage	Mill Pad	157 m^2	300071-L-016
10	Power Station Fuel Yard	Mill Pad	TBD m^2	TBD
11	Water Treatment Plant	Mill Pad	TBD m^2	TBD
12	First Aid Facility (included with Admin Building)	Mill Pad	60 m^2	300071-L-020
13	Entry Station	Mill Pad	36.7 m^2	300071-L-015

Table 18.2 Buildings and Structures

Construction and Code Requirements

Buildings will be pre-engineered construction with structural steel frames, steel girts, purlins and intermediate structural members. Walls will be constructed of insulated metal wall panels and the roof will be a metal standing seam roof system. Each building will come complete with doors, windows, Heating Ventilation and Air Conditioning (HVAC), electrical, plumbing, fire detection and suppression systems.

Buildings and structures will be constructed to codes and standards which are presently in force and adopted by Montana's Department of Labor and Industry.

Administration Building/Mine Dry

The administration building and mine dry are combined into one building to minimize the amount of disturbed land and to promote communication between the various operations teams. This structure is designed as a single-story building. It is sized to accommodate key personnel from each of the operations groups. The administration space includes eight offices, 51 cubicles, reception area, conference room, copy/printer areas, break room, IT/communications room, storage, and rest room facilities. Building facilities are designed to accommodate the largest expected staff and guest presence. The estimated dimensions of this area of the building are 37.4 m x 22.5 m giving a footprint area of approximately 840 m^2 (9,040 ft^2).

The mine dry is designed to meet the needs of a diversified work force and provide storage, changing and washing facilities for both the mine and process operations teams, technical and administrative staff, and guests. It has dedicated facility accommodations for a combined male and female workforce, small and medium meeting rooms, a mustering area for pre-shift operational and safety discussions, laundry facility, mechanical room, and storage space. The locker/dry/shower facilities can accommodate 150 shift personnel, 50 male and 25 female workers per shift; additionally, there are locker and shower facilities for 100 technical and administrative staff and guests, 70 male and 50 female. The approximate overall dimensions of the dry portion of the building are 37.4 m x 22.5 m providing an estimated footprint area of 840 m^2 (9,040 ft^2).

Included within the mine dry footprint is a first aid treatment center. This part of the building is 30 m^2 (323 ft^2). The facility provides a first aid examination room, consultation office and restroom facilities. It is located in close proximity to the mine and process areas so that medical attention can be administered very quickly following an incident.

Laboratory

The laboratory building is approximately 39 m X 16 m for a total of 585 m^2 (6,300 ft^2). The building is split into wet and dry work areas with an airlock passage separating each. It will house rooms for sample receipt and storage, sample preparation and drying, weighing, assaying, and testing facilities with supporting mechanical/electrical rooms and chemical storage facility. There will be additional office space, break room and rest rooms for the plant metallurgist, metallurgy and lab supervisors and personnel. The building is maintained with a dust collector covering the sample preparation area and dedicated HVAC units between the wet and dry sections.

Heavy Vehicle Shop and Storage

The heavy vehicle workshop is sized to perform minor repair service to the mine equipment. Major repairs will be performed off-site. The workshop will be outfitted with a five-bay maintenance area to repair and maintain the mine fleet and other mobile equipment. Three bays will be devoted to heavy mobile equipment repairs and maintenance. One bay will be configured to accommodate electrical maintenance and welding repairs. One bay will be devoted to accommodating light vehicle repairs and maintenance. The end of the building houses the lube storage and waste oil collection facilities and a separate designated space is provided as a maintenance tools and materials stores area. There is a space allotted for a supervisor's office, literature and record library, and restroom facilities. The workshop is nominally 54 m x 42 m, for a total of 2,270 m^2 (24,400 ft^2). A transformer, electrical room and compressor room will be in separate buildings constructed adjacent to the workshop.

In the vicinity of the mine workshop is a tire servicing area and storage laydown. The tire change is a 23 m x 37 m (75 ft x 121 ft) slab. Storage will be a fenced compound that will store bulk items and equipment as necessary.

Heavy and Light Equipment Wash Pad

This is approximately a 22 m x 20 m (72 ft x 66 ft) concrete pad and wash rack. The pad is split into two lanes on either side of the rack to accommodate heavy mine equipment along one lane and light truck units in the other. This facility includes water supply, tanks, pumps, and dirty water collection and processing. The pad will be near to the mine workshop to allow equipment to be cleaned before being moved into the workshop for service.

Entry Gate and Weigh-Scale

An entry area including a guard house and weigh-scale will be located at the main entrance to the property for traffic and inventory control. The gate house is approximately 6 m x 6 m (20 ft x 20 ft).

18.7.2 *Fuel, Lubricants and Dispensing Station*

Diesel Fuel

The Project will use diesel fuel primarily for the mine trucks, service trucks, emergency generators and the fire protection system. Gasoline will also be used on-site to support other surface vehicles. The Project will store a one week supply of diesel fuel using two, double-walled, Aboveground Storage Tanks (AST's) with a total combined capacity of approximately 100,000 L (26,000 gal). There will be several smaller diesel storage tanks placed adjacent to their applicable diesel-powered equipment. The two large diesel storage tanks and gasoline tank will be located in the primary fuel farm area situated on the west side of the heavy equipment workshop. The planned fuel-storage tanks for the Project start-up phase will include:

- 50,000 L (13,000 gal) double-walled tank (diesel);
- 50,000 L (13,000 gal) double-walled tank (diesel);
- 1,900 L (500 gal) double-walled tank (gasoline);
- 15,140 L (4,000 gal) tank (diesel generator);
- 950 L (250 gal) tank (diesel fire pump).

A fuel dispensing station will be located adjacent to the primary fuel farm to allow for fuelling of both heavy and light vehicles. All fuel tanks and the fuel dispensing station will be located in a spill containment area on concrete pads. The spill containment capacity will be no less than 110% of the total tank storage capacity. The fuel dispensing will be equipped with automatic shut-off mechanisms to prevent over-fuelling and spillage. Manual fire suppressant equipment will be installed at the fuelling station.

Additionally, a fuel and lubricant truck will be used to dispense fuel to mobile equipment and the smaller dedicated service diesel storage tanks.

Lubricants

Various oils, lubricants and anti-freeze necessary for servicing the mine operations equipment will also be stored on the property adjacent to the heavy equipment workshop. These fluids will be stored in refurbished cargo containers with a gross floor area of 100 m^2 (1,070 ft^2). This storage facility will house approximately 7,500 L (2,000 gal) or a two-week supply of lubricants, coolants and waste fluids. The surroundings will be protected from spills via a HDPE-lined, bermed containment storage facility.

The lubricant storage facility will be furnished with loading/unloading arms and pumps. This storage facility will also contain air-operated transfer pumps for supplying lubricants to the dispensing reels located in the nearby heavy equipment shop service bays.

Waste oil and spent coolants will be picked up and removed by a licensed hazardous waste disposal contractor for recycling or off-site disposal.

Propane Storage

Propane tanks will be used to temporarily store propane on the construction portal pad to seasonally heat mine air prior to the completion of the first ventilation raise. Operationally, propane will be stored at the ventilation raises once they have been constructed, to seasonally supply the heaters to the UG fresh air intakes.

18.7.3 Explosives Storage

The main explosives storage facility is located UG off the main ramp and is protected from vehicle traffic. This facility is equipped with a 4.5 t (5 ton) jib crane that will facilitate delivery of explosive products from raised platforms to the ANFO loaders.

18.7.4 Sewage Treatment

Domestic wastewater from lavatories, toilets and showers on the surface mine site will be treated and disposed using a subsurface effluent discharge system. The main system will be composed of a collection system, septic tank, distribution box, manifold, and a series of subsurface networks for effluent distribution. The Montana Standards for Subsurface Wastewater Treatment Systems, Circular DEQ 4, 2013 will serve as the guide for the design and sizing of this system.

The area designated for this system is predominantly due south of the mill pad and administration/dry building facility locations. A pumped flow method of effluent distribution will be used to discharge from the septic tank to the subsurface treatment system.

The wastewater volumes generated at the site surface are determined based on the workforce size per day and wastewater flow values as presented in Montana's Circular DEQ-4. The resulting average daily wastewater flow from the site is 29,440 Lpd (7,740 gpd). The average daily wastewater flow by facility is shown below in Table 18.3.

Facility	Wastewater Flow per Capita		Persons (Max)	Total Wastewater Flow	
	gpd	Lpd		gpd	Lpd
Administration Building	13	50	68	890	3,400
Mine Truck Shop	15	57	20	300	1,140
Mill Maintenance Shop	15	57	20	300	1,140
Laboratory	15	57	15	225	860
Mine Dry	50	190	120	6,000	22,800
First Aid Facility	13	50	2	25	100
TOTAL				**7,740**	**29,440**

Table 18.3 Wastewater Flow Estimate

Based on the expected rate of effluent flow from the septic tank and online soil infiltration data (which are consistent with data collected during siting studies for mine dewatering UIG systems), approximately 1,325 m^2 (14,250 ft^2) of absorption field will be required. The Project will need to construct a trench field measuring approximately 165 m x 61 m (540 ft x 200 ft) to provide the necessary absorption capacity.

18.7.5 Fire Protection System

The firefighting system is composed of one electric fire water pump, one diesel fire water pump, a jockey pump, and controls will be installed. Dedicated fire mains with hydrants will be provided at the process plant, ancillary buildings, truck shop complex, and primary crushing area. Fire extinguishers will also be provided throughout the facilities. Fire hose reels and cabinets will be installed throughout the process plant building and truck shop. Sprinkler systems will be installed in the warehouse, the main office, and the truck shop complex. Fire alarm systems will report to the plant control room, which will be manned 24 hours a day.

The Project will also have a fire truck on-site to contend with fires that may occur.

18.7.6 Communications

Communications to the Project will be provided via a new fibre cable to the facilities. The cable will be spliced to the existing network cable that passes the site on the north side of Sheep Creek County Road and will be buried adjacent to the site access road to a communications shelter at the plant site. WAN & LAN, the Telephone System and IT needs will be distributed from the communication hut to the rest of the facilities.

18.7.7 Paste Plant

Thickened plant tailings from the tailings thickener will be pumped to a 300 m^3 capacity paste filter feed tank. Variable speed paste filter feed pumps will transfer the material at a feed density of 55% to 60% solids to two conventional vacuum disc filters, with a third filter installed as a standby unit. The design has allowed for 85% of the thickened tailings to be fed to the filters, with the remaining 15% of the underflow being recombined with the filter discharge and binders at the paste mixer to generate the paste and avoid the requirements of additional water dilution.

The paste will then be pumped via duty and standby pumps and insulated pipelines to either the mine for UG paste backfill or to the CTF for deposition. A tailings production profile is presented in Section 20.7.1.

18.7.8 Site Security

A four-strand barbed-wire fence will surround the core Project facility area. The purpose of this fence is to keep cattle out of the active mine areas while allowing property owners maximum use of their land for cattle grazing. The fence will be placed on either side of the main access and construction access roads to lane them off for worker, vehicle, and cattle safety. The main access road to the Project will have a lockable gate at the Sheep Creek county road. This gate will remain open during all active periods of operation. The main access road will have another gate and a guardhouse on the SE end of the mill facility pad. The guardhouse will be located along the barbed-wire perimeter fence described above. This gate will be either monitored or manned 24-hours per day during operations to control entry onto the site. Because the operating schedule is anticipated to be seven days per week, 365 days per year, mine staff will provide security. In the event of a longer term closure the company may elect to hire a watchman.

All lined ponds including the CWP, PWP, TWSP, CTF and the CTF and PWP foundation drain ponds will have 2.4 m (8 ft) high wildlife fences constructed around them. The NCWR will have a barbed-wire fence surrounding the reservoir to prevent cattle from damaging the shoreline and diversion ditches. Chain-link fences, 1.8 m (6 ft) tall, will also surround the four ventilation raise collar areas and the PWS well location.

18.8 Other Surface Infrastructure

18.8.1 Construction Laydown Area

During the construction phase of the Project, at least three construction laydown areas will be available for use by the contractor. A 1.3 ha (3.1 acre) construction laydown area will be provided on the west end of the mill facility. A temporary construction concrete batch plant will be located on the NW corner of this pad. Sealed concentrate containers may be stored on this pad while awaiting shipment during operations. Spare parts and materials that do not require protection from the elements will also be stored in the laydown area. After the major construction effort is completed this laydown area will be removed and replaced with the paste plant facility. Additional laydown areas will be available for use either at the WRS facility site (4.1 ha, 10.2 acres) and near the eastern margin of the borrow pit (1.0 ha, 2.5 acres).

18.8.2 Portal Pad

The portal pad will be one of the early structures constructed at the site and was designed by Tetra Tech (Cawlfield, 2020). Initially the pad will support pre-production mining efforts for the development of the primary decline ramp and subsequently it will support mining operations. During the pre-production time frame the portal pad will be used for staging temporary facilities that support the pre-development work. These temporary facilities will include an office/dry/change house; a mobile equipment maintenance garage/wash-pad/fuel storage and fuelling station building; shop/warehouse with a power supply area (with generators and air compressor); storage facilities; a potable water tank; a sanitary holding tank; a temporary propane tank; laydown area; and parking for equipment and personnel.

As mining operations are initiated, these facilities will be replaced by permanent structures that will be constructed on the mill pad location of the site. During the remainder of the mine operation, the portal pad will be an open surface upon which mine trucks and other vehicles circulate, with designated drive, parking, and equipment laydown areas.

The portal pad will have a total disturbance area of approximately 2.8 ha (6.9 acres) and is designed to have a finished upper working surface area of about 2.0 ha (4.9 acres). The flanks of the portal pad will consist of compacted, 2.5:1 H:V fill slopes. The pad will require both cuts into undisturbed ground and compacted fills to create a nominal level working surface area. A safety berm will be constructed along the south edge of the pad with a low point located in a topographic re-entrant on the upper surface of the pad. During mine operations, the entire surface area of the portal pad will be compacted road-base/gravel, or small, local concrete slabs. The western side of the portal pad near the jaw crusher stockpile will contain a single HDPE 60 mil (1.5mm) liner underlain by 15 oz/yd2 non-woven geotextile.

All portal pad surface area drainage will be collected as contact water and directed to the CWP. The top of the portal pad will be constructed with a gently sloping upper surface to the south that allows for gravity-flow drainage. Lateral flow along the south edge of the portal pad's upper surface will be directed in lined ditches along the safety berms to the reentrant where it will enter a lined slope drain. A diversion ditch constructed from the toe of the fill materials will route runoff to the water section of the CWP. A separate run-on diversion ditch to be located north of the portal pad will be constructed early in the Project.

18.8.3 Waste Rock/ROM Stockpile

The WRS stockpile was designed by Knight Piesold (Magoon et al, 2017) and can store up to a maximum of 500,000 t (551,155 tons) of waste rock. A total of 884,344 t (974,822 tons) of UG waste rock will be generated over the life of the mine, all of which will eventually be placed in the CTF. A proportion of the UG waste rock has the potential for acid generation and metal leaching and therefore the MOP requires that all waste rock be treated as potentially acid generating (see Section 20.2.4 – Environmental Geochemistry). All UG waste rock will be co-disposed with the tailings in the CTF during mining operations. The WRS stockpile was originally designed to be a temporary facility to be used early in the mine construction and development for approximately 2.5 years. However, the Final EIS (MDEQ, 2020) recommends the Agency Modified Alternative (AMA) to make the WRS facility stay open through the life of the mine in order to store enough waste rock to generate 81,785 m^3 (106,971 $yards^3$) of cemented tailing paste for use as backfill in certain UG mine workings in closure (see UG mine closure details in Section 20.10.7 – Site Specific Activities Related to Permanent Closure – below for additional details).

Waste rock on this stockpile will be stacked over a 4.1 ha (10.2 acre) area to a maximum height of 15 m (50 ft) in 5 m (16 ft) lifts with 2.5 m (8 ft) benches. Additional waste rock material will be placed on top of the upper liner in areas where mine trucks will establish routinely travelled access ramps onto the pad.

The WRS pad will have a 100 mil (2.5 mm, 0.1 inch) HDPE geomembrane with a minimum 300 mm (12 inch) thick layer of excavated and prepared sub-grade bedding material comprised of waste rock above the liner and granodiorite below it for protection from the mine fleet traffic during waste rock placement.

Approximately 294,988 t (325,168 tons) of waste rock generated during production mining will be placed directly into the CTF as lenses of waste along the flanks of the CTF haul ramp and comingled with and covered by cemented paste tailings.

The WRS pad and liner will be sloped to the south towards an outlet pipe along the south edge of the pad. This outlet pipe will transfer collected precipitation and seepage through the waste rock pile from the top of the liner to the double-lined CWP adjacent and to the south of the mine portal pad. Water from the WRS pad will be gravity drained to the CWP by a 200 mm Solid NT12 PE pipe. Water will be temporarily stored within the CWP during the period of use of the WRS pad, prior to its automatic pump transfer to the WTP.

18.8.4 Process Water Pond

The PWP has been engineered by Knight Piesold (Magoon et al, 2017) and will store water needed for milling, paste backfill and cemented tailings operations. Water stored in the PWP includes direct precipitation, contact water, water from the mine dewatering, water from the CTF water reclaim system, and water separated in the thickening process. If there is excess capacity, the overflow will be pumped directly from the PWP to the RO water treatment facility for treatment and then released to the UIG's. The PWP is located NW of the CTF. The PWP is sized to store approximately 420,000 m^3 (549,340 $yards^3$) with operational capacity of 120,000 to 200,000 m^3 (157,000 – 262,000 $yards^3$). The additional 220,000 m^3 (288,000 $yards^3$) will allow storage of surface water from storm events.

The pond is designed as a double-lined facility composed of two layers of 100 mil HDPE geomembrane with a geonet layer in between. Any seepage into the geonet will be directed via gravity to a sump and pump reclaim system situated at a low point in the pond basin. An underlying sub-grade bedding layer will be installed to protect the lining system.

The embankment of the PWP is proposed to be homogeneous rockfill embankment with upstream and downstream slopes of 2.5H:1V. The crest width is proposed to be 10 m to allow working space for pipelines and traffic. The embankment height varies, up to a maximum of approximately 23 m. Most of the embankment fill is to be sourced from the PWP basin. The material is expected to consist of fresh to moderately weathered rockfill. The embankment is designed with a 2 m freeboard.

18.8.5 *Brine/Contact Water Pond*

The BCWP has been engineered by Tetra Tech (Cawlfield, 2020) and is a single pond which consists of two cells separated by a central berm and identified as brine cell and contact water cell. The primary purpose of the BCWP is to store the brine by-product generated from the RO water treatment process, to store captured water that was in contact with disturbed areas (mill, portal pad, and temporary waste dump catchment areas, etc.) during the construction, pre-production and operations phases, and initially during closure. Each of these generated waters will be transferred and stored within their respective cell of the BCWP. Storage of both the brine and contact water in these cells is temporary.

The brine will be added to the tailings thickener as means of disposal within the cemented paste material either UG or in the CTF. The effect on concrete properties from high concentrations of chloride, sulphate, and other deleterious ions in the brine are expected to be minor with no effect to the final strength or structure of the cemented tailings. The contact water will be pumped to the water treatment plant for treatment and released via the UG infiltration gallery.

The brine cell is located on the west side of the BCWP and will utilize a double-liner system with an integrated seepage collection and recycle system. The design capacity is 21,000 m^3 (5.5 M gal) of brine storage with an additional capacity of 4,000 m^3 (1 M gal) for operational variances or direct precipitation from a 1 in 200-year storm event. A 1 m freeboard allowance is included above the maximum water level.

The contact water cell is located on the east side of the BCWP and it is a double-lined cell. This cell has a design capacity for 70,000 m^3 (18.5 M gal) of water. 35,000 m^3 of this storage is for captured water from a 1 in 200-year storm event. The remaining 35,000 m^3 is provided to store water from UG mine dewatering in the event of a surge or if the water treatment plant is not operational. A 1 m freeboard allowance is included above the maximum water elevation.

18.8.6 *Non-Contact Water Reservoir*

The NCWR facility was designed by Knight Piesold (Magoon et al, 2017) and is part of the appropriation permit requirements in SRA's mitigation plan to address depletion of surface water flows associated with the consumptive use of groundwater in its operations. The NCWR will store water during the irrigation period of the year to off-set consumptive use during the non-irrigation months of the year. The NCWR has been designed with an average seepage rate of approximately 273 m^3/d (50 gpm). For the purposes of the MOP application, the capacity of the NCWR was established as 360,000 m^3 (470,860 $yards^3$) of water on an annual basis. The MDNRC will review SRA's draft mitigation plan and will determine the ultimate volume of water that needs to be stored in the NCWR.

The embankment of the NCWR will be a homogeneous rockfill embankment with upstream and downstream slopes of 2.5H:1V. The crest will be 10 m in width to allow for maintenance traffic. The embankment fill will be sourced from the excavation and shaping of the CTF basin. The upstream slope of the embankment will be lined with 100 mil HDPE geomembrane to minimize seepage into the embankment. The natural slopes of the basin will remain unlined.

The NCWR design includes a spillway at the south abutment with the purpose to prevent overtopping of the embankment and to safely route a 1 in 200-year design storm event through the NCWR and discharge it to the downstream wetlands. Additionally, a diversion channel is incorporated upstream of the NCWR which will intercept and divert runoff around the facility and discharge to the environment, downstream of the embankment. The channel is sized to safely convey a 1 in 100-year storm event.

18.8.7 Treated Water Storage Pond

During the evaluation of the water management plan in the EIS process, it was determined that a TWSP was proposed as a contingency. The purpose of the TWSP is to store treated water from the water treatment plant when this water does not meet the seasonal effluent limits for total nitrogen required by the surface water discharge permit. The TWSP is proposed to remain operational throughout the life of the mine.

The TWSP was designed by Hydrometrics (Zieg, 2018; Hydrometrics, 2018a) and is a lined facility located to the east of the CTF. The pond has been designed to store up to 204,000 m^3 (54 M gal) of treated water plus 5,400 m^3 (1.4 M gal) of additional capacity to contain direct precipitation from a 1 in 200-year storm event providing sufficient temporary storage capacity between the months of July to September. A 1 m freeboard allowance is included above the maximum water elevation.

18.8.8 Cemented Tailings Facility

The CTF was designed by Knight Piesold (Magoon et al, 2017) to store approximately 55% of all tailings generated by the mill over the active mine life and 100% of waste rock produced by the mining effort. The CTF has a storage capacity of 4.3 M m^3 (5.6 M $yards^3$). This capacity includes 3.56 M m^3 (4.7 M $yards^3$) of cemented tailings and 350,000 m^3 (460,000 $yards^3$) of waste rock material. The additional capacity is for temporary storage of storm water from a PMF event.

The CTF basin will have a double-liner system comprising a geonet layer between layers of 100 mil HDPE geomembrane. A foundation drain system will be located beneath the CTF liner system to collect groundwater flow and potential seepage beneath the CTF. This water will flow, by gravity, to a foundation drain collection pond located downstream and east of the embankment. Additionally, an internal basin drain will be located above the liner system to collect the tailings bleed water in the water reclaim system and seepage collection sump where, a reclaim pump system is for the transfer of collected water to the PWP.

Water from storm events will be temporarily stored in the CTF and later transferred to the PWP, as available storage capacity allows.

The embankment of the CTF is to be constructed with homogeneous rockfill material with upstream and downstream slopes of 2.5H:1V. The embankment is to have a crest width of 10 m (33 ft) to allow working space for pipelines and maintenance traffic. The maximum embankment height is approximately 46 m (150 ft) downstream side. The embankment fill is to be sourced from the CTF basin. The material is expected to consist of fresh to moderately weathered rockfill.

18.9 Site Support Equipment

Site support equipment provides support to mining and process operations at the Project site. A list of site support equipment is provided in Table 18.4. The support equipment is based on similar fleet sizes and equipment types utilized at other mining operations.

Equipment Description	Quantity
Light Vehicles & Personnel Movement	
Pick-up Trucks	8
Site Services Equipment	
Hiab Truck – 5 T	1
ROM Loader (CAT980)	1
Skid Steer Loader	1
Concentrate Loader (CAT950)	1
Forklift – 3T (Warehouse)	1
Mobile Crane Truck – 20 T	1
Misc. Small Equipment	
Mobile Air Compressor (5,200 L/min @ & Bar)	1
Diesel Welder – 7.2kVa 180 amp	2
Light Plants	4
Crane	
Crane (55 T)	1
Rescue Vehicles	
Ambulance	1
Fire and Rescue Truck	1

Table 18.4 Site Support Equipment

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19. MARKET STUDIES AND CONTRACTS

19.1 Introduction

Cu concentrate produced from the Black Butte Copper Project mining and milling operation will be transported off site using trucks and rail to an available west coast port, and shipped to Cu smelters for further refinement or a blending facility. The marketing study further contemplates options for sales to smelters in North America.

19.2 Market Studies (Strategy)

Wood Mackenzie (2020) produced a Cu market assessment report that includes a smelter study and Savage Services (2019) produced a Cu concentrate supply chain report to support the FS. Most of the information presented below is from these two studies.

19.3 Concentrate Specifications

Based on Wood Mackenzie's analysis, the Cu concentrate that was produced during metallurgical test work for the FS is a medium Cu grade product. It will likely attract premium to long-term Treatment and Refining Charges (TCRC's) because of penalties on deleterious elements. With an estimated analyzed As grade of 0.69%, the Cu concentrate product exceeds maximum Chinese import limits of 0.5% As and cannot be sold into China as a standalone product. However, given the comparatively low content of other deleterious elements, the attracted penalty can be minimized by blending with a concentrate that is low in As, Bi and Hg but high in other impurities.

The two main markets available to the Cu product are to smelters with capacity to treat high As bearing concentrates and to the blending market via traders.

The concentrate specifications shown in Table 19.1 are based upon metallurgical test work carried out during the FS.

		76% UCZ + 24% LCZ Blend Composite		
Element	Unit	Analyzed Result	Minimum	Maximum
Cu	%	21.8	20	26
Au	g/t	0.21	0.1	0.3
Ag	ppm	17	9	53
As	ppm	6,900	3,000	12,000
Al	%	0.11	0.05	0.12
Al	%	0.11	0.05	0.12
Sb	ppm	215	200	539
Bi	ppm	250	50	350
Cd	ppm	6	4	8
Cl	%	0.01	<0.01	0.03
F	%	0.01	<0.01	0.02
Pb	ppm	815	402	3,200
Hg	ppm	5.2	<1	9
Co	ppm	1,090	764	1,880
Se	ppm	5	<0.8	12.2
Si	%	1.02	0.36	1.28
Zn	ppm	548	350	980

Table 19.1 Expected Cu Concentrate Specification

19.3.1 Concentrate Off site Logistics

As requested by SRA, Savage Services (2020; 2019) have developed and evaluated the estimated cost for a truck-to-rail model with the transfer of containers from truck-to-rail ('Transloading') taking place in either Townsend, MT or Livingston, MT. In evaluating these two options, Savage Services has determined that Townsend is the optimal location for a Transload terminal and estimated this complete supply chain to cost $105.20/t to the Port of Everett. This price reflects the cost determined by the Savage Services team as a feasible and efficient supply chain solution designed to optimize capital assets and safeguard against bottlenecks and/or supply chain failures.

In summary, Savage Services (2020; 2019) determined the primary components illustrated in Figure 19.1 for the Black Butte Copper Project site concentrate logistics solution.

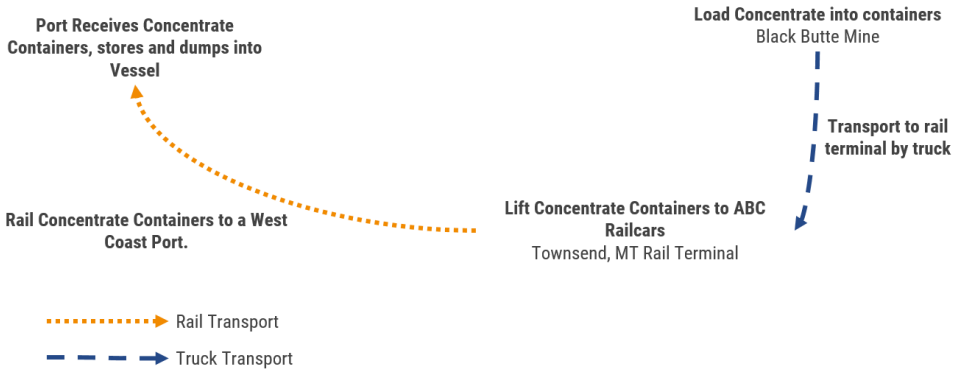


Figure 19.1 Components of the Truck/Rail Model

Mine-Site Loading

Truck loading at the mine will include the following activities and will be performed entirely by mine labor and equipment:

- Removing lids from containers,
- Loading concentrate into the containers (to a 24.4 t target),
- Placing lids on the loaded containers.

Upon loading at the mine, it is expected that containers will remain fastened to the pin trailer, meaning there is no need for container lifting equipment at the mine itself (not to be confused with container lid lifting equipment). For this to be true, it has been assumed that mine loading can take place upon the arrival of empty containers (i.e. “just in time”). All product loaded into containers will need to be weighed at the mine to prevent the need for double handling further down the supply chain. To maintain the high environmental standards of the mine, it is critical to Savage Services that once a container is closed and latched there will be no handling of the Cu concentrate until loading of a vessel at a west coast port (i.e. only the handling of containers). To ensure this is the case, it is vitally important that mine labor have the equipment and tools necessary to verify 23.6 t (52,000 lbs) of concentrate is loaded into a single container, with little variance (preferably +/- 1%).

Road Haulage Route

The haul route is estimated be to 105 km (65 miles), one way. The haul route is comprised of mine haul road, County road 119, US-89 and US-12. Local ingress and egress routes to/from the truck/rail transloading terminal are not included. As such, it is anticipated that the round-trip cycle time will be 195 minutes per truckload, of which 20 total minutes are included for loading/unloading. For the purposes of this study, Savage Services (2020) modelled a two shift per day, seven day per week trucking operation. While this model could require additional labor owing to scheduling, it more importantly reduces the amount of necessary equipment and enhances the utilization rate of tractor/trailer assets. Including driver pre/post trip,

Savage Services expects one shift to equate to ~10.5 hours and three complete loads per truck. At two shifts per day each truck in the system has a daily capacity of 142 t, or an annual capacity of ~52,000 t. This recommended tractor/trailer configuration operating seven days per week with two shifts per day would require a minimum of four haul units operating in the system. Note that the truck fleet operations typically require additional haul units as contingency for service work, mechanical issues, weather, etc.

SRA plans on optimizing truck moves from the mine to Townsend by fully leveraging a 131,000 lb maximum Gross Vehicle Weight (GVW) of the State of Montana. Equipment setup is comprised of the following:

- • One tractor (19,000 lbs assumed);
- • Two pin-chassis type trailers (9,000 lbs assumed for each trailer, total of 18,000 lbs);
- • One converter dolly required to haul doubles (4,500 lbs assumed);
- • Two concentrate containers (6,130 lbs assumed for each container, total of 12,260 lbs).

The above setup equates to equipment totalling 54,760 lbs, meaning the maximum concentrate payload expected to be achieved under Montana's prevailing 131,000 maximum allowable GVW is 76,240 lbs, or 17.3 t per container (38,120 lbs in each container). At a payload capacity of 53,880 lbs per container, this scenario 'light-loads' each container by 15,760 lbs (71% capacity utilization).

The cost per tonne of moving concentrate in containers from the mine to a rail terminal in Townsend, MT is estimated at $11.60/t. A less efficient five day per week operation would increase the per tonne rate by ~$1.00-$1.50/t.

Concentrate Containers

Rigid top opening half height containers will allow the movement of concentrate in compliance with the high environmental standards of the mine and function according to the requirements of the port. The Townsend option will require at least 1,200 containers, which is the base number of containers to support the operation. More containers would be necessary with the double trailer road haul. This total provides ~72 containers that would be considered Safety Stock, which would allow the logistics to absorb variations in transit times, containers that are held in transit for repairs and some measure of make-up capacity in the event that there is a delay in production or offtake. Costs used in all analyses assume that the containers are purchased rather than leased at an approximate savings of $1.00/t. Container payload is 24.4 t (53,880 lbs) at a maximum. The volume capacity of the container is 16.6 m^3, meaning depending on the density of the concentrate being loaded, the container's volume will be 67%-81% filled. The Savage Services (2019) study contemplates each container being loaded to 23.6 t (52,000 lbs). With the double trailer option (Savage Services, 2020) study, containers will be loaded to 17.3 t (38,120 lbs).

The container design key features include:

- Sloped internal side walls to minimize hang-up;
- Internal floor to wall to corners shall be rounded;
- No internal bracing or gusseting;
- Suitable for tippler operation – rotated 360 degrees;
- No exterior hang-up points;
- Flow gussets;
- Special internal painting;
- Integrated lid lifting;
- Interchangeable lids – Weather tight but vented;
- Construction suitable with IMDG BK2.

Figure 19.2 Photograph of a Container

Tractor/Semi-Trailer Configuration

Savage Services proposes a tractor/semi-trailer configuration for transporting the containers between the mine and the rail terminal. Each haul unit will consist of a four-axle tractor (with lift axle) coupled with a tri-axle container trailer, which is a configuration chosen to allow for the maximum payload of a container. Savage Services anticipates that the legal maximum allowable GVW for the haul routes will be greater than the anticipated fully laden gross weight of each haul unit.

The above-described haul unit configuration will allow for operating efficiency and simplicity while ensuring optimum payloads are in compliance with the current prevailing weight laws governing commercial vehicles in the state of Montana.

Rail and Haulage Route Option Alternative

For purposes of this study, Savage Services evaluated two potential locations for a rail terminal at the direction of SRA: Townsend and Livingston. After evaluating Livingston as a potential terminal site Savage Services recommends eliminating it as a favourable option due to infrequent rail service. Discussions with MRL revealed Townsend would likely receive five days per week service (switches) while Livingston would receive something much less.

Townsend is served by the MRL, which is a shortline based in Montana. This shortline is a Class II regional railroad that operates over 1448 route km (900 miles) of track in Montana and Idaho. It interchanges with BNSF Railway, a Class I railroad, at several points across its network, including at Helena, MT. Townsend is the closest rail station to the mine, at 105 km (65 miles). The MRL owns ~5 ha (12 acres) of land adjacent to their mainline, which has been the assumed location of the Townsend based rail terminal previously cited. Helena, MT is the "serving" yard for the Townsend area. Railcars will be stored there before coming to Townsend. For purposes of this study we have assumed that as cars arrive in Helena they will be released to the terminal in Townsend and will therefore not experience any demurrage in the Helena yard.

Savage Services' research indicates the optimum economics can be achieved by utilizing a configuration of eight concentrate containers per railcar, each container carrying 17.5 t (38,570 lbs) net weight. This equates to a total net payload of 140 t (308,570 lbs) per rail car. Concentrate container tare weight is estimated to be 2,780 kg (6,130 lbs), meaning the gross weight of each fully loaded container is estimated to be 26.4 t (44,700 lbs).

Ports

The Port of Everett, Washington has been selected as the preferable port for receiving containers of Cu concentrate for loading into vessels. Savage Services estimates the cost of all port services from receiving railcars of concentrate containers to vessel loading and returning of empty containers is ~$29.73/t.

These port services include the following:

1. Receiving articulated rail cars of concentrate containers at the port.
2. Removing containers from the railcars and placing them in a 'near dock' storage yard.
3. Transportation of the containers from storage yard to 'under hook' for vessel loading.
4. Lifting and dumping of containers into the hold of a vessel (this work, and associated costs of approximately $5.40/t, may be assumed in the ocean carrier rate).
5. Transportation, storage and loading of empty concentrate containers back on to articulated rail cars.
6. Releasing of railcars to the railroad.

This method of transportation and handling will limit bulk product handlings and loss along the supply chain, decrease particulate emissions at the port and reduce potential for product contamination. The quoted price for the Port of Everett includes the cost of purchasing a Ram spreader/revolver for the handling of the containers used in this process, though it should be noted the port has optionality when it comes to the type of equipment used in the vessel loading process. A photograph of a typical ram tipper is shown in Figure 9.3.

Figure 19.3 Photograph of Ram Tipper

The Port of Everett indicated no issues accepting or safely handling mine concentrate. Further, the high-level nature of the Savage Services' report precludes Savage Services from bifurcating port fees in a meaningful way. Owed to unique ocean carrier contracts that may be in place with certain ports, it is difficult to determine with accuracy what fees fall on users/carriers without knowing the parties involved or when title of the product changes hand. Lastly, it should be recognized that other port options exist on the west coast. The Port of Everett was selected due to its indicative cost advantage.

Ocean Freight

Wood Mackenzie's indicative bulk freight rate estimates from western seaboard ports to potential customers are detailed below in Table 19.2:

Smelter	Country	Freight Rate ($/t concentrate)
Onsan	South Korea	12 - 16
Pasar	Philippines	15 - 19
Dahej	India	26 - 30
Chinese smelters	Shanghai	13 - 17

Table 19.2 Data from Wood Mackenzie (2020) Indicative Bulk Freight Rates to Key Customers

Ocean port of origin is one of the few elements of this supply chain that SRA, or a supply chain logistics company like Savage Services, could put out for competitive bid or tender on a year to year basis for potential cost savings. As the Black Butte Copper Project reaches a final investment decision and a formal program is launched there may likely be an alternative solution that presents added economic benefit. Regardless, the optionality SRA has when it comes to ocean port of origin presents an opportunity to de-risk the supply chain as contingency plans can be more robust.

19.3.2 Price Forecasts

Price forecasts used in this FS have been based on the analysis completed by Wood Mackenzie (2020). These prices are detailed in Section 22.2.5.

19.3.3 Payable Revenue

The prices at which Cu concentrates are sold to smelters are based on three factors: the metal content of Cu, Au and Ag, penalties for deleterious elements, and payment terms for smelting and refining services. In the absence of a terminal market price, concentrate pricing is established through a process of negotiation between buyer and seller. While the terminal market price is used to determine the gross value of a Cu concentrate, the net value of the Cu concentrate is derived after the deduction of the negotiated charges for treatment and refining.

19.3.4 Offtake Terms

SRA has begun the process of identifying preferred offtake partners for the Cu concentrate and has had discussions with many potential customers.

Based on these discussions and consistent with Wood Mackenzie (2020), it is expected that concentrates will be sold to smelters or traders on normal industry terms. It is expected that there will be potential for a penalty charge applied to As in the Cu concentrate, with an allowance of $2 to $7 per dry tonne of Cu concentrate applied in the FS economic modelling. There is a lesser potential for additional penalty charges applied to Bi (up to $1.50 per dry tonne of Cu concentrate of grades over 0.02%) and Hg (up to $0.10 per dry tonne of Cu concentrate of grades over 5 ppm) in the Cu concentrate. However, the existing mine models do not contain the predicted Bi and Hg contents in the Cu concentrate.

19.4 Contracts

There are no marketing agreements in place as of the date of the FS.

19.4.1 Deeds

There are no deeds in place as of the date of the FS.

19.4.2 Other Material Contracts

There are no other material contracts in place as of the date of the FS.

19.5 Cu Concentrate Demand

Wood Mackenzie (2020) forecasts global refined Cu consumption to steadily increase from 24 Mtpa in 2020 to 33.5 Mtpa by 2040, equivalent to a Compound Annual Growth Rate (CAGR) of 1.7%. China will continue to feature as the world's largest consumer of refined Cu, accounting for 45% of global demand throughout the forecast period. Future growth in Cu after China, is expected from India and the rapidly developing South East Asian nations, as investment into Cu intensive sectors continues to flow into these countries.

Following a peak in base case Cu mine production in 2023, additional mine production will be required to replace output from mines that are forecast to close and to meet growing demand for Cu. The combination of growing demand and declining base case production implies a shortfall in identified mine output that steadily increases to 7.4 Mtpa by 2040, 4.9 Mtpa of which will be required by Greenfield projects.

US-China trade war tensions in 2019 continued to suppress investor and consumer confidence and Cu prices remained volatile. As the year progressed, it became clear that the US-China trade war resulted in a deterioration in the global macro-economic landscape and negatively impacted Cu demand for the year. Cu prices for the year averaged at $ 2.72/lb ($6,000/ t) down 8% from average 2018 prices.

Global total Cu consumption contracted for the first time in a decade in 2019, slipping by 0.4% to 29.6 Mtpa. This reflected the deterioration in the global macro-economic landscape, the recession in the automotive industry, and a slowdown in Chinese economic activity and decline in demand in the mature Asian economies (which are heavily reliant on China as their key trading partner), as well as lower offtake in the US and Europe.

Notwithstanding the challenges facing the global economic landscape, we expect refined demand to recover over the short to medium term, expanding at a CAGR of 2.1% from 23.5 Mtpa in 2019 to 26.1 Mtpa in 2024. This equates to an average increase of 514 ktpa of refined Cu. This will be underpinned by the introduction and ramp-up of new semis capacity in Southeast Asia, Europe, the Middle East and Russia and the Caspian.

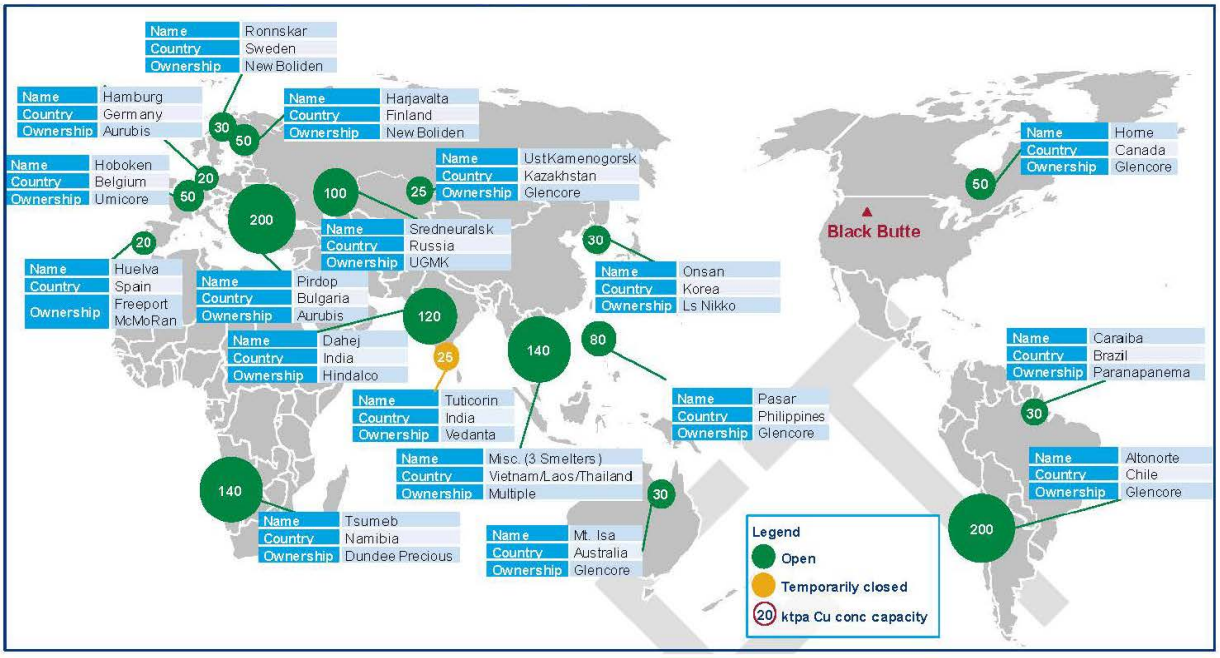
Looking further ahead, Wood Mackenzie (2020) forecasts global refined consumption to steadily increase to 33.5 Mtpa by 2040, equivalent to a CAGR of 1.7%. China will remain the world's primary player in the Cu industry for the foreseeable future. The sheer scale of Chinese Cu consumption and the smelting industry it has built to support itself over the last two decades, means that despite the current slowdown of its economy, China continues to remain the world's largest single consumer and producer of refined metal through to 2040.

Beyond 2020, India and the Association of Southeast Asian Nations (ASEAN) are expected to dominate refined Cu consumption growth as investment into Cu intensive sectors continues to flow into these countries.

In India, strong growth in Cu demand will be supported by the continued building of downstream facilities as well as favorable demographic trends, urbanization, and rising income per capita. After India, the rapidly developing ASEAN countries will also experience strong growth in refined Cu consumption, albeit from low bases.

19.6 Available Smelting and Refining Options and Costs

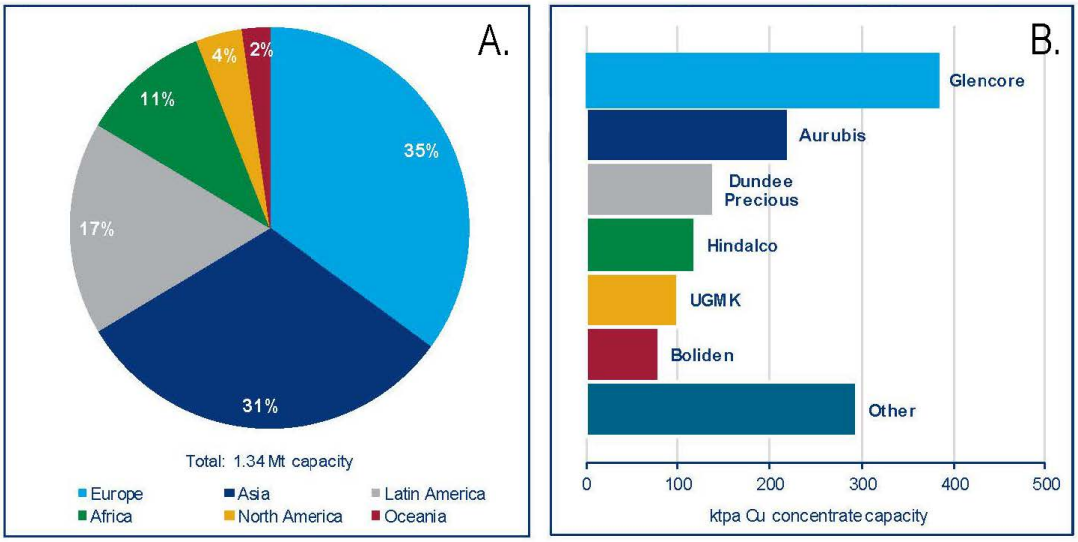
Based on Wood Mackenzie's 2020 analysis and the expected Black Butte Copper Project concentrate specifications listed in Section 19.3 above, the following smelters identified in Figure 19.4 have been identified as potential customers.



From Wood Mackenzie (2020)

Figure 19.4 Location of Possible Smelters to Treat the Cu Concentrate

Glencore operates four possible smelters across four regions with a total treatment capacity of 385 ktpa. Glencore's Horne operation in Quebec, Canada is the closest to the Black Butte Copper Project which has previously treated Cu concentrate from Barrick Gold Corporation's Phoenix Project in Battle Mountain, Nevada. Other operators of note are Aurubis who operate smelters at Pirdop and Hamburg with a combined capacity of 220 ktpa. The Altonorte smelter in Chile and Pirdop in Bulgaria are the largest smelters with each having the ability to treat up to 200 ktpa of concentrate. Dundee Precious Metals Tsumeb smelter in Namibia is also another alternative. Figure 19.5 shows the distribution of potential customer smelters by region and ownership.



From Wood Mackenzie (2020)

Figure 19.5 (A) Possible Smelter Capacity by Region (B) Possible Smelter Capacity by Ownership

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20. ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 Introduction

The following section provides an overview and summary of site conditions, and the environmental studies completed to support the baseline characterization, permitting process and closure planning, as well as the evaluation of potential environmental and social impacts. In accordance with CIM guidance (Waldie et al., 2018) for NI 43-101 compliant FS technical reports, Section 20 provides:

a) A summary of the results of any environmental studies and a discussion of any known environmental issues that could materially impact the issuer's ability to extract the mineral resources or mineral reserves;
b) Requirements and plans for waste and tailings disposal, site monitoring, and water management both during operations and post mine closure;
c) Project permitting requirements, the status of any permit applications, and any known requirements to post performance or reclamation bonds;
d) A discussion of any potential social or community related requirements and plans for the Project and the status of any negotiations or agreements with local communities; and
e) A discussion of mine closure (reclamation) requirements and costs.

The following summary of environmental conditions, potential impacts, and permits is based on the work of numerous technical consultants. INTERA Inc. performed a technical review of the studies thought to have the greatest potential for environmental impacts and operational risks (water and geochemistry). All information included in this section is assumed to be complete and accurate.

The following summary is extracted from the extensive environmental studies that were summarized in the Draft and Final Environmental Impact Statements (EIS) (MDEQ, 2019 and 2020) and the Mine Operating Permit (MOP) (Tintina Montana, 2017), with additional information (Zieg, 2018; Hydrometrics, 2018a) developed to refine some of the details related to water management.

20.2 Baseline Studies

Baseline studies of the Project site were performed for the following aspects of the Project. The detailed reports are provided in the MOP and its numerous appendices.

- Surface Water Hydrology;
- Air Quality;
- Cultural and Tribal Resources;
- Groundwater Hydrology;
- Geology and Geochemistry;

- Land Use and Recreation;
- Visuals and Aesthetics;
- Socioeconomics;
- Soils;
- Noise;
- Transportation;
- Vegetation;
- Wetlands;
- Wildlife; and
- Aquatic Biology.

20.2.1 Physical Baseline

The Project is located on private land in the Little Belt Mountains, with elevations that range between 5,600 and 6,100 feet above mean sea level (Figures 4.1, 18.1, and 20.2). The climate is characterized as cold, semi-arid "steppe' according to the Köppen-Geiger climate classification scheme. The site climate includes hot summers and cold winters which usually experiences snowfalls. Winters can also be dry while summers can be wet. This type of climate is often subject to major temperature swings between day and night, sometimes by as much as 20°C (approximately 36°F) or more. The topography is moderately sloped with open woodland consisting of Douglas fir on the ridgetops and aspen and willow along the drainages. Much of the lower valley areas are grasslands used for cattle and wildlife grazing.

Surface Water

Surface and groundwater investigations for the Project were conducted by Hydrometrics (2017). Details of the Baseline Water Resources Monitoring and Hydrogeological Investigations are provided in Appendix B of the MOP.

The Project is in the middle portion of the Sheep Creek drainage (see Figure 20.1). Sheep Creek, a fifth-order stream, flows out of the Little Belt Mountains approximately 27 km (17 miles) upstream from the confluence with the Smith River, which in turn is a tributary to the Missouri River. Sheep Creek drains an area of 502 km^2 (194 $miles^2$) and runs approximately 55 river km (34 river miles) from its headwaters down to the Smith River. The Project area is approximately 31 river km (19 river miles) above the confluence with the Smith River. Sheep Creek flows in a meandering channel through a broad alluvial valley upstream of the Project site and enters a constricted bedrock canyon just downstream of the Project site (Hydrometrics 2017). A number of named and unnamed tributaries flow into Sheep Creek, including Little Sheep Creek and Coon Creek in the immediate vicinity of the Project (see Figure 20.1). The Holmstrom Ditch in the vicinity of the Project was constructed in 1935 to divert water from Sheep Creek for irrigation and continues to operate seasonally (Hydrometrics, 2017).

Water resources baseline monitoring for the Project began in 2011. Quarterly flow and water quality monitoring has been conducted at numerous sites since 2011. In 2014, two additional monthly flow measurement locations were added along Sheep Creek. The highest stream flows in Sheep Creek (Stations SW-1) occur from mid-May through mid-June, when flows exceeded 2.8 m^3/s (100 cfs). Annual peak flows captured in the data record ranged from over 5.7 m^3/s (200 cfs) in 2015 to just above 22.7 m^3/s (800 cfs) in 2014. Following the high-flow period, flows recede to an average monthly flow of 0.4 to 0.9 m^3/s (15 to 30 cfs) by late summer. Winter base flow was determined to be approximately 0.4 m^3/s (15 cfs) across the monitoring period.

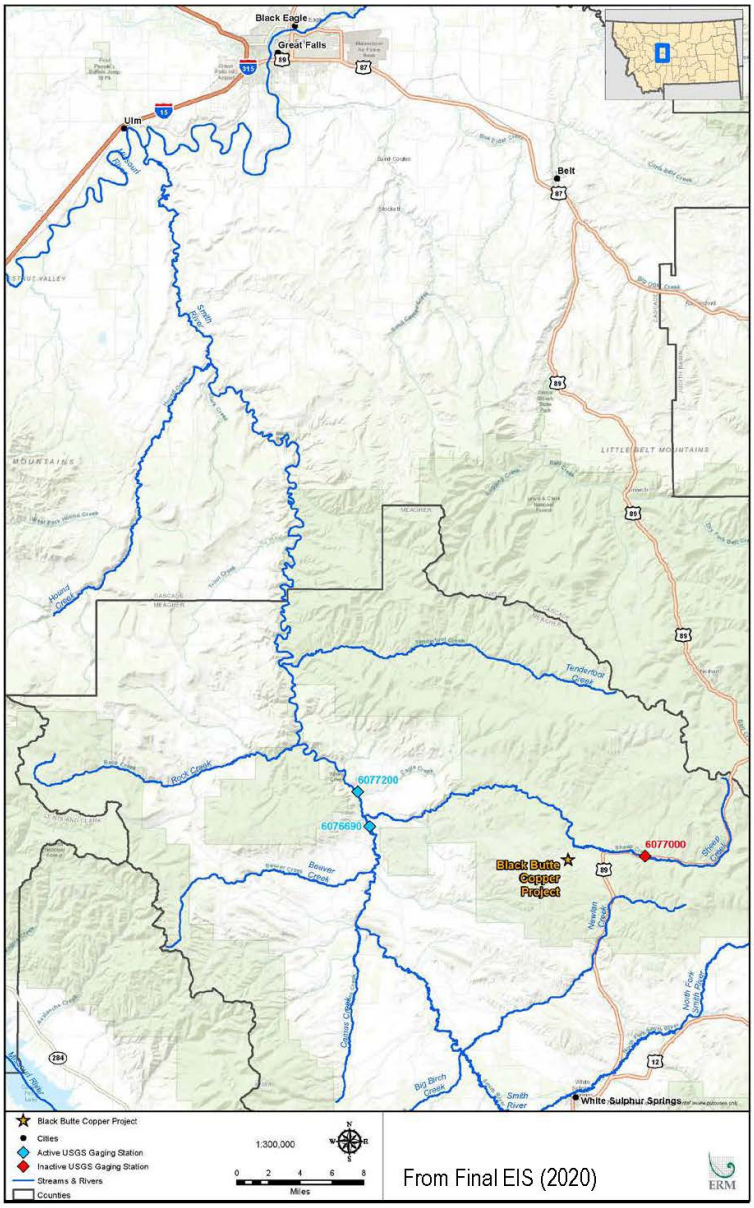


Figure 20.1 Plan Map Showing Rivers and Gaging Stations in the Project and Region

Baseline surface water monitoring commenced in 2011 at 11 locations. Currently, surface water monitoring for water flow and water quality is conducted at nine surface water locations, in addition six further sites are monitored for water flow only. Surface water results show near neutral to slightly alkaline pH values (6.8 to 8.7), and low to moderate specific conductance (49 to 497 micro mhos per centimetre). Calcium and bicarbonate dominate the major ion chemistry of surface waters. Metals data show infrequent exceedances of MDEQ water quality standards for total iron and dissolved aluminium, most commonly during periods of high flow. Sheep Creek is included in MDEQ's 303(d) list of impaired streams (MDEQ, 2020d) for dissolved aluminium and E. coli, with sources listed as grazing in riparian zones, disturbances associated with human activities, and natural sources.

In addition to the stream flow monitoring, baseline investigations identified 10 seeps and 16 springs in the Project area (see Figure 20.2). Generally, the smaller springs and seeps are located in the ephemeral headwater channels of small tributary streams. These form small boggy areas with limited flow that are generally re-infiltrated into the channels within a few hundred feet of the source. Of the identified springs, six have been developed to supply livestock watering tanks. Spring flow rates in the Project area ranged from no flow during certain dry or frozen periods in the year to greater than 6.3 L/sec (100 gpm).

Groundwater

Water resource baseline monitoring and hydrologic investigations for the Project have been carried out since 2011 and are ongoing. Information on site hydrogeology, groundwater conditions and groundwater chemistry are presented in Appendices B and U of the MOP Application (Tintina Montana, 2017) and subsequent updates.

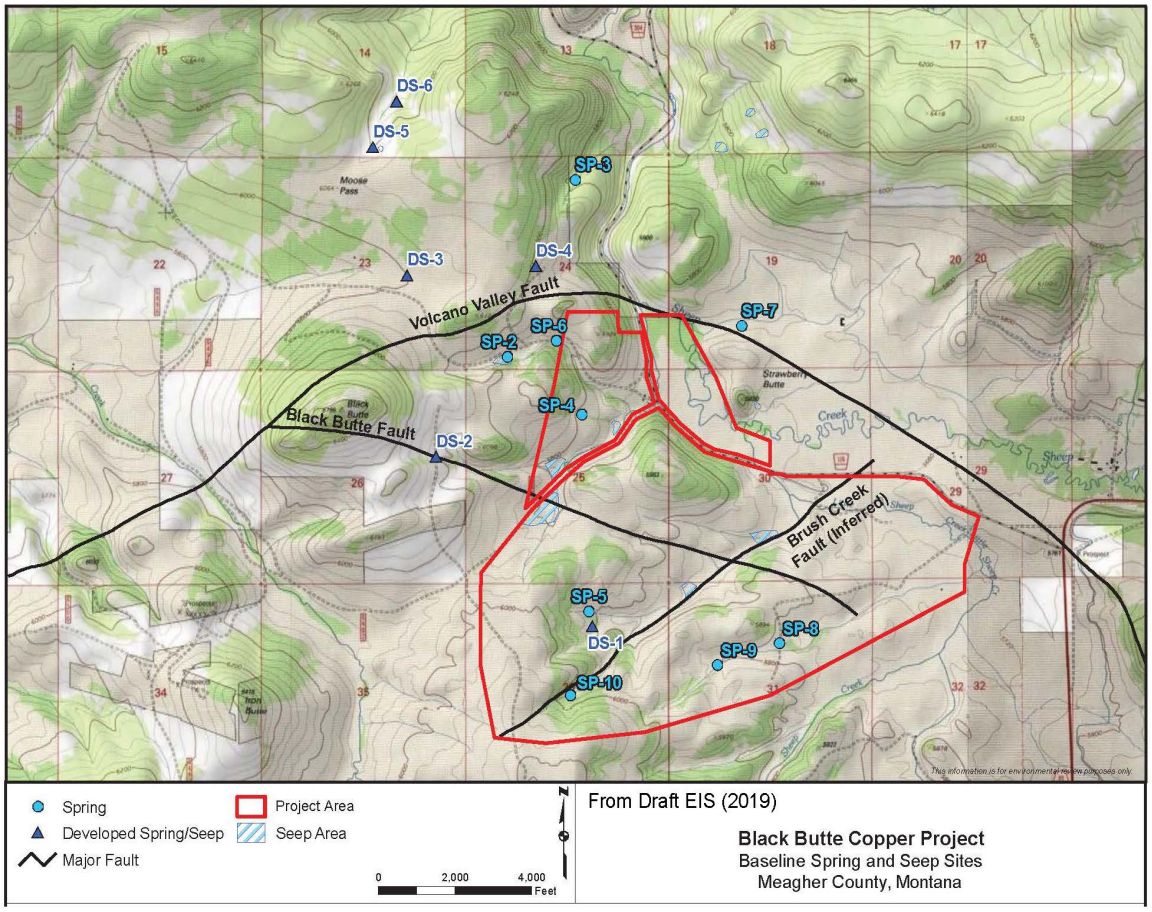


Figure 20.2 Surface Springs and Seeps Map

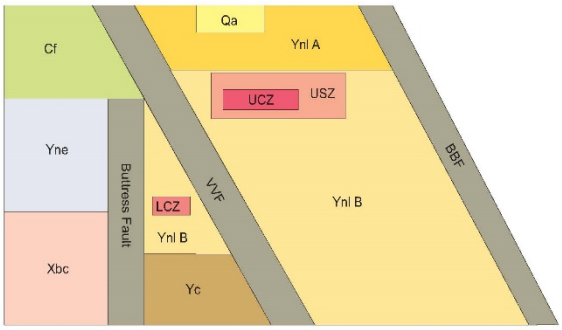
Unit	Description	Thickness (ft.)	Hydraulic Conductivity* (ft./day)	Storage Coefficient	Source of Hydraulic properties
Geologically-Based Hydrostratigraphic Units					
Sheep Creek Alluvium (AL)	coarse-grained sand and gravel alluvium	17	200	0.2 to 0.35	slug test; literature
Flathead Sandstone *(Cf)*	sandstone bedrock	100	10^{-5} to 1.5	NA	literature
Lower Newland Formation Shallow *(Ynl A)*	calcareous and bib-calcareous shale and siltstone bedrock	30-50	1 to 2.3 GM 1.5	1 x 10^{-4} to 8 x 10^{-6}	pumping test
Upper Sulphide Zone	highly mineralized zone	30-150	0.01 to 0.7 GM 0.08	6 x 10^{-5} to 9 x 10^{-5}	pumping test
Upper Copper Zone	Cu mineralized zone				

Unit	Description	Thickness (ft.)	Hydraulic Conductivity* (ft./day)	Storage Coefficient	Source of Hydraulic properties
Lower Copper Zone	Cu mineralized zone	30-50	1.9×10^{-4}	NA	pumping test
Lower Newland Formation deep (*Ynl B*)	dolomitic and non-dolomitic shale and siltstone bedrock	150 in FW; to 2,000	0.001 to 0.007	NA	pumping test
Chamberlain Formation Shale (*Yc*)	siliceous, locally arenaceous shale	500	0.001 to 0.007	NA	assumed
Neihart Formation Quartzite (*Yne*)	recrystallized sandstone	800	Low; N.A.	NA	assumed
Crystalline Bedrock (*Xbc*)	metamorphic crystalline rock	To depth	10^{-3} to 10^{-1}	NA	literature
Structurally Defined Hydrostratigraphic Units					
Volcano Valley Fault	fault; clay gouge core; variable associated fracturing	150	7.1×10^{-4} to 1.5×10^{-5} avg. 2.8×10^{-5}	NA	permeameter testing
Black Butte Fault		10 – 14			assumed
Buttress Fault		5			
Brush Creek Fault		44			

*GM=geometric mean value; used when properties ranged several orders of magnitude; FW=footwall; N.A.=not available; avg.= arithmetic average value; used when properties were within the same order of magnitude; * hydraulic conductivity values determined from the aquifer testing described in the Hydrometric's Baseline Water Resources Report (2017).*

Table 20.1 Hydraulic Properties of Hydrostratigraphic Units

Groundwater flow is controlled by the hydraulic characteristics of the geological units described in Part 7 and structural features (faults). Figure 20.1 provides a generalized cross-section through the mine, which is represented in the conceptual groundwater model as the Hydrostratigraphic Units (HSU) shown in Figure 20.3 (Hydrometrics, 2016a). Table 20.1 from the MOP summarizes the physical and hydraulic characteristics of the principal HSU's that were evaluated during the FS. Hydrogeologic investigations and monitoring focused on the first five HSU's. The principal HSU's consist of the alluvium in the Sheep Creek drainage, and four units within the Lower Newland Fm (Ynl A, USZ, Ynl B, and LCZ). The highest hydraulic conductivity of the principal HSU's is in the alluvium at 61 m/day (200 ft/day). The HSU's within the Lower Newland Fm show a general decreasing trend in hydraulic conductivity with depth (0.3 to 0.7 m/d or 1 to 2.3 ft/day for Ynl A to 5.8×10^{-5} m/d or 1.9×10^{-4} ft/day for the LCZ). Faults running through the mine site were also evaluated as potential controls on groundwater flow, either as barriers to flow or preferential flow paths for groundwater flow (Table 20.1). An additional fault (Fault #1) was identified after the MOP application was submitted. The faults have much lower hydraulic conductivities than most of the HSU's, indicating they are most likely barriers to groundwater flow rather than preferential pathways.



LEGEND

Qa	Quaternary Alluvial Deposits
Cf	Flathead Sandstone
Ynl A	Lower Newland Fm. (shale above the USZ)
Ynl B	Lower Newland Fm. (shale below the USZ)
USZ	Upper Sulfide Zone
UCZ	Upper Copper Zone
LCZ	Lower Copper Zone
Yc	Chamberlain Fm. (shale)
Yne	Neihart Fm. (quartzite)
Xbc	Metamorphic Crystalline Basement rock
VVF	Volcano Valley Fault
BBF	Black Butte Fault

Figure 20.3 Hydrostratigraphic Units

Groundwater monitoring has included measurements of surface water flow, groundwater-level elevations, and water temperatures. In addition, surface and groundwater samples have been collected and chemically analysed for various properties following protocols described in the "Water Resource Sampling and Analysis Plan" (Hydrometrics, Inc. 2016). The groundwater component of the monitoring program involves quarterly (or in some cases less frequent) measurements of water levels in 34 monitoring wells and piezometers, and collection of water samples from 29 monitoring wells. The locations of these wells and piezometers are shown on Figure 20.4.

Groundwater isoelevation maps were generated for both the regional and mine development areas. The regional map (Figure 20.5) was generated from SRA's well network and 20 domestic wells available from the Montana's Groundwater Information Center (GWIC). Due to the limited number of wells throughout the regional area, groundwater elevations were also estimated using the elevation of perennial streams. A potentiometric map of the mine site is presented in Figure 20.6, based on water level data collected from SRA's network of monitoring wells and piezometers.

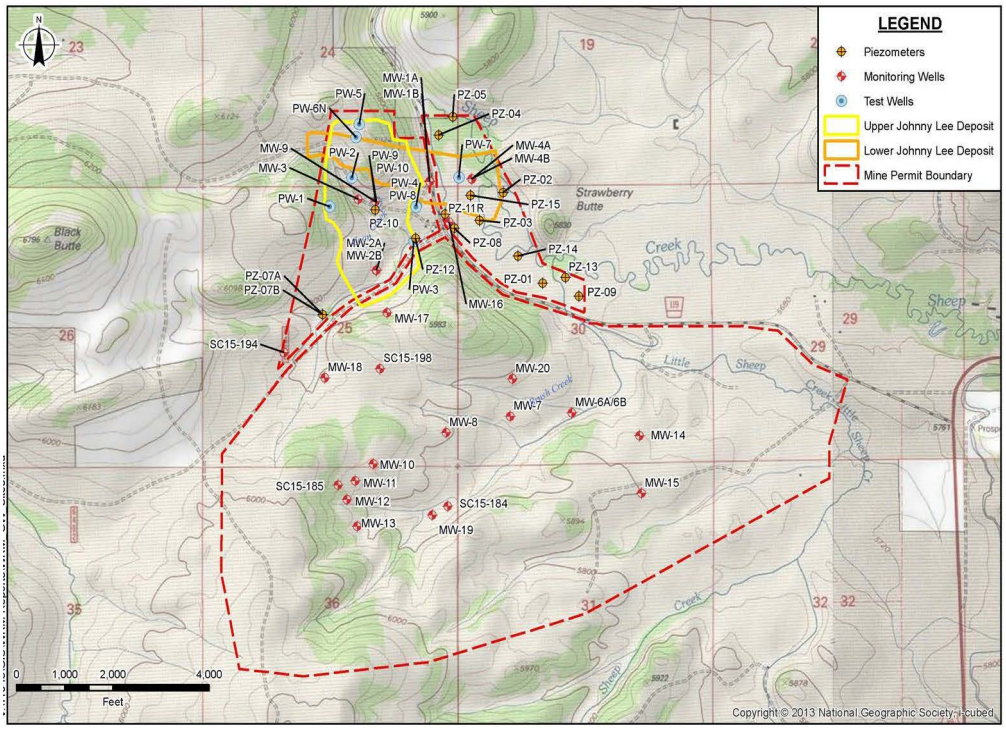


Figure 20.4 Groundwater Monitoring Sites

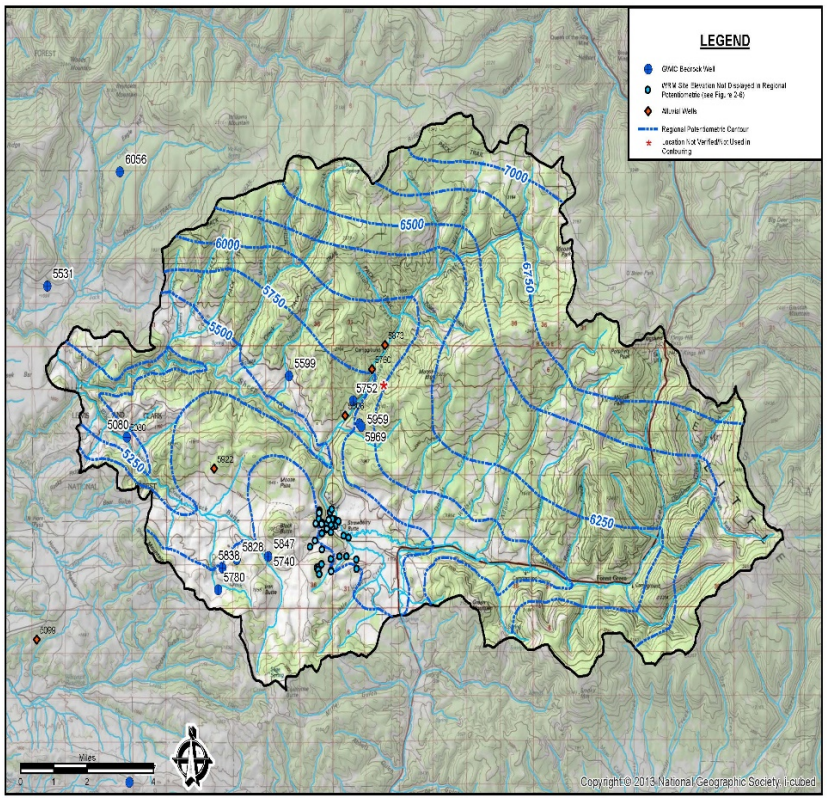


Figure 20.5 Regional Potentiometric Surface Map

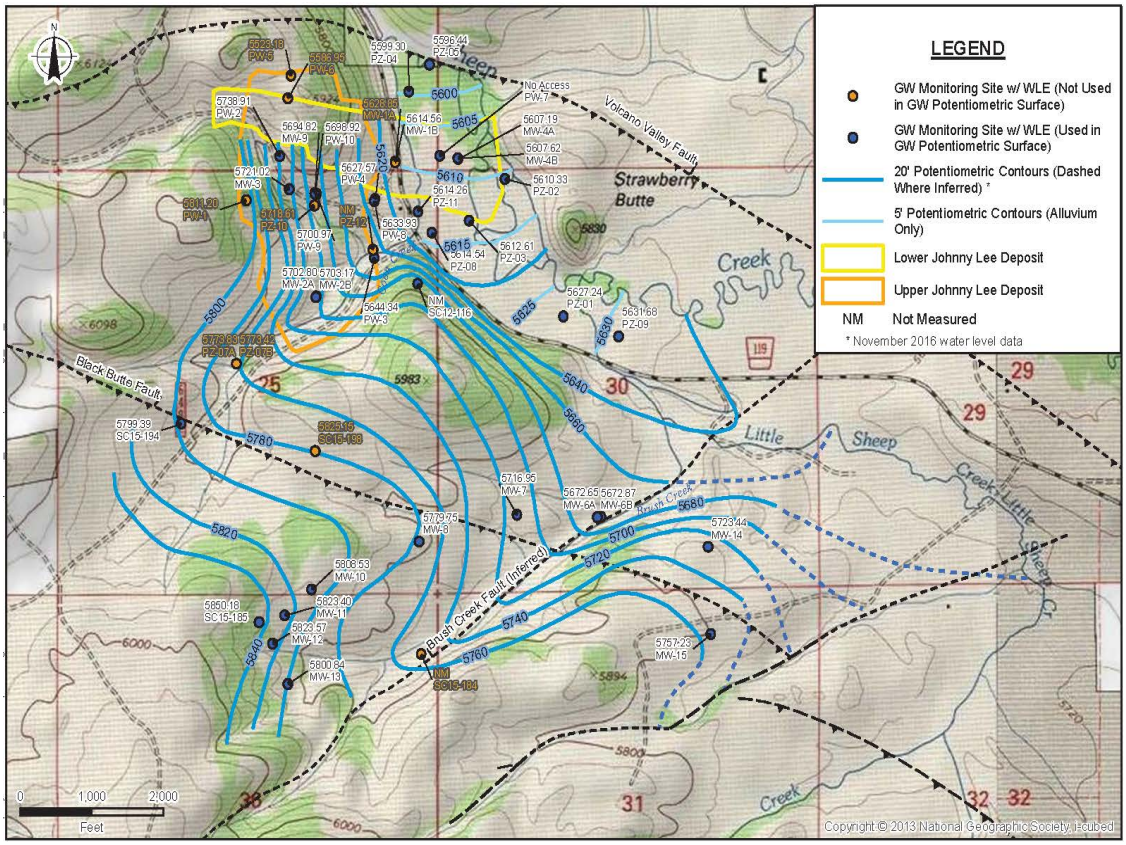


Figure 20.6 Mine Site Potentiometric Surface Map

Figure 20.6 depicts the bedrock potentiometric surface in the Lower Newland Formation, as well as elevations of the water table in the shallow alluvial system. The potentiometric surface shows an eastward trending flow direction in the bedrock groundwater system within the Project area consistent with the general topographic trend. Based on the potentiometric contours for the bedrock hydrologic system, hydraulic gradients range from 0.04 in between Brush Creek and Little Sheep Creek to 0.1 in the UCZ area. Groundwater in the Sheep Creek alluvium generally flows parallel to Sheep Creek. The hydraulic gradient in the alluvial system is relatively flat (0.008) through most of the monitoring area and then increases slightly to 0.013 in the northern portion of the valley. The increased gradient and near surface water level elevations in this area indicate that the alluvial groundwater system discharges to surface water as the alluvium thins and then pinches out against the less permeable bedrock, which forces the water upward where Sheep Creek flows over the bedrock rise and downstream into the canyon.

Recharge takes place in upland areas and groundwater flow converges toward the major drainages, including Sheep Creek, Moose Creek, Little Sheep Creek, and Black Butte Creek (Hydrometrics, Inc. 2017).

Groundwater within the Sheep Creek alluvium is in direct hydraulic communication with the Sheep Creek stream channel, as evidenced by the correlation between flow in the perennial streams across the mine site and groundwater levels adjacent to the stream channels. Where alluvium is not present, the stream is in direct or indirect hydraulic communication with bedrock. Except for peak stream levels during May and June, the Sheep Creek water level is typically lower than groundwater levels in the adjacent alluvium and bedrock, and thus receives groundwater discharge. Due to these processes, Sheep Creek is generally a gaining stream within the watershed, with significant base flow supported by groundwater discharge.

Groundwater quality varies depending on depth and proximity to the mineralized zone. A summary of groundwater exceedances is presented in Table 20.2 (MDEQ, 2020). Groundwater in shallow alluvial wells and shallow bedrock wells is calcium/magnesium bicarbonate type water with near neutral pH and moderately low dissolved solids, with generally low to non-detectable concentrations of dissolved metals. Water quality parameters that exceeded regulatory limits are listed in Table 20.2.

Wells completed in the USZ have the highest concentrations of dissolved solids and sulphate compared to the other hydrostratigraphic units. The pH of water in the USZ ranges from 6.04 to 7.31 s.u.

Grouping	Geology	General Water Type	Wells	pH	Total Dissolved Solids	Exceedances	Comments
Alluvium/ Overburden	Qal	Calcium/ magnesium bicarbonate	MW-1A, MW-2A, MW-4A	6.24 to 7.66	176 to 302 mg/L	▪ Arsenic, barium, iron, lead, magnesium, and thallium above hhs in MW-1A. ▪ Thallium above hhs in MW-2A	▪ High turbidity in MW-1A may be responsible for elevated metals concentrations in this well. ▪ Sulphate concentrations are relatively low (from 8 to 51mg/L).
Shallow Bedrock	Ynl A Ynl B Above USZ	Calcium/ magnesium bicarbonate	MW-1B, MW-2B, MW-4B, MW-6A, MW-6B, MW-7, MW-8, MW-9, MW-10, MW-11, MW-12, MW-13, MW-14, MW-15, PW-1, PW-2, PW-3, PW-8, PW-9, PW-10, SC15-184, SC15-185, SC15-194, SC15-195, SC15-198	6.02 to 8.27	54 to 548 mg/L	▪ Antimony above hhs in WM-08 ▪ Arsenic above hhs in MW-1B, NW-2B, MW-9, PW-8, PW-9. ▪ Iron Above secondary standard in MW-1B, MW-2B, MW-9, MW-10, MW-11. {W-1, PW-2, PW-3, PW-9. ▪ Lead above hhs in PW-8. ▪ Manganese above secondary standard in WM-1B, MW-6B, MW-7, MW-8, MW-9, MW-10. MW-11, PW-1, PW-3, PW-8, PW-10, SC15-185. ▪ Strontium above hhs in PW-10. ▪ Thallium above hhs in MW-1B, MW-2B, MW-9, PW-8.	Sulphate concentrations range from 1 to 247 mg/L

Table 20.2 Summary of Groundwater Quality

Groundwater in the LSZ is a sodium/potassium bicarbonate type water with alkaline pH (10.77-11.58 s.u.), and lower concentrations of sulphate than the shallow HSU's. Trace constituents detected above the reporting limit include barium, iron, lead, manganese, molybdenum, strontium, and zinc. None of the trace constituents exceeded the groundwater human health standard.

Soil

Soil investigations for the analysis area were conducted by WESTECH Environmental Services, Inc. (WESTECH) in 2015, and encompassed a total of 1,363 ha (3,368 acres), including the approximately 764 ha (1,888 acres) of the MOP Application Boundary. Methods and results are provided in Appendix E of the MOP. The baseline soil survey included 30 soil survey locations with soil quality samples from 25 locations. Data recorded at each location included: field drainage class; slope range; parent material; vegetation and land use; topography and position; aspect; surface runoff; erosion; permeability; horizon types; depths and thickness; color and texture; coarse fragment content; carbonates; clay films; effervescence; roots; and structure. Laboratory analyses were performed on selected physical and chemical characteristics of the soils, including: particle size analysis; percent rock fragments; percent organic matter; salinity/conductivity; and chemical properties including soil pH and total concentrations of arsenic, cadmium, copper, lead, and zinc.

The baseline soil survey identified 18 soil types. The distribution of soil types is shown in Figure 20.7. The predominant soil type is a loam (Cheadle channery loams), a soil with roughly equal proportions of sand, silt, and clay.

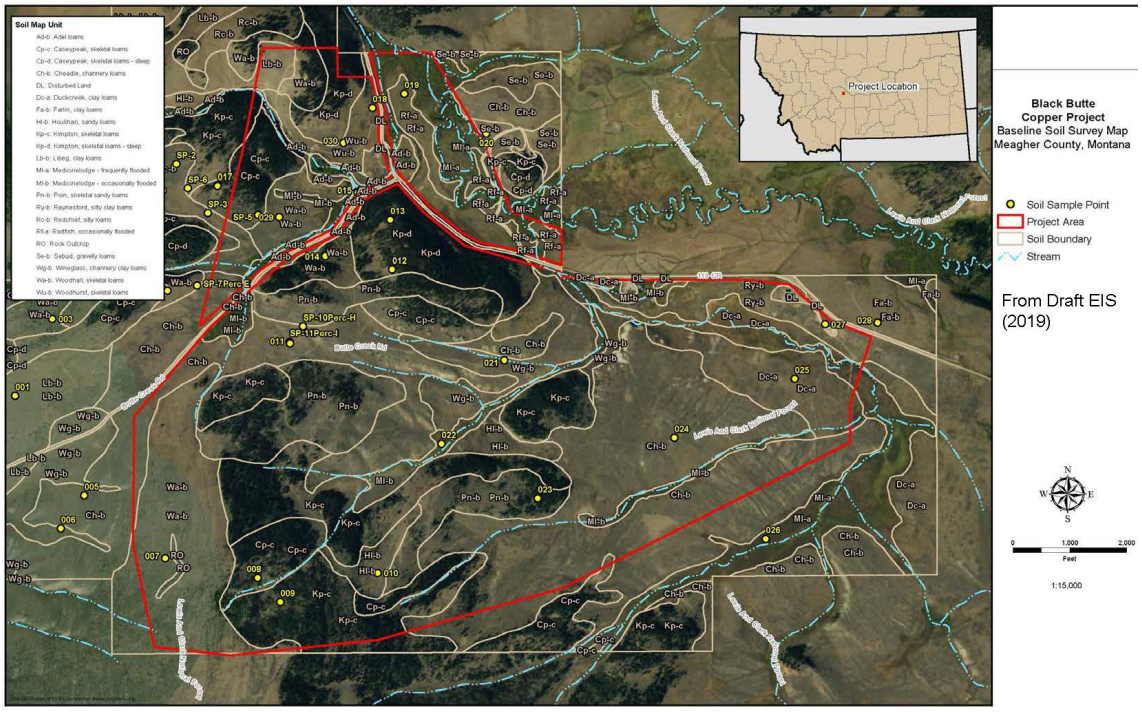


Figure 20.7 Baseline Soil Survey Map

Air Quality

Meagher County, where the Project is located, is classified for air quality purposes as "Unclassifiable or Better than National Standards" for all criteria pollutants (40 CFR 81.327 and MDEQ). This classification indicates that MDEQ has not monitored the area's air quality but that, based on the Department's experience, the area is presumed to meet the ambient air quality standards.

There are no significant sources of air pollution in the vicinity of the Project area. The nearest significant source is the Graymont Indian Creek Lime Plant located approximately 74 kilometres by air, (46 air miles) SW of the mine. White Sulphur Springs is approximately 24 km (15 miles) south of the mine site and does not have any significant emitting sources.

The mine Project will include a variety of air pollutant emission sources consisting of diesel-fuelled stationary engines, gas-fired heaters, mined material handling equipment, fugitive dust sources, and vehicle operation. The Cu ore mining activities will be completely underground, and the mine is mechanically vented at three locations to maintain a safe working atmosphere. These vents will be sources of air emissions, primarily combustion gases from explosives, vehicle exhaust and from gas-fired vent air heaters. Particulate Matter from UG operations is not expected to exit from the vents at significant rates. Aboveground material handling activities will also cause air emissions, primarily fugitive dust and emissions from combustion of motor fuels (diesel and gasoline) used to operate mining vehicles (e.g., haul trucks), stationary equipment, portable equipment, and support vehicles.

Quantitative modelling was conducted to evaluate the potential air quality impacts of the Proposed Action, including the impacts of underground and aboveground stationary sources. Air dispersion modelling was performed primarily to quantify concentrations of regulated pollutants resulting from stationary and fugitive source emissions, and these results were compared to federal and Montana ambient air quality standards.

Land Use and Recreation

The analysis area for land use encompasses the Project area for the mining facilities and adjacent lands.

Northeastern Meagher County is a rural area with the nearest major population area the City of White Sulphur Springs, approximately 24 km (15 miles) to the south of the Project area. Large-lot residential properties, ranches, and cabins are present along US-89 between the City of White Sulphur Springs and the Project area. Private citizens own all the property in the Project area which contains the Johnny Lee copper resource which SRA intends to for mine. Land uses are predominantly agricultural and include primarily livestock grazing and hay production. In addition, recreational uses including fishing and big game hunters use the Sheep Creek drainage.

The Bar Z Ranch and Short Ranch own 100% of the surface and mineral rights of the lands containing the proposed mine and related facilities. SRA has lease agreements with each of these owners of the approximate 764 ha (1,888 acres) within the proposed Project area, current land use is principally livestock grazing (94%) with the balance dedicated to hay production. A portion of Bar Z Ranch (approximately 1.5 ha or 3.7 acres) is located within the Project area (see Figure 20.2).

There are no public recreation opportunities located within the Project area. Bar Z Ranch, located within the Project area, offers lodging and private fly-fishing expeditions along multiple waterbodies including Sheep Creek and the Smith River (Fly Fishing Montana, 2017). Public recreational opportunities in the surrounding area include hiking, camping, fishing, hunting, boating, snow skiing, and river floating. In addition to hiking and camping, there are boating and fishing opportunities on Sheep Creek, Smith River, Newland Reservoir, Lake Sutherlin, and Bair Reservoir. There is one infrequently used picnic area along Sheep Creek located approximately 7.7 km (4.8 miles) upstream of the Smith River confluence on private land. The MOP provides a detailed list of campgrounds and hiking trails within 24 km (15 miles) of the Project. Hunting near the Project area includes elk, deer, black bear, mountain lion, and bobcat.

Visuals and Aesthetics

Plan maps of the surrounding area and proposed facility locations were used to determine viewing angles of the Project to generate visual resource projections from three perspectives, including from US-89 about 800 m (0.5 miles) south of the junction with the Sheep Creep county road looking W/NW; on the Sheep Creek county road about 800 m (0.5 miles) west of the core sheds looking SW; and Butte Creek Road about 100 m (300 feet) NE and downhill of the divide between Sheep and Butte Creek looking NE.

Transportation Resources

The baseline transportation study in the vicinity of the Project area as well as the city of White Sulphur Springs and major routes within Meagher County was evaluated by Abelin Traffic Services in April 2018.

US-89 and county roads provide the primary access to the Project area as previously described in Section 5.1 of Chapter 5. US-89 lies east of the Project area and is the only paved road in the vicinity. County or US Forest Service roads bound and traverse the Transportation Analysis area. Highways US-89, US-12, and MT-360, along with secondary Montana Highway 294 (MT-294) comprise the main highways in Meagher County. Within the Transportation Analysis area, approximately 35 km (22 miles) of unpaved roads traverse the property. The established county gravel roads are located in the north half of the Transportation Analysis area and include Sheep Creek Road and Butte Creek Road.

There are several railroad lines near the site which may serve to transport Cu concentrate. The nearest active railroad line to the Project area is a MRL line 64 km (40 miles) W/SW of White Sulphur Springs at Townsend, MT. In addition, a BNSF line lays 92 km (57 miles) east of White Sulphur Springs near Harlowton, MT.

Information on local air transportation provided in the MOP reports one public airport located 5 km (3 miles) south of White Sulphur Springs, approximately 31 km (19 miles) due south of the Project area. The airport serves transient general aviation (62%) and local air traffic (38%). Great Falls, Helena, Bozeman, and Billings, MT all have regional airports with scheduled passenger services suitable for travel to the Project area.

Noise

Baseline ambient noise monitoring was conducted on September 2013 in general accordance with the American National Standards Institute (ANSI) S12.18-1994, *Procedures for Outdoor Measurement of Sound Pressure Level* (ANSI, 1994). Details of the noise evaluation are provided in Appendix J of the MOP. Noise measurements were taken at four locations ranging from 0.8 to 3.2 km (0.5 to 2 miles) from the proposed mine portal location. Noise levels at each of the four locations were typical for sparsely populated rural areas. Dominant noise sources during the daytime consisted of vehicles, haul trucks from the Black Butte Iron Mine, ATV's, and occasional air traffic. Dominant night time noise sources included flowing water in Sheep Creek, breezes, and traffic along US-89. Average noise levels ranged from 22 to 48 dBA.

20.2.2 Biological Baseline

Vegetation

The baseline vegetation surveys were conducted by WESTECH in May, June, and July 2015 and are summarized in Appendix H of the MOP application (Scow, 2017). 185 sample plots were surveyed throughout the vegetation analysis area. The results of the surveys indicated there are five habitat and community types within the vegetation analysis area:

- Upland grassland (23%). Much of this is altered grassland sub-community type which includes rangeland and cropland for hay within the Sheep Creek floodplain;
- Upland shrubland (41%);
- Conifer forest and woodland (15%);
- Lowland altered grassland (4%); and
- Riparian and wetland (RW) (7%).

There are no federally listed Threatened and Endangered (T&E) plant species in Montana; however, Montana does maintain a list of Species of Concern (SOC), which are species that are rare, threatened, and/or have declining populations and as a result are at risk or potentially at risk of extirpation in Montana. The most recent list of T&E vegetation for Meagher County reported 24 SOC species within the Meagher County element data. Of these, only one was identified within the analysis area: long-styled thistle (Cirsium longistylum). No federal listed species were reported within the vegetation analysis area.

Twelve state, county, and problematic listed noxious weed species were noted within the vegetation analysis area during the 2014 to 2015 baseline vegetation surveys. Of these 12 species, the three most common noxious weeds were Canada thistle (Cirsium arvense), common houndstongue (Cynoglossum officinale), and musk thistle (Carduus nutans).

Wetlands

The baseline wetland and waterbody surveys were conducted by WESTECH in August and September 2014 and were summarized in Appendix C-1 of the MOP Application (WESTECH, 2015a). Wetland analysis was performed within the Project lease hold (3,110 ha or 7.684 acres) using methods described in the 1987 USACE Wetland Delineation Manual. The 2014 wetland and waterbody baseline survey identified 122 ha (328.8 acres) of wetlands within the wetland analysis area (Figure 20.8). The largest wetlands and wetland complexes were associated with the herbaceous meadows and shrub wetlands within the riparian areas surrounding Sheep Creek and Little Sheep Creek. Smaller and sometimes isolated wetlands were associated with the headwaters of the wetland analysis area wetlands and waterbodies.

During the 2014 wetland surveys, the primary wetland functions were rated using the Montana Wetland Assessment Method rating system, which ranges from Category I (high quality wetlands that provide potential habitat for listed species) to Category IV (small or isolated wetlands that lack diversity and provide little wildlife habitat). The Little Sheep Creek Wet Meadow and the Sheep Creek Spring Tributary assessment areas are rated as Category I, primarily because of the fen features located within these assessment areas. The six Category II assessment areas are rated as Category II rather than Category I because of the lack of fen features within these wetlands. The six Category III assessment areas are rated in this category primarily due to their decreased function compared to the other categories, which lowered their rating.

SRA requested an Approved Jurisdictional Determination (JD) from the United States Army Corp of Engineers (USACE) as part of the Section 404 permitting process. The October 3, 2017 Approved JD determined that most of the wetlands delineated within the analysis area were jurisdictional (a total of 132.5 ha or 327.4 acres) and, therefore, will require authorization via Section 404 of the Clean Water Act for any proposed dredge or fill impacts to these wetlands. The Approved JD also determined that the six small, isolated wetlands totaling approximately 0.5 ha (1.3 acres), were not jurisdictional and, therefore, will not require Section 404 permit authorization to impact these wetland features.

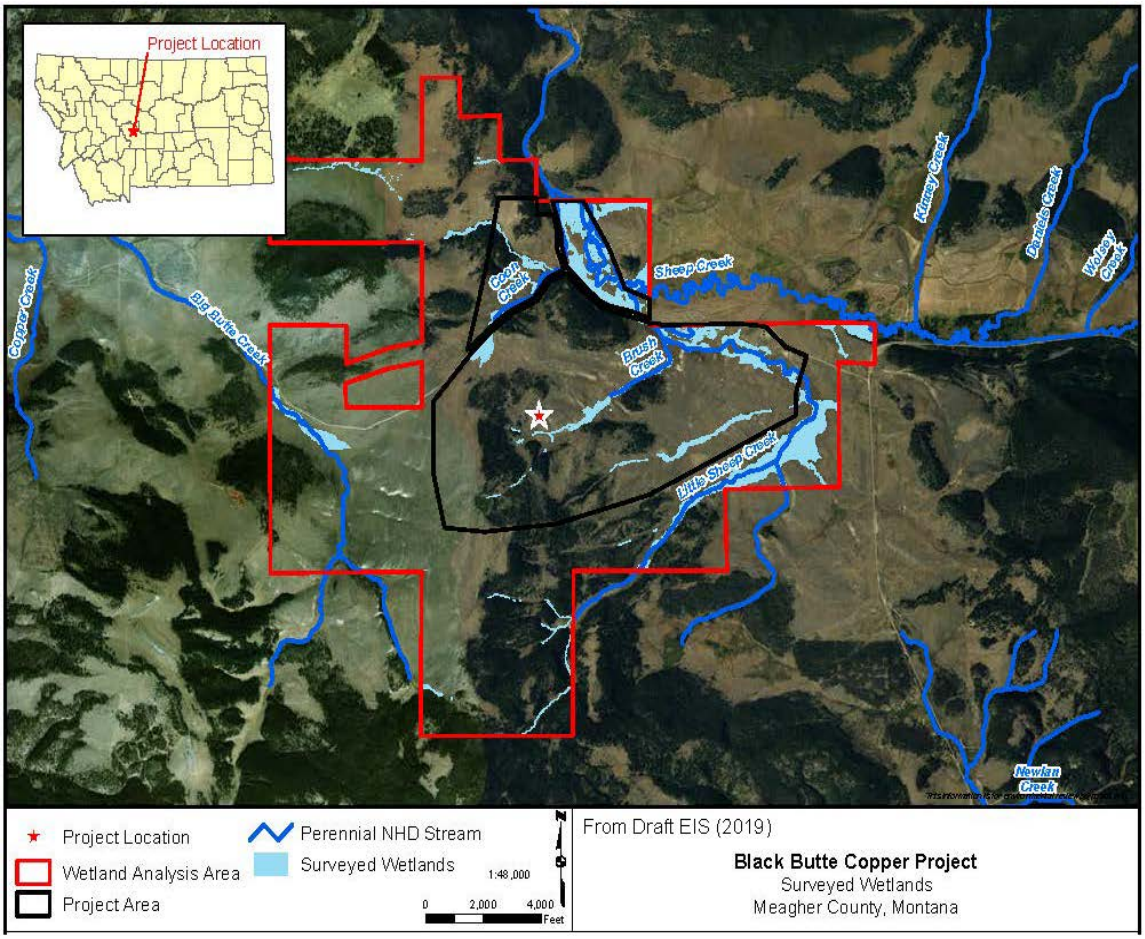


Figure 20.8 Surveyed Wetlands Map

Aquatic Biology

The baseline aquatic surveys and assessments of streams were conducted by Montana Biological Survey during the fall of 2014, the spring and summer of 2015, and all seasons in 2016. Annual and seasonal aquatic monitoring has continued in 2018 and 2019. Results are summarized in Appendix G of the MOP Application. The results from these seasonal baseline surveys are summarized below and included the assessment of fish, mussel, macroinvertebrates, periphyton and stream habitat at sites in the Project Area of the Sheep Creek drainage basin. Montana Biological Survey issued an updated report in February 2018 (Stagliano, 2018) that includes additional 2017 seasonal survey results and included Moose Creek as an additional reference reach.

As previously stated in Section 20.2.1, the entire Black Butte Copper Project study area lies within the Little Belt Mountains in the Middle Rockies Ecoregion. The Sheep Creek watershed upstream from the Project area drains approximately 202 km^2.

Tenderfoot Creek, a 64 km (40 mile) long tributary to the Smith River has a total watershed area of 281 km^2 and was chosen as the off-site control reach. An estimated watershed area of 203 km^2 (78 $miles^2$) is drained above the reference reach. Montana Biological Survey (2017) determined that the watershed areas upstream of the Sheep Creek Project area sites and Tenderfoot Creek reference reaches are nearly identical.

The surveys used a Before, After, Control (upstream and off-site reference) and Impact (BACI) (within and 7.2 km downstream of the proposed mine activity) sampling design. A total of 26 fish surveys were performed and 34 macroinvertebrate and 10 periphyton samples were collected during the visits. Biological community integrity was calculated for the 26 fish surveys using Observed/Expected Models (O/E), while the macroinvertebrate and periphyton samples were assessed with MDEQ Multimetric Indices (MMI) (Teply and Bahls, 2006; MDEQ, 2012).

Stream habitat geomorphology in the survey area is dominated by riffle and runs at all sites with Sheep Creek, Little Sheep and Tenderfoot Creek averaging 85%, 73%, and 75%, respectively. Of the eleven sampling reaches evaluated in the study area, six were found in Proper Functioning Condition with a stable trend, and five were deemed Functional at Risk. Sites were ranked Functional at Risk because they either had riparian habitat altered by cattle or by human stream manipulation. Highest site integrity scores were recorded at Sheep Creek upper and lower reaches and Tenderfoot Creek. The survey determined that the riparian habitat of the lower reference reach on Tenderfoot Creek to be moderately degraded.

Seven fish species and one hybrid (four native/four introduced) from 5,031 individuals were collected at 10 sites during the 26 stream surveys. Rocky Mountain sculpin comprised the highest proportion of total individuals collected (74%) and had 100% occupancy. Other native species included mountain whitefish, longnose dace and white sucker and mountain sucker. Rainbow trout were the dominant salmonid by numbers at almost all of the Sheep Creek sites. Rainbow and brook trout were collected at nine of the ten sites in total while brown trout were detected at seven of the 10 sites. Coon Creek upstream from the county road was determined to be fishless in 2015. No fish SOC were identified during any of the surveys.

A total of 145 macroinvertebrate taxa were reported from the macroinvertebrate survey. No Montana SOC invertebrates were collected. Overall, Sheep Creek MMI scores averaged 62.6 and ranks slightly impaired by MDEQ standards. Both Little Sheep Creek sites were ranked impaired by the MDEQ MMI with scores of less than 63.

A total of 167 unique diatom and algae taxa were reported from the 10 periphyton assessment samples. No periphyton species are listed as SOC in the state. Based on Diatom assessment, one site at lower meadow Sheep Creek had the highest probability (82%) of impairment followed by another site at Sheep Creek at 61%. The other Sheep and Little Sheep Creek sites had a 40% or less chance of being impaired. The Tenderfoot Creek reference sites were ranked least likely to be impaired (<20%). Aquatic benthic macroinvertebrate and periphyton communities at virtually all sites are exhibiting some signs of nutrient enrichment (lowered biotic integrity), likely due to livestock use.

All stream reaches were visually surveyed for amphibians or reptiles during all visits. Observed species included the western toad (a Montana SOC species), and the Columbia Spotted Frog (one site each at Sheep Creek and Little Sheep Creek).The terrestrial garter snake was observed at one site along the banks of Tenderfoot Creek during a summer survey.

Wildlife

The baseline terrestrial wildlife resources in the Project vicinity were evaluated by WESTECH Environmental Services, Inc. for four seasons in 2014-2015 and were summarized in Appendix F of the MOP Application (WESTECH, 2015b). The surveyed area encompassed approximately 2,141 ha (5,290 acres) from the Sheep Creek bottomlands south and west through the adjacent uplands and encompassed the permit area and associated facilities areas. Incidental observations near the study area were also recorded.

Wildlife habitats in the study area are a function of geology, climate, topography, soils, vegetation, and land use. Wildlife habitat components can be broadly defined as food, water, cover, and space. The type, quantity, and distribution of these components determine the kinds of wildlife present in a given area. A list of major terrestrial wildlife species potentially occurring in the study area was derived from general literature sources and species definitions and/or codes in Montana Fish, Wildlife and Parks (MFWP) (MCA 87-2-101; 87-2-102; ARM 12.2.501; Foresman, 2012). The list was further refined by legal status (listed, proposed or candidate species under the Endangered Species Act of 1973; USFWS, 2015), occurrence status (resident vs. migrant), and whether or not preferred/primary habitats were available in the study area.

The study recorded a total of 83 species that included: zero amphibians, one reptile, 20 mammals, and 62 birds. Although the area has limited habitat availability for some species, all of the species recorded during the study were expected based on habitat availability. The total number of species is likely low because many species are difficult to observe by the methods employed during the evaluation. Nevertheless, the study area is considered to support good wildlife species richness. This evaluation recorded no amphibians. However, a Columbian spotted frog and a common garter snake were recorded during the aquatic sampling.

Big game species observed during the study included: pronghorn, elk, mule deer, white-tailed deer and black bear. Black bear were occasionally reported from spring through autumn, although denning in the study area was not observed. The only upland game species observed during the study was the dusky grouse which is considered uncommon. 11 species of raptors were recorded that included: bald eagle, golden eagle, red-tailed hawk, ferruginous hawk, rough-legged hawk, northern harrier, sharp-shinned hawk, northern goshawk, American kestrel, great horned owl, and great gray owl. The study located no nests of any raptors in the area.

The USFWS (2015) identified three terrestrial wildlife species that are listed or proposed candidates for listing under the Endangered Species Act for Meagher County: Canada lynx (listed threatened), greater sage-grouse (candidate), and Sprague's pipit (candidate). The study area does not have the preferred habitat for the Canada lynx and the probability of a sighting in the area is considered to be very low. The USFWS's (2014) delineated Designated Critical Habitat for the Canada lynx in Montana does not include Meagher County. The greater sage-grouse is considered to be a sagebrush-dependent species. There are no known sage-grouse leks (display sites) within 16 km (10 miles) of the study area. The habitat of the study area does comprise suitable Sprague's pipit habitat.

Montana has established lists of vertebrate animal SOC and eleven such species were recorded in the study area. These lists comprise three categories: SOC are "…considered to be "at risk" due to declining population trends, threats to their habitats, and/or restricted distribution." Potential SOC are "…animals for which current, often limited, information suggests potential vulnerability or for which additional data are needed before an accurate status assessment can be made." Special Status Species "…have some legal protections in place but are otherwise not recognized as federally listed under the Endangered Species Act and are not Montana Species of Concern." The identified SOC identified in the baseline survey consist of one Special Status Species, two Potential SOC, and eight SOC with details included in MOP Appendix F.

20.2.3 Cultural (Archeology) and Tribal Resources

Cultural resources include the locations of human activity, occupation, or usage of the environment that contains sites, features, structures, objects, or landscapes that may have important tribal, historic, or archaeological values. The Project is located on private land and there is no federal regulatory involvement; therefore, the federal laws relating to the protection of cultural resources (e.g., Section 106 of the National Historic Preservation Act) do not apply. The Montana Antiquities Act, which applies to activities conducted on state-owned land, also does not apply.

Montana Environmental Policy Act (MEPA) requires identification of known cultural resources within a Project area and a disclosure of what the potential impacts might be to those resources. This consists of a summary of the results of a file search conducted with the Montana States Historic Preservation Office (SHPO). In addition to the file search, SRA conducted cultural resource inventories on areas targeted for mine disturbance to identify cultural resources that may be eligible for listing in the National Register of Historic Places (NRHP) to support the MOP Application. The inventories were conducted by Tetra Tech in 2011, 2012 and 2015 and summarized in Appendix I of the MOP application. These inventories were conducted under the same standards as required by federal law and followed guidelines provided by SHPO. Historical cultural resources identified in the vicinity of the Project include homesteads, ranches, and refuse dumps.

The cultural resource inventory identified a total of 14 prehistoric and six historic sites within a 607 ha (1,500 acre) survey area. The prehistoric sites consist of the Sheep Creek Surface Stone Quarry District site and 13 separate lithic scatter sites that consist of surface scatter of cultural artefacts and debris that consist entirely of lithic (i.e. stone) tools and chipped stone debris. The historic sites consist of two roads, two log cabins, two irrigation ditches, a mine shaft, and a sheep herder's cairn.

Four of the lithic scatter sites occur within the proposed mine facilities and an additional four of the sites occur 8 to 20 m (25 to 65 feet) from the proposed mine facilities. In 2017, as a result of the USACE tribal/Indian Community consultation process and site visits with four invited tribes (Blackfeet, Fort Belknap, Confederated Salish and Kootenai, and Crow), SRA voluntarily moved the access road and the nearby alluvial conveyance pipeline to avoid one of the impacted lithic scatter sites.

SRA in the Final EIS voluntarily agreed that construction of the mine facilities will either mitigate or avoid any cultural resources eligible for listing in the NRHP. Archeological excavations at several of the impacted lithic scatter sites in 2012 and later in 2019 and 2020 occurred mostly prior to the Phase 1 Development surface construction. As a result of the surface excavation and archeological work done to date, the following sites have been recommended for eligibility for listing in the NRHP:

- Three of the identified individual lithic scatter sites that might be impacted by the proposed mine facilities;
- The Sheep Creek Surface Stone Quarry District site, and
- The sheepherder's cairn.

SRA has committed to avoiding the sheepherder's cairn site during construction. The remaining unexcavated lithic scatter sites that will be impacted by the proposed mine facilities will either be voluntarily avoided or mitigated prior to mine construction.

20.2.4 *Environmental Geochemistry*

The geochemistry of the mine waste rock, tailings and exposed surface within the mine are a major factor in the evaluation of the potential impacts of the mine on the environment during operation and closure. Waste rock geochemistry was characterized by Enviromin and summarized in Appendix D of the MOP application.

The proposed mining operation will mine approximately 8.8 Mt (9.7 Mst) of Cu-enriched rock with an average grade of 2.6% Cu and 0.88 Mt (0.97 Mst) of waste rock. Approximately 8.0 Mt (8.8 Mst) of raw tailings will be produced over the life of the Project. The tailings will be thickened and sent to a paste plant where a mixture of cement and slag will be added to the tailings as a binder. The product, called cemented paste tailings, will be piped either to the UG mine to backfill workings or to a double-lined tailings basin called the CTF. Approximately 55% of the cemented tailings paste produced by the Project will be stored in the CTF, with the remaining 45% used to backfill production workings during the sequential mining of drifts.

An extensive geochemical characterization program was conducted to support the MOP application to determine the acid generation and metal leaching potential of the waste rock, tailings and potentially reactive surfaces for all formations that will be intercepted by the mine, with a focus on the ore zones.

Waste Rock

The acid generation and metal release potential of waste rock, construction rock, and tailings to be produced by the Project have been characterized using static test (Acid-Base Accounting [ABA], multi-element analysis, Net Acid Generation [NAG], and static leach tests) and kinetic methods. Mineralogical analyses of metal residence and asbestiform mineral analyses were also completed. Table 20.3 provides a summary of the anticipated waste rock lithologies (as percentage of total waste rock) as well as the number and types of geochemical tests. These waste rock lithology percentages will be refined as the mine plan evolves. Sufficient samples of each waste lithology were collected to statistically evaluate their geochemical variability, including 5,642 samples of the four dominant rock types for metals content, and 138 ABA and NAG analyses for acid generation potential. Results of the ABA and NAG tests indicate that the majority of Ynl B and Ynl A waste rock (90%) are unlikely to form acid, while the USZ and lower zone FW waste rock have an uncertain acid generation potential or are likely to generate acid.

Material type	Litho-types	Description	Waste Rock % Tonnage	ICP	ABA/ NAG	SPLP	Mineralogy	Asbestos	HCT
Waste Rock Materials	LZ FW	Silicified shale and debris flow	35	550	15	0	0	1	1
	Ynl B	Lower Newland shale and conglomerates	32	1412	34	2	1	2	2
	USZ	Lower Newland upper sulphide zone	28	2,542	41	2	1	2	2
	Ynl A	Undifferentiated Lower Newland	4	1,138	48	2	1	2	1
	Total Dominant Waste Rock Samples[1]		**99**	**5,642**	**138**	6	3	7	**6**
	Additional Waste Rock Samples[2]		**<1**	**1,855**	**37**	3	1	4	**2**
	All Waste Rock Samples[3]		**100**	**7,497**	**175**	9	4	11	**8**
Near-surface Materials	Ynl Ex	Near-Surface Lower Newland shale	<1	108	10	--	--	1	1
	Tgd	Tertiary Granodiorite	<1	76	8	--	--	1	1
	Total Excavation Tonnage		**NA**	**184**	**18**	--	--	2	**2**

[1] ***Total waste rock*** *tonnage over the life of the mine equals 706,525 tonnes (778,810 tons) as reported in the MOP Application (Tintina Montana, 2017). These waste rock lithology tonnages will be refined as the mine plan evolves. A total of 7,497 ICP analyses of waste rock were evaluated.*

[2] ***Four waste rock*** *types will be mined above 1% of total tonnage; 5,642 ICP analyses were evaluated for these units.*

[3]*Additional waste rock units were characterized representing less than 1% of tonnage; 1,855 samples were evaluated for these units.*

Table 20.3 Geochemical Testing of Major Waste Rock and Near-Surface Materials by Lithotype

Although asbestiform minerals are highly unlikely to occur in the rock units in the Project area based on the site lithology, asbestiform mineral testing was included in the characterization work completed for all waste rock units. No asbestiform minerals were identified in any lithotype.

Kinetic tests of waste rock acid generation and metal release potential were conducted following ASTM method D5744 for Humidity Cell Tests (HCT's). The tests were run for 24 to 88 weeks, with the time period determined by the results of the tests. All HCT's were conducted on composites of subsamples from the individual lithologies, determined by a statistical analysis of static test results. Kinetic testing has shown evidence of sulphide oxidation in the four dominant waste rock units. However, consistent with the static test results and the presence of abundant carbonate mineralization, acid generation in waste rock HCT's was limited. Furthermore, metal release from waste rock HCT's was varied. The Ynl A and Ynl B released relatively low concentrations of a few metals (with nickel and thallium exceeding groundwater standards in the initial weeks of testing). In contrast, the USZ released strontium and thallium at concentrations exceeding groundwater standards throughout the test, with additional metals (notably Cu, Pb, and Ni) exceeding groundwater standards.

Tailings

Static test and kinetic tests were also performed on tailings produced from metallurgical pilot tests and paste tailings from pilot thickener tests (Table 20.4). Tailings were mixed with 2 to 4% binder, consisting of equal parts of Portland cement Type I/II and slag from Lafarge. The addition of cement provides structural strength in support of drift and fill mining methods, and to change the physical properties of the material to a stable, non-flowable material with low hydraulic conductivities (approximately 10^{-9} m/sec) in both surface and UG settings. While tailings with cement/slag binder was the only amended tailings material that underwent geochemical testing, Lafarge Type F fly ash was also evaluated as a potential binder. Combinations of tailings with different binders may change over the life of mine but should not significantly change the geochemical characteristics of the cemented tailings nor how they are managed.

Tailings Test Material(s)	ABA	NAG	ICP metals	Sat. HCT	Unsat HCT	Diffusion Test
Straight (Raw) Tailings	X	X	X	X	X	-
Paste Tailings 2%	X	X	X	-	X[1]	[2]
Paste Tailings 4%	X	X	X	-	X[1]	X
Paste Tailings 4% and Waste Rock	-	-	-	-	X[1]	X

[1] *Unsaturated (Unsat) HCTs conducted on intact cement paste cylinders,*

[2] *An attempted test of 2% cemented paste tailings could not be completed.*

Table 20.4 Black Butte Copper Project Tailings Treatments and Related Testing

Kinetic tests of raw (non-amended) and cemented paste tailings were completed using a variety of methods. Table 20.5 summarizes the tailings characteristics, testing methods and conditions, and the various operational scenarios represented by each kinetic test of tailings. Cemented paste tailings cylinders were tested (without crushing) in conventional HCT's to simulate subaerial weathering. They were also subjected to diffusion tests to simulate diffusion through cemented backfill in saturated UG workings. Raw, non-amended tailings were also tested using both sub-aerially and in a modified, saturated test, to represent dry stack surface placement and subaqueous impoundment deposition scenarios, respectively.

Action Scenarios	Facility Represented	Tailings Characteristics	Test Method
Proposed	Backfilled Paste in flooded workings	4% binder	ASTM C1308 diffusion test
	Cement paste in CTF, subaerial weathering, routine operations	2% binder	ASTM method D5744 (HCT)
	Cement paste in CTF, subaerial weathering, final closure lift	4% binder	ASTM method D5744 (HCT)
Alternative	Saturated tailing e.g., subaqueous impoundment	Straight (Raw)	Modified ASTM method D5744 (saturated HCT)
	Subaerial weathering, e.g., dry stack tailing pile	Straight (Raw)	ASTM method D5744 (HCT)
Additional*	Cement paste in CTF, subaerial weathering	4% co-disposed with waste rock	ASTM method D5744 (HCT)
	Backfilled Paste in flooded workings	4% co-disposed with waste rock	ASTM C1308 diffusion test

**Geochemical testing of paste tailings mixed with ROM was conducted to evaluate previously considered scenarios that are no longer pertinent to the mine operational plans. These tests are described here for completeness but are not relevant to compliance assessments or modelling. See MOP Appendix D for data. ASTM = ASTM International; HCT = Humidity Cell Test*

Table 20.5 Tailings Characteristics, Kinetic Test Methods, and Facility Scenarios

ABA and NAG tests indicate that the tailings would have a strong potential to generate acid regardless of cement addition. The HCT's indicate that all of the cemented paste tailings samples had potential to oxidize and to release sulphate, acidity, and metals if left exposed to air and water. Importantly, this was not observed immediately in test cells, and the rate of weathering in a humidity cell is recognized to be significantly greater than in the field, since the increased surface area and weekly air/water cycles increase the sample reactivity. The cement binder provides structural stability and some alkalinity but does not neutralize all the acid produced by sulphide oxidation.

20.2.5 Socioeconomic Description

Information on demographics and socio-economics of the local communities was evaluated to determine the availability of employees for the mining operation and the potential socio economic impact of the mine on the communities.

Meagher County is sparsely populated by both Montana and US standards. The land area is 6,194.8 km^2 (2,391.82 miles^2) and the population density is 0.8 people per mile^2, while the average for Montana in 2010 was 7.0 people per mile^2. Between 2010 and 2014, the population decreased from 1,891 to 1,853. The US Census Bureau reports that migration out of the county is greater than migration into the county, and the number of births has also decreased.

Table 20.6 lists the population of White Sulphur Springs and other nearby towns and their distance from the Project site. Figure 20.1 shows locations of towns. More than 142,000 people live less than 161 km (100 miles) from the Project site.

Town or City	Population	Distance from White Sulphur Springs
White Sulphur Springs	925	0
Billings	108,869	164
Great Falls	56,690	64
Bozeman	41,660	95
Helena	29,943	91
Livingston	7,245	87
Lewistown	5,867	130
Townsend	1,942	57
Three Forks	1,903	92
Harlowton	974	82
Belt	604	35
Martinsdale	530	51
Ringling	45	36

Table 20.6 Population of Towns and Distance from Project Site

The demographics data for the county indicates that 25% of the population is over 65 and 24.6% is under the age of 18. Approximately 50% of the county's population is of employment age, compared to 56% for the state.

Meagher County is rural and the main industry is farming and ranching. Table 20.7 shows the industries in the county and trends in employment between 2001 and 2011. The total number of people employed in Meagher County in 2011 was 697.

Total Employment (Number of Jobs)	2001	2011	Change 2001-2011
Farm	227	179	-48
Retail trade	76	107	31
Real estate and rental and leasing	8	37	29
Administrative and waste services	24	na	na
Educational services	5	11	6
Arts, entertainment, and recreation	52	78	23
Accommodation and food services	109	84	-13
Other services, except public admin.	68	59	-9
Government	180	145	-35
Total Employment (%)			
Farm	19.5%	13.5%	-21.1%
Retail trade	6.5%	9.5%	34.2%
Real estate and rental and leasing	0.7%	8.1%	40.8%
Administrative and waste services	2.1%	na	na
Educational services	0.4%	0.8%	373.2%
Arts, entertainment, and recreation	4.4%	5.9%	42.9%
Accommodation and food services	9.4%	6.4%	-13.7%
Other services, except public admin.	5.8%	4.5%	-13.2%
Government	15.5%	14.1%	-16.1%

Table 20.7 Meagher County Employment by Industry, 2001-2011

The unemployment rate is an indication of the potential available employees. Meagher County and Montana reported lower than average unemployment rates for June 2015 of 3.5 and 4%, respectively, compared to the US unemployment rate of 5.3%.

Income is reported by the US Census as “per capita” and household. The per capita takes the total income for the county, state, or country and divides it by the total population for an indication of the income per person. The household income is reported as the median household income. Table 20.8 reports these numbers for Meagher County, Montana, and the US, along with the percent of the population that is considered below the poverty level.

Income Level	Meagher County	Montana	US
5-year (2009-2013) average per capita income in past 12 months (2013 dollars)	$20,288	$25,373	$28,155
Median household income, 2009-2013	$38,182	$46,230	$53,046
Persons below poverty level, percent, 2009 - 2013	13.6%	15.2%	15.4%
US Census 2015a, 2015b			

Table 20.8 Per Capita and Household Income

20.3 Environmental Issues and Impacts

20.3.1 Current Liabilities

SRA has conducted exploration under Exploration License #00710 issued by the MDEQ. Regulations include the bonding of exploration disturbances to ensure reclamation is completed. SRA currently has an obligated bond of $137,365 for completion of the reclamation of the 2018/2019 Phase 2 and earlier drill programs. These obligations will be released when the reclamation is completed by SRA and inspected and approved by the MDEQ. In addition, there are approximately 37 monitoring wells/test wells, and one water well, and 15 piezometers currently in place that will ultimately need to be removed during closure and reclamation.

20.3.2 Potential Impacts

Potential short and long-term impacts caused by mining activities were evaluated from several perspectives: impacts to the environment during operation and closure, issues or impacts that could materially affect the mine's ability to extract the mineral resources or access mineral reserves, and socio-economic impacts.

Potential impacts to the environment were addressed in detail in the EIS (MDEQ, 2020) and are summarized below.

Surface and Groundwater

Operation of the mine dewatering system is believed to significantly lower groundwater levels around the mine and somewhat reduce base flow in nearby creeks based on water modelling work. Springs and seeps within the area should not be affected, as they appear to be fed by local perched systems that are not connected to the groundwater system. Operation of the alluvial Underground Infiltration Gallery (UIG) will increase groundwater discharge to Sheep Creek in the vicinity of the UIG, partially compensating for the decreased base flow caused by mine-dewatering. The NCWR will recharge groundwater beneath this pond, partially compensating for the decrease in base flow caused by mine-dewatering. Decreases in stream flow will be largely offset by the Water Rights Mitigation program.

Contact groundwater in post-mine voids would migrate via shallow bedrock toward discharge zones mixing with non-contact groundwater. Transport of chemicals dissolved in contact groundwater would be retarded by the process of adsorption, and groundwater discharging to Sheep Creek would not affect its water quality. Some of the seeps and springs located within the anticipated mine drawdown cone were developed for stock use (Figure 20.1). If springs and seeps located within the mine drawdown cone are connected to the deep groundwater system they may experience decreased flow, and some might dry up. However, many of the springs and seeps appear to be connected to perched groundwater bodies and also may only flow seasonally; these would not likely be directly affected by creation of the groundwater drawdown cone in the deeper HSU's. The mine will have to provide replacement water for any springs that are being put to

beneficial use and are depleted by dewatering. Vegetation and wildlife may be affected at the springs or seeps depleted by dewatering, to limit these effects there is a monitoring program and mitigation plan to maintain wetlands throughout the Project area. Spring flow would be anticipated to re-establish when shallow groundwater recovers to baseline conditions, within two years after the cessation of dewatering.

Air Quality

Quantitative modelling was conducted to evaluate the potential air quality impacts of the Proposed Action, including the impacts of underground and aboveground stationary sources. Air dispersion modelling was performed primarily to quantify concentrations of regulated pollutants resulting from stationary and fugitive source emissions, and these results were compared to federal and Montana ambient air quality standards.

Predicted impacts for criteria pollutants at all offsite locations comply with health-based Montana and federal primary standards, which are protective of ambient air quality. Project emissions of haze precursor pollutants are sufficiently below regulatory thresholds to not warrant evaluation of haze/visibility impacts. Modelled concentrations for the Proposed Action were predicted to be less than the regulatory Significant Impact Level (SIL) at all locations within the basin. Consequently, those impacts were judged to be negligible in extent and magnitude.

Stream Water Quality and Flow

Potential impacts to surface water quality and flow are summarized in Table 20.9 (MDEQ, 2020). Overall, the anticipated impacts are expected to be minimal, as summarized below.

- Minimal surface disturbance will result in insignificant impacts on surface runoff.
- Diversion of water to the NCWR falls within existing leased water rights (pending review and approval of the MDNRC).

Secondary impacts on base flow of Sheep Creek as a result of mine dewatering and disposal of treated water to the UIG are expected to be insignificant and to partially offset one another. A more significant impact upon base flow will be possible for Coon Creek, with the total reduction in Coon Creek estimated to be approximately 70% of the steady state base flow. Impacts to Coon Creek will be mitigated by pumping water from the NCWR into the headwaters of Coon Creek to augment flows within 15% of the average monthly flow (Hydrometrics 2018c).

Water Supply

A critical issue in the ability of a mine to process ore is the availability of water supply. The mine will depend entirely on groundwater produced from the UG workings at an average rate of 32 L/sec (500 gpm). (Hydrometrics, 2019). To address the potential risk of insufficient water supply, Hydrometrics (2019) performed an evaluation of the basin water balance using the worst-case estimates for mine dewatering and precipitation capture.

Project Phases	Project Facilities/Activities	Notes
Mine Construction (Phases I and II; Project Years 1 - 4)	Surface disturbance affecting runoff	Surface disturbance is less than 1% of local watershed area. Best management practices and the relatively small percentage of the total area (<1%) of stream and wetland features will be impacted through surface disturbance during construction.
	Diversion of stream flow to the NCWR	Based on existing leased water rights along Sheep Creek (pending review and approval by the MDNRC).
	Mine dewatering	Simulated base flow depletion for all streams except Coon Creek within surface base flow measurement error (±10%), Coon Creek base flow reduction will be offset with water from the NCWR and through an agreement with the water rights' holder to utilize the water rights (pending approval with the MDNRC).
	Underground infiltration gallery	Partially compensates for the potential loss of base flow in Sheep Creek.
Mine Production (Phase III; Project Years 5 – 15)	Surface disturbance affecting runoff	Surface disturbance is less than 1% of local watershed area.
	Diversion of stream flow to the NCWR	Based on existing leased water rights along Sheep Creek.
	Mine dewatering	Simulated base flow depletion within surface base flow measurement error (±10%).
	Underground infiltration gallery	Partially compensates for the potential loss of base flow in the Sheep Creek.
Post-Mine Period (Mine Closure and Post-Closure, Phase IV)	Surface disturbance affecting runoff	Surface disturbance is less than 1% of local watershed area.
	Diversion of stream flow to the NCWR	Not required after consumptive use of groundwater stops.
	Mine dewatering	Base flow depletion is expected to cease within two years after dewatering stops.
	Underground infiltration gallery	No discharge to UIG after UG mine is closed and water treatment no longer necessary.

Table 20.9 Potential Impacts Regarding Surface Water Quality and Flow

Water Disposal

The current mine plan only considers disposal of excess water via the UIG. The sensitivity analysis performed by Hydrometrics (2019) estimates that the UIG will be capable of disposing of all treated water even under the maximum water production scenario. Additional options for disposal of treated water were identified by Hydrometrics (2019a) and include deep well disposal, land application, and forced evaporation.

Soils

All suitable topsoil and subsoil within the recommended salvage depths will be removed prior to commencing construction activities. To the extent practicable, salvage activities will be timed to avoid periods of wet or saturated soil to limit soil compaction. If possible, soil removed from an area will be hauled directly to, and used to reclaim, another previously disturbed area (thus eliminating the need for prolonged storage). Soils removed during road and diversion ditch construction will be concurrently used to revegetate adjacent cut and fill slopes. A soils specialist will be present on-site during initial soil salvage activities to provide oversight of soil suitability and establish salvage guidelines for specific soil types or landscape features.

It is expected that soil thickness will vary considerably throughout the proposed disturbance area, although soil volumes are estimated to be sufficient to support the proposed reclamation. However, in the event of a shortage of cover soil, soils containing coarse fragments in excess of 50% by volume will be screened and salvaged for use in reclamation so that no offsite topsoil will be required. Coarse material will be used as fill during reclamation and closure.

If revegetation problem areas are encountered, the cause will be identified. If the cause appears to be related to soil infertility or toxicity, a soil testing program will be implemented. Appropriate remedial actions will be taken to correct any significant problem identified by MDEQ.

Land Use

Impacts on land use will include the loss of use of approximately 126 ha (311 acres) of ranching/livestock grazing and hay production lands due to construction and operations of the Project. These direct impacts will last about 19 years through mine construction, operations, closure, and reclamation. No direct impacts on land use for lands adjacent to the Project area will occur. No conflicts with adjacent land uses are anticipated given that there are no zoning restrictions in this area. No direct impacts on recreation will occur in the proposed disturbance footprint.

Wetlands

Potential impacts to wetlands will result from construction of the access and/or service roads, the CTF, and the wet well proposed to be constructed for diverting and piping Sheep Creek spring runoff water.

Filling or excavation of wetlands would result in permanent direct impacts to wetlands. The wetland impact analysis identifies wetland type, total area of direct impact, percent of analysis area, and the wetland name to be impacted by the Project. Installation of the CTF, the wet well for the Sheep Creek water diversion, and associated mine facility access and service roads would result in approximately 0.34 ha (0.85 acres) of permanently impacted wetlands from fill and dredging activities.

Waste Rock and Tailings

The waste rock and tailings have the potential to impact surface and groundwater if not managed to minimize their exposure to the environment. The predicted environmental impacts of rock geochemistry based on the interaction of groundwater and the exposed reactive surfaces are discussed in the water resources section. To protect shallow groundwater and surface water from potential acid generation and metal leaching of sulphidic mine waste or tailings, the mine wastes will be managed to isolate the wastes from the surface environment.

Socio-Economic

As required by Montana statute, a Hard Rock Mining Impact Plan (HRMIP) has been negotiated with local government entities (Meagher County, the City of White Sulphur Springs and School District #8) and approved by an independent HRMIP Board through the Montana Department of Commerce. This plan projects the fiscal impacts the proposed mineral development will have on the affected units of local government based on the best available information. The Impact Plan also identifies mitigation measures, as well as adjustment and amendment provisions to mitigate actual impacts through payments deposited upfront into an escrow account upon receipt of the Record of Decision (ROD). These funds are held and used in the lieu of future taxes.

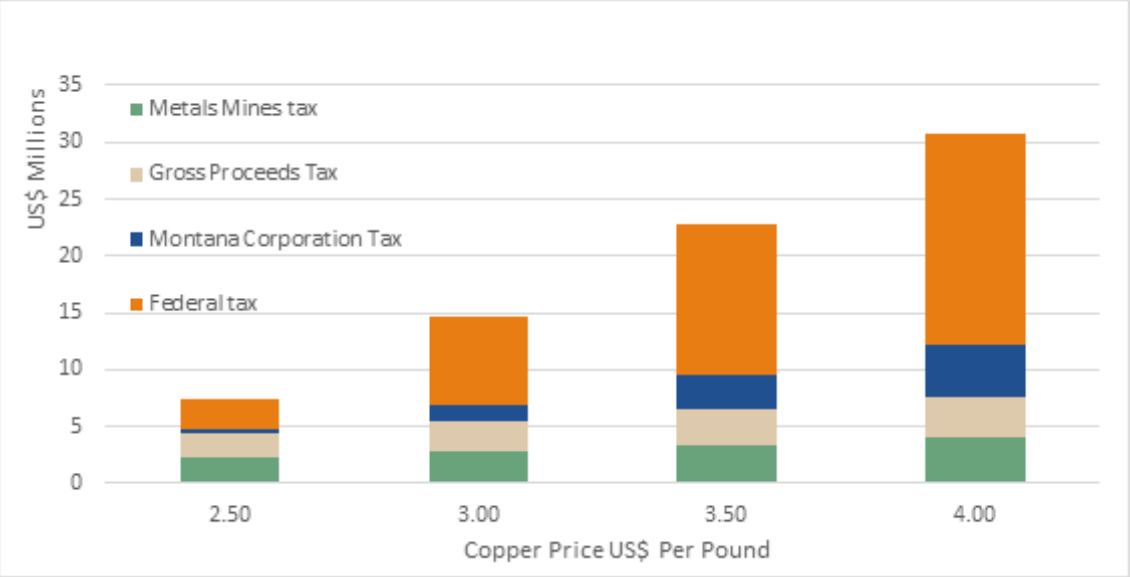
Total projected impact payments during mine construction to the start of mining are $437,000. Meagher County is projected to experience increased net operating costs in the Sheriff's Department and Planning Services during the first three years of mine construction and development projected at $338,700. A method for tax crediting is proposed. The City of White Sulphur Springs is projected to experience net operating costs in planning services projected to total $98,300.

No net impact costs are projected for White Sulphur Springs School District #8 during the mine construction time.

Increased Local Government Revenue

Meagher County's taxable value is projected to increase by an estimated $6.7 million after the Black Butte Copper mine is constructed and in commercial production with a taxable value projected at $8.2 million at peak Cu production in approximate Year 7. The current taxable value of Meagher County is $9 million, so this will almost double the County's taxable value.

Figure 20.9 represents estimated taxes for the Black Butte Copper mine at different price points based on 400 tonnes per day Cu concentrate production.



** *As presented in the 2019 Black Butte Copper Community Presentations*

Figure 20.9 Estimated Average Yearly Taxation

In the HRMIP, it is projected over $4 million per year will be paid in metal mines license tax to the State of Montana. Current Montana Statutes require 35% of the metal mines license tax be distributed to affected counties. The HRMIP estimated over $1.4 million per production year to Meagher County and at least $6M plus in the Hard-Rock trust account over the LOM. During production years, this is the primary source of funding available to affected local governments to mitigate impacts from a significant reduction in workforce or closure of the mine.

Metal Mines Gross Proceeds will be paid to Meagher County at 3% of the annual gross proceeds and at a tax rate of 3%. Gross proceeds by statue is defined as the monetary payment or refined metal received by the mining company from the metal trader, smelter, roaster or refinery.

Other local property tax will be in the form of business equipment taxes and expanded opportunities ancillary to the Black Butte Copper Project.

20.4 Legal Framework

The MOP Application was designed to meet the requirements of the Montana Metal Mine Reclamation Act (MMRA)(Title 82, Chapter 4, Part 3, MCA) and the rules and regulations governing the Act. Additional permits (Section 20.8.1) for water rights were obtained through the MDNRC.

Compliance with the applicable legal requirements is demonstrated by the MDEQ's approval of the following: Final MOP, Air Quality Permit, National Pollutant Discharge Elimination System (NPDES) and construction storm water permit. A Draft EIS was published by the MDEQ on March 11, 2019, as required under MEPA, Section 75-1-201, *et seq*., MCA, and finalized on March 13, 2020. Subsequently, the MDEQ issued a ROD for the mine on April 9, 2020, identifying MDEQ's decision, the reasons for the decision and special conditions surrounding the decision and its implementation.

A detailed list of all the Project permits and plans is provided in Table 20.10.

	Permit/Plan	Agency	Status
Mine Operating Permit	Exploration Licence	MDEQ: Air, Energy and Mining Div, Hard Rock Mining Bureau	Approved
	Environmental Impact Statement – Record of Decision	MDEQ: Air, Energy and Mining Div, Hard Rock Mining Bureau	Approved
	Hard Rock Mining Operating Permit	MDEQ: Air, Energy and Mining Div, Hard Rock Mining Bureau	Approved
	Full Project Reclamation Bond	MDEQ: Air, Energy and Mining Div, Hard Rock Mining Bureau (Increment 1 Approved 8/14/2020)	In Progress
Water Quality	Montana Pollution Discharge Elimination System Permit	MDEQ: Water Quality Div., Water Protection Bureau	Approved
	Montana Groundwater Pollution Control System Permit	MDEQ: Water Quality Div., Water Protection Bureau	Approved
	General Permit - Storm Water Discharges Associated with Const Activity	MDEQ: Water Quality Div., Water Protection Bureau	Approved
	Spill Prevention, Control and Countermeasures Plan	MDEQ: Permitting and Compliance Div., Waste and Underground Tank Management Bureau	Submitted
Water Rights	Certificate of Water Rights/Groundwater Appropriations	MDNRC, Water Rights Bureau.	In Progress
Water – Other	Public Water Supply Permit	MDEQ: Water Quality Div., Public Water and Subdivisions Bureau	Not Initiated
	Sewerage Disposal	Meagher Country Health Department	Not Initiated
Wetlands Streambeds	Clean Water Act Section 401 Permit	USACE, MDEQ: Water Quality Div., Water Protection Bureau	Approved
	Clean Water Act Section 404 Permit	USACE	Approved
	MT Streambed Preservation Act 310 Permit	Meagher County Conservation District, MDEQ: Water Quality Div., Water Protection Bureau	Approved
	MT Streambed Preservation Act 318 Permit	Montana Fish, Wildlife and Parks, MDEQ: Water Quality Div., Water Protection Bureau	Approved
Aquatics	Aquatics Monitoring Program	MDEQ: Air, Energy and Mining Div, Hard Rock Mining Bureau; Montana Fish, Wildlife and Parks	Submitted
Dam Safety	Dam Safety/Hazard Evaluation	MDNRC, Water Resources Div., Dam Safety Bureau	In Progress
Tribal Communications	Associated with 404 Permit	US Army Corps of Engineers	Completed
Air/Noise	Air Quality Permit	MDEQ: Air Quality Bureau (Amended on August 10, 2020)	Approved
Hard Rock Mining Impact	Hard Rock Mining Impact Plan	MT Department of Commerce, Community Development Div., Hard Rock Mining Impact Board	Approved
Power Transmission Line	Mt Major Facilities Siting Act	MT Public Service Commission	In Progress
Invasive Vegetation	Weed Plan	Meagher County Noxious Weed Management	Approved
Cultural Resources	Historical Preservation Act	MT State Historical Preservation Office	In Progress
Emergency	Emergency Response Plan	MDEQ: Air, Energy and Mining Div, Hard Rock Mining Bureau State Fire Marshal	Submitted
Mining Operations	Notification of Commencement of Operations	US Mine Safety and Health Administration	In Progress
	Hazardous Waste ID	US Environmental Protection Agency, US Department of Energy	Not Initiated
	FCC Radio Licenses	Federal Communications Commission	In Progress
	Explosives Permit	Bureau of Alcohol, Tobacco, Firearms and Explosives	Not Initiated

Table 20.10 List of Permits Required, Plans Requiring Submission and Acts for Compliance.

All activities proposed in the MOP and all surface disturbances will occur on privately owned ranch land (Figure 20.2). SRA has entered into agreements with surface, mineral, and water rights owners on approximately 3,110 ha (7,684 acres) of private lands, and also controls 525 mining claims contiguous with the fee simple (leased) lands. Figure 20.3 shows the proposed mine permit boundary and the location of the Johnny Lee deposit relative to the lease lands. SRA has established its legal right to explore for and develop mineral resources in the Mine Permit Area by virtue of its mineral rights lease agreement with the mineral rights' owners.

20.5 Environmental Management Systems

The need to perform in compliance with even more stringent environmental legislation and to satisfy different stakeholders demands and pressures, is progressively leading the mining sector towards the implementation of Environmental Management Systems (EMS). SRA plans to develop an EMS for the mine to facilitate compliance with environmental regulations and ensure that corporate environmental policies are adopted and followed.

An EMS is a framework that helps an organization achieve its environmental goals through consistent review, evaluation, and improvement of its environmental performance. The assumption is that this consistent review and evaluation will identify opportunities for improving and implementing the environmental performance of the organization. Basic elements of an EMS include the following:

- Review of the organization's environmental goals and performance;
- Analyzing its environmental impacts and legal requirements;
- Setting environmental objectives and targets to reduce environmental impacts and comply with legal requirements;
- Establishing programs to meet these objectives and targets;
- Monitoring and measuring progress in achieving the objectives;
- Ensuring employees' environmental awareness and competence; and,
- Reviewing progress of the EMS and making improvements.

A key component of an EMS is the site-specific Quality Assurance Project Plan, which provides detailed descriptions of the procedures, personnel, processes, equipment, data quality objectives, data review and many other details that ensure that site data generated from measurements and samples meet Project requirements for completeness and representativeness.

20.6 Occupational Health and Safety

SRA's Health and Safety Policy outlines the Company's commitment to our prime goal, "Everyone goes home safe every day". We are committed to establishing a successful safety program through engaged Leadership and a true 'Team' culture.

As SRA is fully committed to identifying, assessing and controlling risks associated with its operations, a Mine Safety Plan has been developed in accordance with Federal and State Regulation regarding OSHA and MSHA safety requirements. The Mine Safety Plan is intended to deliver a high standard of Health & Safety performance, allowing the Management Teams to meet Health & Safety Policy objectives, Legislative requirements and drive continuous improvement processes.

SRA ensures that the Mine Safety Plan is clearly communicated and summarized to all employees, contractors and visitors via Site Induction before any work commences. The Mine Safety Plan is a public document placed in accessible locations for employees and business partners to utilize. This plan is regularly reviewed and updated in accordance with 30 CFR, in consultation with the Mine Safety Committee. Contributions from stakeholders are taken into consideration.

The Mine Safety Plan sets measurable targets against industry best practice, defining a consistent structured approach to effective Health & Safety Management.

20.7 Waste (Tailings) Management

Approximately 8.0 Mt of raw tailings will be produced over the life of the mine. The tailings will be thickened and sent to a paste plant where cement, slag, and/or fly ash could be added to the tailings as a binder. The product, called cemented paste tailings, will be piped either to the UG mine to backfill workings or to a double-lined tailings basin called the CTF. Approximately 55% of the cemented tailings paste produced by the Project will be stored in the CTF, with the remaining 45% used to backfill production workings during the sequential mining of drifts.

20.7.1 Tailings Properties

ABA and NAG tests indicate that the tailings will have a strong potential to generate acid regardless of cement addition. The HCT's indicate that all of the cemented paste tailings samples had potential to oxidize and to release at least some sulphate, acidity, and metals if left exposed to air and water. Importantly, this was not observed immediately in test cells, and the rate of weathering in a humidity cell is recognized to be significantly greater than in the field, since the increased surface area and weekly air/water cycles increase the sample reactivity. The cement binder provides structural stability but does not completely neutralize sulphide oxidation.

20.7.2 Tailings Production Profile

Based on the mine reserves listed in Section 15, a total of 8 Mt (8.8 Mst) of raw tailings will be generated from the milling process shown in Table 20.11 below. Binder will be added to the raw tailings for both underground and surface disposal. Over the LOM the tailings will be split 55% and 45% between surface deposition into the double-lined CTF or transported back UG as cemented tailings paste backfill, respectively.

Year	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9
Raw Tailings Generation (in tonnes)	189,175	1,070,363	1,060,467	1,069,574	1,085,325	1,105,162	1,102,549	940,254	377,618

Table 20.11 Total Tailings Generated over the LOM Showing Tailings Quantities.

20.7.3 CTF Operation and Tailings Delivery

As listed in Table 20.3, the waste rock types that will be mined and ultimately disposed of in the CTF include the FW of the LSZ, Lower Newland Formation dolomitic shale (Ynl B) and portions of the USZ. The LS FW will be exposed in UG workings and temporarily in active stopes. Some waste rock will also be stockpiled for approximately two years on a lined surface pad prior to being co-disposed with cemented tailings early in mine life. Once the temporary WRS pad is reclaimed, all of the waste rock, including the rock to be mined from the LZ FW during development, will report directly to the CTF for use in constructing the foundation drain and ramp. Waste rock produced after the CTF begins full operations will be encapsulated in cemented paste tailings in the lined CTF impoundment to both minimize the amount of contact water and limit the influx of oxygen. This will delay the potential onset of acid generation in waste rock, as well as reduce the volume of water that might require treatment.

During mine operation, tailings produced from the flotation plant will be blended with cement/binders to create cemented paste tailings. The mine will use a DF mining method, placing 45% of produced tailings mixed with 4% cement and binder as backfill into mined out UG stopes and access headings during operations. The remaining tailings (approximately 55%) will be amended with as much as 2% cement and binder, and transferred as paste into a double lined CTF. The operational plan for the CTF is to utilize an internal sump to rapidly transfer any water from the CTF to the PWP, providing for little or no water storage on the facility.

20.7.4 CTF Monitoring

Proper operation, monitoring, and record keeping of the CTF and other water and waste management facilities are included in the Tailings Operations, Monitoring and Surveillance (TOMS) Manual. This TOMS Manual was prepared by Knight Piesold in 2017 and developed to comply with Montana State Law as described in MCA Title 82 Chapter 4 Part 3 Section 79 (82-4-379). The principal objectives of the TOMS Manual include:

- To describe the roles and responsibilities of SRA site personnel, third party consultants, and regulators for the management of the facilities;
- To identify the operation, maintenance, surveillance, and inspection requirements;
- To define training requirements for those involved in the operation of the facilities, and
- To provide details on the emergency processes, plans, and procedures.

The TOMS Manual is currently in the design and permitting phase and is not yet in operation and will be updated as development of the mine progresses.

20.8 Water Management

Water will be managed using several facilities during the life of the mine. The water management ponds including the BCWP, TWSP, NCWR, and PWP, and associated foundation drain ponds previously described in Section 18. The primary source of water for use in the mill will be from dewatering of the UG mine workings; this water will be stored in the PWP. Three separate water supply systems consisting of a process water supply, fresh water supply, and potable water supply will be required for the Project and are summarized in the sections below. A RO water treatment plant capable of treating up to 32 L/sec (500 gpm) will be needed throughout mine construction, UG production mining, and during UG mine closure. Figure 20.10 and Figure 20.11 below show the average annualized water balance schematic during mine operations (Zieg, 2018).

20.8.1 Water Supply/Water Rights

The primary source of mill process water will be recycled water from the PWP, which will be stored in the process water tank. Mill make-up water will be provided by mine dewatering.

Fresh water for the mill (from the fresh/fire water tank) will be obtained from the PWP or mine dewatering and be treated through media filtration near the mill. The mill fresh water will be treated through a separate WTP from the RO WTP and will be used for other milling purposes. Water supplies for the mill and the water treatment/storage system were designed to comply with restrictions by the MDEQ for water that will be discharged under the MPDES permit. The MDEQ requires that no process water will be comingled with water that is treated and then discharged to the MPDES outfall (alluvial UIG). Process waters include water from the PWP, CTF, or mill. The mine water supply, circulation, treatment and discharge systems are shown in Figure 20.10. Potable water (from the potable water tank) will be sourced from a water supply well (PW-6) that is located approximately 200 m north of the secondary escapeway identified in Figure 20.2 and will be treated as necessary for human consumption.

Water stored in the NCWR will be used to offset potential adverse effects to surface water rights from the consumptive use of groundwater used in the mining process. Water will be transferred to Sheep Creek, Coon Creek, and Black Butte Creek to offset the stream depletions. In addition, water will be allowed to seep from the NCWR through its unlined bottom to groundwater and the downstream catchment. Seepage from the NCWR is expected and is intended to offset a portion of mine site water consumptive use. Analyses indicate an average seepage rate of less than 3.2 L/sec (50 gpm). Because the reservoir will contain natural surface water, it will not have the potential to chemically degrade groundwater.

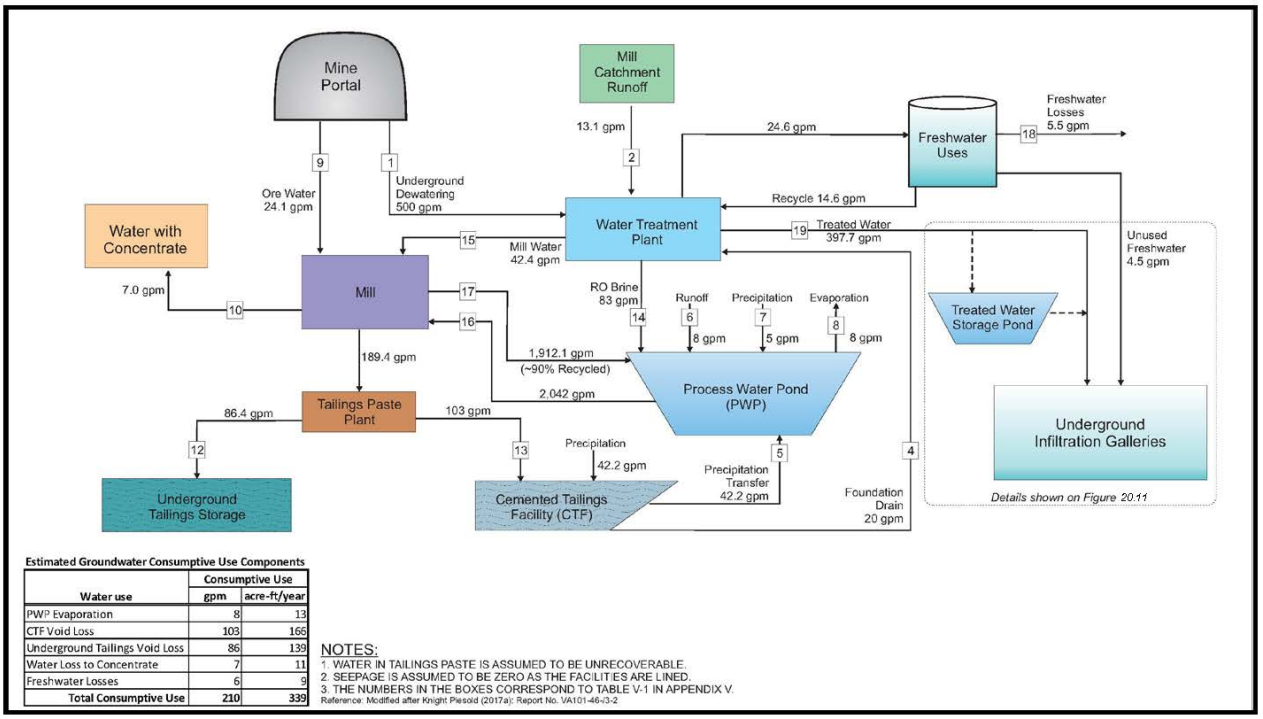


Figure 20.10 Average Annualized Water Balance During Mining Operations

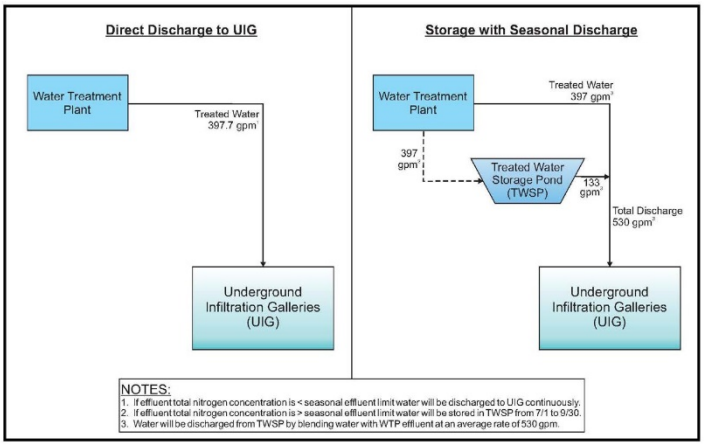


Figure 20.11 Average Annualized Water Balance During Mining Operations (Part 2)

20.8.2 *Water Balance*

A water balance was prepared for the mine operation based on average values at year six of mine operation.

This section details the sources of information, the integrated function of the principal components and results of a sensitivity analysis of the water balance based on potential extremes of water excess or scarcity. The annualized water balance during mine operations shown in Figures 20.10 and 20.11 are based on average values for precipitation and temperature, a calculated volume of groundwater from the mine dewatering system and other assumptions and limitations. As the mine plan is refined and additional details develop on mill water demand and system operation, the water balance will evolve and be refined.

Process Water Pond (PWP)

- Minimum Operational Capacity: 80,000 m^3 (2,825,173 ft^3) (Tintina Montana, 2017);
- Maximum Operational Capacity: 200,000 m^3 (7,062,933 ft^3) (Tintina Montana, 2017);
- Normal operations cannot send water from PWP to WTP (only allowed in closure).
- Exception for 10-year, 24-hour storm event or greater (this analysis does not consider storm events);
- Precipitation from November through March falls as snow and accumulates as snowpack until the spring, when it melts during April and May. Therefore, the precipitation that accumulates between November and March will report to the PWP during April and May;
- Discharge of RO Reject to the PWP is equal to 15 to 20% of water treated by RO;
- Current MPDES permitting requires some raw water needs for the mill to be sourced from the PWP and treated via separate treatment system in the mill.

Treated Water Storage Pond (TWSP)

- Sized to store average flow rate (2,207.6 m^3/day or 0.9 ft^3/s) from WTP from July through September;
- Maximum Storage Capacity: (not including storm events): 204,000 m^3 (7,204,192 ft^3);
- Precipitation from November through to March falls as snow and accumulates as snowpack until the spring, when it melts during April and May. Therefore, the precipitation that accumulates between November and March will report to the TWSP during April and May (Hydrometrics, 2018a);
- Water stored in TWSP from July 1 through September 30;
- Average inflow rate: 2,167.8 m^3 day (0.89 ft^3/s);
- Water discharged from TWSP from October 1 through June 30;
- Average discharge rate: 725 m^3/day (0.3 ft^3/s);
- A water storage year is considered as the period of July through June.

Alluvial Underground Infiltration Gallery (UIG)

All process water will be treated to applicable water quality standards (except for the seasonal standard for total nitrogen) prior to discharge via the alluvial UIG. The water treatment plant is described in Sections 18.4.4 and 20.8.3.

- Excess water from the mine not used in the milling or mining process will be discharged to the groundwater system using the alluvial UIG;
- Discharge allowed from October 1st through June 30th at the MPDES outfall (Outfall #001);
- Discharge to UIG is contingent on the total nitrogen concentration and season (Figure 20.11). When total nitrogen concentrations in the effluent are less than the seasonal effluent limit, water will be discharged directly from the water treatment plant to the alluvial UIG year round. When the effluent nitrogen concentration exceeds the seasonal discharge limit, the water will be stored temporarily in the TWSP;
- Discharge includes treated water from WTP and water stored in TWSP;
- Average discharge rate of 25 L/sec (398 gpm) to the UIG, and seasonal discharge of 33 L/sec (530 gpm).

Cemented Tailings Facility (CTF)

For the purpose of the water balance estimate, all precipitation falling on the CTF was assumed to report immediately to the PWP. The water balance does not attempt to simulate the delayed movement of water from the CTF due to the possible accumulation of snow over the winter and a spring freshet.

Sensitivity Analysis

Since the water balance uses average values, and water supply, storage and disposal are critical aspects of the mine operation, a sensitivity analysis was performed to evaluate the effects of extreme conditions (Hydrometrics, 2019). These included average, minimum, and maximum mine dewatering rates and abnormally wet, dry, and average precipitation, and their effects on water storage in the PWP and TWSP. These two storage facilities and the alluvial UIG have been identified as the limiting factors of the water balance with respect to water supply, storage, and discharge capacity. The mean cases, based on the average conditions for operations during year six of mining as described in the MOP application, are presented in the previous Section, Figures 20.10 and 20.11 showing the daily flow volumes under average conditions.

Deviations from the mean case are expected throughout the LOM, resulting from variability in precipitation and mine inflow rates. The effects at storage-limiting components (PWP, TWSP, and UIG) were evaluated by conducting analyses to estimate the operational storage conditions under several scenarios described below.

Three scenarios (mean conditions, wet year, and dry year) were developed by Hydrometrics (2019) to evaluate the effects of precipitation on the storage response at the PWP and the TWSP. The effects of precipitation extremes were evaluated using the following conditions:

- Wet Year (95th Percentile): climatic conditions to be exceeded once every 20 years, on average;
- Dry Year (5th Percentile): climatic conditions expected to be exceeded 19 years out of 20, on average (i.e. volumes will not exceed these values more than once every 20 years).

Two scenarios were developed to evaluate the effects of mine dewatering on storage at the PWP and the TWSP, considering the UIG discharge criteria. The mine dewatering rate was varied based upon water storage capacity and discharge criteria to identify maximum and minimum average dewatering rate thresholds. The High Rate dewatering scenario is defined as dewatering at an annual average rate greater than the mean case, where the cumulative storage in the TWSP is approximately equivalent to the extreme maximum storage capacity.

The Low Rate dewatering scenario is defined as dewatering at an annual average rate less than the mean case and achieving the consumptive use requirements of the Project which are limited by the groundwater beneficial use water right. The water right allows for groundwater from mine dewatering to be used and consumed for milling and mining purposes at an average flow rate of 1,145 m^3/day. The use of water in the mill is currently limited to groundwater from the portal; however, incidental capture of precipitation on the CTF and PWP may be used in the mill as this water is not allowed to be discharged per the Draft MPDES permit. The annual average net precipitation to the CTF and PWP is about 257 m^3/day. This results in a dewatering rate of approximately 889 m^3/day to meet the Project requirements. Therefore, the effects on storage volumes of the PWP and UIG for the Low Rate dewatering scenario were evaluated at a dewatering rate of 889 m^3/day.

Overall, the monthly water balance analysis show that the water storage and discharge facilities are appropriately sized to manage the 95th and 5th percentile annual precipitation. However, the TWSP is limited in size such that it can only facilitate an increase of 105% in the average dewatering rate prior to exceeding the operational capacity at the end of September. The water storage and disposal criteria limit the mine dewatering rate to a maximum annual average of 2,862 m^3/day. The discharge capacity of the UIG is capable of emptying the TWSP prior to seasonal storage resuming at the beginning of the subsequent July. The low dewatering analysis indicates that at a dewatering rate of 889 m^3/day (32% of projected), there is sufficient water for the mill and mining processes. As noted in the results for the Wet Year and High Dewatering Rate scenarios, additional water from the PWP will need to be treated in the mill for freshwater needs. Additionally, the analysis indicates that operating the UIG at the maximum discharge rate is ideal for managing storage in the TWSP.

20.8.3 Water Treatment

A WTP will be used during construction, operations, and closure to treat mine process water and contact water. Figure 20.12 and Figure 20.13 show the proposed design (Tintina Montana, 2017; Kimball, 2017). Each phase will have different design flows and raw water quality. The treatment processes will include an oil and grease skimmer, clarifier, filtration, and RO system to remove contaminants. The concentrated RO reject (i.e., water that does not pass through RO membranes for treatment; also called brine) will be stored in a CWP brine cell during construction. During operations, brine will be stored in the PWP, diluted and recycled back to the mill. Liquid and solid treatment residuals (i.e., materials or constituents that are filtered out by the RO membranes) will be disposed onsite using the PWP and CTF, respectively. The RO permeate (i.e., water that passes through RO membranes or filters for treatment) that meets discharge requirements will be discharged to an alluvial UIG system or reused. The UIG will be functional at the onset of mine development and before the dewatering of mine workings begins.

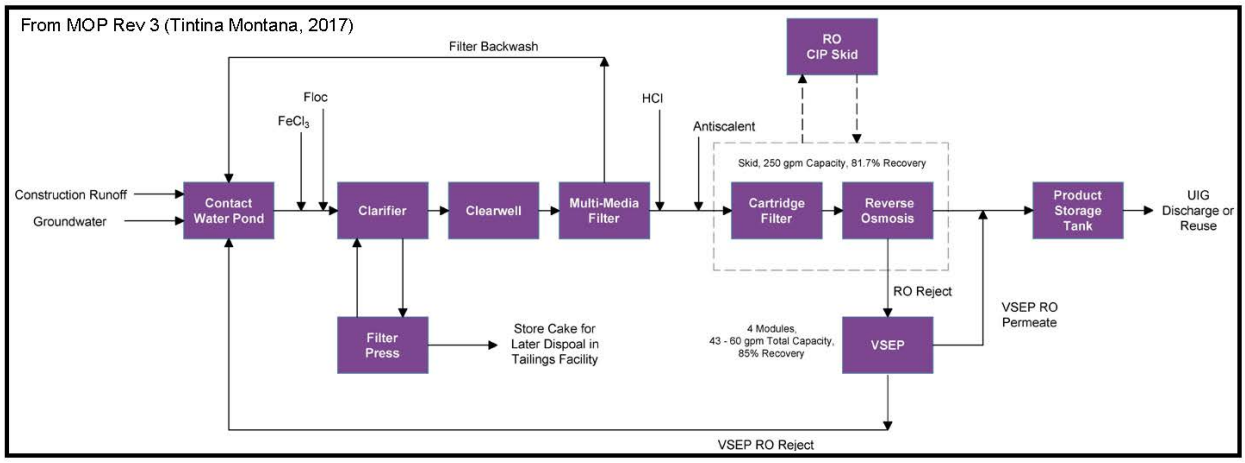


Figure 20.12 Water Treatment Construction Phase Process Flow Diagram

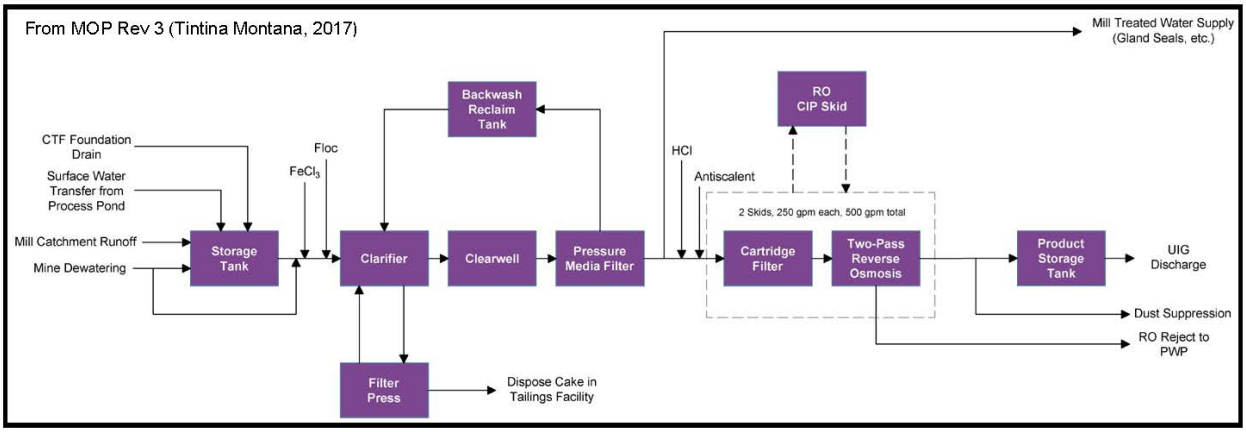


Figure 20.13 Water Treatment Operational Phase Process Flow Diagram

20.8.4 *Treated Water Disposal*

Treated water from the WTP will be discharged to the shallow groundwater alluvium via an alluvial UIG (MPDES outfall #001) which will be located adjacent to Sheep Creek. The UIG will receive an average of approximately 25 L/sec (398 gpm) of treated water from the WTP. The maximum total design discharge capacity of the UIG is 37 L/sec (575 gpm).

Discharge of treated water will be contingent on the total nitrogen effluent limit as described in the Integrated Discharge Permit Application Narrative (Hydrometrics, Inc. 2018). The ”Integrated” permit includes discharges from treated water from the mining and milling processes (MPDES) and storm water associated with industrial activities (Industrial Storm Water is described in the following section). Storm water which has come in contact with mine drainage, mined materials, or process waters, referred to as contact water, is collected and treated (RO plant) prior to discharge under this MPDES outfall.

When total nitrogen concentrations in the effluent are less than the seasonal effluent limit, water will be discharged directly from the water treatment plant to the alluvial UIG year round at an average rate of 25 L/sec (398 gpm). However, if the effluent nitrogen concentration exceeds the seasonal discharge limit, the water will be stored temporarily in the TWSP from July 1 to September 30. Starting October 1, the stored water would be routed back to the WTP and blended with the WTP effluent prior to discharge to the alluvial UIG, with an average discharge of 33 L/sec (530 gpm) (Figure 20.11). The depth to the groundwater table in the UIG area once the mine has been developed will be approximately 2.4 to 4.0 m (8 to 13 ft). The planned location of the UIG is outside all designated wetland areas, and its length is oriented perpendicular to the groundwater flow direction.

20.8.5 *Storm Water*

Storm water management for the Project will also use the Hydrometrics (2018) integrated permitting approach as described above. This will be implemented by permitting the 13 applicable storm water outfalls in the MPDES discharge permit, using the regulatory requirements of the Multi-Sector General Permit for Storm Water Discharges Associated with Industrial Activities. Numerous mine facilities identified in the site facilities map are covered by the storm water regulations including power plant (substation) areas and reclaimed area not released from reclamation bonds.

Storm water management will include diverting storm water run-off around disturbed areas, or by collecting run-off from contact water areas for treatment prior to discharge. The majority of storm water run-off at the Project site will be controlled by diversion around disturbed soils. Hydrometrics (2018) derived the storm water flow rates through SEDCAD modelling of the 10-year, 24-hour storm event that included 5.8 cm (2.28 inches) of precipitation within a 24-hour period. Flow calculations were completed using a SCS Type II storm event curve distribution.

20.8.6 Erosion Control Methods and Best Management Practices

During the construction, operations, and closure phases several erosion control techniques, methods or features will be used. These are listed below and defined in the Integrated Discharge Permit Application Narrative (Hydrometrics, Inc., 2018): vegetation management and revegetation, mulching, rolled erosion control products, slope roughening, recontouring, silt fencing, temporary sediment traps and sediment basins, filter bags, flocculants, collection ditches, diversion ditches, culverts, and water bars.

Erosion control Best Management Plans (BMP's) will be implemented prior to and during construction at the Project. Erosion control BMP's reduce erosion by stabilizing exposed soil, or by reducing surface run-off flow velocities. There are generally two types of erosion control BMP's: source control BMP's for protection of exposed surfaces, and conveyance BMP's for control of run-off. Examples of BMP's that will be implemented are included in both the Integrated Permit application and are from "Water Quality BMP's for Montana Forests" (MSU Extension Service, 2015).

20.9 Monitoring

Three general monitoring programs will be conducted throughout the mine permitting period and during the life of the mine including: ongoing baseline monitoring, operational monitoring, and post-mining monitoring and are all described in the MOP Application. Monitoring programs will be developed and implemented under the auspices of a site-specific EMS.

Baseline monitoring has been initiated and will continue throughout the mine permitting period to assess environmental conditions prior to surface disturbing activities. This monitoring characterizes existing conditions at the site and provides a baseline against which to identify potential impacts to resources. Ongoing monitoring programs during the permitting period include climate, water quality and quantity, aquatic resources, and environmental geochemistry.

Operational monitoring is designed to identify potential environmental impacts to natural resources. The frequency of monitoring is stipulated by the MDEQ (2020a) and intended to detect potential impacts in a timely manner and trigger the implementation of operational changes and/or mitigation measures. Operational monitoring will include: water quality and quantity and compliance of facilities (described in the section 20.9.2 below); monitoring of the waste and water management facilities that will be included in the TOMS manual and partially described above in Section 20.9.1; facility geotechnical monitoring (described in the section below); waste rock geochemistry; air quality; wetlands; noise; reclamation; and terrestrial and aquatic resources (described below in Section 20.9.3).

Once mining operations are complete and final reclamation and closure plans have been executed, post-operational and closure monitoring plans will be developed in consultation with MDEQ and implemented to monitor potential impacts. This monitoring will continue until such time that the MDEQ determines that the frequency and number of sampling sites for each resource can be reduced or that closure objectives have been met and monitoring can be eliminated. Typically, these plans focus on long-term water quality and quantity monitoring, effectiveness of erosion control, and monitoring of vegetative success to document post-closure conditions at the former mine site. This monitoring program will also include facility closure, and aquatic resources. The post-operational monitoring will continue until the mine is certified by MDEQ as being fully reclaimed by meeting closure goals and all bonding release milestones have been met.

20.9.1 Geotechnical Monitoring

Geotechnical monitoring will include survey monuments on the crest and downstream slopes of all embankments and inclinometer measurements. The monuments will require initial quarterly surveying and then at regular semi-annual intervals to indicate any settlement or movement of the embankments. Additional monitoring at the BCWP, PWP and CTF will include the monitoring of the pore pressures within the foundation drain (if any), basin drain system, and wet well sump and pump system; and of the vibrating wire piezometers that will be installed during operations. Longer term monitoring frequency will be optimized during operations based on an observational approach. Details of the inspection and monitoring of the facilities (including frequency) will be included in the TOMS Manual which will be periodically updated during operations as additional information and data is collected and reviewed.

20.9.2 Water Monitoring

Monitoring of water resources at the Project site was initiated in 2011 and is ongoing on a quarterly basis. Monitoring during the operational and closure period of the mine will include water quality of streams, seeps, wetlands and groundwater, groundwater levels and stream flows. Groundwater monitoring wells will be added to the existing network at locations down gradient from water-storage facilities to allow quarterly sampling of water quality. The results of the sampling will be used to evaluate potential impacts to groundwater.

Monitoring will continue after closure to ensure no unforeseen impacts are occurring. Monitoring will continue until the MDEQ determines that the frequency and number of sampling sites for each resource can be reduced or that the closure objectives have been met and monitoring can be eliminated.

20.9.3 Terrestrial and Aquatic Effects Monitoring

A proposed aquatic resource sampling plan during construction and the operational mine life has been defined in the MOP Application in Appendix G-1 and updated in 2019 by the Montana Biological Survey (Stagliano, 2019) and submitted to the MDEQ – Hard Rock Mining Bureau for their review and subsequent review by MFWP. Sampling is proposed to be done annually during construction (a total of a three-year period, three events) and during the first year of operations and then every year throughout the operational mine life (13 remaining operational years, six events).

Revegetation monitoring and erosion control will be conducted throughout construction, operations, closure, and post closure.

20.9.4 Reporting

A report will be submitted annually describing post-operational closure water quality monitoring results, discussing results, QA/QC measures, reclamation and erosional problems, remedial measures taken (if any), and documenting reclamation/closure success. Post-operational monitoring will occur until such time as the mine is certified as fully reclaimed and all bonding release milestones are met, or as determined in the post-operational monitoring program to be developed in conjunction with MDEQ.

20.10 Mine Closure Plan and Bond Requirements

The mine closure plan and bond requirements are identified in the following sections below and are described in the MOP Application. The Agency Modified Alternative (AMA) as presented in the Final EIS relates to additional requirements for underground mine closure that are discussed below in the Section 20.10.7. (Site Specific Closure Activities). The activities related to implementing the AMA will need to be finalized with the MDEQ and included in the Project Bond cost estimation.

20.10.1 Closure Objectives and Categories

The reclamation and closure plan is structured to meet the requirements of the Montana MMRA (MCA § 82-4-301). Three categories (periods) of reclamation and closure have been developed for the Project and are described in separate sections below:

- Contemporaneous construction reclamation and operational maintenance,
- Short-term temporary closure (less than one year), and
- Permanent closure.

The overall objectives of these categories of reclamation and closure are similar and include:

- Recontouring and revegetation of disturbed areas and landforms that have been modified by the construction of facilities and throughout life-of-mine operations;
- Stabilization of disturbed areas using erosion and sediment control BMP's, and revegetation measures to prevent air and water pollution;
- Surveying and monitoring the pond water elevations and pond volumes (water inflows and outflows) to make certain that none of the ponds exceed their design capacity and at the same time leave additional pond storage capacity for additional possible water inflows from either the probable maximum flood event or the design 1 in 200-year storm event (depending on the facility);
- Confirming and documenting that the volumes of the water and brine in the ponds don't exceed the facility design standards at the time of closure (including monitoring of the various volumes of water and brine generated from the WTP;
- Monitoring programs will continue during construction, operations, temporary closure and in permanent closure until closure objectives have been met; and
- Noxious weed control.

20.10.2 Closure Criteria (Disturbed Lands Reclamation Compliance)

The permanent reclamation and closure plan will be compliant with applicable reclamation requirements set forth in MCA 82-4-336 for the reclamation of disturbed lands. The applicable requirements of MCA 82-4-336 with which the permanent reclamation and closure will comply will consider the site-specific conditions and circumstances. Reclamation activities will be completed not more than two years after completion or abandonment operations in accordance with MCA 82-4-336(3) unless an alternative timeframe is allowed by the MDEQ or if SRA requests an extension from the MDEQ.

20.10.3 Reclamation Bond Requirements

The MDEQ Hard Rock Mining Bureau is responsible for calculating the amount of performance bond for reclamation of the Project based on regulations included in the Montana MMRA (Title 82, Chapter 4, Part 3, MCA 2019 82-4-338(1)). The purpose of the bond is to ensure the availability of funds sufficient to perform reclamation in the event of a default by the operator. The posting of the financial assurance payable to the State of Montana is a precondition to issuance of a MOP. The amount of financial assurance is based on the estimated cost to the State to ensure compliance with the Clean Air Act of Montana, the Montana Water Quality Act, and the MMRA including the reclamation plan set forth in the MOP application. The MDEQ estimates the Project Bond after the ROD is awarded to SRA. The Project performance bond is separate from the Exploration bond that is related to surface activities related to exploration activities.

Once the reclamation plan has been agreed upon, the MDEQ Hard Rock Section staff will prepare a bond calculation assuming site-wide reclamation, and a closure plan that would be executed by a third party contractor under MDEQ supervision. Bond levels are reviewed annually during operations, and every five years the bond amount is recalculated and the amount held by the State is supplemented if necessary to ensure that adequate funds are available for closure of the site.

SRA worked with SRK Consulting using the Standardized Reclamation Cost Estimator (SRCE) version 2.0 spreadsheet software to calculate the closure cost associated with the Project as stated in the MOP Application (Revision 3) document, information included in subsequent MOP updates (Zieg, 2018) and information in the Final EIS (MDEQ, 2020). The estimated closure cost excludes the cost for implementing the single AMA (see section below called "Site Specific Closure Activities Related to Permanent Closure" for AMA details) that still needs to be determined and approved by the MDEQ. The full Project bond cost estimate is for the LOM that includes the early phase activities, all other surface mine facilities and the UG mine as described in the sections below called "Permanent Reclamation and Closure Activities" and "Site Specific Closure Activities Related to Permanent Closure". It includes the maximum Project buildout and permitted surface disturbance totalling 126 ha (310.9 acres) of Project surface disturbance. The costs of the reclamation and closure activities include concurrent reclamation liabilities early in the life of the Project until such time that relevant facilities are reclaimed and SRA proposes potential liabilities that could be encountered in the event of a default in addition to the liabilities of the full buildout. The default mine closure scenario is assumed to be the result of economic factors which require the operator (SRA) to declare bankruptcy and includes an early two month "care and maintenance" period that includes water treatment using the existing equipment at the site. The default mine closure scenario also includes:

- a starting PWP water volume equal to 160,000 m;
- no water or brine in the BCWP at the start;
- implementing all the proposed rinse cycles to the four separate litho-units (LSZ, Ynl-B, USZ, and Ynl-A) using the maximum number of rinse cycles as defined in the MOP;
- installing all the 14 hydraulic plugs and barriers (including reclamation material backfill) and monitoring wells as defined in the MOP, and
- treating all the remaining water in the PWP.

SRCE (McIntosh, 2020) calculated an amount of $24.3M for the full Project Bond (LOM) mine closure activities (excluding the AMA costs that are still to be determined) that will be required to be settled by SRA to the MDEQ prior to the start of the Phase 2 Development construction activities (see below discussion). This estimated SRCE closure cost excludes any indirect costs (engineering, design and construction plan, contract administration, contractor overhead and profit, and contingency) that will be added by the MDEQ likely to be 36% of the total estimated cost.

The ROD document (MDEQ, 2020a) separates the Mine Development Construction activities into two Phases with five stipulations. SRA requested the MDEQ consider an incremental bonding payment approach with two distinct bonding increments (Incremental Bond 1 and Incremental Bond 2 that correspond to the Phase 1 and Phase 2 Development Construction activities described in the ROD, respectively). MDEQ agreed to the incremental bonding approach as defined by the ARM 17.24.140 and on May 19, 2020 calculated the Incremental Bond 1 amount to be $4,653,348 (MDEQ, 2020c) that includes 36.4% indirect costs. As of the publishing date of the Feasibility Study Technical Report, the MDEQ is still estimating the Incremental Bond 2 amount that will include the cost of implementing the AMA as described in the Final EIS (MDEQ, 2020).

20.10.4 Contemporaneous Construction Reclamation and Operational Maintenance

Construction reclamation takes place to stabilize disturbed areas during and shorty after mine facility construction and is extended to include operational programs to maintain optimum reclamation performance over time. Operational reclamation is focused upon construction disturbances associated with facilities that will remain in place throughout LOM operations (i.e. cut and fill slopes, downstream embankments, pipe line trenches, access roads and erosion control ditches and berms), or those that occur peripheral to and between constructed mine facilities. Interim reclamation of temporary or construction roads, embankments, soil stockpiles, ditch cuts and fills, trenches, surface water control structures, and other disturbances not inherently stable will occur during the first appropriate seeding season following construction. Reclamation of disturbed areas will be carried on throughout operations to the maximum extent practicable.

20.10.5 Temporary Closure Activities

Short term temporary closure reclamation will occur if the mine enters into a period of temporary inactivity for a time period of up to one year. At that time, evidence of intent not to abandon the Project and to resume operations will be submitted to MDEQ (in accord with ARM 17.24.150). SRA will request that deferred implementation of the permanent closure plan be approved by MDEQ. Short-term temporary closure reclamation and site protection will include: continued underground mine dewatering, continued treatment of water through the WTP (and properly disposing of the brine), stabilizing site-wide drainage facilities, prevention of unnecessary erosion by stabilization and revegetation of any existing disturbances, maintaining site access, maintaining water quality sampling and monitoring/reporting, maintaining the site weather station, providing site security by maintenance of fencing for all of facilities (including the ponds, ventilation raises, and the mill area), protection of equipment, and preparation and implementation of a facility inspection programs. All infrastructure required to resume mining will remain in place in this scenario. If the mill is operating during a temporary closure scenario, the following other tasks will be implemented: gate and lock the mine portal entrance to minimize wildlife and people from entering the UG mine, and removal of all explosives from the site using a licensed contractor.

20.10.6 Permanent Reclamation and Closure Activities

Permanent reclamation and closure will occur at the end of mine life and would also be initiated if a temporary closure extends beyond a period of one year. It includes reclamation of all disturbances and removal or closure of facilities. All buildings, related equipment, and surface infrastructure at the mine site will be dismantled and removed. No waste rock would be left on the surface in closure. The details of the permanent reclamation and closure plan are included in the MOP Application and summarized below.

Post-mining general construction measures will include facility removal, landform restoration, surface reclamation, and closure activities. The goals of these post mining construction measures are to achieve long-term stability of each reclaimed facility site and the remaining CTF embankment, to use all of the remaining reclamation material stored in the two designated stockpiles (north and south), to develop a self-sustaining productive vegetative cover, and to ensure long term protection of the environment with respect to water quality and erosion.

Reclamation of facility areas will also include removal and off-site disposal or recycling of liners, filling of excavated basins, and reshaping of the ground surface. This will be done by placing embankment or other appropriate fill materials to create a self-draining surface approximating the pre-mining topography (except for the CTF) that will provide long-term stability after closure. The upper reclaimed surface of each facility will be capped with approximately 68.5 cm (27 inches) of soil (topsoil and subsoil) and revegetated. Sub-grade bedding material may need to be locally placed above the final (upper) tailings cemented paste layer lens to provide a lower protective layer for HDPE geomembrane placement, depending upon the grade and smoothness of the final upper surface. Surface gradient restoration will be suitable for the post-operation use of rangeland. Grading will minimize the amount of precipitation and run-on that infiltrates into disturbed areas.

Vegetation will be removed during initial construction. Small shrubs and herbaceous vegetation will be salvaged with topsoil. Merchantable trees will be harvested and removed from the site. Non-commercial trees, slash, tall shrubs and small stumps will be chipped and salvaged with topsoil. Larger stumps will be stored at the toe of soil and reclamation stockpiles to aid in erosion control and ultimately for distribution as part of reclamation. The suitability of soil and subsoil proposed for reclamation was determined from physical and chemical data collected during the baseline soil survey.

Revegetation measures in mine closure include soil replacement using the stockpiled topsoil and subsoil, seedbed preparation and seeding with approved seed mixes. Approximately 430,974 m^3 (563,691 $yards^3$) of soil would be salvaged and stockpiled long-term for reclamation activities associated with mine closure, and approximately 233,016 m^3 (304,773 $yards^3$) would be temporarily stored and replaced on site for reclamation of construction activities, including grading, slope stabilization, drainage control, topsoil and subsoil placement, and seeding using an approved seed mix (Zieg, 2018). A 10% buffer was added to the

estimated soil salvage volumes to account for construction buffer zones and a 12% swell factor was used to replicate bulked soil volumes. All disturbed ground will be recontoured and revegetated.

Following removal and/or salvage of facilities, including all above ground piping and other infrastructure, any remaining solid waste will be disposed of in accordance with laws and regulations of the Montana Waste and Underground Tank Management Bureau, and Meagher County. Valuable inert waste such as steel, concrete, plastic or wood will be sold to scrap dealers for recycling. Concrete foundations will be broken up, exposed rebar cut-off, levelled, and buried on the respective facility sites with a minimum cover of 1.2 m (4 ft) of fill material. No landfills for waste disposal of any type are proposed in the MOP Application. Hazardous wastes will be removed from the property by a licensed commercial hazardous waste disposal company.

20.10.7 Site Specific Closure Activities Related to Permanent Closure

The closure of specific mine facilities is discussed in detail in the MOP Application. The closure of the CTF and the UG workings are two of the most important activities and are summarized below.

CTF Closure

Under planned operations and reclamation and closure, permanent closure of the CTF will include the following steps:

- Dewatering (by pumping back any water on the liners or from the water reclaim sump to the PWP or the CWP);
- Pumping of water from the foundation drain pond to the WTP for treatment (if required);
- Shaping and/or filling of the final upper surface of the tailings with various materials to make an even surface prior to installation of the 0.254 cm (0.1-inch or 100 mil) HDPE geomembrane cover which will be welded to the existing liner system;
- Placement of the non-reactive sub-grade bedding and excess reclamation fill materials totalling a minimum of 1.6 m (5.3 ft) in thickness above the HDPE geomembrane cover;
- Reshaping (grading) of the ground to create a new closure topography to achieve a minimum gradient of -0.5% to the east to create a self-draining topographic surface suitable for closure;
- Placement of 0.35 m (1.15 ft.) of topsoil/subsoil cover and establishment of long-term revegetation cover.

Closure of the CTF will maintain the original 2.5:1 slope angle of the east-facing embankment. The CTF embankment will be geotechnically stable (during operations and at closure), and long-term erosional stability and revegetation success will have been established from the previous work completed during construction and operations. The closure goal established for final grading to "blend with existing topography" will be achieved in closure for all mine-related facilities except for the east embankment of the CTF.

Underground Mine Closure

Mine workings would ultimately be sequentially flooded at closure with groundwater as described in the MOP Application (Tintina Montana, 2017). Prior to flooding in particular portions of the mine, the walls of the workings within these zones would be rinsed to remove oxidation products. Rinse water would be collected, pumped, and treated as necessary. Each zone would then be flooded with groundwater and a hydraulic barrier would be installed. In all, 14 hydraulic barriers—both plugs and walls, which are masses of concrete—would be installed in the underground workings. Five of the hydraulic barriers would be installed at the main access ramps, eight in the four ventilation raises (an upper and lower barrier in each raise), and one plug at the mine portal. The primary purpose of the hydraulic barriers is to segment the mine workings based upon sulphide content to facilitate rinsing and improve water management. Mine closure objectives including water treatment as defined in the MOP Application are expected to take up to four years after milling ceases, and once initial facility closure activities have been sufficiently implemented.

The Final EIS (MDEQ, 2020) contains one AMA that recommends additional backfill of the "open" sulphide-mineral bearing mine voids as part of mine closure, as compared to the closure plan in the MOP Application that includes leaving some mine voids "open." The AMA proposes to backfill certain voids (i.e., access openings) with a low hydraulic conductivity material consisting of cemented paste tailings generated from mill processing of the stockpiled ore and/or waste rock at the end of operations. Cemented paste tailings would only be used to backfill certain mineralized mine voids to avoid the potential of degrading groundwater quality in non-mineralized geologic units.

The AMA identified in the Final EIS (MDEQ, 2020) states that 81,785 m^3 (106,971 $yard^3$) of cemented paste tailings would be needed to backfill the unfilled sulphidic access tunnels and ventilation raises (as identified in the MOP). The backfill material would be mixed with cement in a manner that achieves a similar low hydraulic conductivity as is proposed for backfilling of the mined stope areas. Since the volume of stockpiled waste rock source (equivalent to approximately 153,061 t or 168,721 ton) would exceed the proposed volume of the Cu-Enriched Rock Stockpile storage capacity, this Project modification would also need to utilize the WRS pad until the end of operations and backfilling of certain interior UG mine surfaces. The backfilling schedule would be coordinated with activities elsewhere in the mine, so as not to interfere with necessary access, ventilation, and safety for other operations.

The volumes of treatment water to be generated during UG mine closure are derived from assumptions in the MOP Application regarding rinse volumes and numbers of rinses required per lithotype unit to be excavated during mining as well as the remaining PWP water volume. The final concentrated brine stream will be trucked to a licensed off-site facility for disposal. Treated water in closure will be discharged either to the UIG or used for UG mine closure (rinsing/flushing the UG workings as described above). Off-site disposal of brine could begin anytime during the UG mine closure process and may need to be completed prior to starting to close and reclaim the BCWP. During the UG mine closure process, the BCWP and the PWP will always have adequate storage capacity.

20.10.8 Closure Schedule

SRA will initiate final closure reclamation activities within one year of reaching a decision to permanently discontinue mining in the Project area. An extension of the one-year time frame could be requested from MDEQ if circumstances are warranted. If additional mining in the vicinity of the Project area is still under consideration, temporary closure of the site will follow the procedure outlined in in Section 20.10.5 above.

Long-term closure of the site is expected to take approximately two to four years, excluding successful long-term revegetation establishment and water quality monitoring. All major facilities have reclamation and closure plans associated with them, as described in the previous Section, with the exception of the main Project access road (which will be downsized in closure).

The NCWR will be used for mitigation of residual depletion in surface waters during operations and for approximately 20 years after the cessation of mine dewatering (Zieg, 2018). The water design storage capacity of the NCWR required after operations has ceased is much less than what is required during operations at 65,400 m^3 (53 acre-ft); therefore, the NCWR embankment size after facility reclamation will be reduced from 138,000 m^3 (4,873,424 ft^3) to approximately 37,000 m^3 (1,306,643 ft^3). The excess 101,000 m^3 of embankment material “cut” will be used to help reclaim the mine facilities in mine closure. Once the NCWR is not necessary to mitigate flows, the wet well, intake pipeline, transfer pipeline, and embankment will be reclaimed.

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21. CAPITAL AND OPERATING COSTS

21.1 Operating Cost Estimate

Operating costs for the processing plant have been estimated to an accuracy of +/- 15%. The costs are presented in US dollars ($) and are based on prices obtained during the second quarter of 2019 (2Q19) with a review undertaken in second quarter of 2020 (2Q20).

21.1.1 Underground Mine Operating Costs

Operating costs have been developed using the parameters specified in the process design criteria which are listed in Appendix B. Annual operating costs and costs per tonne mined have been developed and are summarized in Table 21.1. Operating costs are estimated to an accuracy of +/- 15%.

- Ore drift development costs related to the drilling, blasting, mucking and haulage of all material from ore drifts including access declines and associated drifts in waste;
- Paste backfill costs related to creation of paste barricades and maintenance of the paste system. All binder costs are included in the processing cost area;
- Mine maintenance costs to maintain the UG mobile and fixed plant equipment;
- Mine general costs for labor for UG operations including the mine department, support equipment and definition drilling.

The operating cost estimate has been developed based on the Project schedule presented in Section 16.

The costs have been compiled from a variety of sources including:

- First principle estimates;
- Suppliers' budget quotations;
- Mine Schedule.

The operating cost estimate is $27.09M per annum or $23.35/t ore mined.

Cost Centre	Annual Cost $ million	Unit Cost $/t
Development	12.46	10.74
Pastefill	1.10	0.95
Mine Maintenance	2.89	2.49
Mine General	10.64	9.17
TOTAL	**27.09**	**23.35**

Table 21.1 UG Mine Operating Costs Summary

Major Operating Items

Table 21.2 and Figure 21.1 illustrate the mining operating cost broken down by major component:

Labor

UG mining staffing levels are built up based on a first principle estimate of productivities and varies throughout the mine life. All on-costs related to providing health insurance, retirement benefits, workers compensation and required payroll taxes as well as overtime requirements are included in the estimate.

Equipment Operations

UG mining equipment operating costs are based on equipment operating hours required for the LOM plan. Costs per hour are sourced from equipment manufacturers and include elements such as:

- Maintenance Parts;
- Lubricants;
- Tires;
- Ground Engaging Tools.

Rebuild costs as well as equipment replacement costs are included in Sustaining Capital and not in the operating cost.

Fuel Costs

Fuel Costs have been calculated based on equipment consumption. The fuel price used is $0.34/L diesel.

Power Costs

Power costs directly for the UG mine include, the primary and auxiliary fans, power for operating equipment, main and secondary pumps as well as ancillary fixed equipment. The power price used is $0.075/kWh.

Propane

Propane will be required to heat the intake air to +2˚C during the winter months for the safety and comfort of workers. The propane cost used is $0.26/L.

Consumables

Consumables usage is built-up based on quantities from the LOM plan. Costs are based on budgetary quotations. Where freight wasn't included in the quotation price, 5% of the quoted price has been included.

Cost Centre	Annual Cost $ million	Unit Cost $/t
Labor	8.36	7.21
Equipment Operations	5.58	4.81
Fuel	0.89	0.77
Power	0.61	0.53
Propane	0.20	0.17
Consumables	10.00	8.62
Other	1.45	1.24
TOTAL	**27.09**	**23.35**

Table 21.2 Mining Operating Cost by Major Component

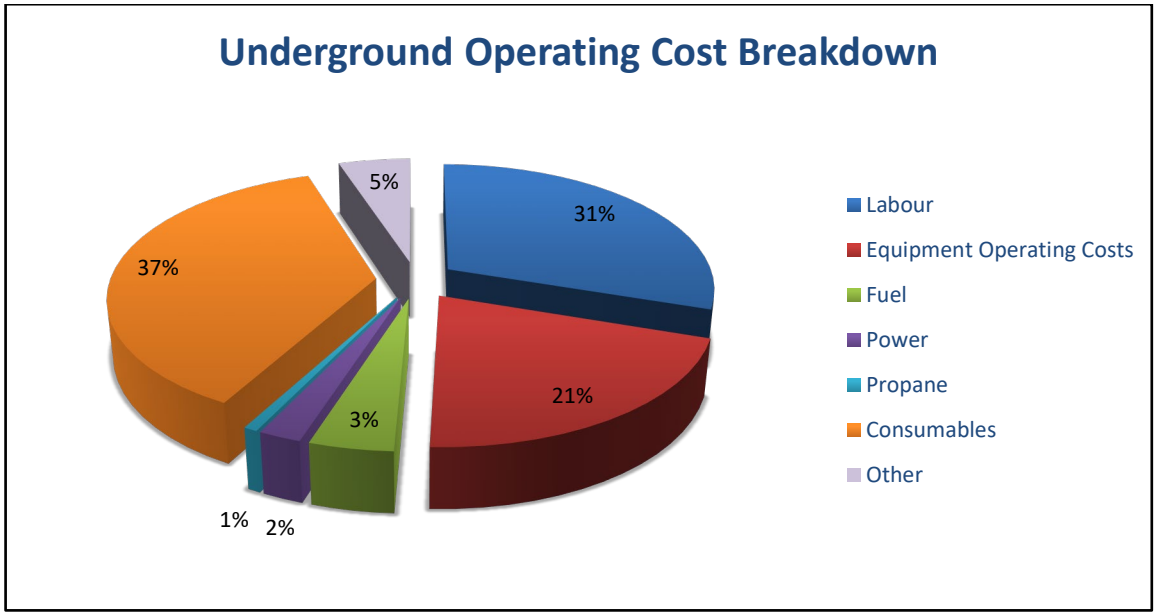

Figure 21.1 Mining Operating Cost Breakdown

21.1.2 Process Plant Operating Costs

Operating costs have been developed using the parameters specified in the process design criteria. Annual operating costs and costs per tonne milled have been developed and are summarized in Table 21.3. Operating costs for the treatment plant have been estimated to an accuracy of +/- 15%.

The costs cover the processing of ore from the ROM pad battery limit. This includes the sections covering crushing, milling, flotation, dewatering, concentrate handling and trucking, site services (power, air, and water), and administration costs.

The operating cost estimate has been developed on the basis of a process plant feed tonnage of 1,204,500 tpa.

The costs have been compiled from a variety of sources including:

- First principle estimates;
- Suppliers' budget quotations;
- GRES data based on similar operations;
- Metallurgical test work results.

The operating cost estimate is $49.97 million per annum or $41.49/t milled.

Cost Centre	Annual Cost $ million	Unit Cost $/t
Processing Costs – Operations		
Salaries/Labor	7.29	6.06
Power	5.95	4.94
Reagents	3.50	2.91
Water Treatment	0.16	0.13
Grinding Media	4.26	3.53
Paste Consumables	4.37	3.63
Other Consumables	2.15	1.79
Maintenance	3.05	2.54
General	0.38	0.32
Crusher Feeding	0.27	0.23
Heating Requirements	0.28	0.23
Processing Costs – Administration		
Salaries/Labor	3.03	2.52
Maintenance	0.59	0.49
Laboratory	0.54	0.45
Concentrate Land Freight	12.67	10.52
General	1.47	1.22
TOTAL	**49.97**	**41.49**

Table 21.3 Annual Processing Costs – Concentrator

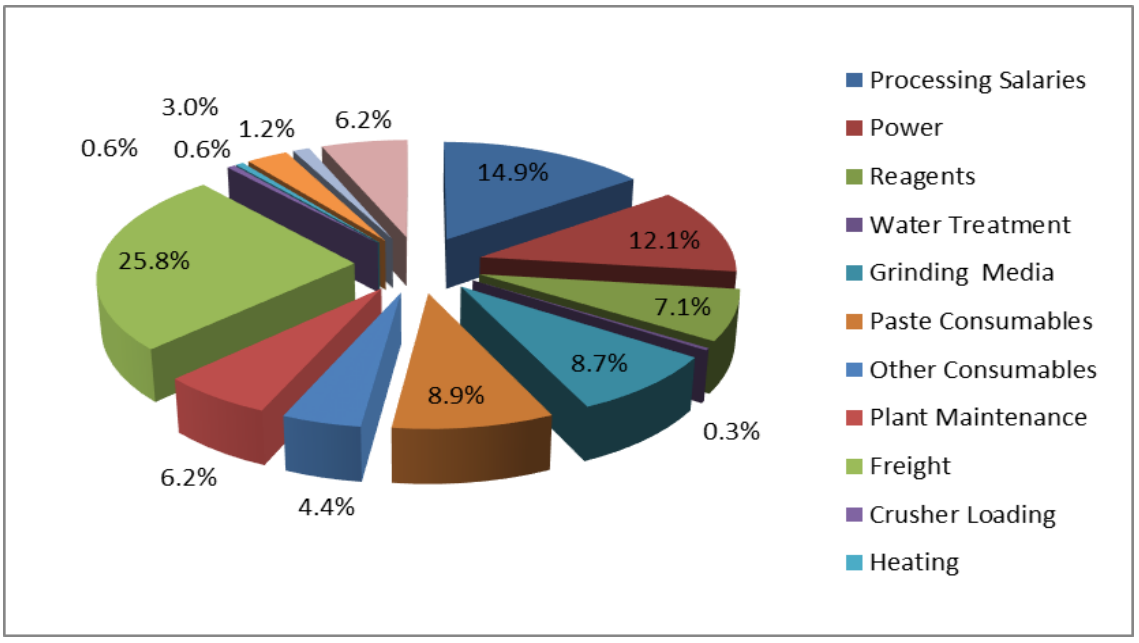

Figure 21.2 Annual Processing Costs - Concentrator

Qualifications and Exclusions

The operating costs presented have been calculated from first principles and budget quotations for supply of chemicals, materials and services. Process plant operating costs are considered to have an accuracy of +/- 15%. The following items have been excluded from the operating cost estimate:

- Royalties – included in financial analysis;
- All head office costs and corporate overheads;
- Exchange rate variations;
- Escalations;
- Project financing costs;
- Interest charges;
- Political Risk Insurance;
- Land compensation/land owners costs;
- Subsidies to the local community;
- Rehabilitation costs;
- Amortisation and depreciation charges.

Concentrate

- Concentrate transport from site by truck in sea containers has been included to port in the operating cost estimates. Costs associated with export transport by sea have been excluded;
- Product marketing costs have been excluded;
- Treatment and refining of the concentrate smelters have been excluded.

Tailings Storage

- Future wall lifts on CTF have not been included within the operating cost.

Environmental

- Plant site rehabilitation costs have been excluded from the operating cost estimate;
- Cemented tailings rehabilitation costs have been excluded from the operating cost estimate;
- Environmental sampling and monitoring costs have been estimated as an incremental allowance to the existing environmental costs and include additional assay costs, additional contracts/consultants costs and general consumables.

Labor

- Overtime allowances have been included and adjusted for Montana State laws to yield an overall adjusted hourly rate. Any additional overtime was not considered as the workforce is employed on a rotating roster;
- Union fees have been excluded;
- The labor schedule and roster are based on seven days on/seven days off shift rosters for operations; nine days on/five days off for technical staff and operations supervision and five days on/two days off for management and administration roles;
- A specific on-cost rate has not been used in the estimate. As health insurance has several levels depending on spouse and spouse with dependent children to a single employee rate; an aggregate rate of 16.6% was determined based on 1/3 basis for employee, employee and spouse and employee with family dependents. Payroll tax is based on federal (FUTA) at 0.06% on the first $7000 and state (SUTA) at 4.1% on the first $33,000. Other components such as social security at 6.2% on the first $123.9K and Medicaid at 1.45% on all income were also allowed for within the on-cost calculations. The only fixed on-cost was for retirement allowance at 2.4% of all primary income;
- No accommodation costs have been allowed for within the operating estimate for camp facilities as the workforce will be drawn from local communities, with the main base being White Sulphur Springs.

Consumables

- The consumption of reagents and other consumables are based on rates from the metallurgical test work and GRES experience;
- All reagent and consumable costs have been sourced from suppliers and are calculated as free-in-store at the Project site.

Utilities

- The power schedule has been based on the equipment list prepared by GRES. GRES has calculated the power draw based on utilization factors;
- The power cost has been based on grid power supply to site at a unit rate of $0.075/kWh.

General and Administration

- General and administration costs have been estimated from other similar scale operations.

Major Operating Items

The major operating costs are detailed in Table 21.3 and consist of the following:

Salaries

The operational labor costs account for 14% of the overall cost to operations for the processing plant and 6% associated with the administration labor costs. These costs are based on data sourced from government departments, HR consultant database information and correspondence with SRA, for Montana and US based mining and processing operations.

As the operation will be locally based a premium was applied to all the salaries to ensure that adequately skilled labor and technical professionals could be sourced and maintained at site. Consequently, the rates applied are in the upper 80% of the range.

As the operations have adopted an attractive seven days on/seven days off arrangement for the process plant labor, the number of employees has increased over a three panel 14 days on/seven days off arrangement.

Cost Category	Number of Employees No.	Base Salary/Wage $/a	On Cost $/a	Total Cost $/a
Processing				
Laboratory	9	519,554	107,464	627,019
Process Plant Management	14	1,342,428	332,565	1,674,993
Process Plant Operations	28	1,593,006	637,385	2,230,391
Plant Maintenance	29	2,227,269	448,634	2,761,902
Administration				
(Including Procurement, HSE)	36	2,426,959	603,924	3,030,883
Water Treatment (Sub-contract)	3	N/A	N/A	481,000
TOTAL	**119**	**8,161,216**	**2,163,971**	**10,806,187**

Table 21.4 Processing Salaries and On-Costs

Power

Power costs have been calculated using the installed power and a utilization factor for the plant against the operational hours.

The main users are the grinding mills and the paste plant facility:

- SAG Mill installed power of 2,000 kW draws 1,566 kW, and equates to 12.6 MWh/a;
- Ball Mill installed power of 4,600 kW draws 3,680 kW, and equates to 29.7 MWh/a;
- Regrind Mill installed power of 1,500 kW draws 825 kW, and equates to 6.6 MWh/a;
- Paste Plant vacuum pumps, air compressors and paste pumps are also a high demand with 11.1 MWh/a of combined usage.

Area Description	Installed kW	Consumed kW	Annual Usage Kwh
Primary Crushing	366	246	1,427,238
Ore Storage and Handling	159	91	461,862
Grinding and Classification	7,722	5,762	45,136,173
Flotation Regrind	2,574	1,521	12,006,023
Concentrate Dewatering and Containerisation	261	162	996,874
Tailings Disposal	149	74	560,755
Reagents	65	42	176,229
Paste Plant	3,577	2,069	11,453,120
Air Services	720	295	2,121,532
Water Services	882	478	2,724,008
Admin, Lighting, Workshops, Misc. #	584	385	2,294,726
Totals	**17,057**	**11,123**	**79,358,541**
Total Cost $			**5,951,891**
Total Cost $/t Ore			**4.94**

Table 21.5 Power by Plant Area

Reagents

Reagent costs have been based on dry powder bulk delivery to site with on-site mixing required, for hydrated lime, cement and ground blast furnace slag – bulk truck facilities have been allocated for typical 20 t deliveries. Some of the reagents such as MIBC, A3477 and W31 will be supplied in liquid form in 1,000 L liquid bulk containers and will not need dilution.

Hydrated lime has been chosen for use as the pH modifying reagent on a safety basis to avoid the interaction and dangers associated with slaking lime from quicklime. A future operational cost saving option to use quicklime will have to be weighed against the payback period for capital expenditure associated with a lime slaking plant.

The unit costs and consumptions used for the reagent cost estimate are summarized in Table 21.6.

Item	Unit	Consumption			Cost		Variable Cost $/t Ore
		Basis	Rate	Annual Total	Unit Cost	Annual Cost	
Reagents							
Hydrated Lime	t	g/t ore	1,320	1,590	188	298,524	0.25
MSP	t	g/t ore	435	524	1,050	550,155	0.46
Dextrin	t	g/t ore	345	416	1,210	502,819	0.42
Areophine 3477	t	g/t ore	129	155	3,341	519,157	0.43
W31	t	g/t ore	102	123	4,800	589,723	0.49
MIBC (AeroFroth 70)	t	g/t ore	136	164	4,651	761,947	0.63
Flocculant Tails Thickener	t	g/t tailings	30	33	2,935	96,515	0.08
Flocculant Cu Cons Thickener	t	g/t conc	30	3	2,935	9,542	0.01
Flocculant Paste	t	g/t tailings	45	49	2,935	144,772	0.12
Anti-scalent	t	g/t ore	10	12	2,500	30,113	0.03
Paste Plant							
Cement	t	kg/t tail	10.4	11,426	183	2,094,396	1.74
Binder	t	kg/t tail	10.4	11,426	199	2,272,361	1.89
Lime	t	kg/t tail	0	0	198	0	0.00

Table 21.6 Reagent Operating Costs

Grinding Media

The grinding media costs are summarized in Table 21.7. Usage rates for grinding media were developed from the bond grinding test Ai figure and supplied by OMC as part of their comminution review. An adjustment has been made to the ball mill usage figure to represent a saving of 30% for the use of hi-chrome (stainless steel) media. This figure is based on industry experience and is a conservative adjustment to the usage rate.

Item	Unit	Consumption			Cost		Variable Cost $/t Ore
		Basis	Rate	Annual Total	Unit Cost	Annual Cost	
Grinding Media							
SAG Mill Balls	t	kg/t ore	0.84	1,007	1,259	1,267,765	1.05
Ball Mill Balls	t	kg/t ore	1.36	1,641	1,776	2,897,174	2.41
Regrind Mill Media (Ceramic)	t	kg/t conc	0.28	30.3	3,050	92,546	0.08

Table 21.7 Grinding Media Operating Costs

The grinding media consumption and cost for the IsaMill™ was based on 2.5 mm MTX ceramic media for Cu concentrate regrind application from database information.

Concentrate Freight

Concentrate freight is by far one of the largest costs to the operating cost and has been based on $105.21/wmt to port and loaded onto the ship. Based on the mass balance physicals the operation will produce 133,276 wmt per annum at 10% moisture content. This equates to $12.7M per annum or $10.52/t.

21.1.3 Operations Management and Manning

The operating philosophy of the processing plant has been based on a continuous 24 hour operation with two 12 hour shift rotations. The processing plant will use a seven days on/seven days off, four panel shift roster to achieve the production targets. Technical based personnel will be based on a nine days on/five days off roster with a back to back arrangement for supervisors and superintendent roles. Senior management and administration will be based on a five days on/two days off arrangement. All roles are locally based with no fly-in or out option.

Organization

The Project is to be operated with a departmental structure as shown in Figure 21.3. The total direct workforce requirement for the process plant operation is 119 employees. The processing structure provided is considered adequate to maintain and operate a plant of this size and complexity.

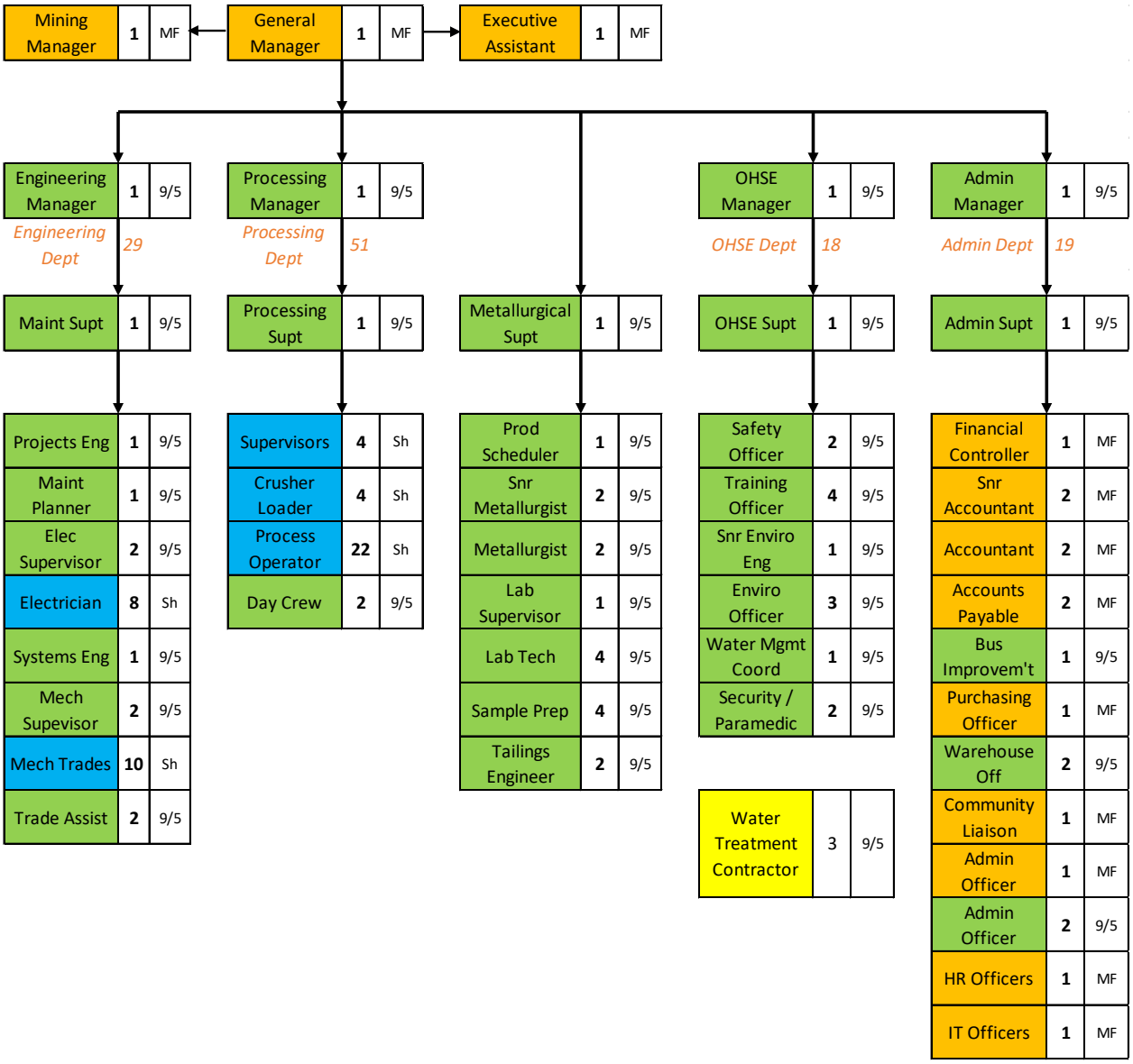

Figure 21.3 Processing Organizational Structure – (Excludes Mining Structure)

21.1.4 *Operating Cost Overall*

The combined mining and processing costs for the Project, given in Table 21.8, are $77.06M per annum and $64.84/t (mined ore to mill). The overall operating cost as a function of Cu lbs is $1.41/lb Cu produced.

Cost Centre	Annual Cost $ million	Unit Cost $/t
Processing Costs – Mining		
Labor	8.36	7.21
Equipment Operations	5.58	4.81
Fuel	0.89	0.77
Power	0.61	0.53
Propane	0.20	0.17
Consumables	10.00	8.62
Other	1.45	1.24
Sub-Total	**27.09**	**23.35**
Processing Costs – Operations		
Salaries/Labor	7.29	6.06
Power	5.95	4.94
Reagents	3.50	2.91
Water Treatment	0.16	0.13
Grinding Media	4.26	3.53
Paste Consumables	4.37	3.63
Other Consumables	2.15	1.79
Maintenance	3.05	2.54
General	0.38	0.32
Crusher Feeding	0.27	0.23
Heating Requirements	0.28	0.23
Processing Costs – Administration		
Salaries/Labor	3.03	2.52
Maintenance	0.59	0.49
Laboratory	0.54	0.45
Concentrate Freight	12.67	10.52
General	1.47	1.22
Sub-Total	**49.97**	**41.49**
Total	**77.06**	**64.84**

Table 21.8 Combined Operating Costs

21.2 Capital Cost Estimate Summary

21.2.1 Introduction

The Project capital cost estimate has been developed for the FS based upon an EPCM approach for the construction and commissioning of the Project facilities. This includes mine plant and infrastructure, the process plant and infrastructure, general mine infrastructure and roads.

The estimate includes all the necessary costs associated with process engineering, design engineering and drafting, procurement, construction and construction management, commissioning of the operational facilities and their associated infrastructure, mining services establishment, first fills of reagents, consumables and spare parts to design, procure, construct and commission all of the facilities required to establish the Project.

The estimate is based upon preliminary engineering, quantity take-offs, budget price quotations for major equipment, and bulk commodity costs achieved recently on similar projects undertaken in the minerals processing industry. Unit rates for installation were benchmarked to those achieved recently on similar projects undertaken in the minerals processing industry.

The estimate pricing for the major equipment was obtained predominantly during 2Q19, the bulk material rates and labor were updated during 2Q20 and any equipment quoted in foreign currencies has been converted using the foreign exchange rates in Table 21.9 to be presented in USD$. The capital cost estimates presented in this document are considered to have an overall accuracy of ± 15% based on the following:

- Developed engineering quantities from preliminary calculations and layout drawings;
- Budget quotations obtained for major items and site-based contract works;
- The capital cost estimate was broken down using a conventional Work Breakdown Structure (WBS) with plant areas (i.e. crushing, grinding and classification, flotation, filtration and concentrate handling, etc.) as sub-categories;
- In addition, the capital cost estimate was broken down into commodity components (i.e. earthworks, concrete, structural, mechanical, piping and electrical, etc.).

The quantity of labor-hours and therefore costs for engineering design, procurement, construction management and commissioning were all estimated from first principles using in-house data and experience gained from similar projects.

21.2.2 Capital Cost Contribution

The capital estimate for the Black Butte Copper Project has been developed by the following contributors:

- GR Engineering Services: process plant, plant infrastructure, non-process infrastructure and compilation of overall capital estimate;
- Mining Plus: mine infrastructure, mine fleet and mine ancillary costs;
- Knight Piesold: Earthworks quantities and rates;
- Hydrometrics: Treated Water Disposal system;
- Fergus Electric Co-op: HV Overhead Power Transmission Lines and HV switch station;
- IDE: Water Treatment Plant;
- Sundt: Site supervision, labor and construction equipment rates;
- MK Weeden Construction Inc.: Bulk Earthworks;
- SRA: owner's costs.

21.2.3 Foreign Exchange Rates

The following rates were used in the estimate for conversion to USD$.

Currency	Exchange Rate
AU$	1.48
€	0.90
CA$	1.34

Table 21.9 Foreign Exchange Rates

21.2.4 Capital Cost Estimate Summary

The capital cost estimate includes all costs before the commencement of production. The capital costs are summarized in Table 21.10. Capital cost details are included in Appendix E.

Area	Capital Cost $M
Mining	$65.1
Site Infrastructure (including Engineering & Procurement)	$91.4
Mineral Processing	$72.7
Construction Management, Construction Equipment and Owner Costs	$20.5
Contingencies (mine, process, infrastructure and owners costs)	$25.0
Total Project	**$274.7**

Note: *Capital cost details for each Area in this table are included in Tables 21.11, 21.12, 21.13, 21.14, and 21.15.*

Table 21.10 Total Project Capital Cost Summary

21.2.5 Capital Cost Estimate Summary – Mining

The total UG mining capital cost is $105.0M (Table 21.11) and includes project capital and sustaining capital and excludes contingency. The total UG mining capital cost of $65.1M excludes contingency and also excludes $1.4M of indirect costs. Sustaining Capital costs total $38.6M. Of the $38.6M Sustaining Capital, approximately 47% is mobile equipment purchases or rebuilds and a further 39% is capital development. Owner's costs of $0.4M for mine development are excluded from Table 21.11 and included in the Area referred to as "Construction Management, Construction Equipment and Owner Costs" in Table 21.10.

Area	Capital Cost $M
1100 UG Mobile Equipment	$27.0
1200 UG Stationary Equipment	$3.1
1300 Capital Development	$42.7
1400 UG Infrastructure and Services	$6.7
1500 Capitalized Operating Costs	$22.5
1600 Indirects	$3.0
Grand Total	**$105.0**

Table 21.11 Total Mining Cost Summary

Underground Mobile Equipment

The UG mobile equipment cost area comprises the cost to purchase as well as rebuild and replace the mobile equipment fleet throughout the mine life. Budget costs were received from equipment OEM's, quantities and timing of purchases was determined from equipment productivities calculated from first principles and the LOM schedule.

Underground Stationary Equipment and Services

The stationary equipment and services area includes the cost of purchasing and installing all major stationary equipment, including pumps and fans, as well as sustaining costs over the LOM. Budgetary quotes were received from stationary equipment manufacturers sized to meet the design requirements.

Capital Development

Any operating expenses in the initial pre-production period were capitalized. For discussions on the operating cost estimate see Section 21.1. This line item is not net of any revenue accrued during the pre-production period.

Underground Infrastructure and Services

Designs of UG infrastructure including service bays, portal development and explosives magazines were prepared. Material quantities were determined from the designs and then budgetary quotes were received for equipment and material to equip these excavations. Costs for installation of main services including backfill piping, communications systems and main electrical feeder were included.

Capitalized Operating

Any operating expenses in the initial pre-production period were capitalized. This included mine department salaries, power and heating costs as well as contractor indirect costs. For discussions on the operating cost estimate see Section 21.1. This line item is not net of any revenue accrued during the pre-production period.

Indirects

Indirects include costs such as freight, commissioning, first fills as well as owner's costs related to further engineering studies.

21.2.6 Capital Cost Estimate Summary – Infrastructure

Table 21.12 details a summary by area of the capital cost estimate for the design, construction and commissioning of the site infrastructure facilities.

Area	Capital Cost $M
200 Earthworks	
Cemented Tailings Facility	$18.0
Portal Pad, Brine Pond and Contact Water Pond	$0.0
Waste Rock Storage Pad	$2.1
Process Water Pond	$7.2
Cu Enriched Stockpile	$0.2
Reclamation Materials Stockpile North	$0.1
Reclamation Materials Stockpile South	$0.1
Non-Contact Water Reservoir Earthworks	$1.3
Treated Water Storage Pond Earthworks	$2.9
Fibre Cable Route	$0.1
201 Roads	$1.8
203 Fencing	$0.5
393 Water Treatment Plant	$5.4
394 Water Disposal System	$2.7
410 Fuel Storage and Distribution	$0.7
430 Plant Administration Buildings & Offices	$4.9
450 Emergency Power	$2.7

Area	Capital Cost $M
460 Laboratory	$1.5
470 Power Reticulation – Infrastructure	$13.3
490 Communications	$1.1
Grand Total (Excluding Tax)	**$66.7**

***Note**: Capital costs for the PWP, WTP (393), and water disposal system (394) in this table are listed in Area referred to as "Mineral Processing" Capital Costs of Table 21.10. All other Areas listed in this table are referred to as "Site Infrastructure" Capital Costs of Table 21.10.*

Table 21.12 Site Infrastructure Capital Cost Estimate Summary

Earthworks

The estimate is based upon preliminary engineering and quantity take-offs from the drawings using bulk commodity costs tendered for the Project and achieved on similar projects undertaken in the minerals processing industry.

Road Upgrades

The estimate is based upon preliminary engineering and quantity take-offs from the drawings using bulk commodity costs achieved on similar projects undertaken in the minerals processing industry.

Fencing

The estimate is based upon preliminary engineering and quantity take-offs from the drawings using bulk commodity costs tendered for the Project and achieved on similar projects undertaken in the minerals processing industry.

Water Treatment Plant

IDE (2020) provided a budget estimate for a turnkey, design and construct, water treatment plant at the required flowrate based on the available water samples.

Fuel Storage and Distribution

The estimate is based on diesel and propane storage facilities quoted on recent projects for similar sized equipment - tanks, filtration and pumping equipment.

Plant Administration Buildings and Offices

The estimate is based upon preliminary engineering and quantity take-offs from the building drawings using bulk commodity costs achieved on similar projects undertaken in the minerals processing industry.

Emergency Power Generation

The estimate includes two 2,500 kV 13.8 kV Caterpillar 3516CHD diesel generators as backup emergency power generation as quoted by Wagner Power Systems.

Laboratory

Tenders were received from recognised turnkey laboratories for the design, supply, construction and commissioning of a laboratory to process the types and quantities of samples to efficiently undertake the mine grade control and plant metallurgical sample analysis.

Power Reticulation - Infrastructure

Budget costs were received from Fergus Electric Co-op for the design, supply and installation of the high voltage overhead power transmission line to the plant and high voltage switchyard.

Communications

The communications area makes allowances for external coms, fibre cable to the facilities, mobile radio, telemetry, WAN & LAN, a telephone system, IT needs and a communications shelter based on similar projects.

21.2.7 Capital Cost Estimate Summary – Mineral Processing

Table 21.13 details a summary by processing area of the capital cost estimate for the design, construction and commissioning of the new processing facilities before the commencement of production excluding contingency.

Area	Capital Cost $M
Direct Costs	
200 Earthworks	
Plant Bulk Earthworks and Roads	$0.8
310 Crushing and Screening	$4.3
320 Coarse and Fine Ore Storage & Handling	$3.1
330 Grinding and Classification	$13.0
331 Pebble Crushing & Conveying	$1.3
336 Flotation	$5.4
344 Regrind Circuit	$3.6
345 Sampling and Analysis	$0.9
351 Concentrate Thickening and Filtering	$3.5
360 Reagent Mixing and Distribution	$1.4
370 Power Reticulation - Plant	$11.7
390 Water Storage and Reticulation	$1.4

Area	Capital Cost $M
400 Tails Thickening	$1.4
405 Paste Plant	$9.3
406 Paste Pipeline to Decline	$0.5
407 Paste Pipeline to CTF	$1.0
420 Air Services Supply and Reticulation	$0.5
499 Plant Piping	$5.0
Grand Total (Excluding Tax)	**$68.2**

Note: *Direct costs for the Paste Plant (405), Paste Pipeline to Decline (406), and Paste Pipeline to CTF (407) in this table are listed in Area referred to as "Site Infrastructure" Capital Costs of Table 21.10.*

Table 21.13 Processing Capital Cost Estimate Summary

Estimate Basis

The capital cost estimate presented in this study relates to capital works required to construct a new processing plant and support infrastructure facilities. Design criteria and flow sheets for the process plant and infrastructure were developed using historical data, metallurgical test work data, and in house experience.

From the developed processing route, plant equipment selections were made and plant layouts were developed. Sufficient preliminary engineering design was undertaken to ensure the functionality of the proposed layouts, suitability of equipment specifications, and to enable construction material quantities to be estimated within the desired level of accuracy.

The estimate is prepared on a commodity basis and reported by area. Details of the structure of the estimate are provided by commodity in this section.

Current market pricing for equipment, labor and bulk rates are incorporated into the estimate. The installation rates include all charges necessary to deliver the requirements of the Project.

Earthworks

Plant and infrastructure site earthworks quantities were estimated off the plant layout. An overall bill of quantities for the Project was compiled and a costed schedule of rates was used for the estimate. The basis assumes a single sub-contractor on multiple work fronts performing all the nominated earthworks for the Project (roads, plant, pipelines, CTF and water pond works).

Concrete

The concrete quantities were calculated for each area from the general arrangement drawings, layout drawings and preliminary designs developed for the Project.

Written quotations for all in market rates from local contractors were used against the calculated bill of quantities for concrete supply and installation. The rates used are inclusive of a batching plant and are reflective of the overall quantity of the scope.

Labor hours and rates applied to the concrete works durations for material and installation distribution were estimated based on current industry standards.

Structural Steelwork and Platework

Quantities were calculated, on an area by area basis, from the general arrangement drawings and layout drawings developed for the Project.

Market rates from recent similar projects were used against the calculated bill of quantities to give a schedule of rates pricing for all-inclusive supply. The rates utilized for structural steelwork includes heavy, medium, and light members, conveyor gantries, and trestles. The rates utilized for plate work includes small bisalloy lined bins and chutes, rubber lined chutes, hoppers and various other. Separate rates are used for grid mesh flooring, hand railing, and stair treads.

The supply rates include materials supply, shop detailing, fabrication, surface preparation, final painting in the shop, fasteners and identification tagging.

Equipment

The process design criteria were used to develop the mechanical equipment list that defines the requirements and sizes of all the mechanical equipment, plate work and tank items. Specifications and data sheets were developed for all major equipment.

Written budget quotations from enquiries accompanied by engineering specifications and data sheets were requested from recognized suppliers for all of the following major equipment in the plant:

- Primary crusher;
- Apron feeders;
- Dust collectors;
- Pebble crusher;
- SAG mill;
- Ball mill;
- Vibrating screens;
- Regrind mills;
- Flotation cells ;
- Flotation blowers;
- Hydrocyclones;

- Tailings and concentrate thickeners;
- Concentrate filters;
- Sump, slurry and process pumps;
- Paste pumps.

Costs derived from recent projects completed to beyond FS study level and preliminary vendor enquiries were used to estimate costs for the following major items of process equipment:

- Lime and binder silos;
- Conveyor components;
- Flocculant plant as a package;
- Lime storage and mixing system;
- Samplers and analysers;
- Weightometers;
- Compressed air equipment;
- Weighbridge;
- Non process buildings and steel framed buildings (workshops and offices).

Piping

The piping estimate was factored against the total plant capital cost estimate and benchmarked against similar recent projects.

Overland piping quantities have been calculated from process requirements and geographical locations of the water sources and the location of the CTF for the tailings. Database rates provided by piping suppliers for various specification pipe and valves and fittings were applied to the developed quantities.

Electrical and Instrumentation

The electrical, instrumentation and control quantities have been compiled from the Project scope, single line diagrams, layouts, equipment list, and load list. The instrument list was developed from the Piping and Instrumentation Diagram (P&ID's) for this plant.

Cable and material quantities were estimated based on layout, switch room locations, equipment specific requirements and drive requirements, and schedules produced.

Database pricing has been used for all electrical components (i.e. switch rooms and MCC's).

Transport and Freight

Major equipment suppliers were requested to provide pricing to transport equipment to site. Where suppliers have not included transport, an allowance based on the estimated size or weight of each item has been allowed.

Sea freight and packing charges have been included to transport all imported materials and equipment from place of manufacture to site. These costs are included in the supply cost of the item.

Transport for fabricated steel items has been allowed based on quantities expected to be loaded per truck and anticipated truck price between selected suppliers and site. Where possible, fabricated steel components will be suitably packed onto skids and containerized to minimize handling costs.

Installation Labor

Estimates for installation labor were based on estimated labor hours associated with the equipment and fabricated items to be installed in each area of the plant. The estimated hours for installation reflect the labor force productivity for construction sites for the minerals industry and the application of industry standard labor rates for the type of work involved. The rates were developed with due consideration to the rates actually achieved on recent similar minerals processing projects with input from local construction contractors.

Labor crew rates were built-up including an appropriate mix of supervision, skilled and unskilled personnel. Each crew rate included the costs of mandatory meetings and breaks, small tools, statutory labor costs, PPE, and clothing.

The construction labor rates for SMP and E&IC are based on the current Sundt site installation rates. The rates developed are relevant to the current industry and the location of the Project.

Plant Services

The capital cost estimates compiled for the plant services/infrastructure components of the Project are based on requirements dictated by the current process plant design/capacity and plant layout. Dimensions and details are provided in the mechanical equipment list.

The main scope of work items covered by the capital cost estimate are summarized as follows:

- Process plant office including a change house, crib room, ablution, and a central plant control room;
- Laboratory building, office and equipment;
- First aid facility;

- Workshop and stores sheds and offices;
- Gate house and training facility.

Commissioning

Applicable engineering labor rates have been adopted for the commissioning team. The team will consist of engineers from various disciplines, the relevant vendor commissioning representatives and a team of tradespeople selected form the construction workforce. A travel allowance has been made to transport the dedicated commissioning team to and from the site.

Certain items of major equipment such as the crushers, SAG and ball mills, regrind mills, flotation cells, filters, and the thickeners will require vendor representation on site during final commissioning. A vendor commissioning hire cost and travel allowance has been included in the estimate for this purpose. This expenditure will ensure that equipment warranties are preserved and any early operating issues are resolved ahead of handover.

21.2.8 Capital Cost Estimate Summary – Project Indirects (EPCM and Owner Costs)

Indirect costs were estimated by GRES with SRA providing the Owner's costs. The costs were based on quotes from vendors and historical costs from previous projects. A summary of the indirect costs are shown in Table 21.14.

Area	Total $M
500 Project Management	$3.8
501 Engineering and Drafting	$7.1
502 Site Supervision and Management	$8.8
503 Site Construction Cranes and Equipment	$6.2
504 Site Construction Facilities	$0.6
505 Commissioning	$1.1
570 Mobile Equipment	$2.0
602 Initial Fills	$0.8
603 Spare Parts	$2.9
840 Mobilization/Demobilization/Indirect Costs	$10.3
Owner's Costs – General and Administrative	$4.9
Grand Total	$48.5

Note: *Indirect costs for Site Supervision and Management (502), Site Construction Cranes and Equipment (503), Site Construction Facilities (504), and Owner's Costs – General and Administrative in this table are listed in Area referred to as "Construction Management, Construction Equipment and Owner Costs" Capital Costs of Table 21.10. The italicized Area costs in this table are referred to as "Site Infrastructure" Capital Costs of Table 21.10.*

Table 21.14 Indirect Cost Summary

Cranes and Equipment Costs

Estimates for cranes and equipment costs are based on estimated hours of utilization for major cranes and equipment items associated with installation in each area of the plant. The application of locally sourced standard charge-out rates for the various cranes and equipment types involved were supplied by Sundt. The applied rates do not include fuel; however, this cost has been estimated separately.

Construction Facilities

The estimate includes for EPCM offices for the duration of construction on site. An allowance has been made for the hire, establishment, operation and removal of temporary construction offices, crib rooms and ablutions for the construction workforce. SMP and E&I subcontractor offices and facilities for the durations of these construction works are included. Consumables and services are included for the duration of the construction period.

Construction Fuel

Construction fuel has been allowed for the duration of construction at $0.34/L ($1.29 per gallon). Construction fuel is included in the earthworks and concrete all-in rates.

Mobilization and Demobilization

Estimates of the mobilization and demobilization costs for each of the construction contractors have been included in the capital cost estimates. These costs have been based on data sourced from recent projects and make allowance for materials, equipment and personnel.

R&R Flights, Meals, and Accommodation

It has been assumed that temporary construction accommodation and messing facilities shall be established in the vicinity of the Project to cater for the needs of some of the construction workforce, and that accommodation and messing will be charged at $150 per person per day.

Flights, meals and accommodation have been included for all direct labor and indirect construction personnel for the duration of the construction period. The flights for the EPCM personnel have been allowed for in the estimate.

Engineering, Procurement and Construction Management

The EPCM estimate was developed from first principles for the personnel required to perform all activities associated with the Project, for the planned duration of the Project. The involvement of each team member was individually estimated based on the complexity of the task and benchmarked against the EPCM costs incurred on recent similar projects. The design, procurement, and management aspects of the EPCM work will be undertaken off-site in the offices of the EPCM engineer. The EPCM team will work in collaboration with the owner's team to complete the Project.

The capital cost estimate contains an allowance for a site based construction management team and site supervisors to be engaged for the duration of the site works. Travel, site vehicles, and site accommodation costs for these people have been included in the study estimate.

Initial Fills

First fills and operating stocks are calculated and costed for the Project.

Spares

Commissioning spares are included as 1.5% of the mechanical supply costs.

Mechanical capital spares are included in the estimate as 4% of the mechanical supplied costs.

Electrical capital spares are included in the estimate as 4% of the supplied costs excluding the power station.

Owner's Costs

Owner's costs are included as approximately 2.5% of the total project costs.

21.2.9 Contingency and Escalation

Contingency costs were estimated by Mining Plus for the mine and GRES for the Infrastructure and Process Plant. The summary contingency costs are shown in Table 21.15.

Area	Total $M
Mining Contingency	$6.7
Process Contingency	$11.8
Infrastructure Contingency	$6.1
Owners Costs Contingency	$.04
Grand Total	$25.0

Table 21.15 Contingency and Escalation Cost Summary

Engineer's Contingency as Growth

Engineer's contingency costs and allowances commensurate with the level of design and estimating confidence have been included. GRES normally apply 5 – 10% contingency on the standard plant items, with higher contingency attributable to the less defined infrastructure items (e.g. roads and the raw water borefield). This contingency allowance does not include changes to the process flow sheet, process plant design, or major equipment selections.

Escalation

Escalation is excluded from the estimate.

21.2.10 Capital Cost Estimate Clarifications

The following qualifications apply to the capital cost estimate presented in this Study:

- No allowance has been made for sunk costs incurred by the Principal prior to project implementation;
- No allowance has been made for interest charges or capital financing costs;
- No allowance has been made in the estimate for exchange rate fluctuations with respect to equipment sourced from outside the United States of America;
- The capital cost estimate has been compiled to represent an EPCM project execution by a competent and proven engineering and construction organization capable of delivering within the schedule and required budget accuracy in conjunction with the client's owner's team;
- The EPCM capital cost is representative of the effort required to design, manage and execute the scope not deemed to be turn-key scopes of work that will be managed directly by the owner's team therefore requiring minimal input from the EPCM engineering team. A Project and Construction Manager provided by the EPCM contractor for the duration of the works will be a single combined role and will work in conjunction with the client's counterpart;
- All bulk earthworks materials general borrow/fill material, wear course and select fill material will be sourced at site;
- The main access road to the site is designed to meet the requirements of the Project. Internal roads that service the mine services area and the infrastructure facilities are designed for their application;
- Concrete supply at the site will be by mobilized batching plant with cement and aggregates sourced from local suppliers. Construction water to be provided by the owner for use by the sub-contractors is assumed to be of good quality. Any costs associated with improving the existing construction water source or sourcing good quality water for construction from off-site has not been included;
- Drilling and casing of any bores to provide construction water is not included in the estimate;

- No allowance has been made in the estimate to procure or lease land to locate the temporary construction accommodation and messing facilities.

21.3 Sustaining Capital

21.3.1 Mining Sustaining Capital

Of the $38.6M Sustaining Capital (excluding contingency), approximately 47% is mobile equipment purchases or rebuilds and a further 39% is capital development.

21.3.2 Process Plant and Infrastructure Sustaining Capital

A provision for minor projects of 0.25% of the plant CAPEX was applied over the LOM, equating to $1.08M. Two allowances for mill refurbishment or repairs were also listed as 10% of the mill capital costs, allowing for $0.6M over LOM.

Replacements of surface vehicles was limited over the LOM to $0.23M, while other mobile equipment replacement or major repairs not associated with the main operating cost accounts for $0.39M. A $10M rehabilitation accrual is associated with year 10 of the LOM.

21.4 Closure Cost

Mine closure as described in the MOP Application (Tintina Montana, 2017) is expected to take four years to complete and is estimated to cost $24.3M (McIntosh, 2020). SRA believes that the tasks to implement the AMA in mine closure can be done within this same four year closure schedule; however, additional costs will be incurred as a result that will need to be added to the total mine closure cost. SRA and the MDEQ are in the process of finalizing the AMA tasks and costs at the time of this Report.

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22. ECONOMIC ANALYSIS

22.1 Introduction

The economic analysis for the Black Butte Copper Project has been prepared based on the mining schedules and mining cost estimates prepared by Mining Plus and described in Sections 16 and 21. The process and infrastructure costs were prepared by GRES and from management estimates also described in Sections 18 and 21. An economic model has been prepared in Microsoft Excel using the provided costs and schedules to calculate the LOM economic outputs. The economic model taxation treatment was reviewed by dm-t Certified Public Accountants, Spokane, WA.

The economic analysis has been prepared in accordance with Canadian standard NI 43-101 for Technical Reports. Under this standard, the economic analysis must be based on Mineral Reserves only which is presented herein and termed the "FS Case".

For the FS, SRA has undertaken exploration and drilling to develop an updated Mineral Resource estimate that was reported in October 2019. The Measured and Indicated portions of the Mineral Resource have been used to develop a Mineral Reserve, which in turn provides an initial Reserves-based mine life suitable for project financing. As a result, the total FS plant feed estimate for the UG is reduced by approximately 3.0 Mt compared to the 2013 PEA amount of 11.8 Mt which included Measured, Indicated and Inferred Mineral Resources at that time.

The economic analysis for the FS uses a copper price of $3.20/lb from the start of Cu concentrate production. Higher Cu prices give significantly improved economic outcomes (see Section 22.4).

In addition, there is good potential for the Project in the future to deliver significantly greater tonnages from UG based on both successfully converting Inferred Resources to Measured or Indicated Resources and on the broader exploration potential of the Black Butte Copper Project mineralized system (see Sections 9 and 14).

22.2 Economic Analysis

22.2.1 Financial Model Parameters

Section 24 'Other Relevant Data and Information' includes many of the assumptions that were used in the economic model in addition to the assumptions outlined below.

22.2.2 *Mineral Resource, Mineral Reserve, and Mine Life*

Mineral Resources at the Johnny Lee deposit were updated in October 15, 2019 by SRK Consulting. Mining Plus have used these Measured and Indicated Mineral Resources to calculate a Mineral Reserves from the UCZ and the LCZ totalling 8.8 Mt averaging 2.6% Cu as determined at October 19, 2020. The Mineral Reserves are scheduled to be produced over a LOM of approximately eight years through the 1.2 Mtpa plant. In addition to production LOM there will be approximately two years of construction and four years of mine closure activities.

22.2.3 *Metallurgical Recoveries*

Design work for the FS has been based on a weighted average recovery from the LCT's for equipment selection and sizing. As the LCT's only cover several key composites, a relationship was developed between the LCT's and the cleaner variability tests for the remaining composites. Table 22.1 details the percentage basis from the ore resource and mine plan to yield the design figure.

LCT - Composite	Mine Ratio	Cu% Grade	Cu% Recovery	As (ppm) Final Concentrate
LCT150-BBFT01	22%	25.7	71.6	3090
LCT151-BBFT03	22%	27.1	90.1	3090
LCT152-BBFT04	12%	18.9	75.1	9300
LCT153-BBFT10	22%	25.6	86.3	8740
LCT154-BBFT12	11%	21.8	70.6	17500
LCT156-BBFT20	11%	16.9	77.9	4280
Weighted Average	100%	23.5	80.8	7198

Table 22.1 Locked Cycle Test Results

Metallurgical recoveries used in the economic model are developed from the mining block model for each variability sample using the regression model discussed in Section 13.6. In order to replicate the LCT results for variability samples, the relationship between mineralogy and tested variability Cu cleaner recovery was developed to provide a regression for predicted Cu recovery. This data is then converted to a LCT Cu cleaner recovery.

The economic model has been based on the nameplate production rate using the design recovery and grade for the base case. Mining block modelling has then used the relationships to develop a recovery basis for each month based on each variability sample recovery basis. Based on the variability across the deposit, the economic model generated a LOM recovery of 83.8%. Monthly based variability ranged from 74.9% to 89.3% Cu recovery, which is within the defined range of variability samples test results.

22.2.4 *Smelting and Refining Terms*

A treatment charge and a refining charge (TCRC) are the payments made by the miner to the smelter for smelting and refining services. TCRC's were initially established to reflect the individual costs associated with the smelting and refining process. TCRC levels are set based upon the demand and supply of custom Cu concentrates. Treatment charges are expressed in US dollars per dry metric tonne of concentrate and Cu refining charges are expressed in cents per pound of payable Cu.

Wood Mackenzie's forecast contract TCRC's are presented below in Table 22.2. The treatment and refining charges presented reflect those for international transactions, rather than land-tied, or regional deals where some, or all of the benefits of freight-savings are added to TCRC's.

Item	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030
Treatment Charge $/t (nominal)	82	81	72	78	87	96	100	102	104	106	108	110	112
Refining Charge c/lb (nominal)	8.2	8.1	7.2	7.8	8.7	9.6	10.0	10.2	10.4	10.6	10.8	11.0	11.2
Treatment Charge $/t (2019$)	84	81	70	75	82	88	90	90	90	90	90	90	90
Refining Charge c/lb (2019$)	8.4	8.1	7.0	7.5	8.2	8.8	9.0	9.0	9.0	9.0	9.0	9.0	9.0

Data from Wood Mackenzie (2020)

Table 22.2 Wood Mackenzie Treatment and Refining Charges Forecast

The economic model uses a treatment charge of $90/t and a refining charge of 9.0 cents/lb throughout the LOM.

22.2.5 *Metals Payable in Cu Concentrates*

The Cu, Au and Ag content in Cu concentrates is paid for provided that the Au and Ag contents each exceed a particular level, generally 1 g/dmt for Au and 30 g/dmt for Ag. Normally no payment is made for other metals. The Cu payment formula varies smelter by smelter, but similar conditions exist in the contracts of smelters located within certain regions. As such there are three main regional markets on the smelting side for custom concentrate: Japan and Korea; Europe; and China, India and the Philippines. Typical formulae for each region are outlined in Table 22.3.

Copper				Silver				Gold			
Content	China	Japan	Europe	Content	China	Japan	Europe	Content	China	Japan	Europe
<30% Cu	-1	-1	-1	<30% g/t	None	None	Deduct	<1 g/t	None	None	Deduct
<33% Cu	-1.1	-1.1	-1.1	30-99 g/t	Pay 90%	Pay 90%	30 g/t and	1 – 3 g/t	Pay 90%	Pay 90%	1 g/t and
<36% Cu	-1.2	-1.2	-1.2	100-199g/t	Pay 91.5%	Pay 91.5%	pay	3 – 5 g/t	Pay 94%	Pay 94%	pay
<40% Cu	-1.3	-1.3	-1.3	>200 g/t	Pay 94%	Pay 94%	balance	5 – 10 g/t	Pay 95%	Pay 95%	balance
>40% Cu	-1.4	-1.4	-1.4	-	-	-	-	>10 g/t	Pay 96%	Pay 97-98%	-

Data from Wood Mackenzie (2020)

Table 22.3 Metal Payable in Cu Concentrates

Due to the low values of Ag and Au expected in the Black Butte Copper Project Cu concentrate, no payable metals other than Cu are included in the economic model.

22.2.6 Metal Prices

The 2020 COVID-19 pandemic resulted in highly volatile Cu prices, however the Project economics are driven by long-term Cu prices rather than today's spot prices. Wood Mackenzie (2020) forecasts long-term Cu prices beyond 2025 of $7,275/t ($3.30/lb). The London Metal Exchange (LME) price as of September 15, 2020 is ~6,830/t ($3.10/lb).

SRA has determined a long-term Cu price for the economic modelling and analysis in this FS of $3.20/lb, lower than the Wood Mackenzie forecast.

Table 22.4 shows Wood Mackenzie's forecast Change in Metal Stocks & Stock days while the LME Cu price is SRA's forecast.

Item	2022	2023	2024	2025	2026	2027	2028	2029	2030	2035
Change in Metal Stocks (kt)	340	229	19	NA	NA	NA	NA	NA	13	95
LME Cu Price (2020 US$/lb)	3.00	3.00	3.00	3.10	3.20	3.20	3.20	3.20	3.20	3.20
Stock (days)	71	73	72	NA	NA	NA	NA	NA	65	65

Data from Wood Mackenzie (2020)

Table 22.4 Change in Metal Stocks and Cu Price Forecast

22.2.7 Operating Costs

For the full production years, the combined mining and processing costs for the Project average $77.1M per annum or $64.8/t (mined ore to mill). The overall operating cost as a function of Cu metal pounds is $1.41/lb Cu produced. Refer to Section 21.1.4 for detail.

22.2.8 Capital Costs

The capital cost estimate includes all costs before the commencement of production with a total Project cost of $274.7M.

Area	Capital Cost $M
Mining	$65.1
Site Infrastructure	$91.4
Mineral Processing & WTP	$72.7
Project Indirects (EPCM & Owner Costs)	$20.5
Contingencies (mine, process, infrastructure and indirects)	$25.0
Total Project	$274.7

Table 22.5 Summary of Project Capital Cost

22.2.9 Royalties and Special Mining Tax

The economic model assumes that $10M will be paid to Royalty holders prior to production to reduce the NSR from 5% to 2%. The model also assumes that there will be a $3.9M prepaid royalty balance prior to production.

22.2.10 Closure Costs and Salvage Value

Closure costs have been estimated as part of the financial assurance (Project Bond) estimate work that is presented in Section 20.10.3 of Section 20. For this financial assurance cost estimate, no credit was assumed for salvage value of any mine equipment or materials which is in line with general practices for financial assurance. In reality, in a SRA public company default scenario, an Administrator would be placed in charge of the mining operation who would either try to make the mining operation profitable and/or sell off assets in order to pay off creditors.

22.2.11 Financing

The Project will consider a combination of debt, equity and offtake agreements. The adopted financing solution will be subject to many external factors including but not limited to; capital market support, prevailing macro and micro economic conditions, shareholder support, etc.

No offtake agreements, financing streams or royalties (other than those with the existing landholders) exist over the Project or any part thereof.

22.2.12 Inflation

Inflation has not been factored into revenues or operating costs.

22.3 Financial Results

The main inputs and outputs of the economic model for the FS Case are based on the FS base price deck described in Section 22.2.6. The Project's post-tax NPV at a 5% discount rate is estimated to be $77.6M with an IRR of 13%. Cash Costs (C1) are estimated to be 1.51/lb of Cu. Payback of start-up capital is achieved approximately three years from commissioning.

The Project has been demonstrated to be economically attractive based on the FS Case and the assumptions used. The NPV together with a positive IRR supports the Project's viability and provides a compelling commercial case for immediate development. The recommendations and proposed forward program are discussed further in Section 26.

22.3.1 NPV

The Project has a pre-tax NPV of $124.9M and a post-tax NPV of $77.6M using a 5% discount rate.

22.3.2 IRR

The Project has a pre-tax IRR of $17% and a post-tax IRR of 13%.

22.3.3 Payback

The projected cash flow will allow for the Project construction capital to be paid back in approximately three years.

22.4 Sensitivities

The Project NPV is most sensitive to the commodity price and to mine life extensions on the revenue side, and to the grade (which in turn is driven by dilution considerations) and metallurgical recoveries on the cost side. Other significant factors influencing the Project returns include the upfront and UG development capital and the plant operating costs.

22.4.1 Sensitivity Analysis

Table 22.6 and Table 22.7 show the pre-tax and after-tax impacts, respectively from a 10% decrease or increase in certain drivers.

Sensitivity Variables	Confidence Ranges		$ Millions	Pre-Tax	NPV @ 5%
	Worst	Best	Worst	Best	Point
Cu Selling Price	-10%	10%	$30	$216	$125
Cu Grade	-10%	10%	$33	$214	$125
Cu Recovery	-10%	10%	$33	$214	$125
Concentrate Shipping Costs - Land	10%	-10%	$116	$130	$125
Operating Costs - Mining	10%	-10%	$108	$138	$125
Operating Costs - Process	10%	-10%	$107	$140	$125
Capital Costs - Mining	10%	-10%	$113	$133	$125
Capital Costs - Process and Administration	10%	-10%	$105	$142	$125

Table 22.6 Black Butte Copper Project FS Case Pre-Tax NPV Sensitivity Impacts

Sensitivity Variables	Confidence Ranges		$ Millions	Pre-Tax	NPV @ 5%
	Worst	Best	Worst	Best	Point
Cu Selling Price	-10%	10%	($0)	$153	$78
Cu Grade	-10%	10%	$2	$151	$78
Cu Recovery	-10%	10%	$2	$151	$78
Concentrate Shipping Costs - Land	10%	-10%	$72	$83	$78
Operating Costs - Mining	10%	-10%	$66	$90	$78
Operating Costs - Process	10%	-10%	$64	$91	$78
Capital Costs - Mining	10%	-10%	$69	$86	$78
Capital Costs - Process and Administration	10%	-10%	$62	$94	$78

Table 22.7 Black Butte Copper Project FS Case After-Tax NPV Sensitivity Impacts

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23. ADJACENT PROPERTIES

23.1 Introduction

There are multiple exploration prospects in the vicinity of the Johnny Lee and Lowry deposits. As there are no Mineral Resources on adjacent properties, the QP of Mineral Resource has not reviewed any data associated with these prospects. The following information is provided from SRA and is not necessarily indicative of the mineralization on the Black Butte Copper Project.

At co-ordinates 46°45'28"W; 110°57'14"W, a small scale, open pit iron ore mine (Iron Butte Mine) has been developed, from which oxidized Newland Formation massive sulphide is extracted on an irregular basis and sold to a cement producer in Montana. The Iron Butte mine is located 4.1 km W/SW of the Johnny Lee deposit.

The following Sub-Sections provide a brief summary of prospects located adjacent to the Black Butte Copper Project. Figure 23.1 shows the Cu results from all soil sampling campaigns over the property. The 2016 soil sampling program failed to indicate any significant anomaly. Historic soil sampling outlined significant Cu anomalism at the Johnny Lee deposit and the Copper Creek prospect. Minor surface Cu anomalism occurs at the Lowry deposit, Sawmill Hill prospect, and Butte Creek prospect.

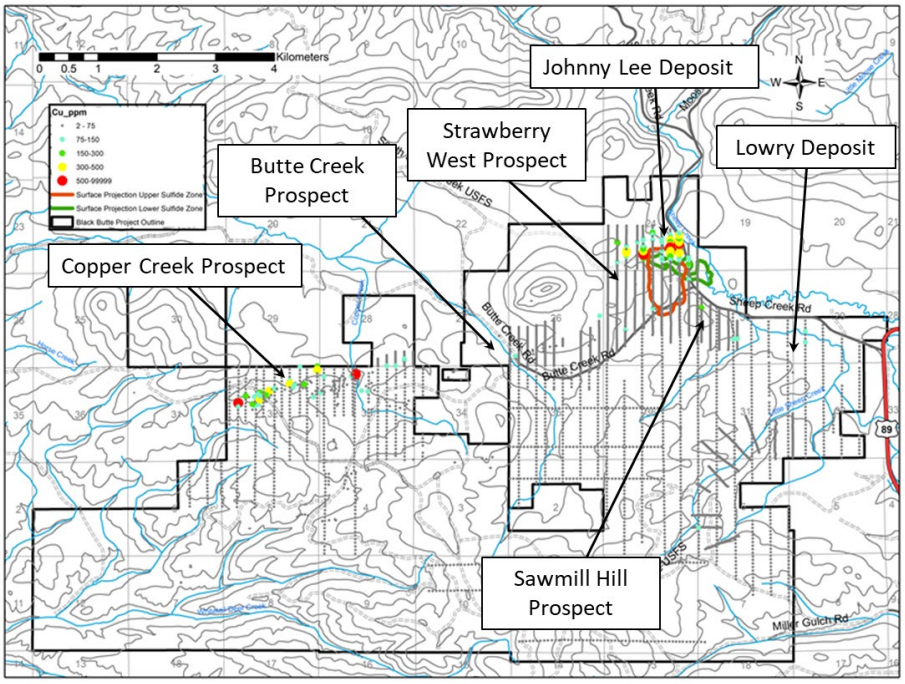

Figure 23.1 Plan Showing Soil Sampling Cu Results, Deposits and Exploration Targets

23.2 Sawmill Hill Prospect

The Sawmill Hill prospect comprises an area of the Johnny Lee USZ measuring 575 m x 600 m in plan view (Figure 23.2), that is untested by drilling. Discontinuous Cu anomalism has been identified in soil samples along the northern and eastern flanks of Sawmill Hill (Figure 23.1).

23.3 Strawberry West Prospect

Four historic drillholes have been completed in the Strawberry West prospect, targeting the down-plunge extension of the USZ in the axial region of the syncline (Figure 23.2). Two of these drillholes intersected significant Cu mineralization in the USZ:

- SC-73: 12.34 m at 2.26% Cu from 441.05 to 453.39 m;
- SC11-095: 6.79 m at 1.77% Cu from 455.53 to 462.32 m.

The mineralized zone is open for 315 m to the east and no drilling has been completed to the west. The mineralization in SC-73 and SC11-095 has been closed off, 80 to 150 m to the south, by drillholes SC-70 and SC11-099 and is truncated by the VVFZ, 70 to 160 m to the north. The zone of mineralization is 750 m down-plunge of the UCZ but is not a continuation of this deposit, rather it represents a discrete zone of mineralization for future studies.

23.4 Butte Creek Prospect

An isolated medium tenor Cu soil anomaly occurs in this area (Figure 23.1) Two historic drillholes (SCC-44 and SC-66) spaced at 500 m intersected anomalous Cu mineralization (0.1% to 0.3% Cu) over narrow (<6 m) intervals.

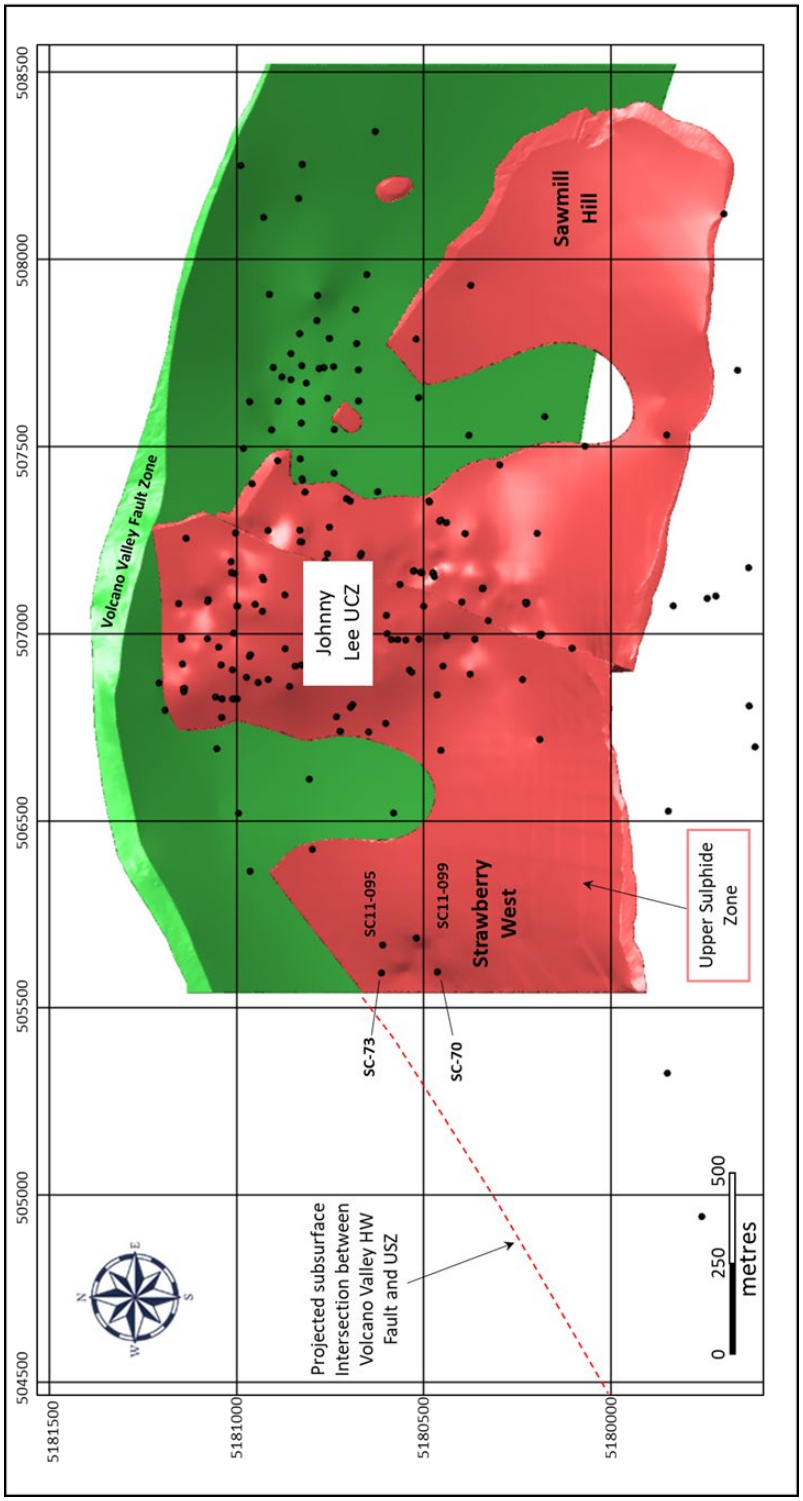

Figure 23.2 *Plan View of the VVFZ And USZ Solids from the Johnny Lee Lithostratigraphic Model, showing the Sawmill Hill and Strawberry West Prospects*

23.5 Copper Creek Prospect

The Copper Creek Prospect comprises a 2,500 m strike length and robust Cu in soil anomaly (Figure 23.1). This surface anomaly coincides with mapped gossan that is interpreted to be oxidized massive sulphide of the Newland Formation. Four widely spaced (580 to 1,750 m) drillholes have been completed in this area but did not intersect strongly anomalous or economically significant Cu mineralization. The source of the soil anomalism has yet to be explained.

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24. OTHER RELEVANT DATA AND INFORMATION

24.1 Project Implementation Plan

It is the intent of SRA to undertake all implementation work on Black Butte Copper Project under an Engineer, Procure and Construction Management (EPCM or EPC) style of Contract.

SRA will establish a client's representative team to manage the requirements for the delivery of the Project.

The capital cost estimate was developed on the basis that the Project would be implemented under an EPCM methodology for the major process plant components, allowing for some form of performance warranty or guarantee to be included.

For the implementation phase, an implementation plan will be developed. The focus of the plan will be to investigate in detail the sequence of the Project to define:

- Overall project timing;
- Project critical path(s);
- Requirements for early works and commitments;
- Areas where significant risk of schedule over-runs exist.

Optimizing the delivery methodology, particularly in respect to interfaces between construction work packages, will be investigated early in the implementation phase of the Project.

The implementation plan will include the following:

- Scheduling of mining and infrastructure works to integrate with the construction of the process plant;
- Scheduling of the process plant works to ensure that plant is commissioned in line with ore availability and development of the mine;
- Definition of the contracting structure to be employed;
- Assessment and definition of resourcing levels required for design and construction.

24.2 Project Personnel (Owner's Team)

SRA will establish a client's representative team to undertake:

- Provision of project management services;
- Coordination of sub-consultants to SRA for the provision of selected infrastructure items;
- Management of project meetings and coordination between the design and the Engineer;

- Provision of site construction expertise for the overseeing and management of site works in conjunction with the Engineer;
- Formulation and management of the mining contract; and
- Liaison with community and authorities for project reporting and approvals.

24.3 Development Methodology

The EPCM contractor will provide a range of project management, engineering, drafting, international and in-country procurement, contract management and commissioning services necessary to provide a complete, safe, quality and technically compliant project. This will also include working with the client's representative team to oversee and undertake all procurement and contract management, construction management, and commissioning and handover requirements. The scope of services will be established as follows:

- Project management and controls;
- Detailed engineering and design;
- Equipment and services procurement and contract management;
- Materials fabrication and delivery;
- Transport and logistics;
- Engineering and drafting;
- Construction management;
- Commissioning management;
- Ore commissioning assistance of the processing plant facilities by the contractor, assisted by the owner's operations team; and
- Client training and handover.

Some of the project scope included in the estimates will be delivered outside the main contract, although the works may still be managed by the contractor. The works outside the main contract may include:

- Design and/or construction of the bulk earthworks including the roads, CTF and water ponds;
- Design and/or construction of the treated water disposal system;
- Establishment of key operations supply and services contracts (e.g. communications, information technology, power and propane supply);
- Purchase and delivery of spare parts;
- Purchase and delivery of first fills and other supplies for operational readiness;
- Procurement of mobile fleet for the mining and processing plant.

24.4 Contracting Plan

24.4.1 Engineering

Engineering design and management will be performed by a multidisciplinary team arranged in a matrix type structure under the guidance of the Project Manager. The Project Design Manager will direct the technical and resourcing aspects of this portion of the work in conjunction with the Project Manager. Functional managers will provide high level technical support as required. All engineering will be carried out by the contractor, typically with specialist support in the following areas:

- Vendor design and construct building packages;
- Hydrogeology;
- Hydrology;
- Geotechnical test work and recommendations;
- Communications;
- Vendor packages including grinding, flotation, filtration and water treatment.

To ensure quality and consistency in design and delivery of the project, engineering will be carried out in line with the contractor's standard procedures as amended to incorporate the requirements of SRA's policies and procedures.

24.4.2 Construction Packages

It is anticipated that the bulk of the site installation work for the processing plant will be performed directly by a preferred construction contractor. Specialist subcontractors may be engaged to perform portions of the works requiring specialist equipment and experience.

Areas where specialist subcontractors may typically be employed include:

- Transport/freight services;
- Bulk earthworks;
- Concrete batching;
- Concrete installation;
- Large process tankage erection;
- Mill liner installation;
- Water Treatment Plant design, supply, installation and commissioning;
- Laboratory design, supply, installation and commissioning;
- High voltage overhead power transmission lines;
- HV Cable terminations,
- Fibre optic cable terminations;
- Overland piping;

- HDPE lining of the CTF and Water Ponds;
- Transportable buildings and offices;
- Steel framed buildings.

The remainder of the works, including the services connection to the buildings, will be performed directly by the installation contractor. The installation contractor will also typically supply cranes and access equipment for the construction.

24.4.3 Concrete Supply

For the FS, it has been assumed that concrete supply will be from a batching plant established onsite.

24.4.4 Equipment and Material Supplies

Equipment and material packages will be competitively tendered except where SRA specifies a particular supplier or make of equipment.

Packages of equipment will be sent out to reputable suppliers, with the tender submissions evaluated and awarded on the basis of technical compliance, schedule requirements and price. Long lead items on critical path for the project will be prioritised.

Materials and fabricated item packages such as platework and structural steel will be sourced locally where possible, and where they are relatively cost competitive. Supply packages will be tailored to meet schedule requirements and fabricator capacities.

24.4.5 Construction Equipment

Major construction equipment will be sourced from local equipment rental providers. It is envisaged the following typical construction equipment will be required for varying durations:

- 250 t crawler or hydraulic crane (for mill installation);
- 150 t hydraulic crane;
- 50 t hydraulic crane;
- 15 t mobile cranes and or lift and carry trucks;
- Telescopic handlers;
- Elevated work platforms;
- Site buses;
- Welding machines;
- Construction vehicles;
- Scaffolding as required.

24.4.6 Cranes

In order to confirm the size of cranes needed for the construction, a lifting review will be undertaken during the next phase. The FS determined that the majority of areas of the plant construction will be able to be achieved with 'typical' cranes for a project of this scale, as lift loads and access to the lift site were considered normal. A 250 t crane has been included for the mill installation.

The preliminary layout of the plant provides adequate space for siting the crane during installation of the mill, and earthworks quantities were allowed for suitable crane pads at the critical crane locations.

24.5 Logistics

Equipment and fabricated items will be sourced from local, interstate and international suppliers. Typically greater than 80% percent of the total capital expenditure for equipment and fabricated items will be sourced within the United States.

24.5.1 International Supply

Internationally sourced equipment has been priced with freight to Seattle and customs clearance all included in the scope of supply. All sea freight will be received through the port and clear customs before being loaded onto trucks for transport to site, or moved to a consolidation yard prior to delivery to site. Payment of any import duties and taxes will be outside the contract and will be managed by SRA.

24.5.2 Interstate Supply

Fabricated items and equipment sourced from interstate will either be transported by road directly to site, or transported to a consolidation yard prior to delivery to site.

A single logistics provider will be contracted for each major interstate area where supply of goods is expected. Smaller one-off interstate logistics may be provided by regular couriers and transport providers.

24.5.3 Local Supply

A local freight carrier will be used to transport materials and equipment to site from locations within the state. Where applicable, loads will be consolidated in the freight carrier's yard prior to transport to site.

The local freight carrier will also be used for the transport of internationally supplied goods from the port to site.

The large items such as the mill shells, thickeners and tanks will be of split construction to facilitate ease of transport to the site and installation.

24.5.4 Site

All equipment and materials will be stored on site in dedicated laydown areas with a materials management system used to receive, store and locate equipment when needed.

24.6 Accommodation

Accommodation will be provided using existing local facilities.

24.7 Construction Temporary Facilities and Services

Prior to commencing work on site, transportable type construction offices will be mobilized and established. Building services will be supplied as follows:

- Power supply will be from temporary diesel generating sets;
- Potable water will be delivered to a dedicated storage tank and will be reticulated to the temporary facilities by the contractor;
- Waste water will be collected in septic tanks, which will be emptied by an appointed contractor to an offsite local disposal area on a regular basis.

The contractor will establish the site-based communication system as part of early works.

Upon the completion of site works, the construction offices will be demobilized and the site made good and rehabilitated.

The temporary facilities will be established close to the processing plant area adjacent to the laydown area. The temporary facilities will consist of:

- Transportable site offices;
- Transportable crib rooms;
- Transportable ablutions buildings.

24.8 Health Safety and Environment

Health, safety and environmental considerations are of paramount importance in the development of the implementation strategy. All work shall be designed and performed in accordance with relevant government, environmental and health and safety regulations. The contractor will commit to a 'no harm, no incident' culture utilizing and employing their own safety management system back-to-back with SRA's safety systems and policies.

A construction safety management plan has been developed as part of the Phase 1 Development Project execution. The safety management plan identifies the Project requirements and the management of safety for the Project. This SRA plan will be periodically updated. Specifically items for the Phase 2 Development construction such as classified plant, inductions, rigging equipment, scaffolding, verifications of competency, licences and such will be logged and registered. The Construction Manager will be responsible for maintaining the registers via the site-based safety officers.

Work will be planned to ensure that it is performed in a systematic and controlled manner, minimizing the risk of damage to the environment. The key environmental risk for construction involves spillage of construction fluids, particularly hydrocarbons, from storage or during construction. Proven procedures will be utilized to prevent such an occurrence. SRA currently has a Spill Prevention, Control, and Countermeasures Plan in place for the Phase 1 Development construction activities that will be periodically updated when necessary.

Safety audits will be conducted throughout the duration of the work by the contractor's safety personnel.

24.9 Project Controls, Planning and Scheduling

A project controls procedure will be implemented with two primary objectives:

- To monitor progress against expectations and provide guidance for when corrective action is required;
- To control and manage change to the core objectives.

Upon commencement of the Project, the contractor will generate a detailed design and construction schedule for the works. The project scope, contract value and schedule shall form the baseline against which progress is measured. Progress will be monitored and earned value management principles used to generate an S-curve based upon calculated actual cost, planned value and earned value.

Progress will be measured against the key deliverables, with the method of measurement being dependent upon the nature of the deliverable. The engineering portion of the work will be against the generation of design documentation including development models and the issuing of drawings and design documents 'Approved for Construction'. The procurement portion will be measured against key milestones for each package such as issue of tenders, package award and receipt of goods. The construction portion will be measured against physical quantities of material installed, and subjective estimates of installation progress and construction labor hours.

Changes to the Project baseline will only occur as variations to the original contract under the contractor's variation management procedure. Regardless of which party instigates the proposed change, the effects to scope, cost and schedule will be fully quantified with the variation agreed by both parties before proceeding.

A construction risk assessment workshop will be held prior to commencement of site works to identify and mitigate potential issues.

24.9.1 Organizational Structure

The Project's organizational structure provides the framework for managing the execution of the various work packages. The organizational structure is largely defined by the contracting model employed, project phase, project size and nature of the work involved. Furthermore, the organizational structure is customized from generic models to meet the unique challenges and requirements of the work package. To ensure the Project is successfully delivered, competent personnel with relevant skills and experience will be identified and employed.

A Project Implementation Plan will be developed as part of project execution and will provide the proposed contractor's organizational structure that will manage each of the project development phases.

24.9.2 Engineering Design and Management

Engineering design and management will be performed by a multidisciplinary team arranged in a matrix type structure under the guidance of the Project Manager. The Project Design Manager will direct the technical and resourcing aspects of this portion of the work in conjunction with the Project Manager. Functional managers will provide high level technical support as required.

24.9.3 Site Construction

The EPCM Project Manager will assume responsibility for overseeing the construction phase of the work. The site support services for the construction team will be provided by installation contractor's safety and administration personnel. The installation construction manager will be supported by a team of discipline specific supervisors, each leading crews of construction personnel. The Owner's team will monitor the performance and independent quality audits of the works.

The installation contractor's site supervision, which will expand and shrink to suit the construction activities, will typically comprise the following:

- Construction Manager;
- Project Engineers;
- Discipline based Supervisors;
- Safety Officer;
- Materials Controller;
- Site Clerk.

24.9.4 *Plant Commissioning and Production Ramp Up*

The act of commissioning involves bringing multiple pieces of plant online in the correct sequence with all safety systems operational to ensure sequences, controls and functionality are correct. A separate commissioning team led by the EPCM contractor's Commissioning Manager will be established to perform this work.

The contractor's Commissioning Manager will manage the pre-commissioning and the dry and wet aspects of the commissioning phase prior to practical completion of the Project being achieved. The contractor's commissioning team will then undertake ore commissioning of the facilities, assisted by SRA's Commissioning Manager and operations team.

The commissioning organizational structure consists of the Commissioning Manager operating under the guidance of the contractor's Project Manager and in coordination with the Construction Managers. Multidisciplinary commissioning teams will be established, consisting of process and discipline engineering personnel used in the design phase of the Project. The commissioning team will also utilize vendor commissioning representatives to ensure vendor equipment is effectively commissioned and warranty terms and conditions are adhered to.

24.9.5 *Pre and Dry Commissioning*

Pre-commissioning involves ensuring all equipment and services are fully installed and mechanically and electrically complete, ready to be run and tested. This includes performing all alignment checks, pressure tests, wiring tests, ensuring that emergency stops and control limits are set, and that the control system functions as intended. It also requires ensuring that all drawings, manuals, functional descriptions and other pertinent data required for equipment no load commissioning are available (and certified). Finally, 'No Load' testing is performed to prove the integrity of the systems.

24.9.6 *Wet Commissioning and Practical Completion*

Wet commissioning will be required to demonstrate the integrity of the circuits at 'steady-state' prior to the introduction of ore. Wet commissioning involves the operation, where possible, of all process circuits with water and with all interlocks and control systems in normal operating mode.

Once pre, dry and wet commissioning are complete, and all required documentation has been provided by the engineering contractor to SRA, the Project will be considered to have reached practical completion.

Commissioning and practical completion will be staged across various plant areas to facilitate ongoing construction activities.

24.9.7 Handover and Ore Commissioning

Following practical completion, sequenced ore commissioning of the facilities will commence under the control of the engineering contractor with assistance from SRA's operations team. The facilities will be handed over to SRA's operations team following successful ore commissioning.

Once the process is stabilized, throughput will be progressively increased until nameplate capacity is achieved and product quality requirements are met.

SRA personnel will be involved in the commissioning process to gain experience on the operation of the plant.

24.9.8 Owner's Management and Interfaces

SRA will establish a Project owner's team to monitor all aspects of project development and implementation.

The SRA Project team will work with the contractor's project management team to ensure the Black Butte Copper Project is completed on time to the agreed scope and project requirements. They will meet regularly to discuss progress and key issues.

During the early works phase, it will be recommended that SRA have a representative located in the contractor's office to monitor progress and facilitate the fast and efficient resolution of queries and required client approvals.

For detailed design and site works the contractor will autonomously ensure that work progresses in a timely and safe manner to required standards as detailed by the project scope.

With regards to safety, the contractor's safety personnel will monitor compliance with safety procedures and requirements.

24.10 Project Schedule

The preliminary implementation schedule has been based on the execution methodology and design presented in the FS Report. The schedule is presented in Appendix F and outlines the delivery of the aspects of the Project covered in the scope of work.

24.10.1 Long Lead Time Items

The following long lead items (Table 24.1), were identified from vendor quotations during the progression of the FS.

Item	Potential Manufacture Location as tendered	Delivery Period (Weeks)	Logistics duration (Weeks)
SAG and Ball Mill	Various major components ex Europe	45-55	6
Flotation Cells	Drives ex Finland Fabricated ex Mexico	38 32	6 2
Regrind Mill	Germany/Canada	36	6
Paste Pumps	Netherlands	32	6
Tailings and Concentrate Thickeners	Mexico	32	3
Apron Feeders	USA	30	4
Vibrating Screens	USA	28-30	4
Paste Filters	Europe	27	6
Water Treatment Plant	Europe	24	6
Concentrate Filters	Italy	24	6
Slurry Pumps	Australia	24	5
Emergency Genset	China	24	4
Jaw Crusher	Finland	22	6-8
Cyclone Clusters	USA	19	1
Pebble Crusher	France	12	8

Table 24.1 Long Lead Time Items

24.10.2 Assumptions and Basis

The preliminary schedule has been based on the following assumptions:

- The schedule commences at Contract Award for the detailed design and construction phase;
- Construction personnel will work on a three week on / one week off roster;
- Work on site will be carried out on a notional 11 hour day, six days per week basis;
- Processing operations will commence approximately 20 months after contract award, with ramp-up to 80% of full production over a period of approximately four months, 100% over eight months;
- Site power distribution will be required to be completed two months prior to ore commissioning, to allow a suitable period for pre-commissioning and wet commissioning.

24.10.3 Schedule Details

The critical path for the Project implementation is the specification, procurement, installation and commissioning of the SAG and Ball mills. The order for commencement of supply is scheduled to be placed within the first four weeks of the Project after award. The duration of this critical path, from contract award to completion of the installation, is approximately 85 weeks.

Mobilization to site is scheduled backwards from the mill delivery date to allow time for mobilization, site preparation, bulk earthworks, civil installation and concrete curing prior to commencement of installation of the mills and other equipment.

The overall schedule for the Project is 92 weeks (contract award to demobilization), with 62 weeks on site.

The preliminary implementation schedule is provided as a summary and critical path in Appendix D.

Milestone	Commence Month Number
Project Award	0
Commencement of long lead procurement	0
Award SAG and Ball Mill Supply	4
Mobilization of road and earthworks contractor	25
Mobilization of concrete contractor	32
Completion of detailed engineering	34
Commencement of SMP works	47
Completion of concrete works	53
Mobilization of E and I contractor	53
SAG and Ball Mill delivery to site	65
Commencement of dry commissioning	81
Completion of SMP works	85
Completion of E and I works	86
Commencement of Ore commissioning	88
Ore Commissioning Complete	90
Operator Training and Handover Complete	92
Demobilization	92

Table 24.2 Key Milestones for Project Implementation

Potential Sources of Delay

The following items have been identified as potential sources of schedule delays:

- Late delivery of vendor data;
- Late vendor deliveries;
- Shipping losses;
- Inclement weather;
- Accommodation availability to sufficiently resource the Project.

24.10.4 Personnel Levels

Head Office

Head office will consist of the following personnel:

- Project management and engineering;
- Procurement and logistics;
- Accounting and cost control;
- Engineering management;
- Discipline engineering (process, civil, structural, mechanical, and electrical);
- Drafting;
- Project support.

Site

Preliminary maximum personnel levels for construction activities at site have been calculated from the installation hours and the schedule. Personnel numbers are based on a work roster of three weeks on / one week off, with work on site carried out for 11 hours per day, six days per week.

Construction manning the processing plant and mine infrastructure will peak at approximately 185 personnel including all construction management, supervision and equipment. Personnel levels for the owner's team, mining contract and bulk earthworks construction (roads, CTF and ponds) have not been included in these construction numbers.

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25. INTERPRETATION AND CONCLUSIONS

25.1 Land and Mineral Leases

Based upon reliance of other experts, all land and mineral leases are determined to be in good order that are included within the Mineral Resource summary. One land parcel, the McGuire parcel, represents approximately five acres in the northern portion of the Johnny Lee deposit and has been excluded from Mineral Resource calculations as SRA does not have an agreement or current rights for production associated with this parcel. Though this land parcel represents a minimal percentage of the Johnny Lee deposit, it is recommended by the authors that SRA seek out agreement or ownership to allow the McGuire parcel to be included in future Mineral Resource calculations.

25.2 Geology and Mineral Resource

Mineral Resources with an effective date of October 15, 2019 at the Johnny Lee deposit consists of two zones: the UCZ and the LCZ. Combined Measured and Indicated Mineral Resources at the UCZ and LCZ total: 10.9 Mt averaging 2.9% Cu totalling 310.9 kt of total Cu metal. There is an additional Inferred Resource of 2.7 Mt averaging 3.0% Cu totalling 79.7 kt of Cu metal. Measured and Indicated Mineral Resources are inclusive of those Mineral Resources modified to produce Mineral Reserves.

The status of exploration activities in the Johnny Lee deposit area have been performed to good industry standards. As the Project has already identified sufficient Mineral Resources, further exploration in the UCZ and LCZ areas are not recommended. The author does suggest testing the Johnny Lee deposit area for the potential of deeper mineralization, as a similar geological environment exists at depth as what currently hosts economic copper mineralization at the Johnny Lee deposit.

Local geology of the region is considered to be prospective for additional Cu and other base metal prospects as evidenced by the known prospects described in section 23. Further afield, the region is prospective for other SEDEX or VMS-like deposits with original targeting focused on Pb, Zn, and Ag mineralization.

Mineral Resources with an effective date of October 15, 2020 at the Lowry deposit consist of two zones: the LMCZ and the LLCZ. Combined Inferred Mineral Resources at the LMCZ and the LLCZ total: 8.3 Mt averaging 2.41% Cu totalling 440.0 kt of total Cu metal.

25.3 Mining and Mineral Reserve

It is the opinion of the QP responsible for the mine design, mine planning, and reserve calculations, Brad Evans, that the level of information and work regarding these issues and estimates are appropriate for a FS and represent good industry practice that align with NI 43-101 reporting. The historic geotechnical database as well as new rock mass testing was reviewed to classify the rock mass and recommend suitable ground support. This data was used to select the mining methods for this study, considering both mechanized CF and longhole. A NSR approach based on Cu grade, recovery, as content and playabilities was used for mine planning and Mineral Reserve estimates. The value of each stope (considering metal price, recovery, smelter charges and transportation) was compared to economic cut-off values, as well as development costs, and was used to determine the Probable Mineral Reserve. A monthly mine production schedule was developed for both development and mine production. The mine designs and mining plans are based upon engineering studies and design appropriate for a FS.

Proven Mineral Reserves are 2.0 Mt averaging 3.0% Cu and Probable Mineral Reserves are 6.8 Mt averaging 2.4% Cu with a total Mineral Reserve of 8.8 Mt averaging 2.6% Cu. Total Contained Cu is 226,100 t. Mining production occurs over an eight year period with a peak production rate of 1.2 Mt per annum.

25.4 Metallurgy and Mineral Process Design

It is the opinion of the QP responsible for the metallurgical test reports and development of the process flowsheet, Deepak Malhotra, that the process design criteria developed by GR Engineering Services was based on the metallurgical test work following standard industry principles. It is the opinion of the same QP that a marketable-grade concentrate may be produced while maximizing Cu recovery from highly variable Johnny Lee massive sulphide deposits. The design criteria was provided to the equipment vendors to size and cost equipment.

25.5 Project Infrastructure

The Project includes the following major infrastructure to support the mine and processing operations: access road, onsite access and service roads, buildings and structures, fresh water supply from wells and mine dewatering operations, Cu enriched rock stockpile, various water management facilities, a CTF, waste rock storage, paste plant, rolling stock and grid power from an incoming high voltage supply line from the SE side of the Project.

Main access to the facility is well established from the east side of the Project off U.S. Highway 89 using Sheep Creek Road which is a maintained county gravel road. Main access will be via a new mine access road off of Sheep Creek Road and the main security gate near the process plant.

25.6 Risks

A risk matrix was developed for the project which identified low, medium and high-risk parameters in the various disciplines (i.e., geology, mining, metallurgy, environmental, etc.) that could have a negative impact on the project economics. Generally, low and medium risks are easy to mitigate whereas high risk items need to be further investigated.

Selected medium risk and identified high risk items are given in Table 25.1. Corrective action plan to mitigate the risks are given in the comments section. However, the company has no control over the litigation suits filed in the courts by several groups that oppose resource development in Montana.

AREA Subarea	Risk	Comments
RESOURCE		
Drilling	Medium	Additional drilling needed in UCZ for metallurgy.
MINING		
Geotechnical	High	Fair-Poor Ground. Good grade control procedures during UG mining required for successful execution (short rounds).
Dewatering	High	Unknown if VVFZ and Fault 1 Faults are water conductors or not. Process plant and mining operations to rely on water recycle plant for UG service water.
METALLURGY/PROCESS		
Geo-metallurgy	High	Blending of feed and even concentrate will be key to plant performance for saleable concentrates.
INFRASTRUCTURE		
Water Supply	High	Water availability from underground as new raw water for makeup will be critical.
Water Management	High	System requires large number of ponds and a complex water balance/treatment/discharge system that will require good monitoring and operational management.
SOCIAL		
Litigator Groups	High	Legal challenge to MDEQ's ROD to prevent/impair project. Such litigation can increase costs and cost delays.
MARKET RISK		
Concentrate Sale Contract	Medium	Need to negotiate sales contract for the concentrate which may have undesirable impurities.
OPERATIONAL RISK		
Recovery underperformance	High	Will affect revenue. Blending of problem ore zones with good ore zones and LCZ mineralization will be critical.

Table 25.1 Selected Medium Risk and High-Risk Parameters and Proposed Mitigation Action

25.7 Opportunities

While preparing the Feasibility Study, there were several areas identified which could potentially enhance the economics of the project. Significant opportunities are as follows:

- Additional exploration and infill drilling and subsequent resource modelling updates could both expand the Mineral Resources and convert Inferred Resources to Measured and/or Indicated Resources at the Lowry deposit. This updated Mineral Resource work could likely increase the life of the project;
- The potential for additional prospects exists in the district proximal to the planned Black Butte Copper Project mine facilities;
- Additional geo-metallurgical test work targeting UCZ and blended UCZ and LCZ ores to improve copper recovery;
- Reduction of paste cement and slag mixture percent will reduce the operating cost;
- Potential of recovering cobalt as a by-product from cleaner flotation tailings;
- Simplification of the logistic cycle for concentrate delivery to port can reduce operating cost;
- Evaluate the use of solar power for partial needs of the facility.

25.8 Conclusions

The FS is technically and economically sound under the present conditions. There are several areas of high risk identified that could negatively impact the project. However, several opportunities have also been identified that could enhance the project economics. It is the opinion of the QP, Deepak Malhotra, that this Project proceed to detailed design and construction using the considerations and recommendations as outlined in the FS.

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26. RECOMMENDATIONS

26.1 Geology and Mineral Resources

26.1.1 Johnny Lee Deposit

Property Ownership

The QP of Mineral Resources recommends that SRA seek to finalize an agreement or ownership over the McGuire parcel in order to secure a continuous mining area and obtain rights to mine all mineralization located as part of the Johnny Lee deposit.

Geology and Mineralization

The QP of Mineral Resources recommends that additional drilling and analytical testing at closer spacing within both the UCZ and LCZ. This will provide improved confidence in grade and geological continuity toward increasing the confidence of Mineral Resources. This may be achieved effectively through UG drilling and development mapping.

Deeper exploration is recommended in the Johnny Lee deposit area as the prospective Cu-hosting zones appear to be present at depth. Additionally, the QP would recommend additional drilling in the Sawmill Hill Prospect area as this may result in additional Cu mineralization between the Johnny Lee and Lowry deposits that would be advantageous for continuous project UG development activities.

Mineral Resources

Upon future acquisition of additional drilling or other data that may provide improved modelling or estimation, it is recommended that an updated geological model and resource estimation occur. Until that time, the current Mineral Resource estimate is considered satisfactory and appropriate for reporting of Mineral Resources at the Johnny Lee deposit.

26.1.2 Lowry Deposit

Geology and Mineralization

Standardization of historic logging data, resolution of intervals, handling of fracture/fault zones, and treatment of pyrite is recommended. The QP of Mineral Resources recommends SRA geologists to undergo additional re-logging of Lowry deposit historical core prior to additional evaluation drilling.

A comprehensive lithostratigraphic 3-D model including structure will provide benefits to mineralized wireframe modelling and understanding of the Lowry deposit.

Drilling

The QP of Mineral Resources recommends the following to help increase confidence in the current Mineral Resources at the Lowry deposit::

- additional infill drilling in the LMCZ and LLCZ to improve the spatial distribution of data by domain;
- detailed lithostratigraphic and structural modelling in 3D; and
- additional recovery and metallurgical studies to refine the existing assumptions on Cu recovery.

In addition to infill for Mineral Resources, it is recommended that future drilling campaigns should aim to target the near-vertical mafic dykes observed in drilling to gain additional information regarding the orientations, widths, and potential for grade dilution of these dykes. It is recommended that a combination of deviated and wedge drilling could provide near-horizontal drilling inclinations to intercept the mafic dykes and near-parallel to expected attitude.

Mineral Resources

Upon further acquisition of drilling data, improved geological modelling and structure, it is recommended that variography and estimation be updated. In general, the spatial continuity determined through variography at the Lowry deposit was limited by data resulting in simplified omni-directional variogram models which are expected to be improved and refined with more data per domain.

26.1.3 Lowry Resource

Additional infill drilling, metallurgy using the established FS process flow sheet, environmental studies including hydrogeological work, and permitting is recommended in order to help convert the Lowry Inferred Resource to Indicated and Measured Classification.

26.1.4 Additional District-Wide Exploration

Several of the identified prospects within the district under current control by SRA have excellent opportunities for testing other sulfide zones within the stratigraphic section. Previous historic exploration drillholes have penetrated significant thicknesses of pyrite-rich zones that indicate a strong and extensive hydrothermal system with additional opportunity for finding base metal enrichment. Future exploration at these prospects could result in additional new base metal discoveries that would be able to utilize most of the planned Black Butte Copper plant facilities and extend the LOM if the economics allow.

26.2 Mining and Mineral Reserves

26.2.1 Geotechnical

Mining Plus recommends further geotechnical investigation to increase the deposit knowledge of the rock mass. This would include shear strength testing of discontinuity sets to determine expected stability of potential discontinuity wedges. In addition, further triaxial testing of all major rock units should be performed to gather a more thorough understanding of the strength properties of the rock. Geotechnical boreholes are still required to be drilled for RAR-1 and FAR-1 and analysis of the resulting core is required to determine site specific properties and required ground support and excavation methods.

In combination with the additional data gathered and the FS mine design, detailed numerical modelling should be performed to determine the expected reactions stress changes and associated reactions of the ground. The results from this modelling and the data gathering should be incorporated into an updated ground support design criterion. Prior to commencing development, a ground support plan to incorporate an observational approach with triggers to adjust support as ground conditions change.

26.2.2 Mining Methods

Alternate extraction methods of the decline should be investigated including the potential use of a continuous miner to potentially increase the decline development rates.

SRA should engage in early discussions with mining contractors to determine the preferred contract style as well as gain more certainty around a large component of the capital cost. These discussions with the contractors considered for the vertical development should also consider alternative sinking methodology for the ventilation raises.

Mining Plus recommends that an investigation be carried out on the viability of using a longhole mining method for the UCZ in addition to the DF. This could potentially lower costs, increase reserve tonnage, decrease the head grade, as well as increase mining rates from the UCZ. Considerations would also include geotechnical stability and effect to project economics.

26.3 Metallurgy and Mineral Process Design

26.3.1 Continuous Improvement

The study has assigned 80.8% Cu recovery to the UCZ ore. SRA management should continue test work to optimize the non-supergene altered UCZ with the objective of improving both Cu/Ag recoveries and concentrate grade/quality (minimize As content).

26.3.2 Cobalt and Silver Enhancement

Co is one of the metals on the US critical material list. The Johnny Lee deposit contains sufficient amounts of Co that can possibly be recovered as a by-product. It is associated with pyrite. The Cu cleaner tailing in the proposed circuit is predominantly pyrite. It is recommended that test work be undertaken on cleaner flotation tailing and combined rougher and cleaner tailing to produce Co rich pyrite tailing. The concentrate will have to be leached to recover Co (hydrometallurgical process).

The Ag that did not float with Cu may be also associated with pyrite. The analysis of Co rich pyrite concentrate will indicate if Ag was also recovered.

26.4 Project Infrastructure

Subject to the Final Mine Operating Permit and project funding constraints, it is recommended that SRA commence basic engineering of the Project in line with the MOP plan, including additional studies and site investigations listed below.

It is recommended that the following be considered in upcoming project phases:

26.4.1 Geotechnical

Build on the current geotechnical database of site investigations to support detailed design for the foundation requirements and to better define the construction material quantities that are available. This work could include:

- Additional geotechnical drilling and test pitting to better delineate overburden, weather bedrock and competent bedrock.
- Additional rock and soil laboratory testing to establish suitability of available construction materials.

26.4.2 Water Management

The subsequent stages of the Project should include the following tasks related to the water management:

- Prepare a storm water pollution prevention plan for the Integrated MPDES permit.

26.4.3 Power Supply

As the Project advances into detailed design the following actions should be considered to secure the supply of power to the site:

- Finalize negotiations with Fergus Electric Cooperative to prepare a binding Letter of Intent.
- Secure easement rights within the identified transmission line right-of-way corridor.

26.4.4 Tailings Management and Waste Rock Storage Facilities

The subsequent stages of the Project should include the following tasks related to the CTF and WRS pad:

- Prepare a detailed tailings/waste rock deposition plan to include contingencies such as seasonal changes, temperature variations, tailings rheology and associated operations costs.

26.5 Environmental Studies and Community Engagement

The success of the Black Butte Copper Project will depend upon meeting all of the State and Federal commitments and obligations as defined in the MOP throughout construction, operations, and mine closure. SRA management needs to set up rigid processes as preliminarily outlined in the MOP to monitor and report the groundwater and surface water quality and aquatic life in the adjacent streams and creeks throughout the Project. It is also recommended for SRA to continue it's strong community engagement and information transparency as mine construction and permitting activities continue.

26.6 Project Implementation

The Project implementation schedule will need to be coordinated to suit the local Montana climatic conditions to best manage the productivity of the various activities to the seasons.

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28. LIST OF ABBREVIATIONS

The following abbreviations may be used in this report.

Abbreviation	Unit or Term
A	Ampere
AA	Atomic Absorption
ABA	Acid Base Accounting
Ag	silver
Ai	Abrasion index
A/m^2	Amperes per square metre
AMA	Agency Modified Alternative
ANFO	Ammonium Nitrate Fuel Oil
ANSI	American National Standards Institute
ARM	Administrative Rules of Montana
ARTCC	Air Route Traffic Control Center
As	arsenic
AST	Aboveground Storage Tanks
ASTM	American Society for Testing and Materials
ATU	Atoll-Textured Units
Au	gold
AuEq	gold Equivalent grade
BACI	Before, After, Control and Impact
BBWi	Bond Ball mill Work index
BCWP	Brine Contact Water Pond
BFP	Belt Filter Press
BHP	Broken Hill Propriety Limited
Bi	bismuth
BML	Base Metallurgical Labs
BMP	Best Management Practices
BNSF	Burlington Northern Santa Fe (railroad)
BRWi	Bond Rod mill Work index
°C	degrees Centigrade (Celsius)
CAGR	Compound Annual Growth Rate
CAI	Cominco American Inc.
($Ca(OH)_2$)	hydrated lime
CCD	Counter-Current Decantation
CF	Cut and Fill (mining)
cfm	cubic feet per minute
cfs	cubic feet per second
CIL	Carbon-In-Leach
CIM	Canadian Institute of Mining, Metallurgy and Petroleum

Abbreviation	Unit or Term
CLA	Composite Length Analysis
cm	centimetre
cm^2	square centimetre
cm^3	cubic centimetre
CMC	Carboxymethyl Cellulose
Co	cobalt
COB	Coarse Ore Bin
CoG	Cut-off Grade
ConfC	Confidence Code
COV	Cut Off Value
Cp	chalcopyrite
CRec	Core Recovery
CRM	Certified Reference Material
CSS	Closed Side Setting
CTF	Cemented Tailings Facility
CTW	Calculated True Width
Cu	copper
CV	Coefficient of Variation
CWi	Crush Work index
CWP	Contact Water Pond
°	degree (degrees)
D60	60% of the particles are finer and 40% of the particles are coarser than D60
dBA	A-weighted decibel
DBA	Database Administrators
DF	Drift and Fill (mining)
dia	diameter
dmt	dry metric tonnes
EIS	Environmental Impact Statement
ekW	electrical kilowatts
EMP	Environmental Management Plan
EMS	Environmental Management Systems
EPA	Environmental Protection Agency
EPCM	Engineer, Procure and Construction Management
FA	Fire Assay
FAR	Fresh Air Recirculation
Fe	iron
FEC	Fergus Electric Cooperative
FEL	Front End Loader
FEMA	Federal Emergency Management Agency
Fm	Formation

Abbreviation	Unit or Term
FS	Feasibility Study
ft	foot (feet)
ft^2	square foot (feet)
ft^3	cubic foot (feet)
FW	Footwall
FWP	Fish, Wildlife and Parks
g	gram
Ga	Giga annum
gal	gallon
g/cm^3	grams per cubic centimetre
g/L	grams per Litre
g-mol	gram-mole
gpd	gallons per day
gpm	gallons per minute
g/t	grams per tonne
GVW	Gross Vehicle Weight
ha	hectare
HCT	Humidity Cell Test
HDPE	Height Density Polyethylene
HMI	Human-Machine Interface
hp	horsepower
HRMIP	Hard Rock Mining Impact Plan
HSU	Hydro-Stratigraphic Units
HTW	Horizontal True Width
HVAC	Heating, Ventilation, and Air Conditioning
HW	HangingWall
ICOLD	International Commission On Large Dams
ICP	Induced Couple Plasma
ICP-AES	Inductively Coupled Plasma – Atomic Emission Spectrometer
ICP-MS	Inductively Coupled Plasma – Mass Spectrometer
IDW2	Inverse Distance Weighted Squared
IDW3	Inverse Distance Weighted Cubed
IFC	International Finance Corporation
ILS	Intermediate Leach Solution
JD	Jurisdictional Determination
JV	Joint Venture
K_{80}	comminution product size (from grinding) through which 80% of particles will pass
kA	kiloAmperes
kg	kilogram
kg/m^2h	kilograms per square metres hour

Abbreviation	Unit or Term
kg/m^3	Kilograms per cubic metre
km	kilometre
km^2	square kilometre
KNA	Kriging Neighborhood Analysis
koz	thousand troy ounce
kPa	kiloPascal
kt	thousand tonnes
kt/d	thousand tonnes per day
ktpa	thousand tonnes per annum
kt/y	thousand tonnes per year
kV	kiloVolt
kW or kw	kiloWatt or kilowatt
kWh	KiloWatt-hour
kWh/t	kilowatt-hour per metric tonne
L	Litre
lb	pound
LCT	Locked Cycle flotation Test
LCZ	Lower Copper Zone
LHD	Long-Haul Dump truck
Lib	Liberated
LLCZ	Lowry Lower Copper Zone
LLSZ	Lowry Lower Sulphide Zone
LMCZ	Lowry Middle Copper Zone
LMSZ	Lowry Middle Sulphide Zone
LOI	Loss On Ignition or Letter Of Intent
LOM	Life-of-Mine
Lpd	Litres per day
Lpm	Litres per minute
L/sec	Litres per second
L/sec/m	Litres per second per metre
LSZ	Lower Sulphide Zone
LUCZ	Lowry Upper Copper Zone
LVA	Locally Varying Anisotropy
LZFW	Lower Zone Footwall
m	metre
M	Million
m^2	square metre
m^3	cubic metre
m^3/d	Cubic metre per day
m^3/h	cubic metre per hour
m^3/s	cubic metre per second

Abbreviation	Unit or Term
m/sec	metres per second
Ma	Mega annum
MARN	Ministry of the Environment and Natural Resources
masl	metres above sea level
MCA	Montana Code Annotated
MCCs	Motor Control Centres
MCE	Maximum Credible Earthquake (event)
MDA	Mine Development Associates
MDEQ	Montana Department of Environmental Quality
MDNRC	Montana Department of Natural Resources & Conservation
MDOT	Montana Department of Transportation
MEPA	Montana Environmental Policy Act
MFWP	Montana Fish, Wildlife and Parks
mg/L	milligrams/Litre
µm	micron or microns
MIBC	methyl isobutyl carbinol
mL	millilitre
mm	millimetre
mm^2	square millimetre
mm^3	cubic millimetre
MME	Mine and Mill Engineering
MMI	MultiMetric Indices
MMRA	Metal Mines and Reclamation Act
MODA	McArthur Ore Deposit Assessments Pty Ltd
MOP	Mine Operating Permit (Application)
Moz	Million troy ounces
Mpa	Megapascal
MPDES	Montana Pollution Discharge Elimination System
MRL	Montana Rail Link
MSHA	Mine Safety and Health Administration
MSP	mono sodium phosphate
MSZ	Middle Sulphide Zone
Mt	Million tonnes
MT	Montana
Mtpa	Million tonnes per annum
MTW	Measured True Width
MW	Million Watts
m.y.	million years
NaCN	sodium cyanide
NAD	North American Datum
NAG	Net Acid Generation

Abbreviation	Unit or Term
NCWR	Non-Contact Water Reservoir
NERC	North American Electrical Reliability Corporation
NGI	Norwegian Geotechnical Institute
NGO	Non-Governmental Organization
Ni	nickel
NI 43-101	Canadian National Instrument 43-101
NN	Nearest Neighbor
NPDES	National Pollutant Discharge Elimination System
NRHP	National Register of Historic Places
NSR	Net Smelter Return
NWE	NorthWestern Energy
OA	OverAll
O/E	Observed/Expected
OK	Ordinary Kriging
OSA	On-Stream Analyzer
OSC	Ontario Securities Commission
oz	troy ounce
% (pct)	percent
P_{80}	Screen size through which 80% of the particles will pass
Pa	Pascal
PAG	Potentially Acid Generation
Pb	lead
PCS	Process Control System
PEA	Preliminary Economic Assessment
PFS	Preliminary Feasibility Study
PLC	Programmable Logic Controller
PLS	Pregnant Leach Solution
PMA	Particle Mineral Analysis
PMF	Probable Maximum Flood
PMP	Probable Maximum Precipitation
POI	Point Of Interconnection
ppb	parts per billion
ppm	parts per million
PV	Partial Validated
PWP	Process Water Pond
PWS	Public Water Supply
Py	pyrite
Q'	Rock Quality Index
QA/QC	Quality Assurance/Quality Control
QEMSCAN	Quantitative Evaluation of Minerals by Scanning electron microscopy

Abbreviation	Unit or Term
QP	Qualified Person
RAR	Return Air Recirculation
RBF	Radial Base Function
RC	Rotary Circulation drilling
RL	Reduced Level (survey term)
RMR	Rock Mass Rating
RO	Reverse Osmosis
ROD	Record Of Decision
RoM or ROM	Run-of-Mine
ROW	Right Of Way
RQD	Rock Quality Description
RTK	Real-Time Kinetic
RTP	Reduction To Pole
S	sulphur
SAG	Semi-Autogenous Grinding (mill)
SCADA	Supervisory Control And Data Acquisitioning
SCH	Schedule (pipe size dimension)
SCM	Semi Cementitious Material
sec	second
SEC	U.S. Securities & Exchange Commission
SEDEX	Sedimentary Exhalative
SEM	Scanning Electron Microscope
Sg	cuprian siegenite
SG	Specific Gravity
SHPO	State Historic Preservation Office
SIL	Significant Impact Level
SIPX	Sodium Isopropyl Xanthate
SMC	SAG Mill Comminution testing
SMP	sodium monophosphate
SOC	Species Of Concern
SPCC	Spill Prevention Control and Countermeasures Plan
SPT	Standard Penetration Testing
SRA	Sandfire Resources America Inc.
SRCE	Standardized Reclamation Cost Estimator
SSC	Sediment-hosted Stratabound Copper
st	short ton (2,000 pounds)
s.u.	sin unidad (without unit)
SWPPP	Storm Water Pollution Prevention Plan
t	tonne (metric ton) (2,204.6 pounds)
T&E	Threatened and Endangered
t/d	tonnes per day

Abbreviation	Unit or Term
Tgd	Tertiary granodiorite
t/h or tph	tonnes per hour
t/m^{2}h	tonnes per square metres hour
Tn	tennantite
TOMS	Tailings Operations, Monitoring and Surveillance
tpa	tonnes per annum
TSF	Tailings Storage Facility
TSP	Total Suspended Particulates
TWSP	Treated Water Storage Pond
t/y	tonnes per year
UCS	Unconfined/Uniaxial Compressive Strength
UCZ	Upper Copper Zone
UG	Underground
UIG	Underground Infiltration Gallery
UII	Utah International Inc.
USACE	United States Army Corp of Engineers
USFS	United States Forest Service
USZ	Upper Sulphide Zone
UTM	Universal Transverse Mercator
UV	Ultraviolet
V	Volts/Validated
VFD	Variable Frequency Drive
VVF	Volcano Valley Fault
VVFZ	Volcano Valley Fault Zone
W	Watt
wmt	wet metric tonnes
WRS	Waste Rock Storage
WTP	Water Treatment Plant
w/v	measure of weight per 100 mL (weight by volume)
XRD	X-Ray Diffraction
y	year
Yc	Chamberlain Shale
Yne	Neihart Quartzite
Ynl	Newland Formation
Ynl L	Lower Newland Unit
Zn	zinc

Table 28.1 List of Abbreviations

28.1 Definition of Terms

The following general mining terms may be used in this report.

Term	Definition
Assay	The chemical analysis of mineral samples to determine the metal content.
Capital Expenditure	All other expenditures not classified as operating costs.
Composite	Combining more than one sample result to give an average result over a larger distance.
Concentrate	A metal-rich product resulting from a mineral enrichment process such as gravity concentration or flotation, in which most of the desired mineral has been separated from the waste material in the ore.
Crushing	Initial process of reducing ore particle size to render it more amenable for further processing.
Cut-off Grade (CoG)	The grade of mineralized rock, which determines as to whether or not it is economic to recover its gold content by further concentration.
Dilution	Waste, which is unavoidably mined with ore.
Dip	Angle of inclination of a geological feature/rock from the horizontal.
Fault	The surface of a fracture along which movement has occurred.
Footwall	The underlying side of an orebody or stope.
Gangue	Non-valuable components of the ore.
G & A	General and Administrative
Grade	The measure of concentration of gold within mineralized rock.
Hangingwall	The overlying side of an orebody or slope.
Haulage	A horizontal underground excavation which is used to transport mined ore.
Hydrocyclone	A process whereby material is graded according to size by exploiting centrifugal forces of particulate materials.
Igneous	Primary crystalline rock formed by the solidification of magma.
Kriging	An interpolation method of assigning values from samples to blocks that minimizes the estimation error.
Level	Horizontal tunnel the primary purpose is the transportation of personnel and materials.
Lithological	Geological description pertaining to different rock types.
LoM Plans	Life-of-Mine plans.
LRP	Long Range Plan.
Material Properties	Mine properties.
Milling	A general term used to describe the process in which the ore is crushed and ground and subjected to physical or chemical treatment to extract the valuable metals to a concentrate or finished product.
Mineral/Mining Lease	A lease area for which mineral rights are held.
Mining Assets	The Material Properties and Significant Exploration Properties.
NSR	Net Smelter Return
Ongoing Capital	Capital estimates of a routine nature, which is necessary for sustaining operations.

Term	Definition
Ore Reserve	See Mineral Reserve.
Pillar	Rock left behind to help support the excavations in an underground mine.
RoM	Run-of-Mine.
Sedimentary	Pertaining to rocks formed by the accumulation of sediments, formed by the erosion of other rocks.
Shaft	An opening cut downwards from the surface for transporting personnel, equipment, supplies, ore and waste.
Sill	A thin, tabular, horizontal to sub-horizontal body of igneous rock formed by the injection of magma into planar zones of weakness.
Smelting	A high temperature pyrometallurgical operation conducted in a furnace, in which the valuable metal is collected to a molten matte or doré phase and separated from the gangue components that accumulate in a less dense molten slag phase.
Stope	Underground void created by mining.
Stratigraphy	The study of stratified rocks in terms of time and space.
Strike	Direction of line formed by the intersection of strata surfaces with the horizontal plane, always perpendicular to the dip direction.
Sulphide	A sulphur bearing mineral.
Tailings	Finely ground waste rock from which valuable minerals or metals have been extracted.
Thickening	The process of concentrating solid particles in suspension.
Total Expenditure	All expenditures including those of an operating and capital nature.
Variogram	A statistical representation of the characteristics (usually grade).

Table 28.2 Definition of Terms

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29. CERTIFICATES

CERTIFICATE OF QUALIFIED PERSON

Deepak Malhotra, Ph.D., SME RM
Principal Metallurgist Pro Solv, LLC
15450 W. Asbury Avenue
Lakewood, Colorado 80228
Email: deepak@rdiminerals.com

This certificate applies to the report entitled: "SANDFIRE RESOURCES AMERICA INC. Black Butte Copper Project Feasibility Study (Johnny Lee Deposit) and Mineral Resource Estimate Update (Lowry Deposit) - Technical Report NI 43-101", effective date October 19, 2020, issued on December 8, 2020.

I, **Deepak Malhotra, Ph.D.,** do hereby certify that:

`

1) I am the President of Pro Solv, LLC, with a business address at 15450 W. Asbury Avenue, Lakewood, Colorado 80228.
2) I graduated with a degree in Metallurgical Engineering, Master of Science in 1973 from the Colorado School of Mines in Golden, Colorado. In addition, I graduated with a degree in Mineral Economics, Ph.D. in 1978 from the Colorado School of Mines in Golden, Colorado. My relevant experience includes working as a metallurgist and mineral economist for a total of 46 years since my graduation with specific expertise in mineral processing, metallurgical testing, and recovery methods. I am a member of the Society of Mining Engineers.
3) I have read the definition of "qualified person" set out in National Instrument 43-101- Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" within the meaning of NI 43-101.
4) My most recent personal inspection of the property which is the subject of the Technical Report, occurred on November 6, 2018.
5) As the primary author, I am generally responsible for all Sections of the Technical Report. In particular, I was the Qualified Person responsible for Sections 13, 17, 18, 19, 21, 22, and 24, and portions of Sections 1, 2, 3, 25, and 26 summarized therefrom, of the Technical Report.
6) I am independent of the issuer, Sandfire Resources America Inc., according to Section 1.5 of NI 43-101.
7) I have had prior involvement with the property that is the subject of the Technical Report. My prior involvement consisted of acting as a Qualified Person for the NI 43-101 Technical Report entitled: "NI 43-101 Technical Report on Mineral Resources – Black Butte Copper Project, White Sulphur Springs, MT, USA" with an effective date of October 15, 2019 and report date of December 6, 2019.
8) I have read NI 43-101, Form 43-101 F1 -Technical Report, 43-101 CP-Standards of Disclosure for Mineral Projects, and confirm that the Technical Report has been prepared in compliance with such instrument, form, and companion policy.
9) As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the portions of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
10) I consent to the filing of the Technical Report with any securities regulatory authority, stock exchange and other regulatory authority and any publications by them, including electronic publication in the public company files on their websites accessible by the public.

Dated this 8th day of December 2020.

"Signed" *"Stamped"*

Deepak Malhotra, Ph.D, SME-RM

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CERTIFICATE OF QUALIFIED PERSON

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Email: eronald@srk.com

This certificate applies to the report entitled: "SANDFIRE RESOURCES AMERICA INC. Black Butte Copper Project Feasibility Study (Johnny Lee Deposit) and Mineral Resource Estimate Update (Lowry Deposit) - Technical Report NI 43-101", effective date October 19, 2020, issued on December 8, 2020.

I, **Erik Ronald**, do hereby certify that:

1. I am Principal Resource Geology Consultant of:
   SRK Consulting (U.S.), Inc.
   1125 17th Street, Suite 600
   Denver, Colorado, USA 80202
2. I graduated with a Bachelor's degree in Geological Sciences from the University of California, Santa Barbara in 1997. In addition, I have obtained a Master of Engineering degree in Geological Engineering from the Colorado School of Mines in 2001 and a graduate certificate in Geostatistics from the Edith Cowan University in 2015.
3. I am a Professional Geologist practicing member of the Professional Geoscientists Ontario (#3050), a Professional Geologist with the State of Wyoming (PG-3568), and a Registered Member of the Society of Mining, Metallurgy, and Exploration (#4129819RM).
4. I have worked as a geologist for a total of 22 years since my graduation from university. My relevant experience includes exploration, mine geology, and resource geology for various companies and mines including over twelve years employed as a geologist with Rio Tinto in the United States and Australia and over two years with SRK Consulting (U.S.), Inc. along with various other roles associated with mining and mineral resources. Recent projects include Mineral Resource modelling and estimation for multiple commodities including copper, iron ore, gold, mineral sands, and industrial minerals.
5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a "qualified person" for the purposes of NI 43-101.

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6. I am responsible for the preparation of Geology and Mineral Resources including: Sections 4 through 12, 14, 23, and portions of 1, 2, 3, 25, and 26 summarized therefrom, of the Technical Report. I visited the Black Butte property on November 6, 2018 for 1 day.
7. I have had prior involvement with the property that is the subject of the Technical Report. My prior involvement consisted of acting as a Qualified Person for the NI 43-101 Technical Report entitled: "NI 43-101 Technical Report on Mineral Resources – Black Butte Copper Project, White Sulphur Springs, MT, USA" with an effective date of October 15, 2019 and report date of December 6, 2019.
8. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the portions of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
9. I am independent of the issuer in accordance with section 1.5 of National Instrument 43-101.
10. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.
11. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated this 8th Day of December, 2020.

"Signed" *"Stamped"*

Erik C. Ronald, MEng, P.Geo (PGO#3050)

CLM MINING
& investment

CERTIFICATE OF QUALIFIED PERSON

Brad Evans, BEng, MAusIMM CP (Mining)

Principal Mining Engineer

CLM Mining & Investment

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This certificate applies to the report entitled: "SANDFIRE RESOURCES AMERICA INC. Black Butte Copper Project Feasibility Study (Johnny Lee Deposit) and Mineral Resource Estimate Update (Lowry Deposit) - Technical Report NI 43-101", effective date October 19, 2020, issued on December 8, 2020.

I, **Brad Evans**, do hereby certify that:

1) I am a Principal Mining Engineer with CLM Mining & Investment with a business address at 35 Mundy Street Bendigo, VIC, 3550, Australia.
2) I graduated with a degree in Engineering (Mining) in 1998 from the University of Ballarat, Ballarat Victoria. Also, my relevant experience includes working as a Mining Engineer for 23 years, 11 of which was on underground mine site in Australia, and 12 of which was as a Principal Mining Consultant working on underground mining projects around the world. I am a member of the Australiasian Institute of Mining and Metallurgy and accredited as a Competent Person in the core discipline of Mining.
3) I have read the definition of "qualified person" set out in National Instrument 43-101- Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a "qualified person" within the meaning of NI 43-101.
4) On November 6, 2018, I visited and inspected the property which is the subject of the Technical Report.
5) I am responsible for Sections 15 and 16, and portions of Sections 1, 2, 3, 25, and 26 summarized therefrom, of the Technical Report.
6) I am independent of the issuer, Sandfire Resources America Inc., according to Section 1.5 of NI 43-101.
7) I have not had prior involvement with the property that is the subject of the Technical Report.
8) I have read NI 43-101, Form 43-101 F1 -Technical Report and 43-101CP-Standards of Disclosure for Mineral Projects. I am responsible for Sections 15 and 16 and portions of Sections 1, 2, 3, 25, and 26 of the Technical Report and confirm that these have been prepared in compliance with such instrument, form, and companion policy.
9) As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the portions of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
10) I consent to the filing of the Technical Report with any securities regulatory authority, stock exchange and other regulatory authority and any publications by them, including electronic publication in the public company files on their websites accessible by the public.

Dated this 8th day of December 2020.

_____*"Signed"*_____ *"Stamped"*

Brad Evans, AusIMM CP (Mining)

CERTIFICATE OF QUALIFIED PERSON

Patrick Williamson, MSc, MMSA QP

Intera Inc., Principal Hydrogeochemist

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This certificate applies to the report entitled: "SANDFIRE RESOURCES AMERICA INC. Black Butte Copper Project Feasibility Study (Johnny Lee Deposit) and Mineral Resource Estimate Update (Lowry Deposit) - Technical Report NI 43-101", effective date October 19, 2020, issued on December 8, 2020.

I, **Patrick Williamson,** do hereby certify that:

1) I am a Principal Hydrogeochemist at INTERA Inc., with a business address at 6707 Winchester Circle, Suite 200, Boulder, Colorado, 80301. Corporate headquarters of INTERA Inc. are located at 600 Great Hills Trail #300w, Austin, TX 78759.
2) I graduated with a degree in Geology, Master of Science in 1987 from the University of Colorado in Boulder, Colorado. In addition, I graduated with a bachelor's degree in Geology in 1982 from the Colorado College in Colorado Springs, Colorado. My relevant experience includes working as a hydrogeologist and environmental geochemist for a total of 33 years since my graduation from the University of Colorado, with specific expertise at mining projects throughout the Americas in water quality, mine water management, waste characterization and management, social license, and permitting. I am a member of the Society of Mining, Metallurgy and Exploration, the Acid Drainage Technology Initiative and the Mining and Metallurgical Society of America.
3) I have read the definition of "qualified person" set out in National Instrument 43-101- Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" within the meaning of NI 43-101.
4) On November 6, 2018, I visited and inspected the property which is the subject of the Technical Report.
5) I am responsible for Section 20, and portions of sections 1, 2, 3, 25 and 26 summarized therefrom, of the Technical Report.
6) I am independent of the issuer, Sandfire Resources America Inc., according to Section 1.5 of NI 43-101.
7) I have not had prior involvement with the property that is the subject of the Technical Report.
8) I have read NI 43-101, Form 43-101 F1 -Technical Report and 43-101CP-Standards of Disclosure for Mineral Projects. I am responsible for Section 20 and portions of Sections 1, 2, 3, 25 and 26 of the Technical Report and confirm that these have been prepared in compliance with such instrument, form, and companion policy.
9) As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the portions of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
10) I consent to the filing of the Technical Report with any securities regulatory authority, stock exchange and other regulatory authority and any publications by them, including electronic publication in the public company files on their websites accessible by the public.

Dated this 8th day of December, 2020

"Signed" *"Stamped"*

Patrick Williamson, MMSA