SANDFIRE RESOURCES AMERICA INC.



SANDFIRE RESOURCES AMERICA INC.
BLACK BUTTE COPPER PROJECT

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

Effective Date: November 5, 2025
Report Date: January 29, 2026

SANDFIRE RESOURCES BLACK BUTTE COPPER AMERICA INC.

NI 43-101 Technical Report

Black Butte Copper Project | Pre-Feasibility Study

Montana

Project Number: 300201

Effective Date: November 5, 2025
Report Date: January 29, 2026

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This certificate applies to the report entitled: "SANDFIRE RESOURCES AMERICA INC. Black Butte Copper Project Preliminary Feasibility Study (Johnny Lee Deposit) and Mineral Resource Estimate Update (Lowry Deposit) - Technical Report NI 43-101", effective date November 5, 2025, issued on January 29, 2026 (the "Technical Report").

I, **Deepak Malhotra, Ph.D.,** do hereby certify that:

1) I am the President of DM Consulting, 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 over 50 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) I have visited and inspected the property which is the subject of the Technical Report on November 6, 2018 and September 14, 2022.
5) As the primary author, I am responsible for all Sections 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, 43-1O1CP-Standards of Disclosure for Mineral Projects, and 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 Technical Report for which I am responsible, contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 29th day of January 2026

_____*"Signed"*_____ *"Stamped"*
Deepak Malhotra, Ph.D, SME-RM

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CERTIFICATE OF QUALIFIED PERSON

I, Berkley J. Tracy, MSc Geology, PG, CPG, PGeo, do hereby certify that:

1. I am employed as a Practice Leader and Principal Consultant (Resource Geology) of SRK Consulting (U.S.), Inc., 999 Seventeenth Street, Suite 400, Denver, CO, USA, 80202.
2. This certificate applies to the technical report titled "SANDFIRE RESOURCES AMERICA INC. Black Butte Copper Project Preliminary Feasibility Study (Johnny Lee Deposit) and Mineral Resource Estimate Update (Lowry Deposit) - Technical Report NI 43-101", effective date November 5, 2025, issued on January 29, 2026 (the Technical Report).
3. I graduated with a Bachelor of Science in Geology from The University of Georgia (UGA), Athens, Georgia in 1998. In addition, I obtained a Master of Science degree in Geology from UGA in 2001. I am a Certified Professional Geologist (CPG #11901) with the American Institute of Professional Geologists (AIPG), a Professional Geoscientist (PGeo #3024) with Professional Geoscientists Ontario (PGO), and a licensed/registered Professional Geologist (PG) in several U.S. states (Georgia #1792, Alabama #1231, South Carolina #2500, Florida #3175, Tennessee #5440, Alaska #243566, Idaho #6971973, Utah #14237954-2250, and Arizona #85349). I have practiced my profession for over 25 years and have held professional registration since 2005. My relevant experience includes base and precious metal exploration, resource geology, three-dimensional (3D) modeling, geostatistical estimation, due diligence reviews, independent audits, planning and supervising geologic logging, sampling, mapping, and feasibility projects, and managing large exploration programs leading to mine development.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I visited the Black Butte property on September 14, 2022, and June 23 to 24, 2025. These field visits allowed independent observation of the property, geology, exploration and sampling procedures.
6. I am responsible for the Geology and Mineral Resources portions of the report, Sections 3 through 12, 14, 23 and portions of Sections 1, 25, and 26 derived therefrom.
7. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.
8. I have had prior involvement with the property that is the subject of the Technical Report. The nature of my prior involvement is related to independent consulting for geological modeling and resource estimation that was the subject of previous disclosure.
9. I have read NI 43-101 and Form 43-101F1 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
10. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 29th Day of January 2026.

"Signed" ***"Stamped"***

Berkley J. Tracy, P.Geo (PGO#3024)

CERTIFICATE OF QUALIFIED PERSON

Shane McLeay, BEng (Hons) (Mining), FAusIMM

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This certificate applies to the report entitled: "SANDFIRE RESOURCES AMERICA INC. Black Butte Copper Project Preliminary Feasibility Study (Johnny Lee Deposit) and Mineral Resource Estimate Update (Lowry Deposit) - Technical Report NI 43-101", effective date November 5, 2025, issued on January 29, 2026 (the "Technical Report").

I, **Shane McLeay**, do hereby certify that:

1) I am a Principal Mining Engineer with Entech Pty Ltd with a business address at 8 Cook St, West Perth, WA, Australia 6005.
2) I graduated with a degree in Engineering (Mining) in 1995 from the Curtin University, Western Australian School of Mines, Kalgoorlie, Western Australia. My relevant experience includes working as a Mining Engineer for 30 years, 15 of which was directly working on minesites throughout Australia, and 15 consulting in Mining Engineering to mining projects throughout the world. I am a Fellow of the Australiasian Institute of Mining and Metallurgy.
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) I have visited and inspected the property which is the subject of the Technical Report on June 25, 2025.
5) I am responsible for Sections 15 and 16 and portions of Sections 1, 2, 3, 25, and 26 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 agree 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 fairly and accurately reflect the 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 29th day of January 29, 2026.

_____ *"Signed"* _____ *"Stamped"*

Shane McLeay, FAusIMM

CERTIFICATE OF QUALIFIED PERSON

Patrick Williamson, MSc, MMSA QP
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This certificate applies to the report entitled: "SANDFIRE RESOURCES AMERICA INC. Black Butte Copper Project Preliminary Feasibility Study (Johnny Lee Deposit) and Mineral Resource Estimate Update (Lowry Deposit) - Technical Report NI 43-101", effective date November 5, 2025, issued on January 29, 2026 (the "Technical Report").

I, **Patrick Williamson,** do hereby certify that:

1) I am a Principal Hydrogeochemist at INTERA Inc., with a business address at 1434 Spruce Street, Suite 207, Boulder, Colorado, 80302. 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) I have visited and inspected the property which is the subject of the Technical Report on November 8, 2018 and September 14, 2022.
5) I am responsible for Section 20 and portions of Sections 1, 2, 3, 25 and 26 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 43-101F1 - 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 agree 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 fairly and accurately reflect the scientific and technical information that is required to be disclosed to make the technical report not misleading.

Dated this 29th day of January 2026

_____"Signed"_____ "Stamped"
Patrick Williamson, MMSA

CERTIFICATE OF QUALIFIED PERSON

Tom Parrott FIEAust CPEng NER RPEQ

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This certificate applies to the report entitled: "SANDFIRE RESOURCES AMERICA INC. Black Butte Copper Project Preliminary Feasibility Study (Johnny Lee Deposit) and Mineral Resource Estimate Update (Lowry Deposit) - Technical Report NI 43-101", effective date November 5, 2025, issued on January 29, 2026 (the "Technical Report").

I, **Tom Parrott**, do hereby certify that:

1) I am a Principal Geotechnical Engineer with Entech Pty Ltd with a business address at 8 Cook St, West Perth, WA, Australia 6005. I graduated with a degree in Geological Engineering in 2000 from RMIT in Melbourne and a Masters of Science (Mining Geomechanics) from the West Australian School of Mines in 2012. My relevant experience includes working as a Geotechnical Engineer for over 25 years post graduate experience in both open pit and underground mines. I have worked in many challenging mines in both Canada and Australia and have extensive experience in operational solutions, ground control selection and analysis, and study work. I am a Fellow of FIEAust, Chartered Professional Engineer (CPEng) and Registered Professional Engineer in Queensland (RPEQ).
2) 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.
3) I have visited and inspected the property which is the subject of the Technical Report on August 18, 2025.
5) I am responsible for portions of Sections 1, 2, 3, 16, 25, and 26 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 portions of Sections 1, 2, 3, 16, 25, and 26 of the Technical Report and agree 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 fairly and accurately reflect the 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 29th day of January 29, 2026.

"Signed" *"Stamped"*

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Tom Parrott, FIEAust CPEng NER RPEQ

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1. SUMMARY

1.1 Introduction and Scope

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

The Technical Report (Report), which presents the results of the PFS, was overseen by Deepak Malhotra of DM Consulting., 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 November 5, 2025, are based on information collected by the various consultants and employees of SRA.

This technical report reverts to a Preliminary Feasibility Study (PFS) from the Feasibility Study (FS) published on December 8, 2020, to reassess and establish an updated resource leading to a new revised plan, which will lay the foundation for proceeding with an updated feasibility study, if necessary. The PFS meets or exceeds all 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 PFS 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

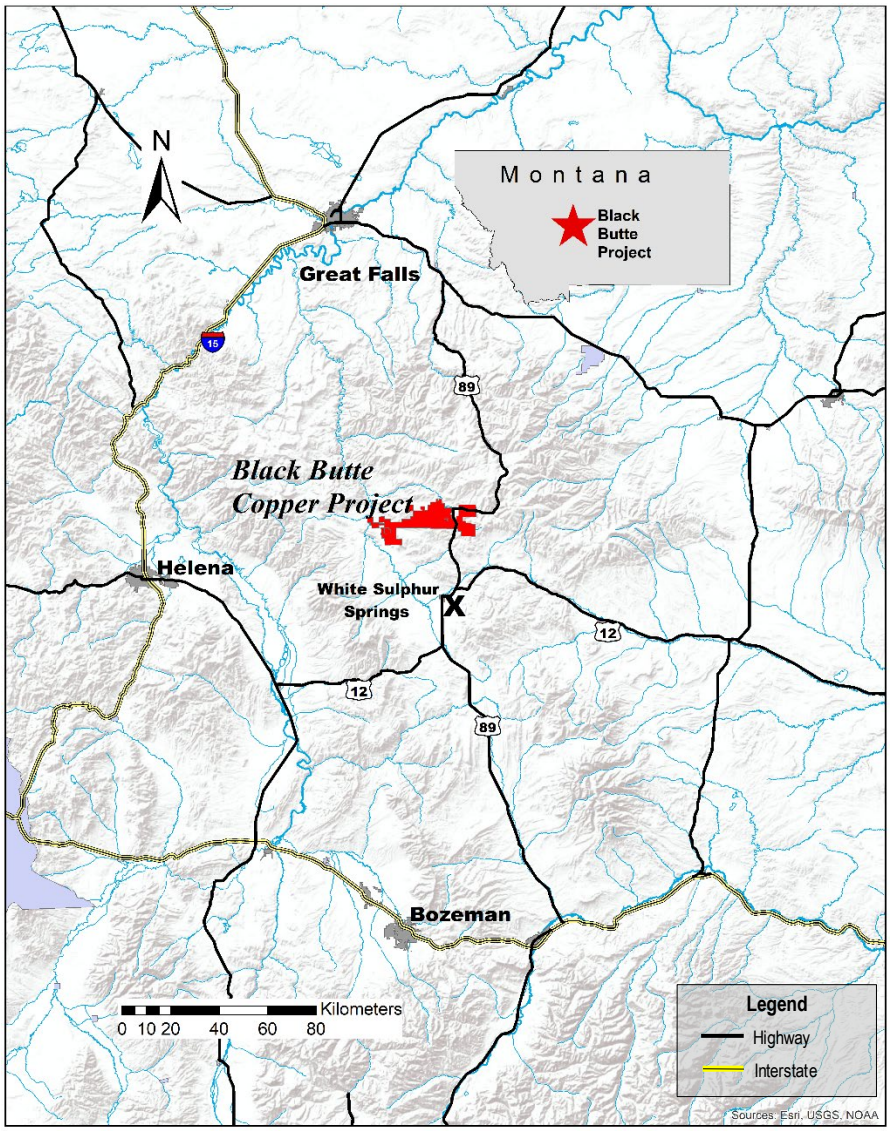
The purpose of this Technical Report (Report) is to present the PFS findings of the Project. The contents of this Report reflect various technical and economic conditions at the time of writing. Given the nature of the mining business, these conditions can change significantly over relatively short periods 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 7,684.3 hectares (ha) (18,986.8 acres) of fee simple lands under mineral lease by Sandfire Resources America (SRA), and 1,016 unpatented mining claims on US Forest Service lands (USFS), leased by SRA, totalling 8,077.8 ha (19,960.4 acres). The Project is located in south-central Montana in Meagher County, 27 km north of White Sulphur Springs (Figure 1.1). This Technical Report includes the Johnny Lee and Lowry copper (Cu) deposits. The Johnny Lee Cu deposit, which forms the focus of this PFS 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.



Source: SRA, 2026 compiled from Esri, USGS, NOAA

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 PFS 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 $75,000 per year in rent (adjusted on August 29, 2024). 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 the 2013 acquisition, SRA completed an additional 128 diamond drillholes at Johnny Lee and an additional 21 drillholes at Lowry 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 (JL UCZ). The north-eastern part of the JL 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 JL UCZ occurs at depths of 40 to 210 m below surface. The JL UCZ is truncated in the north by the Volcano Valley Fault Zone (VVFZ).

Primary Cu mineralization in the JL 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 JL UCZ are a function of both host lithofacies variation and the amount of progressive hydrothermal remobilization that has occurred. The northern part of the JL UCZ has generally had stronger hydrothermal remobilization and exhibits coarse grained, well-liberated copper sulphides and recrystallized euhedral pyrite. The southern part of the JL 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 JL UCZ has created localized zones of coarse grained, well-liberated copper sulphide.

Within the Johnny Lee LSZ is a zone of Cu mineralization that is termed the Johnny Lee Lower Copper Zone (JL LCZ). The JL LCZ is situated below the JL 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 JL LCZ lenses strike EW and dip to the south at 15° to 30°.The JL 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 JL LCZ is almost entirely hosted by chalcopyrite that is coarse-grained in comparison with the northern part of the JL UCZ. Trace amounts of tennantite are rarely observed, typically interstitial to early pyrite. As concentrations in the JL 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 (Mt)	Cu (%)	Metal (kt)
Measured + Indicated Mineral Resources			
JL UCZ			
Measured	2.3	2.0	46
Indicated	13.8	1.9	261
Sub-Total M+I	**16.1**	**1.9**	**307**
JL LCZ			
Measured	0.5	6.5	34
Indicated	2.3	5.1	121
Sub-Total M+I	**2.9**	**5.4**	**154**
Combined JL UCZ + JL LCZ			
Measured	2.8	2.8	80
Indicated	16.1	2.4	382
Total M+I	**18.9**	**2.4**	**462**
Inferred Mineral Resources			
JL UCZ	2.6	1.5	38
JL LCZ	0.8	3.2	26
Total Inferred	**3.4**	**1.9**	**64**

Source: SRK, 2025

Note:

- *The effective date for this Mineral Resource is November 5, 2025.*
- *The Mineral Resources in this estimate were prepared in accordance with the CIM Standards on Mineral Resources and Reserves, Definitions and Guidelines (CIM, 2014) prepared by the CIM Standing Committee on Reserve Definitions and adopted by CIM Council.*
- *Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. Inferred Mineral Resources have a high degree of uncertainty as to their economic and technical feasibility. There is no certainty that all or any part of an Inferred Mineral Resource will be converted to Measured or Indicated Mineral Resources in the future.*
- *To demonstrate reasonable prospects for eventual economic extraction, mineral Resources are reported using an economic NSR cut-off value of US$45.00/t, which is approximately equivalent to 0.52% recoverable Cu, based on long-term Cu price assumptions of US$4.70/lb and variable recovery and mill processing cost assumptions. Variable metallurgical recovery has been estimated in the JL UCZ n a block basis with a consistent 95% Cu recovery applied to the JL LCZ.*
- *There are no known legal, political, environmental, or other risks that could materially affect the potential future development of the Mineral Resources. All Mineral Resources are located within land currently under control or lease to Sandfire Resources America, Inc.*
- *All quantities are rounded to the appropriate number of significant figures to reflect the relative accuracy of the estimate; consequently, sums may not add up due to rounding. Cu assay values were capped where appropriate.*
- *The Mineral Resources for the JL UCZ and JL LCZ were reviewed and approved by Berkley Tracy, P.Geo (PGO#30524) 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 November 5, 2025 – SRK Consulting (U.S.), Inc.

Mineral Resource Category	Quantity (Mt)	Cu (%)	Total Metal (kt)
Measured + Indicated (M+I) Mineral Resources			
Lowry MCZ			
Measured	1.2	2.6	30
Indicated	3.6	2.4	86
Sub-Total M+I	**4.8**	**2.4**	**116**
Lowry LCZ			
Measured	0.1	1.8	1
Indicated	1.7	2.2	37
Sub-Total M+I	**1.8**	**2.1**	**38**
Combined Lowry UCZ + MCZ + LCZ			
Measured	1.2	2.5	31
Indicated	5.3	2.3	123
Total M&I	**6.6**	**2.4**	**154**
Inferred Mineral Resources			
Lowry UCZ	1.3	2.3	30
Lowry MCZ	0.9	2.1	18
Lowry LCZ	0.6	1.7	10
Total Inferred	**2.8**	**2.1**	**58**

Source: SRK, 2025

Note:

- *The effective date of this Mineral Resource is 5 November 2025;*
- *The Mineral Resources in this estimate were prepared in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Standards on Mineral Resources and Reserves, Definitions and Guidelines (CIM, 2014) prepared by the CIM Standing Committee on Reserve Definitions and adopted by CIM Council;*
- *Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. Inferred Mineral Resources have a high degree of uncertainty as to their economic and technical feasibility. There is no certainty that all or any part of an Inferred Mineral Resources will be converted to Measured or Indicated Mineral Resources in the future;*
- *To demonstrate reasonable prospects for eventual economic extraction, Mineral Resources are reported using an economic NSR cut-off value of US$110.00/t, which is approximately equivalent to 1.28% recoverable Cu, based on long-term Cu price assumptions of US$4.70/lb and variable recovery and mill processing cost assumptions;*
- *Metallurgical recovery for Cu has been assigned to the Lowry deposits: 91% for both LUCZ and LMCZ and 94% for LLCZ;*
- *There are no known legal, political, environmental, or other risks that could materially affect the potential future development of the Mineral Resources. All Mineral Resources are located within land currently under control or lease to Sandfire Resources America, Inc.;*
- *All quantities are rounded to the appropriate number of significant figures to reflect the relative accuracy of the estimate; consequently, sums may not add up due to rounding. Cu assay values were capped where appropriate; and*
- *The Mineral Resources for the Lowry Upper Copper Zone (LUCZ), Lowry Middle Copper Zone (LMCZ), and Lowry Lower Copper Zone (LLCZ) were reviewed and approved by Berkley Tracy, P.Geo (PGO#3024) 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 November 5, 2025 – SRK Consulting (U.S.), Inc.

1.6 Mineral Reserve Estimate

A NSR value was calculated for each block in a geological block model based on metallurgical recovery, grade, price, and payability factors. Mine design shapes were generated based on an NSR cut-off value of US$110/t (operating cost). An NSR cut-off value of US$45/t (incremental operating cost) was used to determine whether material required to be mined to access the design shapes should be sent to waste or processed. Detailed mine designs were created with dilution and recovery factors applied, Mineral Resource classifications considered, and other modifying factors and economic viability tested. Summary Mineral Reserve statements for the Johnny Lee Deposit are listed in Table 1.3 below.

Class	Diluted Tonnes (Mt)	Cu Grade (%)	Contained Cu Metal (kt)
Proven	-	-	-
Probable	9.5	2.9	270
Total	**9.5**	**2.9**	**270**

Notes:

1. *The Qualified Person for the Mineral Reserve estimate is Shane McLeay FAusIMM.*
2. *Effective date: November 14, 2025. 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 US$4.70/lb Cu price and NSR cut-off values of US$110/t (operating cost) and US$45/t (incremental operating cost).*
4. *Tonnages are rounded to the nearest 100 kt, metal grades are rounded to one decimal place and metal tonnes are rounded to the nearest 10kt. All units are metric.*
5. *Rounding as required by reporting guidelines may result in summation differences.*
6. *Average LOM metallurgical recovery is 95% for JL LCZ, 78% for JL UCZ, and 86% combined.*
7. *Mineral Reserves are based on the Mineral Resource estimate effective November 5, 2025.*
8. *Mineral Reserves are a subset of Mineral Resources.*

Table 1.3 Mineral Reserve Estimate for the Johnny Lee deposit as of November 14, 2025

1.7 Mining Methods

The Black Butte Copper Project contains four zones, two of which, the Johnny Lee Upper Copper Zone (JL UCZ) and the Johnny Lee Lower Copper Zone (JL LCZ) are the extraction targets and the basis of this Study. These zones are characterized as being reasonably flat lying with relatively narrow widths. All deposits have anomalous Silver (Ag) and Cobalt (Co) mineralization however, Cu is the only economic product considered in this Study.

The Study geotechnical assessment analyzed 4,782 m of drill core, 1,607 structural measurements, and laboratory tests confirming material properties adequate for a PFS. A 3D geotechnical model was developed for use in the Study and future updates.

Maximum unsupported spans were estimated using the Mathews-Potvin method, with drift-and-fill planned for the JL UCZ and cut-and-fill for the JL LCZ, with voids backfilled with cemented paste fill. Expected Q' values guided preliminary span limits, which require review as more data becomes available. Paste fill strength requirements were defined in 2019 studies for the JL UCZ and JL LCZ.

Numerical modelling confirmed the mining sequence's viability but highlighted low-confinement zones in the JL UCZ and the need for updated stress and strength data.

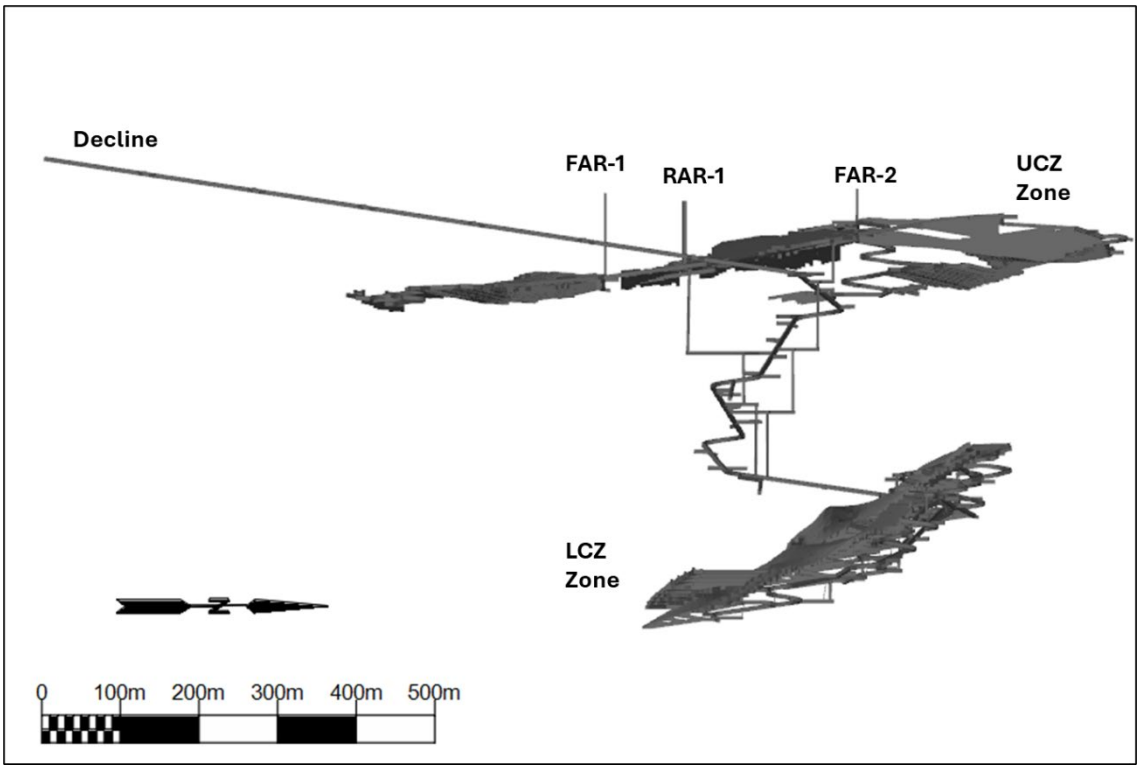
Recommended ground support includes MDX or resin bolts in the initial capital development, friction bolts elsewhere, mesh as primary surface support, fibrecrete for poor ground, and 6 m twin strand cable bolts at intersections.

The mining methods selected are a mix of mechanised Drift and Fill (DF) and mechanised Cut and Fill (CF) with cemented pastefill used for post-mining void support. The UCZ will predominantly use DF and use primary and secondary drives with longhole stripping to improve productivity and cost efficiency. Due to the slightly steeper orebody geometry the JL LCZ will predominantly use CF with multiple sill pillars planned to maximise JL LCZ production.

An underground mining contractor is planned to be engaged to execute the initial 36 months of the mining schedule during which both the JL UCZ and JL LCZ will be accessed and thereafter replaced by an owners mining team. The overall mine life from commencing development of the decline is just under 10 years.

The Johnny Lee deposit will be accessed by a single main ramp driven from surface. The ramp dimensions will be 5.50 m wide by 5.85 m high and excavated with an arched back. The ramp will be excavated at a maximum gradient of -14.3% from the surface and pass to the east of the JL UCZ and then spiral down to the JL 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 60 tonne (t) haul trucks hauling ore directly from stope areas to either a surface ore pad or the surface crusher.



Source: SRA, 2026

Figure 1.2 View of UG Design from the E

The objective of the mine sequence and schedule is to prioritise access to the higher grade JL LCZ and thereafter maximise production with supplemental material sourced from the JL UCZ. Prior to accessing the JL LCZ and during surface infrastructure construction the JL UCZ will be accessed and ore stockpiled for use in commissioning the process plant. As the JL LCZ is depleted production will be predominantly sourced from the JL UCZ.

Grade control drilling will use spacings of ~15 m by 20 m in the JL UCZ and 20 m by 15 m in the JL LCZ, with planned campaigns totalling 110,000 m for JL UCZ and 48,400 m for JL LCZ to achieve Measured Resource confidence.

Table 1.4 shows a mine physicals summary of the year-by-year lateral and vertical development, ore and waste tonnage, mined head grade and paste backfill volume.

Parameter	Total	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11
Lateral Development (km)	**83.4**		1.5	8.8	12.1	13.8	11.6	9.9	8.7	7.9	7.6	1.6
Vertical Development (m)	**1,293**		-	643	386	265	-	-	-	-	-	-
Waste Tonnes (kt)	**1,799**		124	561	507	348	136	42	30	23	24	4

Parameter	Total	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11
Paste Backfill ('000 m³)	**2,331**		-	-	197	196	334	332	383	393	367	129
Ore Tonnes (Mt)	**9.5**		0.0	0.3	0.9	1.2	1.4	1.2	1.3	1.4	1.4	0.4
Ore Head Grade (% Cu)	**2.9**		2.0	1.7	2.8	3.1	3.5	3.7	3.0	2.4	2.2	2.1
Ore Metal (kt Cu)	**270**		0	5	25	37	47	45	40	34	30	7

Table 1.4 Mine Physicals Summary

1.8 Mineral Processing and Metallurgical Test Work

Recent metallurgical test work programs (2024/25) undertaken indicated copper concentrate from the JL LCZ by froth flotation recovered 94.4% to 97.7% (averaging 96.2%) of the copper resulting in a concentrate grading 23% to 28% copper. A discounted copper recovery of 95.2% (accounting for ideal laboratory conditions) was applied to mining blocks in the JL LCZ.

Previous testwork on JL UCZ composites showed a wide range of copper recoveries (61.9% to 91.2%) at concentrate grades of 18.5% to 24.5% copper. Mineralogical investigation of JL UCZ metallurgical composites indicated that copper sulphide liberation was the primary metric that defined metallurgical performance. A variable recovery model was applied to the JL UCZ mining blocks based on the distribution of 113 mineralogy samples results and predicted metallurgical recovery.

Metallurgical test work has demonstrated that there are no reductions in copper recovery to concentrate from JL UCZ or JL LCZ ore by blending and processing the blend using the flowsheet optimized for JL UCZ ore.

Analyses of the copper concentrates from variability testing of JL UCZ composites have reported potentially deleterious levels of arsenic. Various arsenic containing minerals have been identified through mineralogical analysis, with only some associated with copper reporting to concentrate. All samples tested from the JL LCZ showed low arsenic in copper concentrate. Arsenic is forecast to range between 1,900 to 7,520 ppm in copper concentrate over the life of the project.

1.9 Recovery Methods

Metallurgical testwork results were used as the basis for the process plant design criteria and flowsheet development. The test work indicates that the copper in the JL UCZ and JL LCZ can be recovered to a concentrate by crushing, grinding, and froth flotation processes. The JL UCZ ore requires a 38 µm P_{80} primary grind and a 10-15 µm P_{80} regrind of the rougher concentrate to achieve optimized concentrate grade and copper recoveries. The JL LCZ ore achieves optimized recoveries at coarser grind sizes. However, as it will be blended with JL UCZ, the blended ore will be treated using the process as optimized for JL UCZ ore. Metallurgical test work has demonstrated that there are no reductions in copper recovery to concentrate from JL UCZ or JL LCZ ore by blending and processing the blend using the flowsheet optimized for JL 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.

MDEQ issued the Record of Decision for the Mine Operating Permit in April 2020, approving phased mine development and an incremental bonding approach. Incremental Bond 1 and Incremental Bond 2 correspond to Phase 1 and Phase 2 construction, respectively. In accordance with ARM 17.24.140, MDEQ set Incremental Bond 1 at US$4,653,348 on May 19, 2020, and issued the final Hard Rock Mining Operating Permit on August 14, 2020, authorizing Phase 1 activities. Prior to Phase 2 construction, MDEQ will calculate an Incremental Bond 2, which will represent the full project bond amount required for reclamation and closure in the event of operator default. Including the approved MOP, there are 28 permits and plans that have been or 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. To date, 23 permits and plans have been approved, including the MOP, and work has been initiated on one of the five outstanding permits/plans, that being plans for a transmission line to site. The outstanding permits/plans involve potable water, sewage disposal, explosives permit, and hazardous waste identification.

1.12 Capital and Operating Costs

1.12.1 Capital Cost Estimates

The Project capital cost estimate developed for the PFS 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 twenty-nine months of construction and four years of mine reclamation and closure.

A capital cost of US$474 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 ±25% accuracy.

The capital costs are summarized in the table below.

Area	Capital Cost $M
Mining	US$155
Process and Infrastructure	US$293
Owner Costs	US$26
Total Project	US$474

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.
- Procurement and installation of required underground infrastructure.
- 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 were estimated based on applying fixed and variable estimates to the mine plan schedule physicals. Specifically, the costs were based on contractor requests for quotation responses for pre-production activities in the first 36 months of the mine plan, and owner-operator cost estimates in the subsequent production phase, built up from first principle estimates and supplier quotations. Annual operating costs and costs per tonne mined have been estimated. The average underground mining operating cost, excluding capitalised pre-production costs, is US$79M per annum (during the production phase), or US$66/t ore mined.

1.12.3 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. Operating costs for the treatment plant have been estimated to an accuracy of +/-25%.

The processing operating cost estimate has been developed based on a process plant feed tonnage of 1.2 million tonnes per annum. The processing operating cost estimate is US$45 million per annum or US$38/t milled. Economic Analysis

The financial analyses were undertaken based on the following parameters:

- Probable Mineral Reserves of 9.5 Mt averaging 2.9% Cu used in the financial model.
- The mining schedule and mining cost estimates were used for the financial model.
- Copper price of US$4.70/lb.
- LOM Copper recovery of 95% for JL LCZ, 78% for JL UCZ, and 86% combined 8% was used in the financial model.
- The economic model used a treatment charge of $50/t and a refining charge of 5.0 cents/lb. throughout LOM.
- An operating cost of $104/t for Mine to Mill was used for the model.
- Concentrate transport costs of $194/ wmt.
- A capital cost of US$474 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 US$143 M and a post-tax NPV ofUS99 M using an 8% discount rate.
- The project has a pre-tax IRR of 13.63% and a post-tax IRR of 11.3%.
- The projected cash flow will allow for the project capital to be paid back in approximately four years.
- Sensitivity analyses indicate that the project NPV is most sensitive to copper price, recovery, and feed grade.

1.13 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.

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, paste plant filters, and early commencement of site earthworks. The overall schedule for the Project is 29 months (contract award to demobilisation).

Total project labour working on the processing plant and mine infrastructure, site earthworks, owner's team and mining (including contractor and owner's team) will peak at approximately 430. This includes all site construction management, owner's construction supervision, owner's operations team.

1.14 Risks and Recommendations

A risk analyses was undertaken for the Project and categorized into low, moderate, high and extreme risk.

The only high-risk item identified in the study was a failing copper price, leading to reduced revenue. Moderate risks identified included the ability to consistently tight-fill paste, shortage of local skilled mining personnel, local housing availability and uncertainty around tariffs and the impact on capital cost of equipment.

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, further studies at Lowry with the aim to increase reserves and geometallurgy testing for improved copper recovery and further understanding of arsenic mineralisation (to improve blending and reduce arsenic levels in copper concentrate).

1.15 Interpretations and Conclusions

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

1.15.1 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.

1.15.2 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.

1.15.3 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.

1.15.4 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 stable Cu recoveries averaging 95.2% (for its proportion of the plant feed). 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. A metallurgical Cu recovery of 86.1% 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.

UCZ ore is expected to produce a Cu concentrate containing potentially deleterious levels of As. The amount of As that reports to the Cu concentrate is dependent on the proportion of UCZ ore in the blend and the UCZ As mineralisation present. The LOM average arsenic in concentrate is 4,500 ppm, ranging between 1,900 to 7,500 ppm.

Additional blending testwork conducted during the PFS aligned with previous testwork to show there was no adverse effect to either the UCZ or LCZ by blending the two ore sources together.

1.15.5 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.

1.15.6 Mineral Reserve Estimate

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

1.15.7 Mining Methods

Work completed for the PFS indicates that drift and fill (DF) and cut and fill (CF) UG mining is the most appropriate mining method for the Johnny Lee deposit.

1.15.8 Recovery Methods

The PFS 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 10-15 µm P_{80}, Three stage Cleaner Flotation Circuit and Filtering of the Cleaner concentrate to produce a saleable Cu concentrate.

1.15.9 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 PFS 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 PFS level.

1.15.10 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 PFS and notes that they meet or exceed all State standards.

1.15.11 Capital and Operating Costs

It is the opinion of the primary QP of the PFS 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 PFS and in accordance with NI 43-101 reporting.

1.15.12 Economic Analysis

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

1.15.13 Mine Closure

Mine closure is estimated to take approximately 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 (McIntosh, 2020). Additional costs for implementing the AMA tasks 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 Preliminary Feasibility Study (PFS or Study) covering the Johnny Lee deposit at the Black Butte Copper Project located in south-central Montana. Additionally, this report includes the Resource update for the Lowry deposit at the Project.

The quality of information, conclusions, and estimates contained herein is consistent with the level of effort involved in the QPs' services provided to Sandfire Resources America Inc. (SRA), 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 contracts with third-party service providers and relevant securities legislation. The contracts permit 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 are 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, 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 2025.

2.3 Issuer

This report is produced for Sandfire Resources America Inc., a mining exploration and development company based in Montana, USA. SRA was incorporated on 30 July 1998 under the laws of British Columbia and is a subsidiary of Sandfire Resources Ltd., a public company in Australia. SRA is listed on the TSX Venture Exchange with the trading symbol "SFR.V" and on the U.S. OTC Market with the trading symbol "SRAFF". The company is primarily focused on the exploration, development, and mining of its 100%-owned Black Butte Copper Project in central Montana, USA.

2.4 Qualified Persons and Historical Inspections of the Property

Personnel	Company	Expertise	Date(s) of Visit	Details of Inspection
Shane McLeay QP (2025)	Entech Pty Ltd	Mineral Reserves and Mining Methods	June 25, 2025	Inspected drill core, site access, topography, current facilities, and proposed infrastructure locations.
Tom Parrott (2025)	Entech Pty Ltd	Mining Geotechnical Studies	August 18, 2025	Inspected drill core, reviewed logging procedures, topography, proposed infrastructure locations, visible ground conditions.
Deepak Malhotra, QP (2020) (2025)	DM Consulting, LLC	Metallurgy, Infrastructure, and Mineral Economics (Capital Costs and Operating Costs), Overall Study Review	November 6, 2018 September 14, 2022	Toured the Project site to inspect topography, facilities and proposed infrastructure locations, inspected drill core, reviewed logging and sampling procedures.
Patrick Williamson, QP (2020) (2025)	INTERA Inc.	Environmental Studies, Permitting, and Mine Closure	November 6, 2018 September 14, 2022	Visited the mine site, inspected topography, ground conditions and proposed infrastructure locations.
Berkley Tracy, QP (2025)	SRK Consulting (U.S.), Inc.	Geology, Exploration, QA/QC, and Mineral Resources	September 14, 2022 June 23-24, 2025	Toured Project site, core facilities, and logging area. Reviewed drilling, sampling, and logging procedures

Table 2.1 Participants and Inspections

2.5 Report Preparation

This Report was prepared by SRA and overseen by the primary author Deepak Malhotra, with further contributions by external consultants and SRA personnel over the history of the Project. Further contributions were generated by the staff of Sandfire Resources Ltd.

Specifically, the Report has drawn on the following input resources from third-party consulting firms:

- SRK Consulting – Geology and Mineral Resources; mining methods and mine plans, geotechnical assessment, mining infrastructure, Mineral Reserves, mining capital and mining operating costs; overall project economics (2020 FS and 2025 PFS).
- Mining Plus (2020 FS) and Entech (2025 PFS) – 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 (2020 FS and 2025 PFS).
- Base Metal Laboratories (BML) – 2018-2019, 2022-2023, 2024-2025 Metallurgical testwork studies for the deposit.
- INTERA Inc. – Environmental studies, permitting, and mine closure (2020 FS and 2025 PFS).
- Paterson & Cooke USA Ltd. – 2019 Paste fill test work (2020 FS).
- Tierra Group International Ltd. (2020 FS) and Knight Piésold (2025 PFS) – Reviews of the cemented tailings facility and operating plan, and waste and water management.
- Pi Kappa Enterprises, LLC – Infrastructure and waste and water management (2020 FS).
- Hydrometrics Inc. - Surface and ground water hydrology, site water balance support, baseline water quality monitoring (2020 FS and 2025 PFS).
- Enviromin/RESPEC - Environmental geochemistry (2020 FS).
- Savage – Concentrate transport logistics and costs (2020 FS and 2025 PFS).

2.6 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. All statements, other than statements of historical fact regarding SRA or the Black Butte Copper Project, are forward-looking statements. 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. In particular, this Report contains forward-looking statements with respect to cash flow forecasts, projected capital, operating and exploration expenditures, mine life and production rates, potential mineralization and metal or mineral recoveries, and information pertaining to financial and operating performance and mine life at the Black Butte Copper Project. All forward-looking statements in this Report are necessarily based on opinions and estimates made as of the date such statements are made and are subject to important risk factors and uncertainties, many of which cannot be controlled or predicted. Material assumptions regarding forward-looking statements are discussed in this report, where applicable. In addition to such assumptions, the forward-looking statements are inherently subject to significant business, economic and competitive uncertainties and contingencies. Known and unknown factors could cause actual results to differ materially from those projected in the forward-looking statements. Such factors include, but are not limited to: fluctuations in the spot and forward price of metals and commodities (including gold, diesel fuel, natural gas and electricity); the speculative nature of mineral exploration and development; changes in mineral production performance, exploitation and exploration successes; diminishing quantities or grades of reserves; increased costs, delays, suspensions, and technical challenges associated with the construction of capital projects; operating or technical difficulties in connection with mining or development activities, damages to SRA's reputation due to the actual or perceived occurrence of any number of events, including negative publicity with respect to the handling of environmental matters or dealings with community groups, whether true or not; risk of loss due to acts of war, terrorism, sabotage and civil disturbances; the impact of inflation; fluctuations in the currency markets; changes in interest rates; changes in national and local government legislation, taxation, controls or regulations and/or changes in the administration of laws, policies and practices; expropriation or nationalization of property and political or economic developments; failure to comply with environmental and health and safety laws and regulations; timing of receipt of, or failure to comply with, necessary permits and approvals; litigation; contests over title to properties or over access to water, power and other required infrastructure; increased costs and physical risks including extreme weather events and resource shortages, related to climate change; and availability and increased costs associated with mining inputs and labour. In addition, there are risks and hazards associated with the business of mineral exploration, development, and mining, including environmental hazards, industrial accidents and the inability to obtain insurance in certain instances to cover these risks. Many of these uncertainties and contingencies can affect SRA's actual results and could cause actual results to differ materially from those expressed or implied in any forward-looking statements made by, or on behalf of, SRA. All of the forward-looking statements made in this report are qualified by these cautionary statements and SRA and the Qualified Persons who authored this report undertake no obligation to update publicly or otherwise revise any forward-looking statements whether as a result of new information or future events or otherwise, except as may be required by law.

2.7 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, DM Consulting LLC 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.
- Berkely Tracy, Principal Consultant, SRK Consulting (US), Inc. is the QP responsible for Geology and Mineral Resources Sections 4 through 12, 14, 23, and portions of Sections 1, 2, 3, 24, 25 and 26 summarized therefrom, of this Technical Report.
- Shane McLeay, Entech, 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.
- Tom Parrott, Entech, is the QP responsible for the Geotechnical portions of Sections 1, 16, 25 and 26 summarized therefrom, of this Technical Report.

Various external reports, memorandums, articles and other sources of information have been used in preparation of this report to different extents. All sources used to support this report are cited in text and in Part 27 (References) of this report.

2.8 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.

2.9 Effective Dates

The effective date of this Report is 5 November 2025.

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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. Berkley Tracy, 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 (Cook, 2019; Cook, 2019a):

- Cook, J. (2019). Mineral Title Report. Report to Tintina Montana, Inc. from Crowley Fleck PLLP (Attorneys). September, 25, 2019, 11p.
- Cook, J. (2019a). Lease Standing and Ownership of Lands Covering the Johnny Lee and Lowry deposits. Report to Tintina Montana, Inc. from Crowley Fleck PLLP (Attorneys). September, 25, 2019, 1p.

Summary title information is relied upon related to exploration and mining rights is presented in Section 4 of this Technical Report. 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 (Cook, 2019; Cook, 2019a). Additionally, all documentation and details of surface rights have been provided to SRK for QP review by SRA. Based on the reliance of the other identified 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.

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 (MDEQ, 2020a) and Record of Decision (MDEQ, 2020b).
- Greg Bryce of Hydrometrics managed the permitting process for surface and groundwater including water rights. (Hydrometrics, 2016a, 2016b, 2017, 2018a).
- 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).
  - Hydrometrics, Inc. (2016a). *Water Resources Monitoring Field Sampling and Analysis Plan, Black Butte Copper Project*. Prepared by Hydrometrics Inc. for Tintina Montana Inc. Report Dated June 2016. 33p. Appendix U of the MOP Application document.
    - Hydrometrics, Inc. (2016b). Groundwater Modeling Assessment for the Black Butte Copper Project, Meagher County, Montana. Prepared for Tintina Montana Inc. Original Report date November 15, 2015. June, 2016. Appendix M of the MOP Application document.
    - Hydrometrics, Inc. (2017). Baseline Water Resources Monitoring and Hydrologic Investigations Report, Tintina Resources Black Butte Copper Project. Revised Report Dated March 14, 2017. Original report dated August 2015. 200p. Appendices A and B (electronic databases). Appendix B of the MOP Application document.
    - Hydrometrics, Inc. (2018a). Integrated Discharge Permit Application Narrative, Black Butte Copper Project, Meagher County, Montana, Revision 2, (under the Montana Pollutant Discharge Elimination System [MPDES] permit program). Report to Tintina Montana Inc., December 2017 and revised in February and April 25, 2018, 96p.
    - Hydrometrics, Inc. (2018b). Treated Water Storage Pond Memorandum to Jerry Zieg of Sandfire Resources America Inc. October 18, 2018. 2p.
    - Hydrometrics, Inc. (2018c). Black Butte Copper NCWR Diversion on Sheep Creek, Technical Memorandum to Jerry Zieg, Sandfire Resources America Inc. and Craig Jones, MDEQ. August 20, 2018. 2p.
    - Hydrometrics, Inc. (2019a). Monthly Water Balance Analysis – Black Butte Copper Feasibility Study, Technical Memorandum prepared for Sandfire Resources America Inc. August 5, 2019, 13p.
    - Hydrometrics, Inc. (2019b). Black Butte Copper Feasibility Study – Treated Water Discharge Options. Memorandum prepared for Sandfire Resources America, Inc. February 15, 2019.
    - Hydrometrics, Inc. (2021). Outline of Recommended Lowry Hydrological Investigation(s) and Permitting Timelines, Technical Memorandum prepared for Sandfire Resources America Inc. August 5, 2021.

- Hydrometrics, Inc. (2022). 2022 Annual Water Resources Monitoring Report Black Butte Copper (BBC-2022-00188), Technical Report prepared for Sandfire Resources America Inc. June 1, 2023.
- Hydrometrics, Inc. (2025). Preliminary assessment of impacts of the 2025 mine plan changes on groundwater inflow evaluation for Black Butte Copper, Technical Memorandum prepared by Greg Bryce for Sandfire Resources America, Inc. September 3, 2025.

- 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.
  - Kimball, R. (2017). Water Treatment Modelling for the Black Butte Copper Project, Montana. Prepared by AMEC Foster Wheeler. Prepared for Sandfire Resources America Inc. Letter dated May 2, 2017. (update to March 2019 Letter) 4p. In Appendix V of the MOP Application.
    - Kimball, R. (2020). Feasibility level estimate Letter of water treatment and brine processing and disposal for the Black Butte Copper Project, Montana. Prepared by Wood Environment & Infrastructure Solutions, Inc. Prepared for Sandfire Resources America Inc. Letter dated March 25, 2020. 2p.
  - Jon Hall of AQ2 Pty Ltd for design considerations for the hydrogeologic characterization program (Section 20.2.1).
  - Hall, J. (2018) *011e – Proposed Hydro Test Programme (Revised 10 Dec 18)* [internal spreadsheet]. Provided by AQ2 to Sandfire Resources America Inc.
- 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).
  - Enviromin, Inc. (2017a). Baseline Environmental Geochemistry Evaluation: 2015 Operating Permit. November 2015, updated on April 24, 2017 for the Revised Mine Operating Permit Application. Report to Tintina Resources, Inc., Enviromin Inc., Bozeman, Montana, 28p. In Appendix D of the MOP Application.
    - Enviromin, Inc. (2017b). Baseline Environmental Geochemistry Evaluation of Near-Surface Materials: Black Butte Copper Project Revised Mine Operating Permit Application, April 24, 2017, 18p. Report to Tintina Resources. In Appendix D-1 of the MOP Application.
- Numerous consultants from WESTECH Environmental, including (Section 20.2):
- Ken Scow for vegetation studies.
- Scow, K. (2017). *Baseline vegetation inventory: Black Butte Project, Meagher County, Montana*. WESTECH Environmental Services, Inc. Prepared for Tintina Resources.
- Corey Baker for the soil study.

- Baker, C. (2017). Baseline Soils Report Black Butte Copper Project Meagher County, Montana. Report to Tintina Resources, Inc., WESTECH Environmental Services, Inc., Helena, Montana. Revised Report Date March 2017. Original Report dated 2015. 23p. In Appendix E of the MOP application.
- John Beaver, Ken Scow, Lisa Larsen, Dean Culwell, Jessica Allewalt, and Nancy Scow for the wetland delineations.
- WESTECH Environmental Services, Inc. (2015a). Baseline Wetland Delineation and Waterbody Survey, Black Butte Copper Project, Meagher County, Montana. Report to Tintina Resources, Inc., WESTECH Environmental Services, Inc., Helena, Montana. January, 2015. 16p. In Appendix C (C-1) of the MOP Application.
- Patrick Farmer, Ken Scow, Jessica Allewalt, John Beaver, Drake Barton, Dean Culwell, and Corey Baker for the Terrestrial Wildlife Resources evaluation.
- WESTECH Environmental Services, Inc. (2015b). Terrestrial Wildlife Resources Evaluation Black Butte Copper Project, Meagher County, Montana. Report to Tintina Resources, Inc., WESTECH Environmental Services, Inc., Helena, Montana, August 2015. 35p. In Appendix F of the MOP Application.
- David Stagliano of Montana Biological Survey for the aquatic surveys (Section 20.2.2).
  - Stagliano, D. (2018). Baseline Aquatic Surveys and Assessment Summary 2014-2017 of Streams in the Tintina Black Butte Copper Project Area of Meagher County, MT. Report to Tintina Montana Inc., Montana Biological Survey, Helena, Montana, May 2017, updated on February 2018, 46p.
  - Stagliano, D. (2019). Aquatic Monitoring Plan and Assessment Summary for Streams in the Tintina Black Butte Copper Project Area of Meagher County, MT, (2014-2018). Report to Tintina Montana Inc. June 2017, updated April, 2019. Draft copy in Appendix G-1 of the MOP Application.
- Lynn Peterson, John Mueller, and Kirk Miller from Tetra Tech, Inc. for the Cultural Resource Inventory study (Section 20.2.3).
  - Tetra Tech. (2015). *2015 cultural resource inventory of proposed mine facilities and access roads within the Black Butte Copper Project area, Meagher County, Montana* (Report No. #114-710301A). Prepared for Tintina Resources, Inc.
- Sean Connolly from Big Sky Acoustics, LLC for the noise survey and assessment (Section 20.2.1).
  - Connolly, S. (2013). Baseline Noise Survey, Black Butte Copper Project. Report to Tintina Resources, Inc., Big Sky Acoustics, LLC, Helena, Montana, BSA Project #13124, memorandum dated October 7, 2013, 8p. Appendix J of the MOP Application.

- Steve Heck and Jeffrey Bell from Bison Engineeringfor meteorological data (Sections 20.2.1 and 20.8.2).
  - Bison Engineering, Inc. (2018-2023). *Black Butte Copper Project Ambient Air Monitoring Program Quarterly Data Report*. Prepared quarterly for Sandfire Resources America, Inc.

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 Reserves have relied on SRA for all taxation related matters in Sections 21 and 22.

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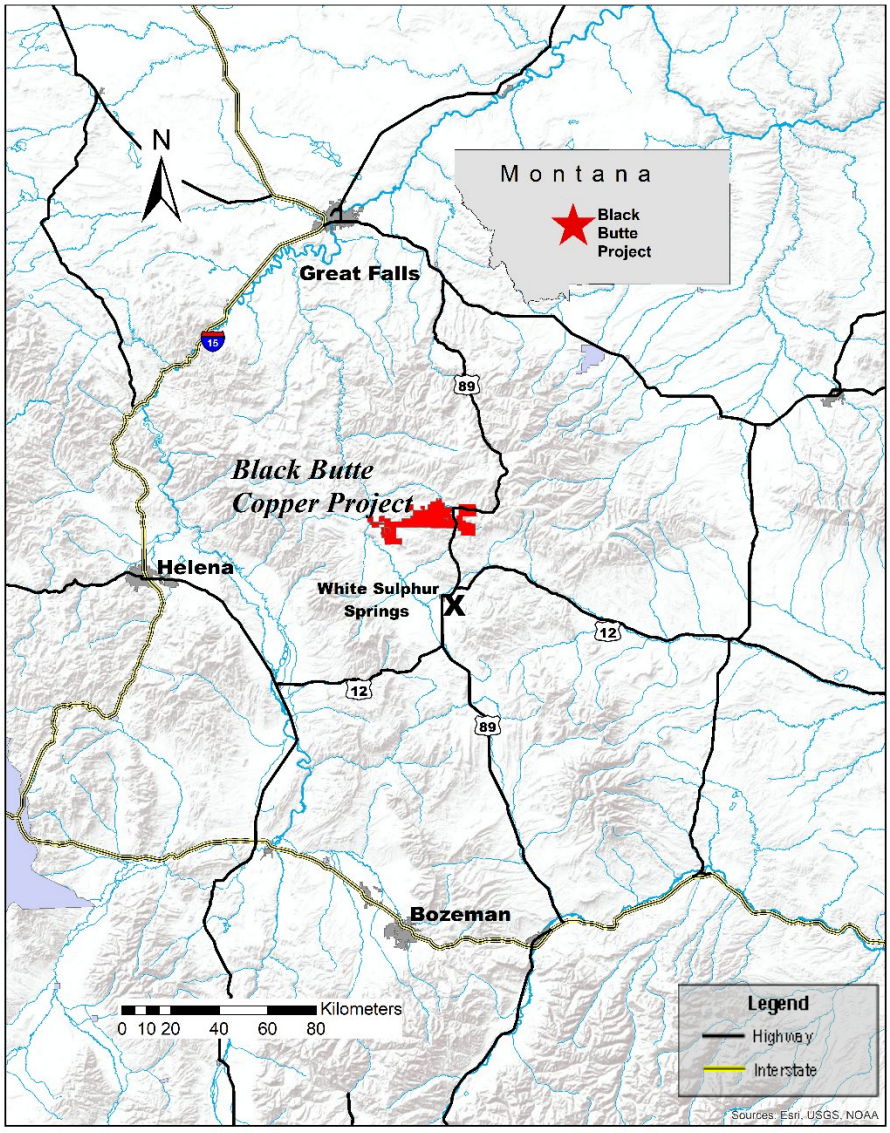
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4. PROPERTY DESCRIPTION AND LOCATION

4.1 Location

The Black Butte Copper Project is located within Meagher County, Montana, USA, approximately 27 km north of the town of White Sulphur Springs (Figure 4.1). US Highway 89 is located 1 km east of the Lowry deposit and 3 km east of the Johnny Lee deposit. (Figure 4.2).



Source: SRA, 2026 compiled from Esri, USGS, NOAA

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 7,684.3 ha (18,986.8 acres) of fee simple lands under mineral lease by SRA through Tintina Montana Inc. (collectively referred to as SRA), and 1,040 unpatented mining claims on USFS lands covering approximately 8,077.8 ha (19,960.4 acres) (Table 4.1, Figure 4.2).

Tract	Surface Estate	Mineral Estate	Agreement Date	Experation Date of Agreement*	Acres	Hectares
Black Butte Mine Property	Tintina Montana Inc.	Hanson, Hanson, Dupea	Dec-20	May 2040	535	217
	Short, Joy	Short (15%), Davis (85%)	Dec-20	May 2040	29.2	11.8
Bar Z Ranch	Bar Z Ranch, Inc.	Hanson, Hanson, Dupea	May-10	May 2040	2,021	818.1
Short, Joy	Short, Joy	Short (15%)	Nov-14	May 2040	2,091	846
		Davis (85%)		May 2040		
Buckingham	Buckingham	Buckingham, Johnston, Bodell	Jun-11	Jun 2041	2,970	1,201.9
Thorson Ranch LLC	Thorson Ranch LLC	100% Thorson Ranch LLC	Oct-18	Oct 2028	3,442	1,393
	Thorson Ranch LLC	50% Thorsen Ranch LLC	Oct-18	Oct 2028	7,615	3,081.7
Patented claims - S26;T12N, R6E surface	Bar Z Ranch, Inc.	Barfuss, Lower & Lower	May-10	May 2040	38	15.4
Patented claims - S26;T12N, R6E minerals	Bar Z Ranch, Inc.	Barfuss, Lower & Lower	Aug-23	Aug 2053	38	15.4
Patented claims - S34;T12N, R6E	Barfuss, Lower & Lower	SRA leases sub-300' from Barfuss, Lower & Lower	Aug-23	Aug 2053	207.6	84.00
24 unpatented mining claims	US. Forest Service	SRA leases sub-300' from GCC Trident LLC	Apr-23	Aug 2053	268.4	108.6
1016 unpatented mining claims	US Forest Service	Controlled by SRA	Various staking dates	Annually on August 31st	19,692	7,969.2

Table 4.1 ***Summary of mineral lands held by the Black Butte Copper Project*** *(*All leases have the option to renew at the end of the term)*

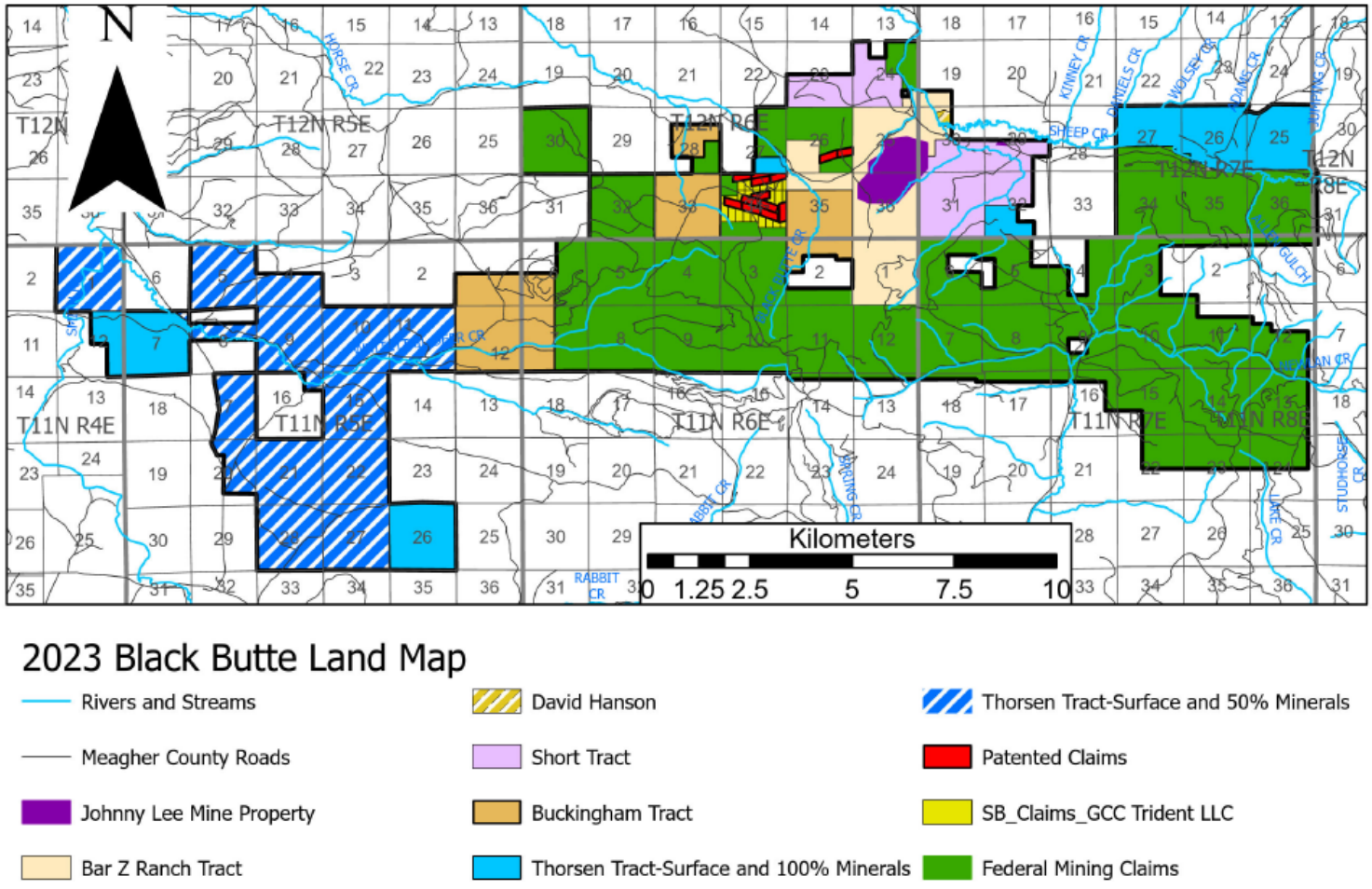
The Black Butte Project controls land and/or mineral rights in all or some part of 75 sections (Table 4.2). The Mine Operating Plan (MOP) application for the Johnny Lee deposit is located on two parcels of fee simple land under mineral lease: The Bar Z Tract and the Short Tract (Figure 4.2, Figure 4.3). 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. Joy Short, a local resident, 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.

Township	Parts or all of Sections
T11NR4E	1, 12
T11N R5E	1,4,5,7,8,9,10,11,12,15,16,17,20,21, 22, 26,27 and 28
T11N R6E	1,2,3,4,5,6,7,8,9,10,11, and 12
T11N R7E	1,2,3,4,5,6,7,8,9,10,11,12,13,14,15,16,17,18,22,23, and 24
T12N R6E	23,24,25,26,27,28,30,32,33,34,35,36
T12N R7E	25,26,27,29,30,31,32,34,35,36

Table 4.2 Legal description of lands of the Black Butte Project

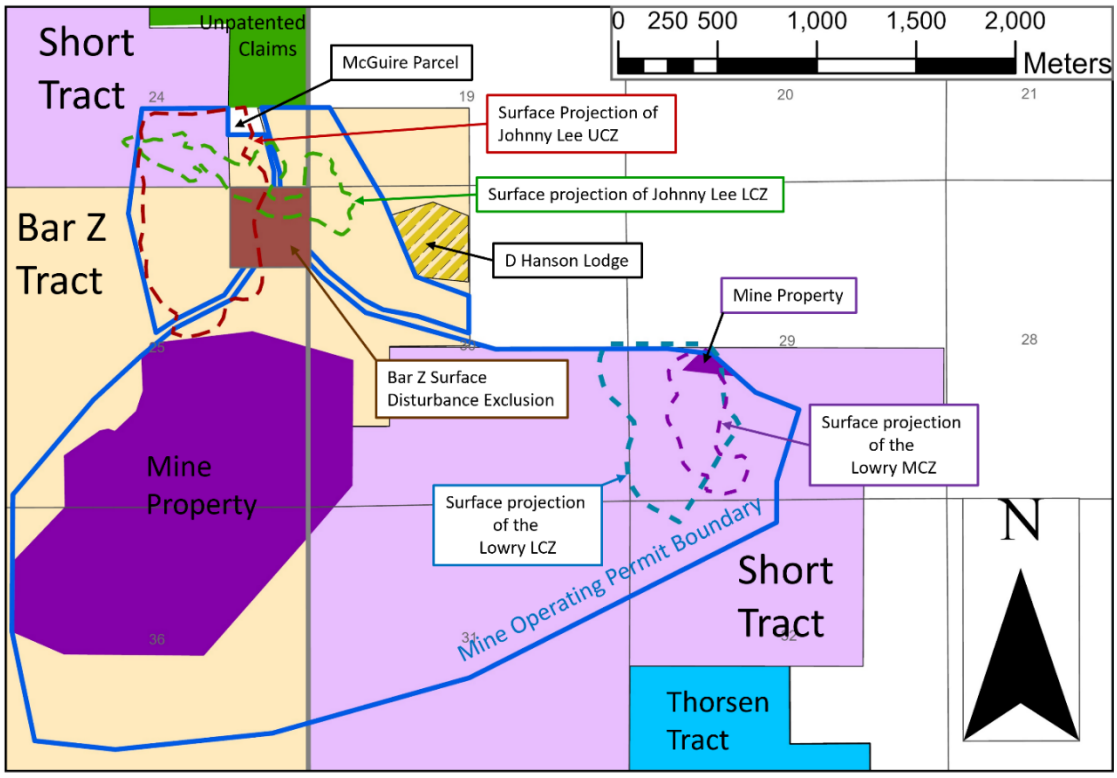
The original provisions of the agreements for both the Bar Z Ranch Tract and Short Tract allowed for, after the publication of a Feasibility Study, the designation of Mine Property for land that would be used exclusively for mining purposes. Mine Property designation is to be utilized for that portion of the leases expected to be occupied by mine facilities including the mine portal, processing area, and the cemented tailings facility, and other ponds and storage areas. The provision provides the landowner the option of selling said Mine Property to SRA, or to receive upgraded lease payments for SRA to have exclusive use of the Mine Property.

After the publication of the Johnny Lee Feasibility study in 2020 (Malhotra and others, 2020), SRA reached an agreement with Joy Short in March 2021 for increased lease payments on 11.7 ha, (29 acres), her portion of the Mine Property. In July 2021, SRA executed an agreement with Bar Z Ranch to purchase 216.47 ha (534.92 acres) of surface for the Mine Property. The Mine Property under these agreements is shown if Figure 4.3. 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. They own 100% of the mineral estate in the vicinity of the Black Butte Copper Project resource area, and 50% of the mineral estate in much of the rest of their ranch (Figure 4.2).



Source: SRA, 2023

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.***



Source: SRA, 2026

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 eight 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.

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 (Cook, 2019; Cook, 2019a).

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 833.25 ha (2,059 acres) owned by the Bar Z Ranch. SRA purchased from Bar Z the surface rights for the Mine Property Area (MPA; 216.48 ha; 534.92 acres) in an agreement dated July 2021, and under the terms of agreement must offer said property back to Bar Z at the end of SRA operations and reclamation. Included in the Surface Lease are surface rights to the Copper Hill (Mineral Survey #10311) and Rio Tinto (Mineral Survey #10304) patented mining claims (Figure 4.2) which combined contribute 13 ha (32 acres) to the total SUA amount. SRA has three separate Mineral Lease Agreements covering 2562 mineral acres of the combined Bar Z tract and MPA with each of the members of the Hanson family who each own equal portions of the mineral interest. The terms are for 30 years which is renewable for subsequent periods of 10 years each. The mineral rights for the Rio Tinto and Copper Hill patented claims are owned by Lower, Lower, and Barfuss with whom SRA has a mining lease agreement executed in August 2023, discussed further below.

In June 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. Arthur Short has passed on leaving Joy Short as sole owner of the surface estate and minerals life estate for the Short tract. 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. In 2021, Joy Short elected to receive increased rental payments in compensation for SRA's designation of 11.82 ha (28.62 acres) as Mine Property. The SUA with Joy Short was amended in August 2024, to achieve an overall rental rate consistent with others in the area. The mineral estate for the Short 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 0.4 ha (5 acres) area adjacent to the Bar Z Tract, on which SRA has no Mineral or Surface Use Agreement. The northeast corner of the JL UCZ projects to surface and outcrops in this area. The portion of the JL UCZ that occurs within the McGuire parcel is excluded from this PFS.

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. The Buckingham tract has no known mineral resources.

The Thorson Ranch tract consists of 4,466.53 ha (11,037 acres) of which 3,081.69 ha (7,615 acres) includes 50% minerals and the remaining 1,384.84 ha (3,422 acres) includes 100% mineral interest. These lands are subject to a Mineral Lease Agreement dated June 2017 and includes lands covering the eastern and western extremities of the leases held by SRA for the Black Butte Copper project as well as two small parcels, one 32.37 ha (80 acre) parcel which lies about two miles west of the Johnny Lee deposit and a 80.94 ha (200 acre) parcel which lies 0.8 km (½ mile) south of the Lowry deposit. The Thorson Ranch tract currently has 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.

GCC Trident LLC leases eleven patented mining claims and controls 24 unpatented Federal lode mining claims in section 34, T.12 N. R.6 E., and mines iron oxide from the eleven patented claims owned by Lower, Lower, and Barfuss. In April of 2023, SRA leased the mineral rights below 300 ft from the unpatented claims held by GCC Trident LLC, and in the agreement GCC agreed to relinquish their rights at depths greater than 300 ft for the eleven patented claims. 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. These annual maintenance fees are currently $200 per claim per year. There are no royalty or lease obligations on these claims. SRA has leased the mineral rights at depths greater than 300 ft under the patented claims from their owners.

In August 2023, SRA executed a mining lease agreement with Lower, Lower, and Barfuss for the mineral interest below a depth of 300 ft for the eleven patented claims in section 34, T.12 N. R.6 E. and the entire mineral interest in two additional patented claims (Copper Hill and Rio Tinto) in section 26, T.12 N. R.6 E.

4.3 Royalties, Agreements and Encumbrances

4.3.1 Bar Z Tract

The Surface Use Agreement required lease payments of $50,000 on signing (2 May 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 the Surface Use Agreement, SRA has the right to designate portions of the property “which are necessary for the construction and operation of a mining and milling operation” as Mine Property and exclude any agricultural or other activity on such lands. The landowner has the options to either require an increased

rental rate from SRA for Mine Property, or transfer title to those lands to SRA in return for compensation 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 to some or all 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). Because these buildings overlie the Johnny Lee UCZ any mine design must ensure their integrity.

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 $75,000 per year and contains the same Mine Property clause as the Bar Z Surface Use Agreement. The landowner may ask for increased rentals or a transfer of title for any lands SRA deems as Mine Property. For a title transfer, 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. 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. The Short Tract also is governed by a Surface Use Agreement for which SRA pays $75,000/year. In addition, 11.6 ha (28.62 acres) of the Short tract are leased as part of the Mine Property Area at a rate of $150/acre/year.

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.

The Thorson Ranch Mineral Lease Agreement (dated June 2017) 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.

The GCC Trident agreement (April 2023) covers 24 unpatented lode mining claims on USFS lands in section 34, T.12 N., R. 6 E and requires a rental payment of $5,000/year. The lease grants SRA rights to minerals at a depth greater than 300 ft. GCC receives no royalty on SRA production from these claims. The agreement has a provision for surface access necessary for SRA's activities so long as it doesn't interfere with GCC iron oxide mining operations.

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 $442,147 for completion of the reclamation of earlier drill programs. SRA has also posted a surety bond of $607,419 for reclamation of planned drilling. These obligations will be released when the reclamation is completed by SRA and inspected and approved by the MDEQ.

Under Mine Operating Permit #00188, SRA currently holds a surety bond of $4,653,347 for reclamation of Phase I Construction activities, including ponds, borrow area and reclamation material stockpiles, roadways, and waste rock storage pad. This bond is insured by Payne-West Insurance.

4.4.2 Required Permits and Status

4.4.2.1 Mine Operating Permit

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 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 4.3. 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, 2017b). 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 13 March 2020.

The MDEQ then issued a ROD on 9 April 2020 (MDEQ, 2020b) 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, 2020b) and in Section 20 of this Report. The Montana Hard Rock Mining Bureau issued the final Mine Operating Permit on 14 August 2020 (MDEQ, 2020d).

On 4 June 2020, conservation groups filed an appeal of the ROD in state district court. On 16 July 2022, the district court ruled in favour of the plaintiffs and vacated the MOP with the provision that Tintina could still complete Phase I construction. In November of 2022, Tintina filed an appeal of the district court ruling to Montana Supreme Court. The MOP was fully reinstated by the Montana Supreme Court on 26 February 2024, in a 5-2 decision. The Montana Supreme Court granted the Company's request for summary judgement allowing Black Butte Copper to move forward with construction.

Table 20.14 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.

4.4.2.2 *Water Rights*

The Company has gained the water rights necessary to mitigate for its consumptive use of water in the project. The Montana Department of Natural Resources and Conservation (MDNRC) requires mitigation for the consumptive use of water for the Project and requires that in a basin closed to granting of any additional water rights such as the Smith River basin, in which the Project is located, one must find a way to retire other water use as part of any mitigation plan. MDNRC approved the Black Butte Copper mitigation plan which involved revision of six existing irrigation water rights under lease to the Company, issuance of a high water right, and issuance of a ground water right. Conservation groups objected to the plan, after which the Company and objectors negotiated stipulations to resolve most of their issues. The conservation groups refused to resolve a remaining issue, whether or not mine dewatering should be considered a 'beneficial use' and thus require mitigation through retirement of other water use. In an initial hearing on the matter, a Hearings Examiner reviewed the objection and ruled that Montana Law dictates that mine dewatering is not a beneficial use, consistent with the MDNRC interpretation in the approved Black Butte Copper mitigation plan, and therefore upheld the MDNRC decision to grant water rights to the Company. Conservation groups appealed the decision to Montana district court, where the reviewing judge dismissed the case. Conservation groups have appealed the dismissal to Montana Supreme Court. The Montana Supreme Court heard oral arguments on the appeal on 29 March 2024 and on 2 January 2025, the Montana Supreme Court in a 5-2 decision ruled in favor of Tintina Montana and the DNRC affirming the district court's determination that mine dewatering is not a beneficial use of water. There are no remaining legal challenges related to the project.

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	

Section	Rules (ARM)/ACT (MCA) Citation
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	
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	

Section	Rules (ARM)/ACT (MCA) Citation
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	
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3.7.1 Introduction	ARM 17.24.116(3)(k); MCA 82-4-336(5)
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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)
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3.7.5 Treated Water Disposition	
3.7.6 Storm Water	
3.7.7 Erosion Control & Best Management Practices (BMP)	
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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)

Section	Rules (ARM)/ACT (MCA) Citation
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)
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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	
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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)
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

Section	Rules (ARM)/ACT (MCA) Citation
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),
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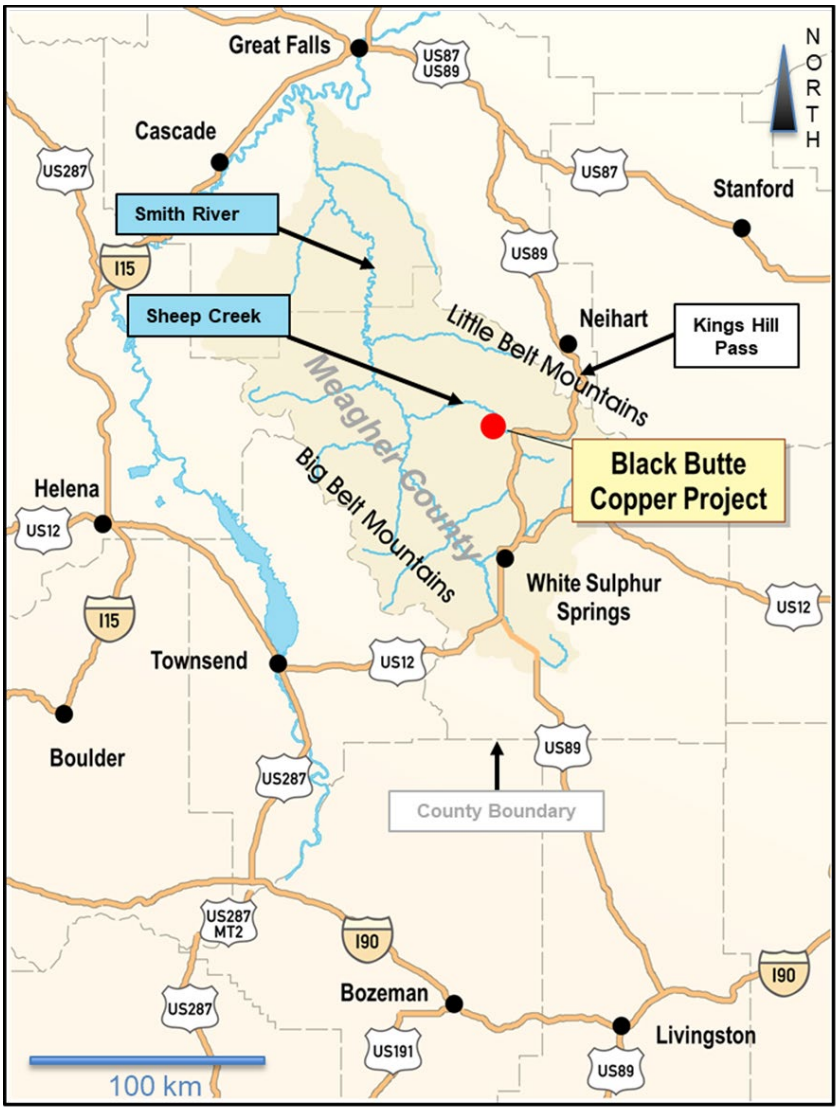
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5. ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

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 a final 4 km on 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.



Source: SRA, 2026

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 955 people (2020 US Census Bureau Decennial Census) and is the county seat of Meagher County, estimated population 1,927. White Sulphur Springs is 122 km from the state capital Helena (population 32,091), 127 km from Bozeman (population 53,293) and 158 km from Great Falls (population 60,442). All 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 m 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 northeast 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 sagebrush 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, respectively, 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. Most recent monitoring results from 2018 to 2023 are summarized in Table 5.1.

	2018				2019				2020				2021				2022				2023			
	Temperature (˚C)			Precipitation	Temperature (˚C)			Precipitation	Temperature (˚C)			Precipitation	Temperature (˚C)			Precipitation	Temperature (˚C)			Precipitation	Temperature (˚C)			Precipitation
	Min.	Max.	Avg.	(mm)	Min.	Max.	Avg.	(mm)	Min.	Max.	Avg.	(mm)	Min.	Max.	Avg.	(mm)	Min.	Max.	Avg.	(mm)	Min.	Max.	Avg.	(mm)
January	-26.3	7.2	-7.2	-	-30.2	7.2	-8.1	0.3	-26.7	4.6	-5.6	-	-27.7	5.6	-7.6	10.7	-34.7	5.4	-8.0	3.1	-34.9	6.1	-9.8	21.3
February	-21.2	2.9	-8.3	-	-36.4	5.7	-16.5	-	-26.4	9.6	-7.7	-	-39.9	7.4	-12.8	21.8	-37.5	6.4	-9.7	5.6	-34.4	4.7	-9.3	4.1
March	-18.2	7.8	-3.9	2.3	-39.3	8.4	-9.1	11.2	-23.8	10.8	-4.3	-	-22.0	12.2	-3.4	7.6	-31.3	14.8	-2.9	7.9	-25.1	4.8	-8.9	9.4
April	-	-	-	-	-18.5	16.7	1.2	36.1	-18.7	20.8	-0.2	38.6	-14.8	21.1	0.7	17.5	-19.6	19.0	-1.9	18.0	-23.1	19.5	-0.8	10.7
May	1.6	22.5	11.0	16.3	-19.9	20.8	5.4	23.4	-7.8	22.4	6.7	73.4	-10.0	18.5	4.6	27.2	-9.8	20.3	5.5	25.7	-4.6	23.1	9.5	90.7
June	-1.7	24.5	12.6	2.5	-2.8	23.6	10.1	-	-1.5	25.3	10.6	-	-2.3	30.1	14.4	6.9	-1.7	28.5	11.1	65.5	-4.5	23.7	11.1	101.9
July	0.8	28.8	15.5	1.0	-3.3	28.0	14.0	3.1	-1.7	28.8	14.0	-	0.1	32.3	18.2	10.9	-0.2	30.2	16.9	12.7	-0.6	30.7	15.1	8.4
August	-3.2	34.0	13.4	12.7	-2.5	28.2	13.9	34.3	-4.3	30.4	14.7	10.2	-2.3	28.9	13.7	61.7	1.7	30.7	18.4	1.3	0.4	30.6	14.6	57.7
September	-5.4	26.5	8.1	47.0	-6.8	27.6	8.7	72.1	-9.8	29.7	9.5	3.3	-8.6	28.1	9.1	1.3	-5.7	32.6	12.4	25.4	-4.7	26.4	9.9	101.9
October	-8.3	18.1	2.0	34.5	-	-	-	-	-31.7	22.2	0.8	34.3	-15.4	23.0	3.7	11.7	-	-	-	-	-29.7	21.3	1.3	15.8
November	-26.5	9.0	-5.6	29.00	-	-	-	-	-19.2	17.5	-2.6	9.4	-19.1	13.5	-0.5	10.4	-	-	-	2.0	-18.3	11.2	-2.1	32.3
December	-27.1	7.4	-7.4	2.3	-	-	-	-	-23.2	9.6	-6.0	3.6	-32.2	13.9	-8.0	3.1	-41.3	6.2	-10.6	10.2	-23.0	11.3	-4.5	5.3

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 underground 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). A new 100 kV power line will be installed by FEC from the existing Northwestern Energy Kings Hill power line, located north of the Bair Reservoir along US-12. The new line will follow approximately 22.5 km of US-12 to the junction of US-89 and then follow US-89 north approximately 24 km to the mine site (see Figure 5.1).

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 high-speed 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 underground 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 has obtained a ground water right and a high water right, contingent on an approved mitigation plan, from the Montana Department of Natural Resources and Conservation (MDNRC).

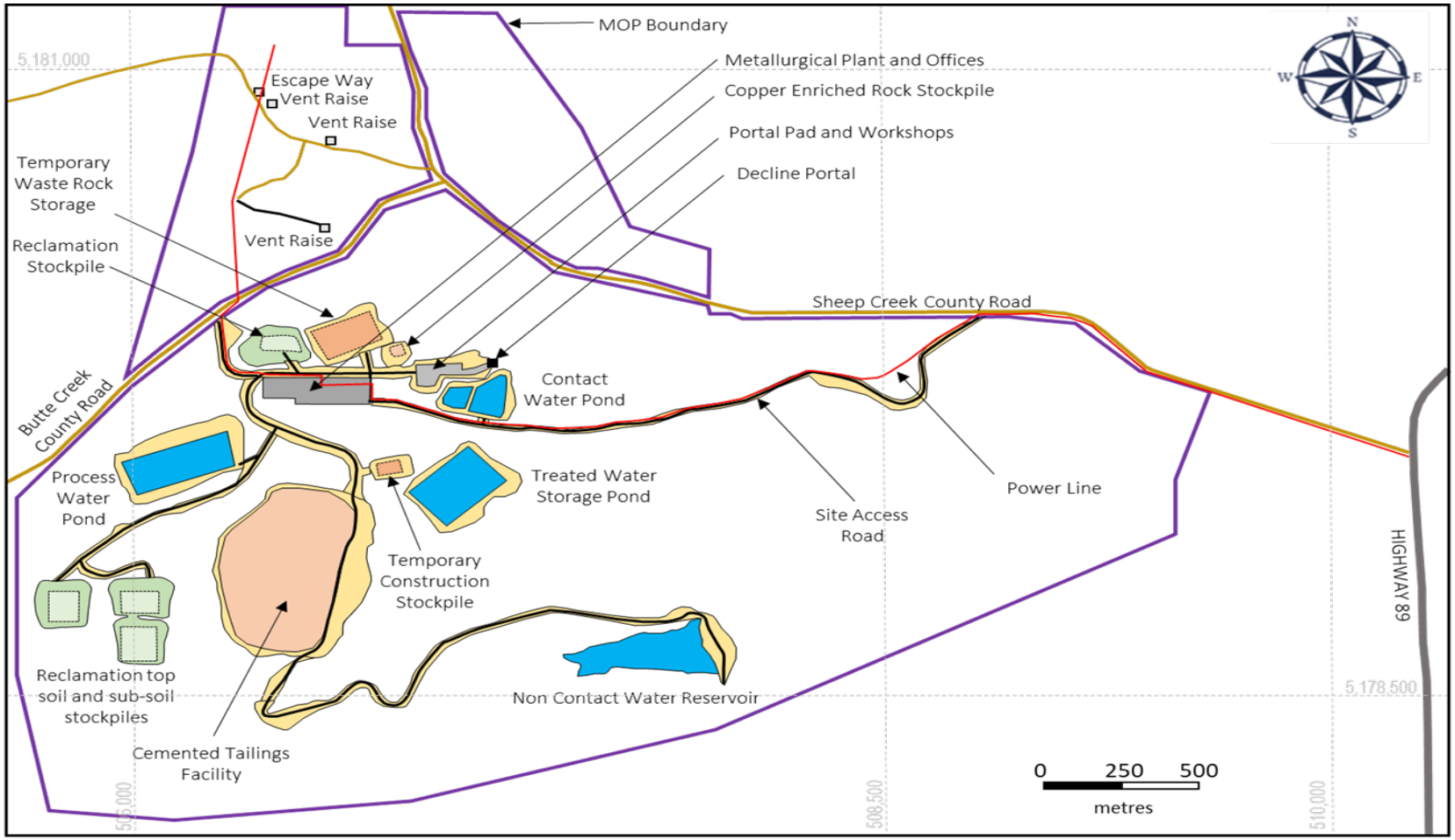
5.7 Surface Ownership

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 450 workers will be required during construction and 270 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.



Source: SRA, 2026

Figure 5.2 Proposed Infrastructure Layout for the Black Butte Copper Project

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6. HISTORY

The first recorded effort toward mineral development on the property was by Messrs. Weir and Tyler, local ranchers, who investigated 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 sunk a 15 m deep shaft on the Volcano Valley fault zone (Pers. Comm., Hanson family) at the same location as the Virginia Mine. He continued to work the property intermittently until 1922. Based on observations in and around the shaft, Sandfire Resources America, Inc. (SRA) geologists interpret the workings to have encountered a small block of weakly copper mineralized rock structurally transported along the Volcano Valley fault from an unknown location presumably at lower elevation further west and entirely oxidized by near surface weathering. The mineralization that forms the Johnny Lee deposit has been named in recognition of Mr. Lee's efforts.

Until 1973, commercial exploration interest was focused on iron contained by ferruginous gossans in the vicinity of the Black Butte 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 4.2). The small-scale development of the iron ore deposits focused on production of iron oxide for industrial use. No copper mining has been undertaken within the project area.

6.1 Prior Ownership and Ownership Changes

The leases that form part of the Black Butte property were originally homestead tracts, railroad lands and patented mining claims 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 consists of unpatented claims on USFS land.

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

Cominco American Inc. (CAI) acquired federal mining claims and leased a number of private 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 joint venture (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 purchased Exxon Minerals' interest in late 1984. 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 in 1995, CAI dropped all the leases. In 2010, Tintina Resources Inc. (Tintina) acquired some of the mineral rights formerly owned by CAI and began data compilation and definition drilling in the resource areas.

Sandfire Resources NL acquired a controlling interest in Tintina in 2014. Tintina Resources was subsequently renamed Sandfire Resources America Inc. The U.S. business entity, 100% owned by Sandfire Resources American Inc., is Tintina Montana Inc.

6.2 Historical Exploration Chronologically by Company

Homestake conducted geologic mapping, soil sampling, and trenching targeting Pb-Zn mineralization in ferruginous gossans. Homestake completed one drillhole, sited over 2 miles southwest of the Johnny Lee deposit, before abandoning the district in 1974.

Anaconda commenced exploration in 1977 and from 1978 to 1979 drilled 12 drill holes, some of which encountered bedded pyritic sulfides, but none encountered economically important concentrations of Zn or Cu mineralization.

Cominco American Incorporated (CAI) staked unpatented mining claims and leased land in the Black Butte area in 1976. CAI completed 14 drillholes in the Black Butte area from 1978 to 1984. Some of these drillholes intersected large thicknesses of bedded pyrite with zones of elevated (>1%) Zn and Cu, but no commercially significant 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 five drillholes recommended by CAI geologists. In the second of these drillholes, SCC-17, UII intercepted both the Upper Copper Zone and Lower Copper Zone of the Johnny Lee deposit. The Upper Copper 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 from 1985 until early 1988 drilling a total of 33 drillholes; at which point they had fulfilled their earn-in agreement for a 50% interest and JV operatorship passed to CAI. In 1990, CAI purchased BHP's 50% interest with no retained royalties. CAI continued to drill the Johnny Lee deposit.

In 1990, CAI discovered the nearby Lowry deposit by deepening an earlier BHP drillhole through the Volcano Valley fault zone where the hole encountered 10 m of 2.8% copper in a well mineralized Lower Copper Zone, similar to the Johnny Lee Lower Copper Zone. As drilling continued for the Lower Sulfide Zone in the Lowry area, CAI drilled 52.1 m of 2.7% copper in a new copper deposit stratigraphically below the Upper Sulfide Zone yet above the Volcano Valley fault, which they called the Lowry Middle Copper Zone. CAI drilled an additional 28 drillholes before abandoning the project in 1995.

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

From 2014 to 2019, SRA focused on gathering environmental, geotechnical, hydrogeological, metallurgical, and geological data to finalize the Johnny Lee Mine Operating Plan, provide data for a Mineral Resource estimate, and complete a feasibility study. In 2020, SRA conducted a 2-D seismic study to generate a three-dimensional geological model of the Johnny Lee and Lowry area.

In early 2021, SRA completed 4,414 meters of core drilling in 8 exploration holes focused in the aera between Lowry and Johnny Lee and to the west of the Johnny Lee Zone. In late 2021, SRA began a drill program to improve the definition of the Lowry Middle Zone and Lowry Lower Zone. This drilling lasted until March of 2022 and completed 8,936 meters of core in 19 drill holes. In December of 2023, SRA began a drill program designed to expand the Johnny Lee Lower Copper Zone resource. This program finished in May 2025, after drilling of 28,099 meters in 79 holes. Total drilling at the time of this writing is 562 holes across the Black Butte district for a total of 189,468 meters.

6.3 Historical Mineral Resource and Reserve Estimates

Historical estimates in this Section are provided to demonstrate the trajectory of the Black Butte Copper Project over time. The historical estimates are relevant in the context of the Project's history and provide a comparison to the current Mineral Resources and Mineral Reserves that are the subject of this Report. A qualified person has not done sufficient work to classify any of the historical estimates as current Mineral Resources or Mineral Reserves. The issuer is not treating any of the historical estimates as current Mineral Resources or Mineral Reserves.

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

Tintina 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 historical estimates are documented below.

SRA does not consider these historical estimates to be current mineral resources, as these have been superseded by the mineral resource estimates of this technical disclosure.

6.3.1 2010 Johnny Lee Mineral Resource Estimate

In 2010, Tintina 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 historical Inferred Mineral Resource (Table 6.1) for the Johnny Lee Upper Copper Zone (JL UCZ) and Johnny Lee Lower Copper Zone (JL LCZ) (Lechner, 2011). A qualified person has not done sufficient work to classify any of the historical estimates as current Mineral Resources or Mineral Reserves. The issuer is not treating any of the historical estimates as current Mineral Resources or Mineral Reserves.

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	2.4	0.1	12.3	166,018	8,618	2,800
Johnny Lee LCZ	Inferred	2,462	4.7	0.1	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, Tintina 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 Preliminary Economic Assessment (Winckers *et al*, 2012). Table 6.2 summarizes the historical 2012 Mineral Resource Estimate. A qualified person has not done sufficient work to classify any of the historical estimates as current Mineral Resources or Mineral Reserves. The issuer is not treating any of the historical estimates as current Mineral Resources or Mineral Reserves.

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	3.0	0.1	16.9	241,769	9,979	4,609
	Inferred	1,257	2.6	0.1	16.4	33,113	1,361	663
Johnny Lee LCZ	Inferred	2,462	4.7	0.1	5.1	116,112	1,315	404
Lowry	Inferred	5,139	2.6	0.1	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 Preliminary Economic Assessment Mineral Resource Estimate

In 2013, Tetra Tech Inc. updated the Preliminary Economic Assessment using a new Mineral Resource estimate that incorporated results from an additional 65 drillholes (Winckers *et al*, 2013). Table 6.3 summarizes this historical 2013 Mineral Resource Estimate. A qualified person has not done sufficient work to classify any of the historical estimates as current Mineral Resources or Mineral Reserves. The issuer is not treating any of the historical estimates as current Mineral Resources or Mineral Reserves.

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	3.0	0.1	16.3	79,380	3,130	1,393
	Indicated	6,250	2.8	0.1	15.5	180,533	8,165	3,249
	Inferred	1,255	2.5	0.1	15.2	31,752	1,270	613
Johnny Lee LCZ	Indicated	2,387	6.4	0.0	4.5	152,863	771	345
	Inferred	205	5.3	0.0	4.1	100,000	45	27
Lowry	Indicated	4,099	2.9	0.1	15.1	120,658	4,082	1,990
	Inferred	801	2.6	0.1	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 Johnny Lee Mineral Resource Estimate

In 2019, the Mineral Resource was updated for the Johnny Lee deposit (Ronald and Malhotra, 2019) and was supervised and reviewed by SRK Consulting (U.S.), Inc. ("SRK"). Table 6.4 summarises this Mineral Resource estimate, which is re-stated with minor edits to the significant digits of grade.

The 2019 Mineral Resource estimate represents the following notable changes from the previously defined Mineral Resource estimate contained within the 2013 Updated Technical Report and Preliminary Economic Assessment for the Black Butte Copper Project, Montana (effective date: 12 July 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, Si, or Au in payable concentrations, and
- Updated cut-off grade used to define the Mineral Resource using recent metallurgical recovery assumptions for copper and updated price assumptions.

The historical 2019 Mineral Resource estimate for Johnny Lee was used for the 2020 Johnny Lee FS. A qualified person has not done sufficient work to classify any of the historical estimates as current Mineral Resources or Mineral Reserves. The issuer is not treating any of the historical estimates as current Mineral Resources or Mineral Reserves.

Mineral Resource Category	Quantity (Mt)	Cu (%)	Total Metal (kt)
Measured + Indicated (M+I) Mineral Resources			
JL UCZ			
Measured	1.4	2.6	36.2
Indicated	8.3	2.3	191.3
Sub-Total M+I	**9.7**	**2.4**	**227.5**
JL LCZ			
Measured	0.6	5.7	32.9
Indicated	0.6	7.9	50.5
Sub-Total M+I	**1.2**	**6.8**	**83.4**
Combined JL UCZ + JL LCZ			
Measured	2.0	3.5	69.1
Indicated	8.9	2.7	241.8
Total M+I	**10.9**	**2.9**	**310.9**
Inferred Mineral Resources			
JL UCZ	2.2	2.3	49.5
JL LCZ	0.5	5.9	30.3
Total Inferred	**2.7**	**2.9**	**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. Copper assay values were capped where appropriate.*
- *Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. 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 copper has been estimated on a block basis in the JL UCZ, averaging 77.4%, with a consistent 94.0% Cu recovery applied to the JL LCZ.*
- *To demonstrate reasonable prospects for eventual economic extraction of Mineral Resources, a cut-off grade of 1.00% copper based on metal recoverability assumptions, long-term copper price assumptions of $3.20/lb, mining costs, processing costs, G&A costs totalling $71/t.*
- *There are no known legal, political, environmental, or other risks that could materially affect the potential development of the Mineral Resources other than those outlined in the Management Discussion and Analyses of the June 2019 Company Quarterly 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 2020 Lowry Mineral Resource Estimate

As part of the Johnny Lee Feasibility Study, the Lowry Mineral Resources were estimated using drilling through 2019. The historical 2020 Lowry Mineral Resource Estimate is provided in Table 6.5. A qualified person has not done sufficient work to classify any of the historical estimates as current Mineral Resources or Mineral Reserves. The issuer is not treating any of the historical estimates as current Mineral Resources or Mineral Reserves.

Mineral Resource Category	Quantity (Mt)	Cu (%)	Total Metal (Cu Mlb)
Lowry MCZ			
Inferred	5.7	2.5	318.7
Lowry LCZ			
Inferred	2.6	2.1	121.3
Combined LMCZ + LLCZ			
Inferred	8.3	2.4	440.0

Source: SRK, 2020

- *The effective date for this Mineral Resource was October 15, 2020. All significant figures were rounded to reflect the relative accuracy of the estimates. Cu assay values were capped where appropriate.*
- *Metallurgical recovery of Cu was 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.*

Table 6.5 Mineral Resource Estimate for the Lowry Deposit as of October 15, 2020

6.5 Historic Production

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

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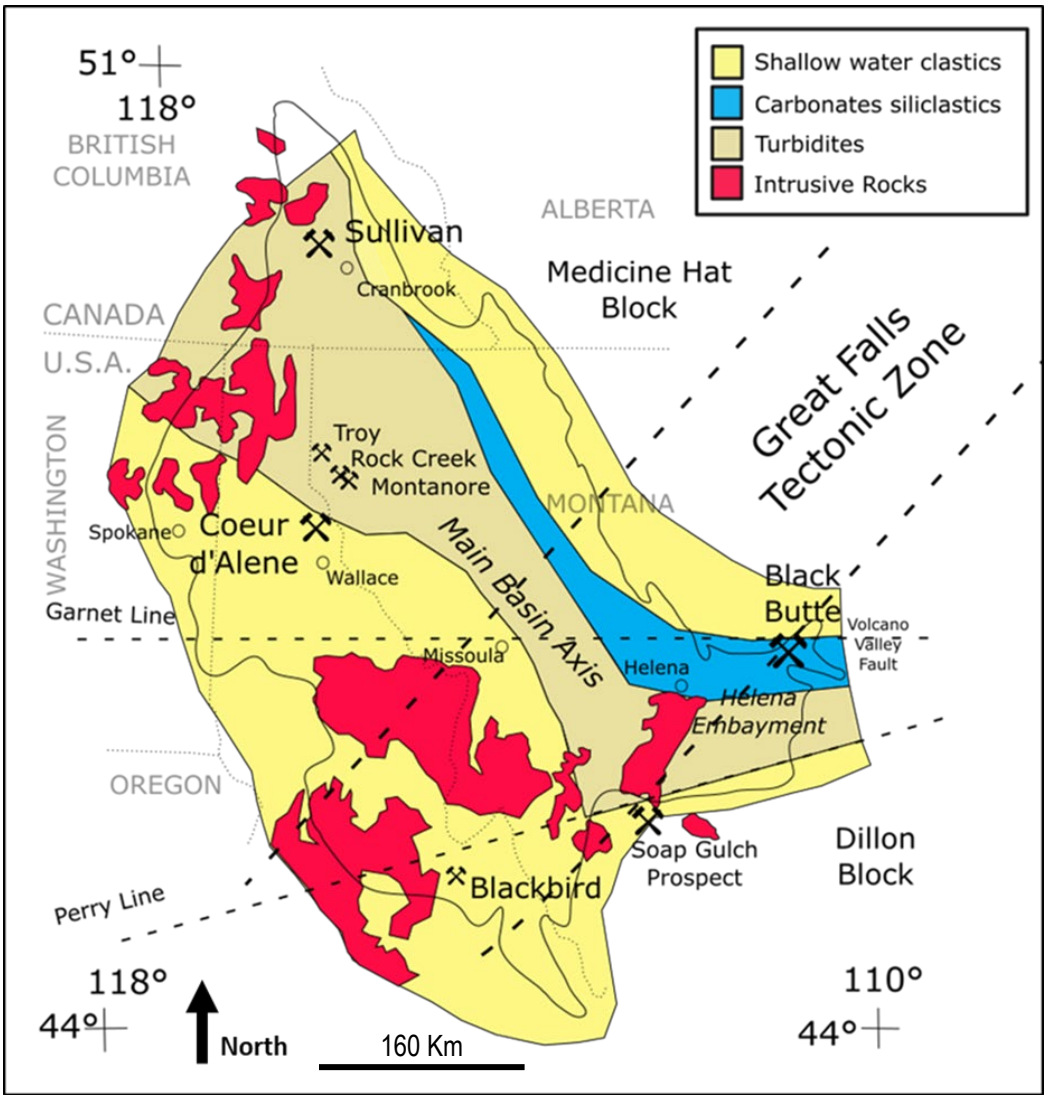
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7. GEOLOGICAL SETTING AND MINERALIZATION

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 or billions of years) 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 southwest Saskatchewan named the Great Falls Tectonic Zone (Figure 7.1).



Source: SRA, 2026 modified from Lydon (2007)

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. Recent U-Pb dating of zircon in a syndepositional volcanic unit in the Lower Belt by SRA marks the beginning of basin filling at least as old as 1,490 Mega annum (Ma or millions of years) (Zieg *et.al* 2024). This basin rapidly filled with a 15 to 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 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.

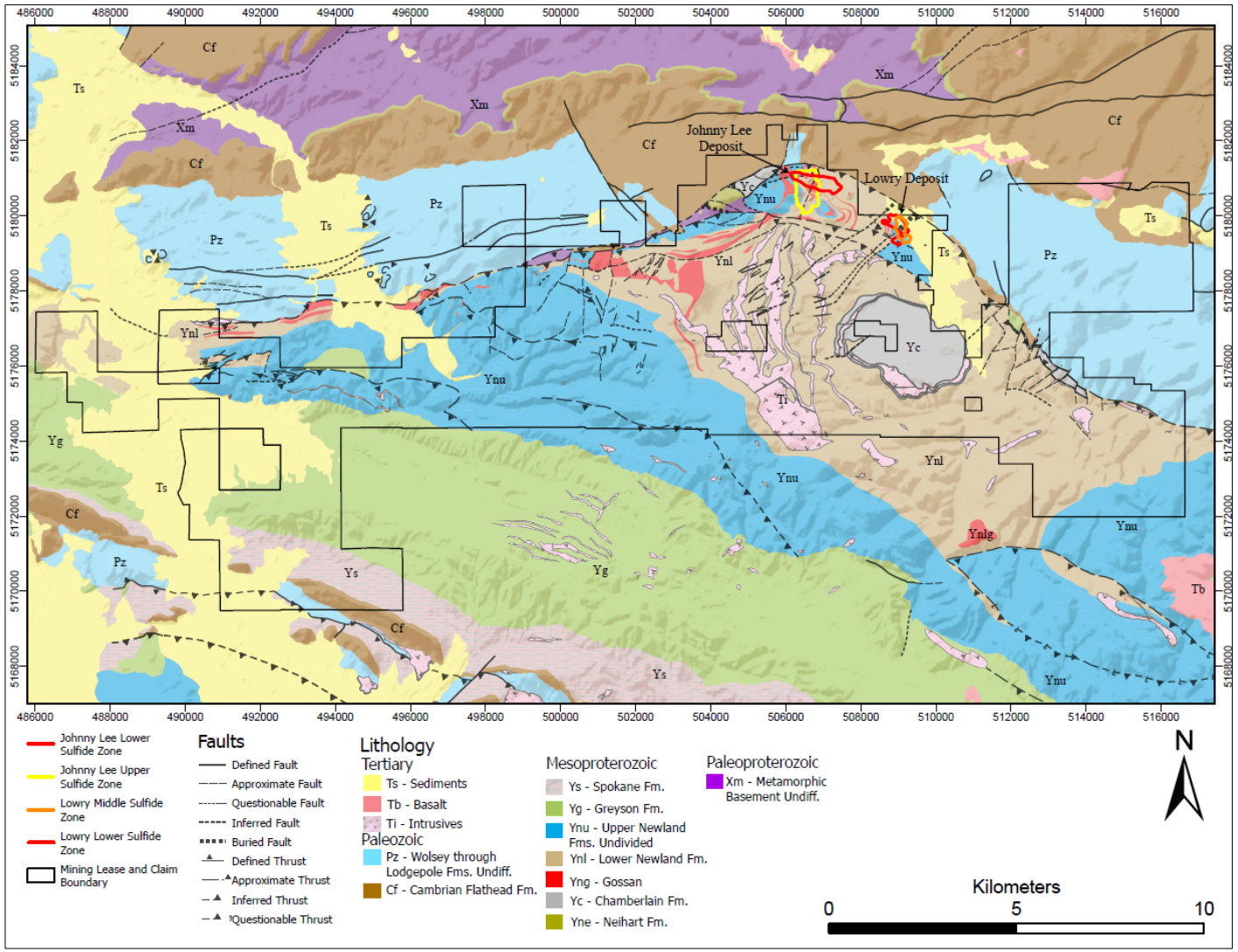
The Belt Basin initially partitioned into at least three sub-basins, the main basin in west Montana, north Idaho, and southern British Columbia and Alberta; the Lemhi basin further south in Idaho; and the Helena Embayment in Montana (Figure 7.1). The Helena Embayment is an EW trending graben bounded by major growth fault zones, principally the Garnet Line on the north, and the Perry Line on the south (Winston, 1986). The Helena Embayment 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).

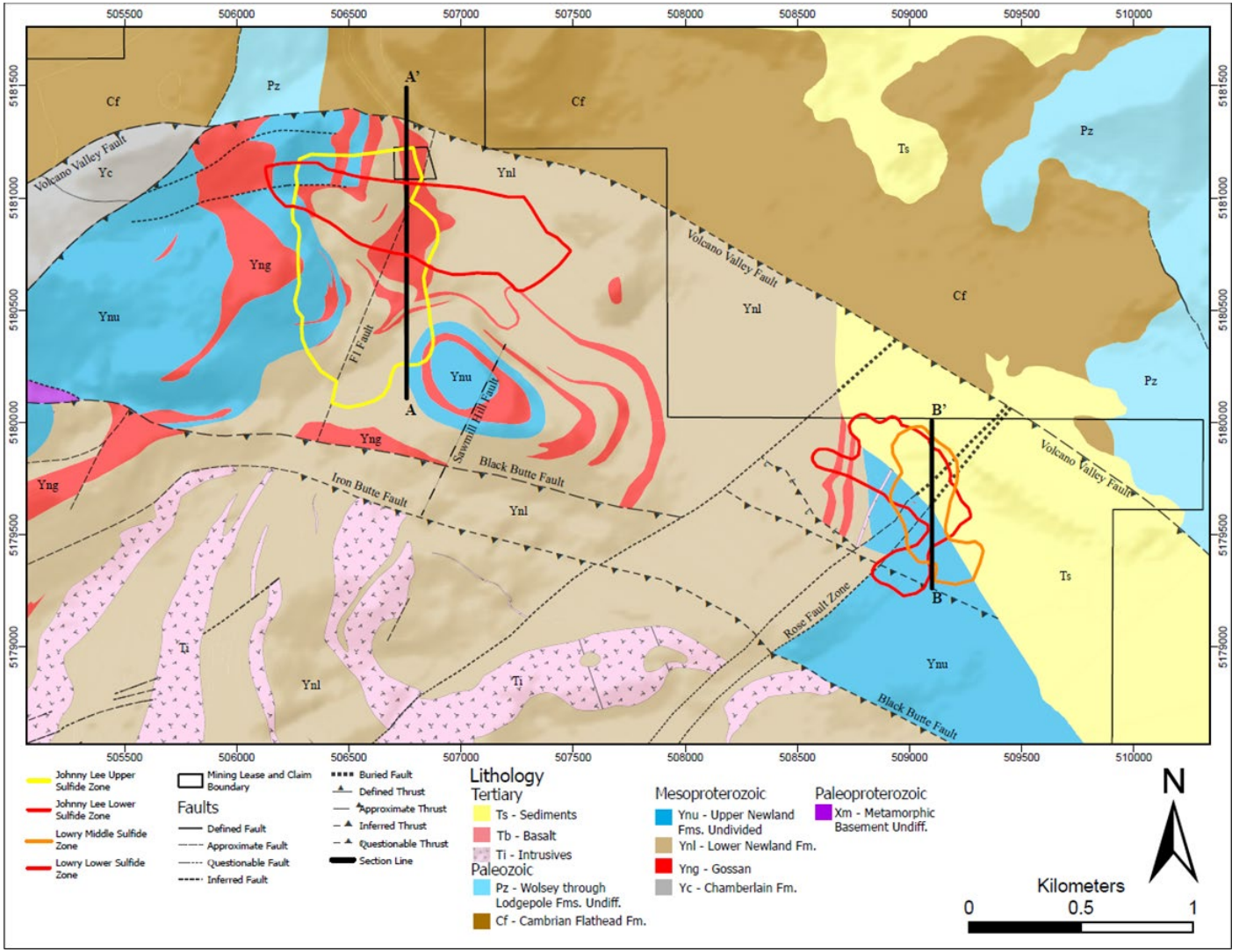
7.2 Local and Property Geology

The majority of the Black Butte Copper Project (BBCP) area is underlain by gentle to moderately southward-dipping rocks of the Belt Supergroup (Figure 7.2). 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. 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. Figure 7.3 shows the local geology around the Johnny Lee and Lowry copper deposits.



Source: SRA, 2026

Figure 7.2 District geological map for the Black Butte Copper Property, showing the property boundary and the locations of the Johnny Lee and Lowry copper deposits

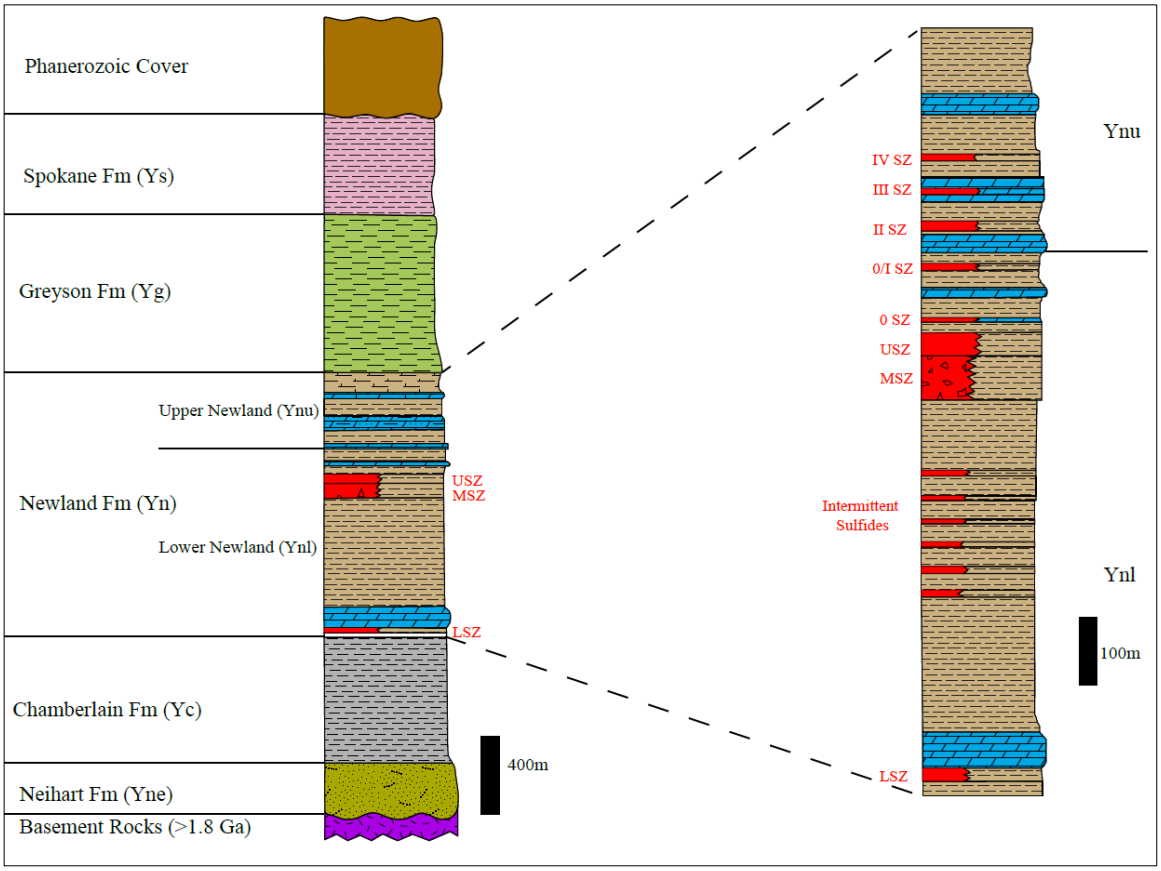


Source: SRA, 2026

Figure 7.3 Geological map of the area of Johnny Lee and Lowry copper deposits

7.3 Lithology and Stratigraphy

The stratigraphy important to the project ranges from a Paleoproterozoic metamorphic basement complex through Lower Belt and Ravalli Group sediments of the Belt Supergroup (Figure 7.4). The Belt Supergroup rocks dip generally south and with greater distance south toward the interior of the Helena Embayment, progressively younger sediments are preserved below the Precambrian – Cambrian unconformity. For this reason, Ravalli Group rocks (Spokane Formation) are not preserved within the BBCP.



Source: SRA, 2026

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

In Figure 7.4, the mineralized horizons are noted as follows: LSZ = Lower Sulphide Zone; MSZ = Middle Sulphide Zone; USZ = Upper Sulphide Zone. Other sulphide zones correspond to names of host stratigraphic units. 0 SZ is the Subzero Sulphide Zone; 0/I SZ = 0/I Sulphide Zone; II SZ = Upper Newland unit II Sulphide zone; III SZ = Upper Newland Unit III Sulphide Zone; and IV SZ = Upper Newland Unit IV Sulphide Zone.

7.3.1.1 *Paleoproterozoic Basement*

Paleoproterozoic basement rocks are exposed very near the Project area and have been intersected in drillholes that penetrate the basal unconformity at the base of the Belt Supergroup. 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).

7.3.1.2 *Mesoproterozoic Neihart Formation*

The lowest unit in the area is the Neihart Formation which unconformably overlies basement rocks (Figure 7.4). While classified as a pre-Belt unit, it shows a gradational contact with the overlying Chamberlain Formation silty shale, a Lower Belt unit. Within the lease boundaries, the Neihart Formation is exposed as fault bounded blocks within the VVFZ (Figure 7.2) but is exposed a short distance north of the property boundary where it rests on Paleoproterozoic rocks. Drilling has intersected the Neihart Formation north of the VVFZ and in one hole south of the Black Butte fault (BBF). Neihart Formation also outcrops along Sheep Creek north of the VVFZ. The Neihart Formation consists of 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. In its type locality at Neihart, 13 miles north of the BBCP, the Neihart Quartzite is described as approximately 270 m thick (Keefer, 1972). It is at least 225 m thick along Sheep Creek north of the VVFZ.

7.3.1.3 *Mesoproterozoic -- Lower Belt Supergroup -- Chamberlain Formation*

The Neihart Formation grades-up into Chamberlain Formation which consists of interlayered carbonaceous silty shale, siltstone, and locally small amounts of sandstone. The Chamberlain Formation is exposed in a fault-bounded block within the VVFZ and in the core of an anticline south of the BBF and southeast of the Lowry area (Figure 7.2).

Drilling has intersected the Chamberlain Formation north of the VVFZ in the Johnny Lee deposit area, where it reaches 180 m thick, and at the north end of the Lowry Zone where it reaches 190 m thick. The upper part of the Chamberlain Formation serves as a termination for most drilling below the Lower Sulphide Zone at both the Johnny Lee and Lowry Copper Zones and is defined by the uppermost silty carbonaceous shale below a transition from Newland Formation shale lithofacies. The Chamberlain Formation records sediment deposition in relatively shallow water (above wave base) conditions.

7.3.1.4 *Mesoproterozoic – Lower Belt Supergroup - Newland Formation*

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

7.3.1.4.1 Lower Newland

The Lower Newland is approximately 800 m thick in the property area, though it thickens considerably to the south. This unit consists mainly of dolomitic to calcareous, and in places carbonaceous shale as well as non-carbonate bearing shale. Beds and even subunits of dolomite, limestone, and conglomerate are common. The shales were deposited as turbidites and microturbudites at the edge of the sedimentary basin. The basal conglomerates of turbidite lithotypes are thickest, most abundant, and coarser grained near the Volcano Valley fault. These conglomerates represent debris flows shed from a shallower water mud shelf north of the resource area. The transition from the shallow water deposition, recorded in the Chamberlain Formation, to much deeper water conditions, as evidenced by Newland Formation turbidites, reflects the onset of significant fault-controlled subsidence of the eastern part of the Belt basin rift. This rifting was the inception of the Helena Embayment part of the Belt Basin. Near the base of the Lower Newland Formation is a dolomite unit called the Hanging Wall Dolomite. Above this, the lower part of the Newland shale has very low carbonate concentrations. Carbonate content increases with higher stratigraphic levels until near its top the lower Newland exhibits fully formed limestone and dolomite units. Numerous layers of pyrite, sometimes in thick accumulations, occur throughout the Lower Newland Formation. These sulphide units range from centimetre-scale layers to units that exceed 100 m in thickness. Shale and dolomite interbeds and conglomerate lenses often occur within the bedded pyrite units.

All economically significant copper mineralization discovered in the Project area to date occurs within massive sulphide units of the Lower Newland. The Johnny Lee Upper Copper Zone (JL UCZ) deposit is hosted by the Upper Sulphide Zone (USZ; locally called the JL USZ when part of the Johnny Lee deposit), and the Johnny Lee Lower Copper Zone (JL LCZ) deposit is hosted by the Lower Sulphide Zone (LSZ; locally called the JL LSZ when part of the Johnny Lee deposit). Similarly, the Lowry Upper Copper Zone (LUCZ) is hosted by the USZ (called LUSZ when part of the Lowry deposit), and the Lowry Lower Copper Zone is hosted by the LSZ (locally called LLSZ when part of the Lowry deposit). Mineralization of the Lowry Middle Copper Zone (LMCZ) is a mineralized solution breccia zone associated with some bedded pyrite in a stratigraphic area that has no comparable mineralization or bedded pyrite at the Johnny Lee Zone (Figure 7.4).

7.3.1.4.2 Upper Newland

The Upper Newland is approximately 350 m thick and consists of dolomite, dolomitic shale, limestone, limey shale, silty shale, non-calcareous shale, and carbonaceous shale. The proportion of carbonate rocks (dolomite and limestone) is much higher than that of the Lower Newland. This carbonate-rich part of the Newland formation represents increased chemical precipitation of carbonate from Middle Proterozoic seawater due to heightened salinities in this portion of the Belt basin, as well as somewhat shallower water depositional conditions, all as part of the filling in of the rift basin after a period of active subsidence. The Upper Newland contains bedded pyrite units at multiple stratigraphic levels, but no economically significant copper mineralization has been identified to date.

7.3.1.5 Mesoproterozoic – Lower Belt Supergroup - Greyson Formation

The Greyson Formation is exposed in the southwest part of the Project area. It consists of non-calcareous silty shale, siltstone, occasional sandstone beds and rare limestone layers. Much of the Greyson shale deposited in a shallow water, above wave base, and represents a post-rift phase of sedimentation. The formation reaches approximately 700 m thick in the region, but no local thickness measurements are available. There are no known bedded sulphides in this formation.

7.3.1.6 Cambrian - Flathead Formation

An extensive sheet of Middle Cambrian Flathead Formation covers much of the area north of the VVFZ, and rests unconformably on Belt Supergroup and older rocks. The Flathead Formation consists of moderately to poorly sorted, fine to coarse-grained sandstone with abundant hematite. North of the VVFZ, drillholes have intersected Flathead Formation unconformably overlying south-dipping Neihart, Chamberlain, Newland and Greyson Formations.

7.3.1.7 Igneous Rocks

Numerous dikes and sills with a wide range of compositions have intruded the Belt Supergroup rocks in the Project area. These are both alkalic and calc-alkalic igneous rocks and similar to intrusive rocks which occur throughout the Little Belt Mountains to the north. These intrusive rocks are all part of the Montana Alkalic Igneous Province. Throughout the Project area, many of the dikes show a strong response to magnetic surveys. Large sills of granodiorite form a partially eroded laccolith called the Miller Gulch laccolith, which is bounded by the Black Butte Fault on the north and Miller Gulch on the south. The CAI/BHP JV obtained a K/Ar age date on the granodiorite of 51 Ma (Figure 7.2). There are numerous examples of other Eocene laccoliths of similar composition near the project area, mainly to the north and west. Intrusive rocks with compositions ranging from lamprophyre (mafic) to granitoid (felsic) intrude thrust faults, high angle northeast trending faults, and along sedimentary bedding in an area bounded by BBF on the south and the VVFZ on the north. Such dikes are very rare in the Johnny Lee deposit area, but more numerous in the Lowry deposit where they invade near vertical northeast-trending structures. Intrusive dikes in the resource areas of the Black Butte Copper Project represent a minor volume with low risk for grade dilution.

7.3.2 Structure

7.3.2.1 General Setting

Two regional crustal structural elements of Proterozoic age intersect to control the geological architecture of the Project area. These are the Paleoproterozoic Great Falls tectonic zone (GFTZ) and the Mesoproterozoic northern margin of the Helena embayment of the Belt Basin (Figure 7.1). The northeastern trending Great Falls tectonic lineament is derived from a suture zone between two crustal provinces, the Medicine Hat block on the north and the Wyoming Province on the south. Metamorphic basement rocks that underlay the project area, which are exposed as thrust slices within the Volcano Valley Fault Zone (VVFZ) and as extensive surface exposures two miles north of the Johnny Lee deposit, are granitic gneiss and

amphibolite attributed to the 1.8-billion-year-old Big Sky orogeny. The northern margin of the Helena embayment was an east-west trending transition from shallow water sediments on the north and deep water, basinal sediments on the south (Godlewski and Zieg, 1984) which cuts across the older Big Sky orogenic suture zone. The Buttress Fault (BF) on the north margin of the Johnny Lee deposit (to be discussed below) is a remnant portion of this feature. The intersection of these two regional crustal features provided the opportunity for a focus of the hydrothermal mineralizing fluids that formed the extensive synsedimentary bedded sulphides of the Black Butte Project area. There is little evidence of further tectonic adjustment along either of these features for the next 1.4 billion years until tectonic compression during the Laramide orogeny caused development of oblique reverse-sinistral faults, such as the Volcano Valley Fault Zone (VVFZ) and Black Butte Fault (BBF). The VVFZ generally followed the trend of BF west of the resource area and arced to the southeast away from the BF in the east part of the Project area. The VVFZ carries Belt sediments of the Helena Embayment over Paleozoic cover rocks which lay unconformably on Belt sediments and metamorphic basement rocks in the north. During Tertiary time, it appears reactivation of northeasterly trending structural elements resulted in small offset off portions of the VVFZ and BF and allowed intrusion of Eocene magmas.

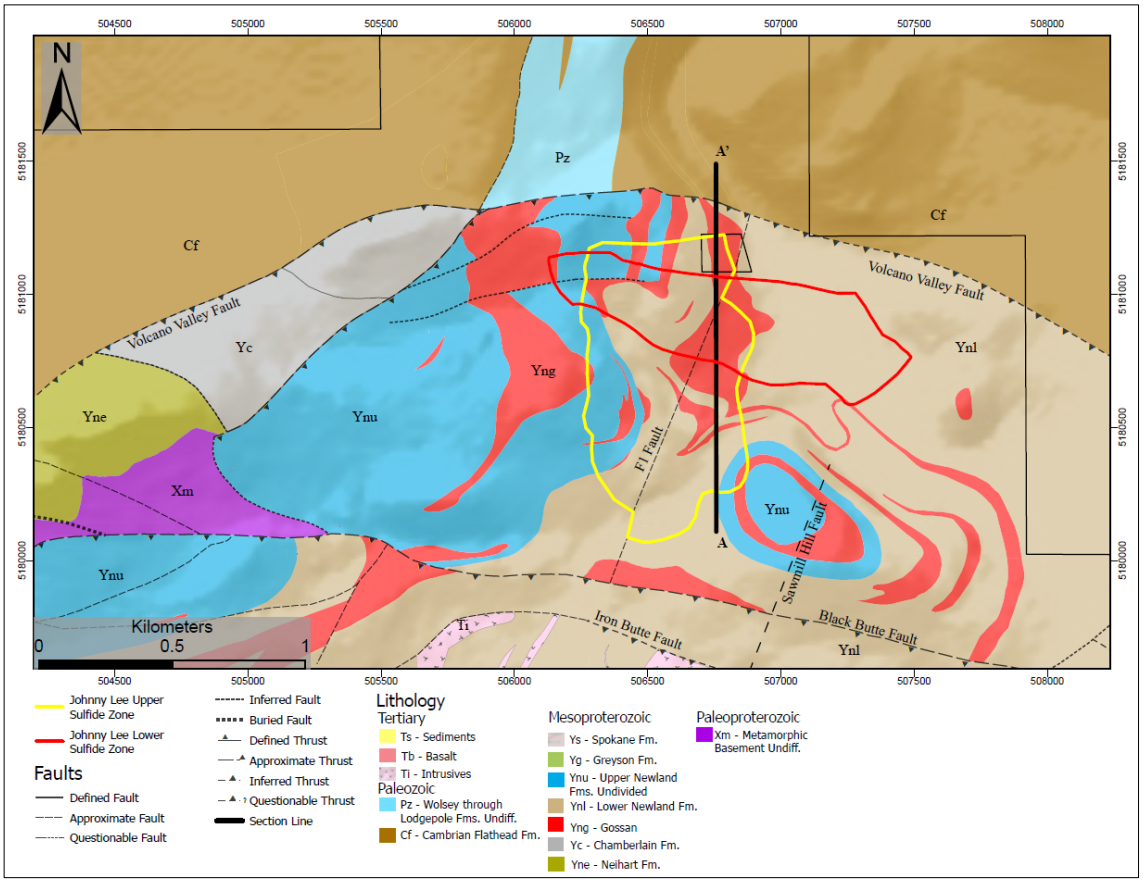
The western segment of the VVFZ lays slightly south of and parallels the BF in the area west of Black Butte (Figure 7.2), where the fault melange includes mega blocks of crystalline basement rocks between hanging wall Newland Formation and footwall Cambrian Flathead sandstone. The central segment of the VVFZ arcs to the north from Black Butte and wraps around the Johnny Lee and Lowry deposit areas before swinging south-eastward to the Newlan Creek area. The BBF splays from the VVFZ on the west end of its central segment at Black Butte, passes south of the Johnny Lee and Lowry deposits, and verges with the VVFZ at the eastern end of the VVFZ central segment. At its northern extent, the VVFZ truncates the upper part of the BF. The eastern segment of the VVFZ follows an easterly trend and carries Newland Formation over successively younger Paleozoic strata eastward. Blocks of Chamberlain Formation form some of the VVFZ fault melange along this eastern segment. The offset along the VVFZ can only be estimated by comparing projections of the Cambrian Flathead sandstone north and south of the fault zone. Estimates of displacement reach 2,500 m for both the eastern and western segments of the VVFZ. Along the central segment of the VVFZ, up to 700 m of this displacement is accommodated by the BBF. Brittle deformation dominates the structural fabric in all faults, though portions of the VVFZ are described as ductile and brittle-ductile.

The JL LCZ and LLCZ resources are bounded on the north by the BF and the south by the VVFZ. A thrust slice bounded on the north by the VVFZ and on the south by the BBF contains all the copper resources in the USZ and MSZ of the Johnny Lee and Lowry deposits. Using these displacement estimates to reconstruct the original distribution of the copper resources, one must conclude that the copper resources in the USZ and MSZ have been carried northeasterly some distance, perhaps more than a kilometre, to their present location above the copper resources in the LSZ.

A three-dimensional (3-D) lithostratigraphic model of the Johnny Lee and Lowry areas, initially constructed in 2019 by SRA using Seequent Leapfrog Geo™ software and upgraded by SRA staff since, allows visualization of significant geological units and geological structures (Figure 7.6). All proposed mine development and infrastructure occur within the boundaries of the 3-D lithostratigraphic model.

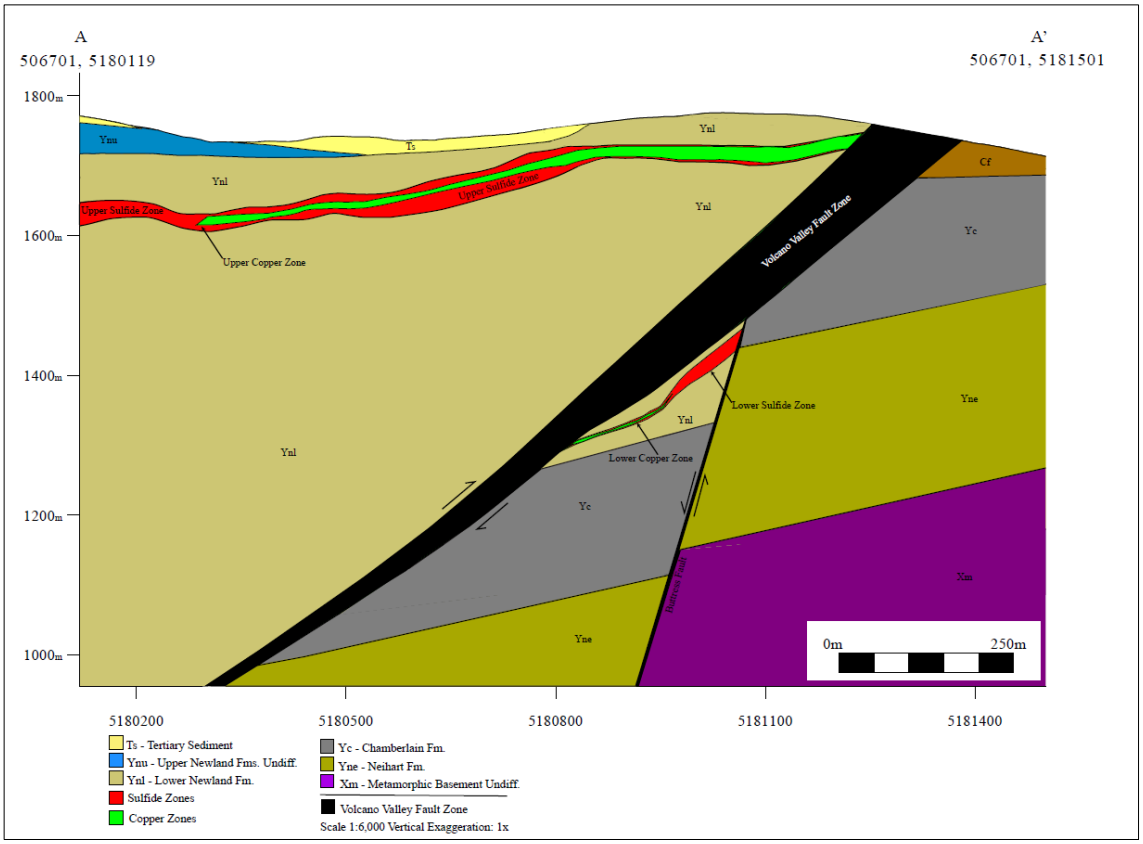
7.3.2.2 Buttress Fault

The Buttress Fault (BF) (Figure 7.6), where drilled in the Johnny Lee deposit area, is a 3 to 5 m wide zone of brecciation and gouge that that strikes east-west, dips 70° south, and is truncated by the shallower angle VVFZ. East of the Johnny Lee deposit and west of Black Butte, Paleozoic sedimentary rocks cover the BF. Its projection in these directions lies north of the surface trace of the VVFZ and is constrained by a series of drillholes which show Chamberlain Formation under Flathead sandstone on the north and Newland Formation under Flathead further the south. Juxtaposition of the Newland and Chamberlain Formations against the Neihart Formation and Proterozoic basement across the BF indicate it has a minimum of 300 m of apparent normal displacement. At the Johnny Lee deposit, the BF truncates the LSZ and forms its northern boundary. The BF projects eastward about 0.5 km north of the Lowry Zone. This fault is Precambrian, and likely a remnant of the fault system that separated the Helena Embayment north margin from a shallower water shelf further north. Subsequent compressive faulting aligned along the trend of the BF. Mapping has revealed other east-west trending Precambrian faults further north.



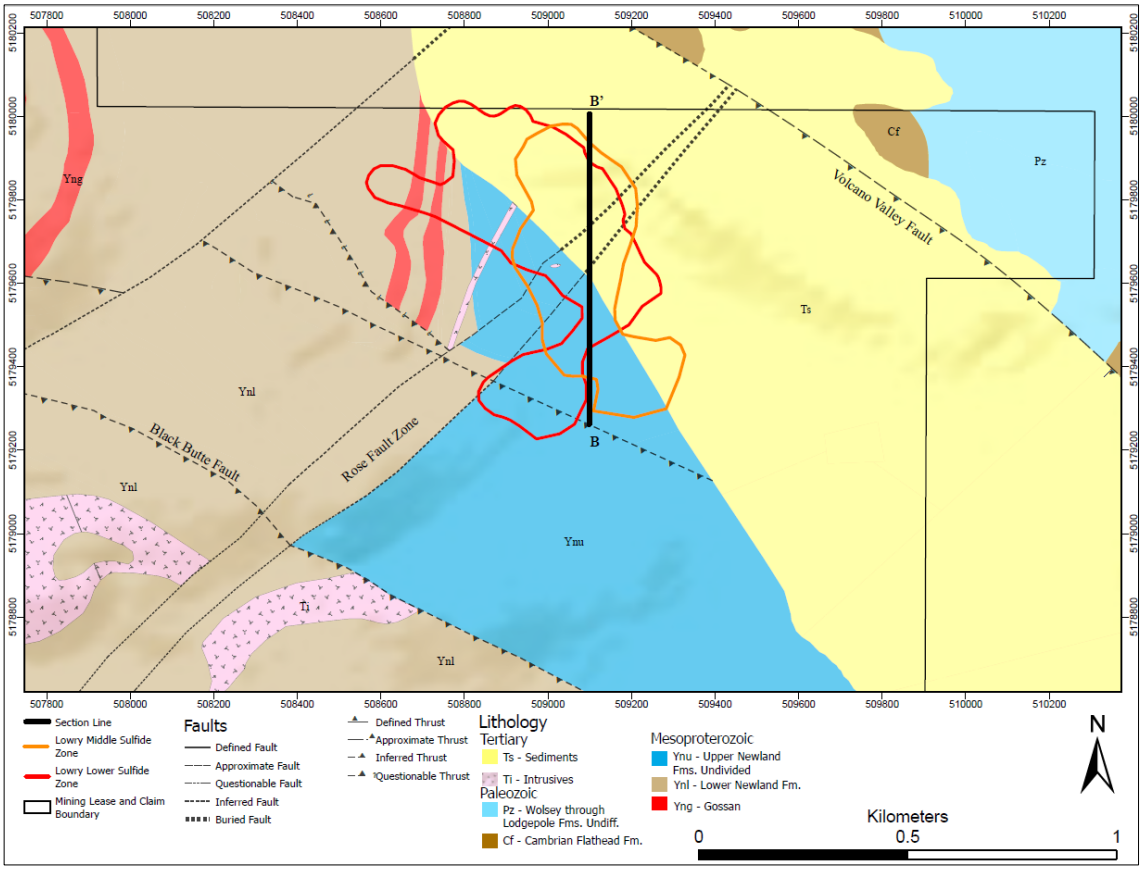
Source: SRA, 2026

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-A' (Figure 7.6)



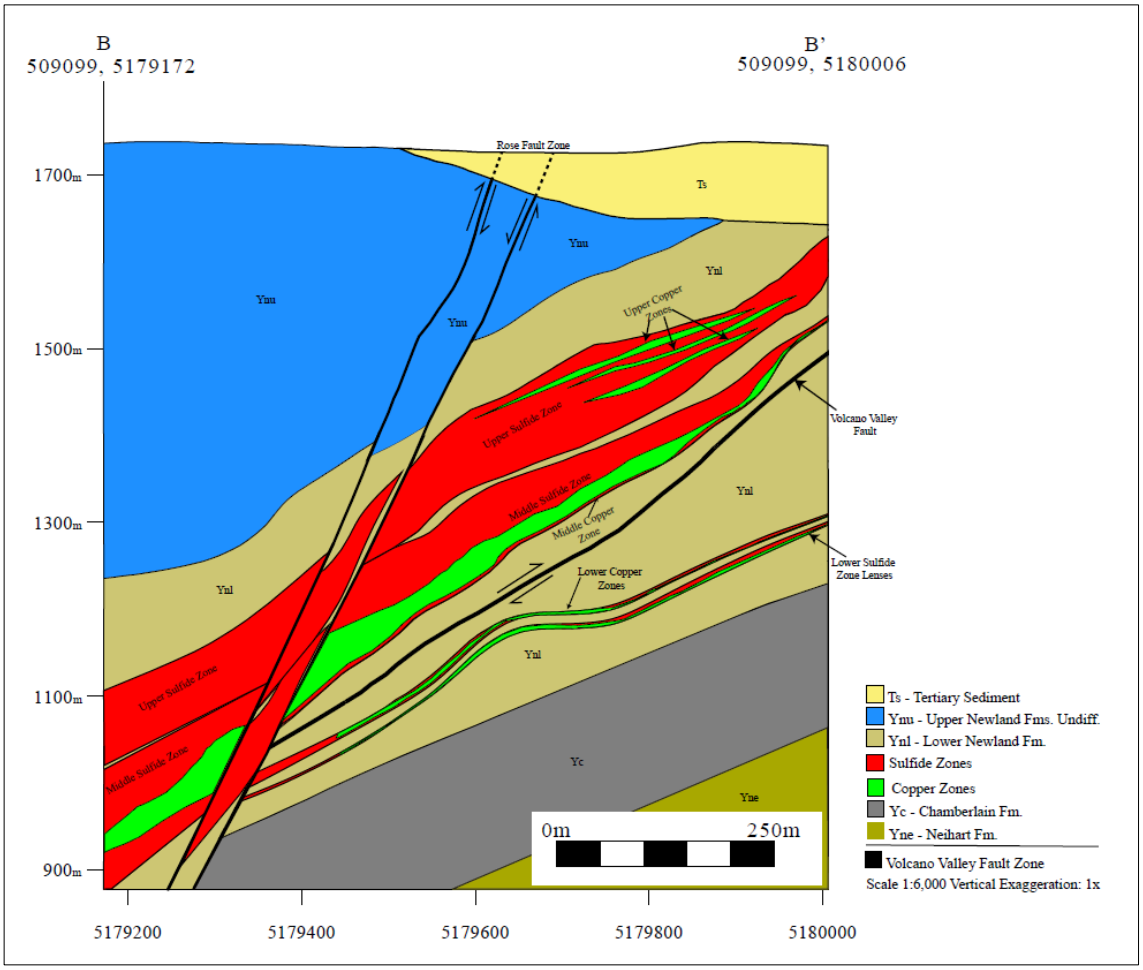
Source: SRA, 2026

Figure 7.6 Cross-section from A to A', view west, through the lithostratigraphic model shown in Figure 7.5



Source: SRA, 2026

Figure 7.7 Geological map of Lowry Deposit



Source: SRA, 2026

Figure 7.8 Cross-section of Lowry Deposit from B to B', as indicated on Figure 7.7

7.3.2.3 Volcano Valley Fault Zone

Within the Project area (Figure 7.2 and Figure 7.3), the VVFZ is an arcuate, W/NW to E/SE striking, southward dipping zone of brittle to brittle-ductile deformation. The Johnny Lee deposit lies inside the top of the arc formed by the VVFZ fault trace, while the Lowry deposit lies further southeast inside the southeast trending portion of the VVFZ surface trace.

In the vicinity of the Johnny Lee deposit, two significant faults termed the Volcano Valley Fault Upper Strand (VVF US) and Volcano Valley Fault Lower Strand (VVF LS) form the edges of the 50 to 170 m wide VVFZ. These faults are characterised by zones of clay-gouge and/or fault breccia (cataclasite) that range in width from 2 to 10 m. Between the two bounding fault strands, the VVFZ contains a tectonic melange of Neihart Formation, Chamberlain Formation and Newland Formation in fault blocks bounded by shear zones. These brittle-ductile shear zones have zones of foliation, quartz-carbonate veining and mineral elongation lineations defined by rod-shaped aggregates of quartz and micaceous minerals. Brittle faults have zones of

clay gouge (typically in shales) or fault breccia (in more competent lithotypes) and often have slickenlines on bounding surfaces. 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). The VVF upper strand truncates the north end of the JL UCZ deposit and the L MCZ and L UCZ deposits. The VVF lower strand cuts off the JL LSZ and forms a south boundary for the deposit (Figure 7.6 and Figure 7.8).

In the Lowry area, the VVFZ is a much narrower zone, generally only about 2 to 5 m wide and in some instances less than one metre wide (Figure 7.8). Only the VVHW fault is well defined in the Lowry deposit area. As at the Johnny Lee deposit, the VVFZ separates the basal Newland Formation with the LLSZ/LLCZ from the upper half of the Newland formation with the LMSZ/LMCZ, LUSZ/LUCZ, and stratigraphically higher sulphide zones. The Volcano Valley fault rises considerably higher above the LSZ in the Lowry area than in the Johnny Lee area and this section includes the Hanging wall Dolomite. This carbonate and enclosing shale show considerably less fracturing, and disruption compared to those above the VVFZ.

The VVF is clearly post-mineral and postdates the Buttress fault. It is the largest representation of Laramide (Cretaceous age) compressive deformation in the region of the Black Butte Copper Project. The VVF aligns parallel to the BF and formed due to the transport of Helena Embayment sediments north against metamorphic basement elevated to a higher elevation north of the BF. It is not, in the strictest sense, a reactivation of an earlier fault in spite of its close proximity to the BF.

7.3.2.4 Black Butte Fault

The divergence of the Black Butte Fault (BBF) from the VVFZ defines the central segment of the VVFZ (Figure 7.2 and Figure 7.3). The BBF splays from the VVFZ on Black Butte and rejoins the VVFZ in the Newland Creek area and carries deep Newland stratigraphy against higher Newland stratigraphy with an offset of at least 700 m. Current mapping south of the Johnny Lee deposit identifies the BBF as two parallel faults spaced 170 to 200 m apart. The northern one extends east from Black Butte, while the southern one is an eastward continuation of the Iron Butte Fault (IBF) which arcs to the southeast to parallel the BBF. This configuration is truncated by the northeast trending Brush Creek fault, east of which the mapping show continuation s BBF as a single fault.

South of the Johnny Lee deposit a single drillhole intersected an 8 m downhole width of brecciation and fault gouge which represents the BBF. It is a curviplanar fault striking and dipping sub-parallel to the VVFZ. It developed during the same deformation event that formed the VVFZ and has similar kinematics. The BBF passes through the vicinity of the planned mill site but shows no evidence of any movement since its original development in the Laramide orogeny and so has no impact on the currently proposed mining and development activity.

7.3.2.5 Iron Butte Fault

The Iron Butte Fault (IBF; Figure 7.2 and Figure 7.3) splays away from the Volcano Valley Fault in the Copper Creek area and trends east through the southern part of Iron Butte, across Butte Creek, and on toward Sawmill Hill, where it abruptly curves from an easterly trend to a southeasterly trend. It then parallels the BBF and finally terminates against the Brush Creek fault (Figure 7.3). It forms a northern boundary to an Eocene granodiorite sill and dike complex which emanate north from a laccolith in upper Miller Gulch.

7.3.2.6 Northeast trending faults

A number of northeast trending faults cut through the area (Figure 7.2). They are concentrated within the area occupied by the central segment of the VVFZ. These lie within the regional northeast trend of the GFTZ and their current disposition may result from re-activation of older (Paleoproterozoic) basement structures.

7.3.2.6.1 Moose Creek Fault

The BF appears offset by about 1,300 m from the Butte Creek area to the Johnny Lee area (Figure 7.2). Strong northeast trending lineations apparent in LIDAR imagery in Black Butte align with the very straight northeast linear occupied by Moose Creek north of the BF and VVFZ. The VVFZ and associated reverse-dextral faults mask any representation of this northeast trending element.

7.3.2.6.2 Coon Creek fault

The east margin of the Johnny Lee deposit follows a northeast trend now manifest as the Coon Creek valley (Figure 7.6). JL USZ mineralization abruptly ends against this feature though concentrated drilling has shown no discernible offset. One must presume that this feature acted as a syn-sedimentary barrier during deposition of the bedded pyrite that forms the JL USZ.

7.3.2.6.3 Sawmill Hill Fault

A mafic dike occupies a northeast trending near vertical structure that passes through the top of Sawmill Hill east of the Johnny Lee deposit (Figure 7.5). Bits of mafic intrusive float, and a strong linear magnetic response, define this feature.

7.3.2.6.4 *Brush Creek Fault*

The Brush Creek Fault (BCF) divides the Johnny Lee deposit area, including Sawmill Hill, from the Lowry deposit area (Figure 7.3). It is mainly defined by the northeast topographic lineament occupied by Brush Creek and discordant bedding orientations of bedrock on either side of the lineament. The interaction between the BCF and the VVF is concealed by valley fill and we have no drill data in that vicinity. Near the CTF area, a single hole intersected the BCF at a low angle where it found a zone of brittle deformation of about 20 m in width.

7.3.2.6.5 *Lowry Dikes*

Two mafic dikes occupy near vertical northeast trending structures in the Lowry deposit. The structures show no offset, and the dikes are not volumetrically significant, but they do interrupt the LMCZ resource.

7.3.2.6.6 *Rose Fault*

The Rose Fault Zone (RFZ; Figure 7.7 and Figure 7.8) trends at a bearing of 44° azimuth through the Lowry deposit and dips -68° southeast. It shows a throw of 80 to 100 m, down to the southeast. The zone is bounded by two subparallel faults; the eastern of the pair shows nearly all the displacement. The VVFZ, and both the LUSZ and LMSZ are displaced by the Rose Fault.

Four kilometres southeast of the Lowry deposit in the Newlan Creek area, a number of other northeast trending structures cut the VVFZ with small amounts of offset. Small faults also cut through the Paleozoic cover rocks north in Green Mountain.

7.3.2.7 *Folding*

SW to NE orientated compression produced E/SE - W/NW trending folds. At the Johnny Lee deposit, this compression has folded the Newland Formation strata in the hangingwall of the VVFZ into a syncline-anticline pair with upright, W/NW to E/SE striking axial planes (Figure 7.2). These folds have a variable plunge that ranges (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).

7.3.2.8 *Other Features*

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.

7.4 Geology and Mineralization of Copper Deposits

7.4.1.1 Overview

Potentially economic copper mineralization has been identified in the Newland Formation in the various sulphide zones in the Johnny Lee and Lowry deposits (Figure 7.3). The Johnny Lee copper deposits include the Johnny Lee Upper Copper Zone (JL UCZ) and the Johnny Lee Lower Copper Zone (JL LCZ). The Lowry copper deposits include the Lowry Upper Copper Zone (LUCZ), the Lowry Middle Copper Zone (LMCZ) and the Lowry Lower Copper Zone (LLCZ). The JL UCZ and the LUCZ are hosted by the much more extensive Upper Sulphide Zone (USZ) and the JL LCZ and LLCZ are hosted by the more extensive Lower Sulphide Zone (LSZ). The USZ and LSZ are district wide bedded pyrite zones which adhere to specific stratigraphic positions – the LSZ at the base of the Newland Formation, and the USZ a relatively short distance (200 m) below the contact between the Upper Newland Formation and Lower Newland Formation. The two are separated by about 700 m of stratigraphy. The LMCZ is hosted by the Middle Sulphide Zone (MSZ) which is of much more limited extent than the USZ and LSZ and is only recognized at the Lowry deposit. It occurs below, and in partial contact with, the LUSZ. The current Mine Operating Permit only covers development and mining of the JL UCZ and JL LCZ.

Cu mineralization in the Newland Formation is largely confined to the USZ, MSZ, and LSZ, but is also known to occur in other sulphide beds stratigraphically above the USZ. This mineralization appears to be sporadic and lower grade than the primary zones, though and is only intercepted in a few drill holes and the nature and extent are insufficiently defined. Shale and conglomerate interbeds within pyrite zones can also host economic grades of copper mineralization.

Fine-grained pyrite beds volumetrically dominate most pyrite occurrences and units dominated by bedded pyrite can reach 100 m thick or more containing greater than 50% pyrite. Zones of greater than 90% pyrite can reach more than 40 m thick. Pyrite beds typically show internal laminations with carbonaceous films in bedding partings interpreted as remains of microbial mattes. Some bedded pyrite zones host layers of fragmented shale and dolomite clasts suspended in a pyrite matrix, and in other cases fragments of bedded pyrite suspended in a pyrite matrix. Soft sediment folding deforms some pyrite bedding. Bedded pyrite as described above forms most of the pyrite in the USZ but can occur in both the LSZ and to some extent in the MSZ.

Complex tubular forms called ‘net texture’ (Graham *et al*, 2012) accompany many bedded pyrite occurrences and typically consist of intergrown sulphide-walled tubes filled with quartz, dolomite, barite, or in some cases open vugs lined with gangue material and chalcopyrite or cobaltite. Occurrences in outcrop adjacent to gossan outcrops appear as clumps of microcrystalline quartz walled tubes arranged along thin beds of dolomitic shale clast conglomerate. Clasts of sulphidic net texture sometimes appear as clasts in otherwise non-sulphidic shale clast conglomerate. This texture is most abundant where bedded pyrite is most abundant. SRA geologists currently interpret net texture as originating as crusts around microbial filaments

growing at hydrothermal vent centres, in most cases replaced by pyrite (McGoldrick and Zieg, 2004). Some, however, propose a hydrothermal vent fluid flow origin (Present *et al*, 2017).

Some pyrite has grown diagenetically in mud or formed hydrothermal sulphide chimneys that collapsed or were compressed by overlying sediment. The LSZ typically shows, above a bedded pyrite base, chaotic zones of collapsed and/or compressed sulphidic chimney remnants and sulphide replacement of earlier coarse-grained dolomite alteration. In general, sulphide textural and field relationships lead to a synsedimentary interpretation of sulphide formation around sea-floor hydrothermal vent centres. In addition, Re-Os data on Black Butte sulphides show the mineralization formed around 1,488 Ma, (Saintilan, 2021), similar to the 1,475 Ma age of formation of the giant synsedimentary Sullivan Zn-Pb-Ag deposit (Slack, 2020) further northwest in the Belt Basin.

The pyrite in the sulphide zones 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 represent the earliest mineralization subsequent hydrothermal activity resulted in increasing recrystallization until the presence of planar crystal faces leads to coarser crystalline pyrite forms (McArthur, 2018).

Cu mineralization in the USZ occurs as chalcopyrite, tennantite, bornite and chalcocite. Chalcocite and bornite occur together, with no chalcopyrite, in a very narrow zone along a northeast trending axis co-incident with the thickest part of the JL USZ and JL UCZ; high concentrations of Sr as celestite and strontianite footwall to the JL UCZ. Bornite and chalcocite occur deeper than chalcopyrite of the JL UCZ in this area and the host rocks show no signs of oxidation. Some have interpreted the chalcocite to be the result of supergene processes; SRA geologists currently believe the area of the bornite-chalcocite occurrences together with the other characteristics mentioned above form a mineral zonation pattern away from a northeast trending hydrothermal vent center for the JL USZ and JL UCZ. The zonation would include chalcocite-bornite assemblage proximal to the vent, then chalcopyrite-tennanite assemblage away from the vent.

In all five copper deposits (JL UCZ, JL LCZ, LUCZ, LMCZ, and LLCZ), 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. SRA geologists interpret copper mineralization as a result of extended progressive hydrothermal activity, though some could result from copper remobilization during a later tectono-thermal event.

Cobaltian sulphides associated with the Cu deposits are generally termed cobaltite for ease of communication but in fact include a broader range of compositions, which consist of very fined-grained grains of Co-Ni sulpharsenides (cobaltite and associated polymorphs); and a range of compositions on the siegenite-linnaeite trend. Generally cobalt minerals are very fine-grained and intimately intergrown with very fine-grained primary pyrite, though areas of coarser grained siegenite occur in some portions of the Cu deposits.

Reddish brown sphalerite occurs in brittle open fractures in the Lowry deposit, but is nearly absent from the Johnny Lee deposit. Masses and disseminations of cream to honey yellow-colored sphalerite occur in subeconomic quantities in sulphide zones stratigraphically above the USZ in both the Johnny Lee and Lowry areas.

Small amounts of galena occur microscopically in the deposits, but are only rarely visible to the naked eye. Galena also accompanies the light-colored sphalerite occurrences.

Several alteration minerals accompany sulphide mineralization. Barite occurs in significant amounts in the JL USZ, and in limited amounts in and near the other Cu mineralized zones. Barite lathes appear randomly oriented within massive pyrite beds, disseminated in host shale and conglomerate layers, as vug-fill, and remobilized in veins and veinlets. The highest barite concentrations occur as a cap above and overlapping the JL UCZ which extends over the entirety of the JL USZ.

Dolomite occurs sporadically disseminated in host rocks peripheral to all five Cu deposits. Spectacular zoned megacrysts of dolomite as well as sheets and disseminations of coarse-grained dolomite occupy the hanging wall of both the JL LCZ and LLCZ.

Silicification accompanies Cu mineralization to varying degrees in all five deposits; it is most concentrated in the JL LCZ, LLCZ, and LMCZ, and less concentrated but still present in the JL UCZ and LUCZ.

Concentrations of Fe carbonate, ranging from Fe-dolomite to ankerite and ultimately siderite, occur peripheral to the L MCZ and LUCZ. It typically replaces earlier thick laminations and thin beds of dolomite that cap microturbidite lamination- and bed-sets.

7.4.2 Johnny Lee Lower Copper Zone

The Johnny Lee Lower Sulphide Zone (JL LSZ; Figure 7.6 and Figure 7.9), located in the FW of the VVFZ and HW of the BF, hosts the JL LCZ. The Johnny Lee Lower Copper Zone (JL LCZ) occurs at depths of 340 to 520 m below surface, strikes approximately east-west EW and dips at 15° to 30° to the south. Interlayered shale and conglomerate overlays the JL LSZ and it rests on a pervasively silicified thick conglomerate below which shale grades into Chamberlain Formation silty shale beds. The JL LCZ deposit lenses are defined by the outer limit of >2.0% Cu mineralization which extends upward into intercalated conglomerate. Minor Cu mineralization also occurs in the silicified conglomerate below the LSZ.

Boundaries to the JL LCZ include the Buttress Fault on the north and the Volcano Valley fault on the south. At the western end of the JL LCZ, the faults merge to cut away the Cu zone. The VVF is post mineral, and the melange between the upper and lower strands of the VVF contain significant amounts of copper mineralized material torn from the LCZ. All holes that intersect the JL LCZ pass through the VVF above the mineralized zone. The Buttress Fault forms a northern boundary to JL LCZ mineralization. The BF dips steeply south and is truncated by the Volcano Valley fault, though there is some indication in surface geology that new movement on the BF after VVF faulting may have caused propagation of new faults upward from the BF through the VVF and supra VVF rock. The Buttress Fault zone appears to range in thickness from 3 to 5 m and shows pervasive clay alteration. It juxtaposes JL LCZ against both Chamberlain Shale and Neihart Quartzite. Its vertical offset is approximately 250 m. It contains no mineralization nor any fragments of early mineralization. The eastern boundary of the JL LCZ is an abrupt cessation of Cu mineralization along a NW trend. Though a thin LSZ continues east, there may have been a NW oriented syn-sedimentary feature that formed a barrier to further Cu mineralization in this direction.

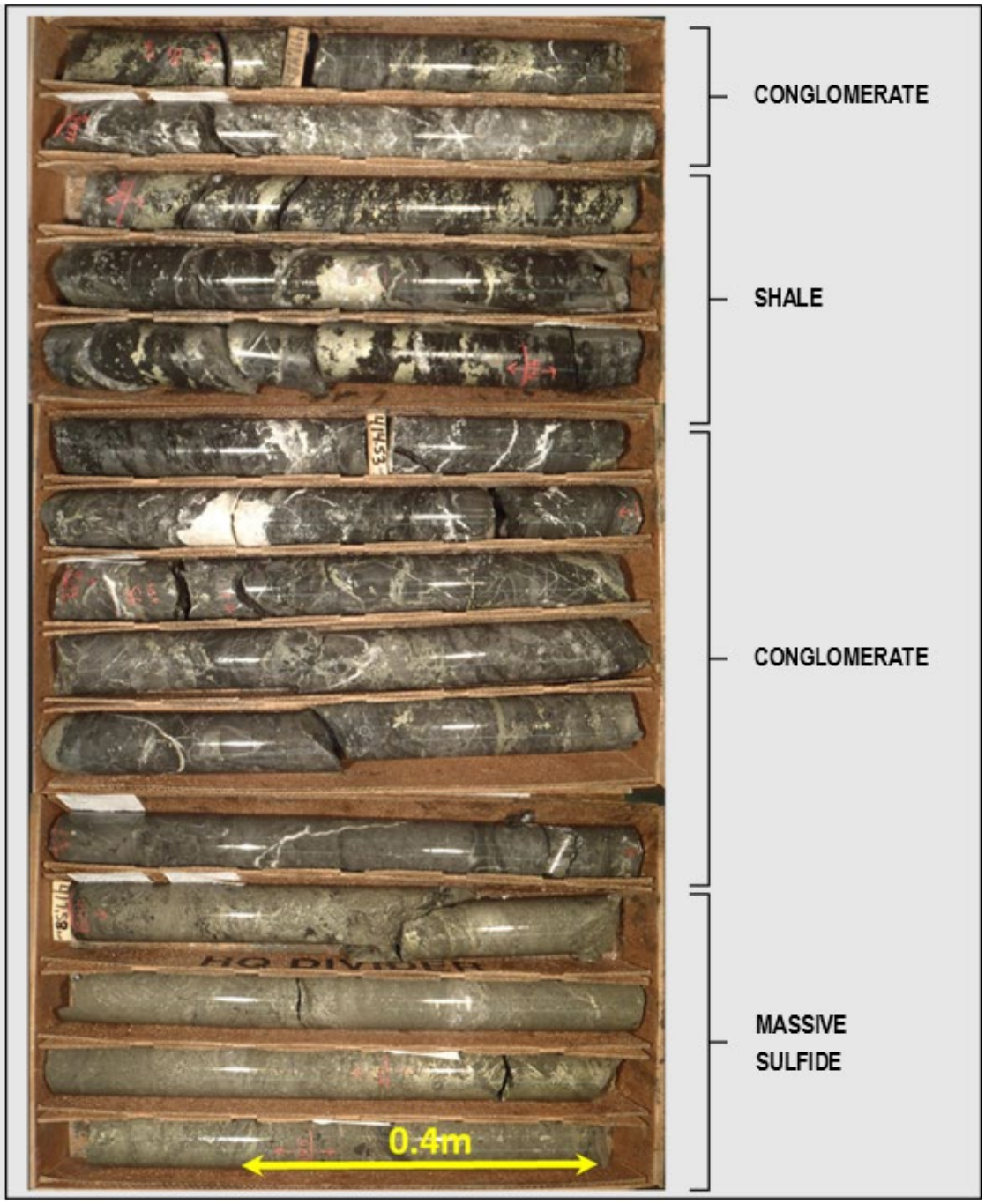


Figure 7.9 Core photographs of drillhole SC11-011 from 411.2 m (top) to 419.7 m (base) showing chalcopyrite hosted Cu mineralization extending from massive sulphide of the JL LSZ into interlayered shale and conglomerate

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

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



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

The historical 2013 Mineral Resource estimate for the JL LCZ (Winckers *et al*, 2013) included estimates for Ag, Au and Co. However, insufficient work has been done to determine the residency of these elements in the JL 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. Johnny Lee Upper Copper Zone.

7.4.3 Johnny Lee Upper Copper Zone

The north-eastern corner of the JL UCZ is exposed on surface (Figure 7.7) and the top of the mineralized zone extends to a depth of 210 m below surface. The high-grade portion of the JL UCZ (>1.0% Cu) is entirely encapsulated by the JL USZ although >0.25% Cu halo mineralization extends in places into shale above and below the JL USZ. (Figure 7.7). A W/NW plunging syncline-anticline pair with a variable strike and dips ranging between 0° and 20° gently folds the JL UCZ. With the exception of its extreme north-eastern corner, the JL UCZ is situated below the level of surficial oxidation.

The Volcano Valley FW Fault rises from the south beneath the JL USZ and truncates the north end of the deposit. (Figure 7.6). Figure 7.20Discontinuous lenses of conglomerate, as well as shale and carbonaceous shale layers ranging in thickness from 1 mm to 2.5 m occur throughout the JL USZ. The proportion of clastic sediment layers relative to sulphide layers is highly variable but shows an overall decrease from north to south.

The bedded pyrite in the JL UCZ exhibits a large degree of textural variation. JL 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 JL USZ commonly contains layers of primitive pyrite (variably recrystallized colloform melnikovite or framboidal pyrite) ovoids (Figure 7.11) that when viewed in section present as small pyrite halos surrounding cores of later barite ± quartz infill (Figure 7.12). These intervals are a form of the net texture described by Graham *et al* (2012).

In contrast to the JL LCZ, where the Cu is hosted almost entirely by chalcopyrite (with rare occurrences of trace tennantite), the Cu mineralogy of the JL UCZ is both mineralogically and texturally more diverse. Chalcopyrite dominates copper mineralogy, reaching up to 15.8% of the total sulphide mineralogy (McArthur, 2018). Tennantite occurs 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 JL 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).

Table 7.1 shows the Cu metal deportment for the northern and southern parts of the JL 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. An average of 3.4% of the Cu occurs in pyrite grains. The contribution to total Cu metal from chalcocite and minor covellite is higher in the southern half of the orebody. 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 7.1 Cu metal distribution for the JL UCZ based on mineralogical study

The mineralogical differences, albeit with significant local variability, between the northern and southern parts of the JL 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 Cu 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 JL 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, net textures, and coarse (i.e. visually observable) chalcopyrite.

Coarse chalcopyrite in the southern half of the JL 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 JL USZ (Figure 7.13). This does occur along bedded pyrite layers but is much more prevalent along net textured zones (Figure 7.14).

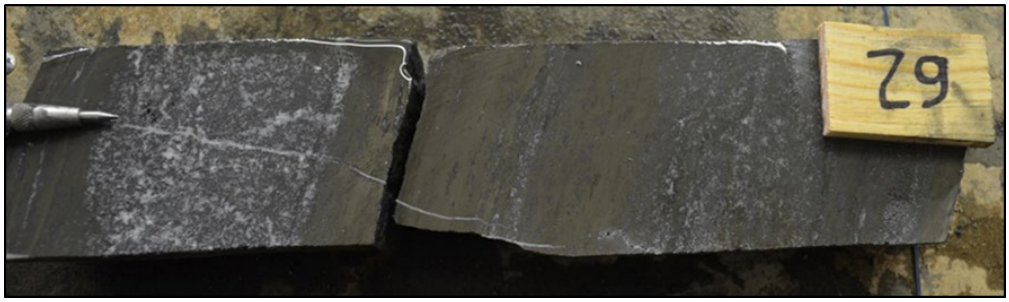


Figure 7.11 Photograph of JL USZ bedded pyrite sulphide containing net texture with light coloured infilling

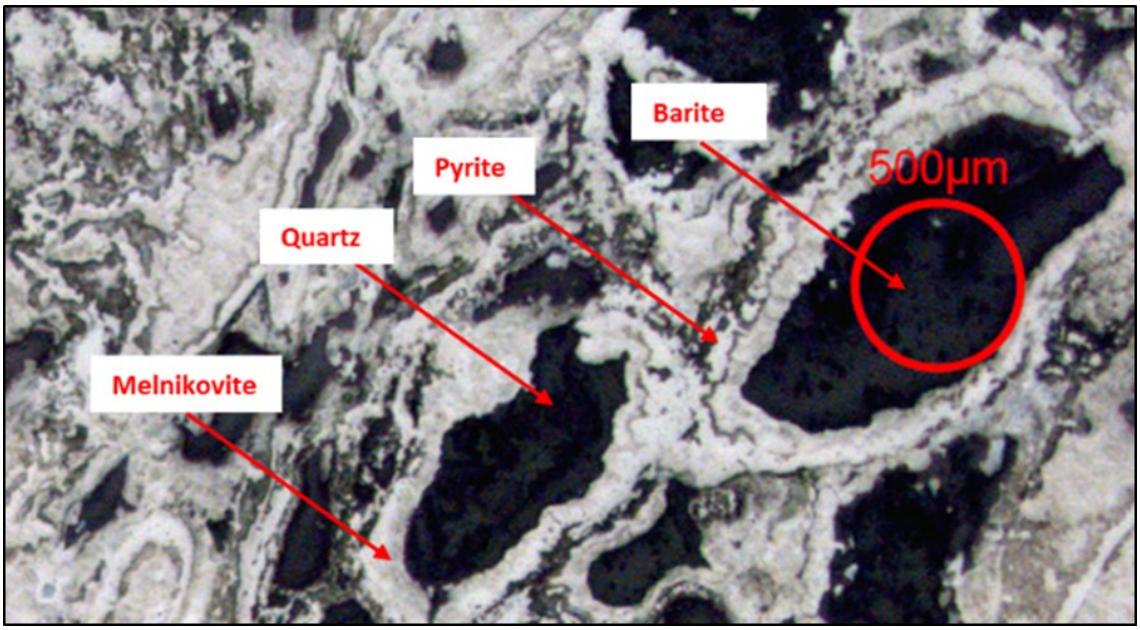


Figure 7.12 Photomicrograph of sample BBPET50 (drillhole SC12-121 at depth 118.2m) showing net texture in JL USZ

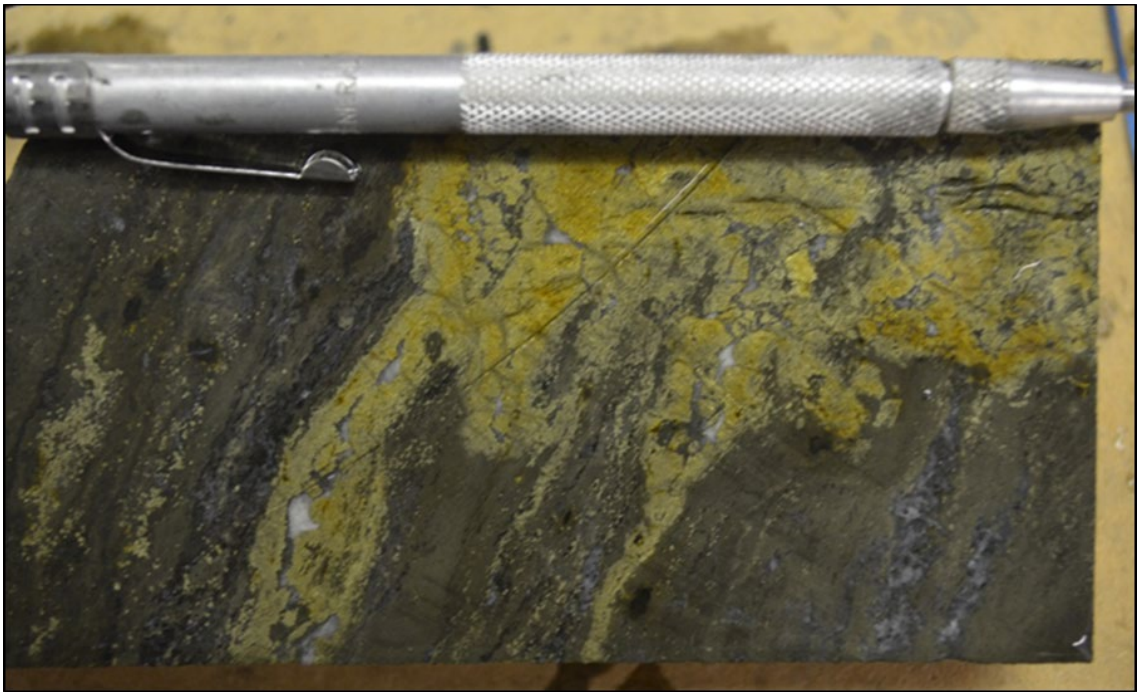


Figure 7.13 Coarse chalcopyrite veinlet cross-cutting layered bedded pyrite with narrow net textured layers in JL UCZ. Note the replacement along certain layers.

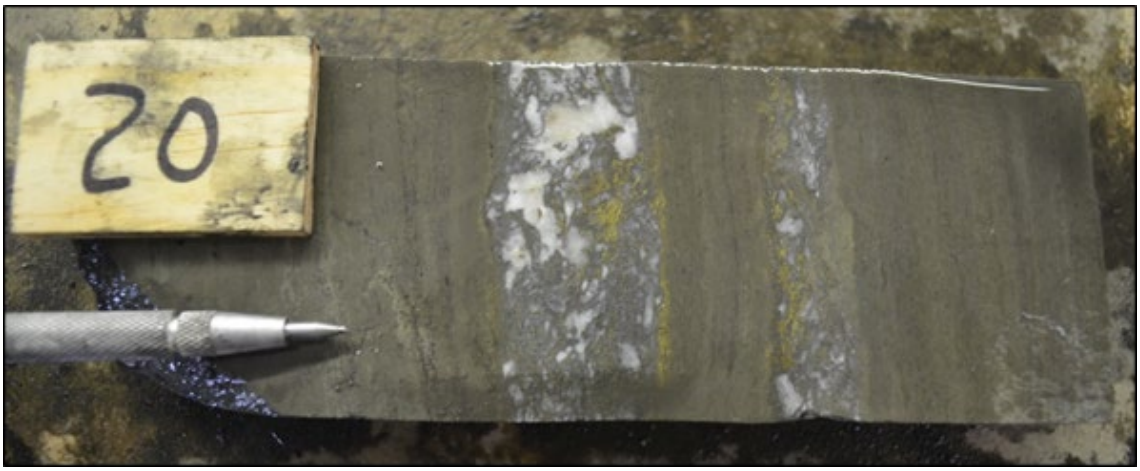


Figure 7.14 Preferential replacement of net-textured layers with chalcopyrite within bedded pyrite of the JL UCZ

Attempts to correlate the proportion of coarse chalcopyrite with Cu grade in the southern part of the JL UCZ failed. Typically, the analyzed grade exceeded the calculated Cu grade from visual estimates of coarse chalcopyrite percentage by a significant amount. In certain drillhole intersections (typically bedded pyrite with no net texture), 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 JL UCZ.

The mineralization in the northern part of the JL UCZ is similar but also exhibits extensive brittle fracture, quartz ± carbonate veining, and hydraulic brecciation. Though some of the coarse chalcopyrite is cross-cut by the fracturing and veining (Figure 7.15), a significant amount of coarse chalcopyrite appears introduced, or remobilized, during the fracturing, veining, and hydraulic brecciation. (Figure 7.16 through Figure 7.19).



Figure 7.15 Coarse chalcopyrite, cross-cut by quartz-carbonate veins in JL UCZ. Core diameter = 63.5 mm



Figure 7.16 Coarse chalcopyrite that is synchronous with hydraulic brecciation, fracturing and quartz-carbonate veins. Core diameter = 63.5 mm



Figure 7.17 Interbedded pyrite and siltstone from the northern part of the JL UCZ. Coarse chalcopyrite and crystalline pyrite have replaced tabular barite crystals. Coarse chalcopyrite also forms veinlets cross-cutting bedding. Core diameter = 63.5 mm

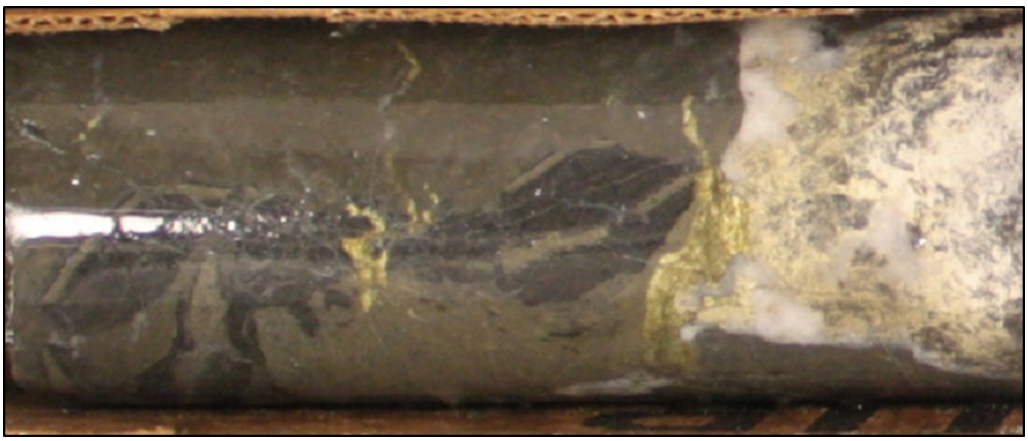


Figure 7.18 Coarse chalcopyrite both synchronous and post-dating quartz-carbonate veining in the northern part of the JL UCZ. Core diameter = 63.5 mm

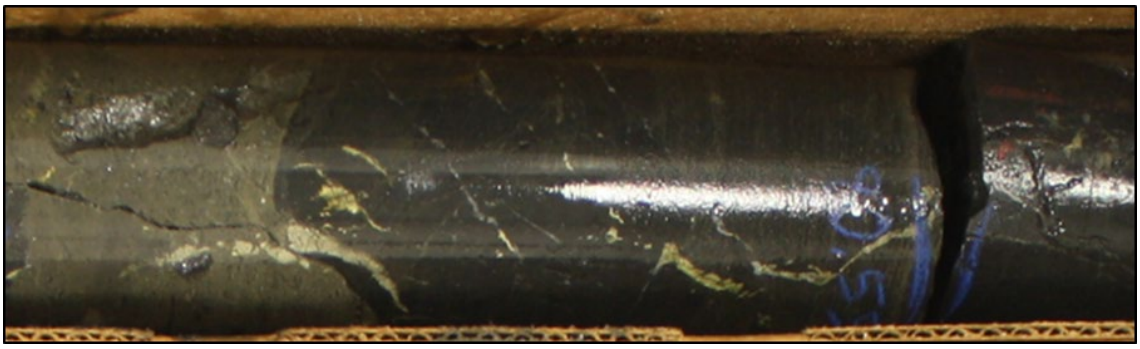


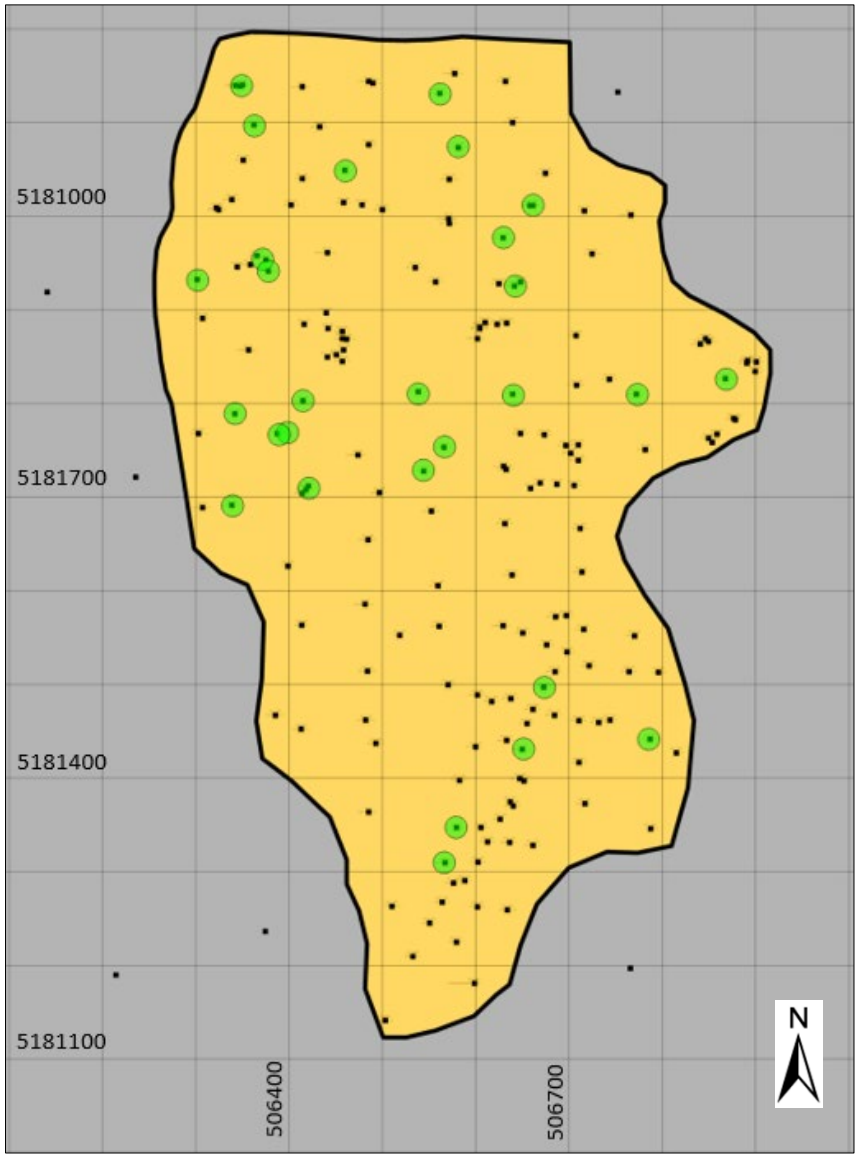
Figure 7.19 Quartz + carbonate + coarse chalcopyrite veinlets cross-cutting siltstone and bedded pyrite in the northern part of the JL UCZ. Core diameter = 63.5 mm

Although some localized quartz-carbonate veining and hydraulic brecciation occur in the southern part of the JL UCZ, it is more consistently developed in the northern part of the JL UCZ (Figure 7.20). The mineralogical differences observed between the northern and southern parts of the JL UCZ are thus mirrored by core observations that reflect post-lithification hydrothermal activity that occurred primarily in the northern part of the JL UCZ.

As only the proportions and grain size of coarse Cu sulphide was estimated from core, the metrics for all other Cu sulphides, and fine-grained chalcopyrite have been constrained using mineralogical investigation (McArthur, 2018). Table 7.2 summarizes the average grain-size of all minerals for the northern and southern parts of the JL UCZ. In terms of average grain-size of Cu sulphide minerals (e.g. chalcopyrite, tennantite, cuprian siegenite, bornite, chalcocite and covellite), those in the southern half of the JL UCZ are smaller than those of the northern half.

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

Table 7.2 Average grain size of all minerals in the northern and southern parts of the JL UCZ based on systematic mineralogy by McArthur (2018)



Source: SRA, 2026

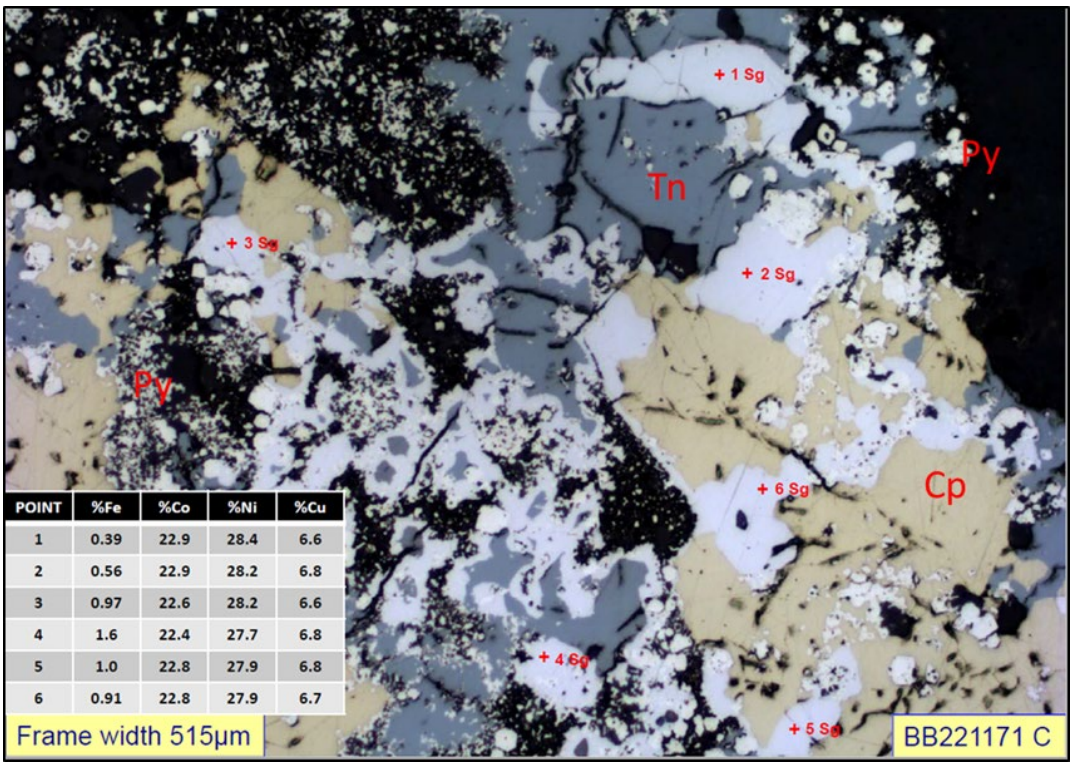
Figure 7.20 Plan view of JL UCZ showing drillhole intersections that have >8% quartz-carbonate veining and brecciation as green dots.

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

The 2013 Mineral Resource estimate for the JL UCZ included estimates for contained Co, Ag, and Au (Winckers *et al*, 2013). Estimated Au grades for the JL 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 JL 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 JL UCZ is pyrite 50.13%; tennantite 0.45%, and cuprian siegenite 0.07% (McArthur, 2018). Approximately 76% of the Co in the JL 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 7.21) 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 7.21 Reflected light photomicrograph of a JL 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, Ag in the JL UCZ appears hosted by late crystalline pyrite and marcasite.

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

Tennantite in the JL 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 JL 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 JL UCZ to be estimated Table 7.3). In terms of average As deportment, the majority occurs within tennantite. The difference between the southern and northern parts of the JL UCZ largely reflects the average lower proportion of tennantite observed in the north.

		Northern JL UCZ		Southern JL 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 7.3 Average As content of main As containing mineral species and proportions of those mineral species in the northern and southern portions of the JL UCZ

7.4.4 Lowry Deposit

The Lowry deposit lies 1.5 km east of the Johnny Lee deposit and consists of copper resources in three pyrite-dominated sulphide zones. The Lowry Upper Copper Zone (LUCZ) lies within the Lowry Upper Sulphide Zone (LUSZ) at the same stratigraphic level as the Johnny Lee USZ. The Lowry Middle Copper Zone (LMCZ) lies within the Lowry Middle Sulphide Zone (LMSZ) no more than 50 m below the LMSZ. Both

zones lie above the VVFZ. The Lowry Lower Copper Zone (LLCZ) lies at the base of the Newland Formation and below the Volcano Valley fault. See Figures 39 and 40 for a plan map and section.

7.4.5 Lowry Upper Copper Zone

The Lowry Upper Copper Zone (Figure 7.8) is defined by 27 holes spaced roughly 50 to 100 m apart, and visual examination of the drill core from these show that the LUCZ shares its characteristics with the JL UCZ. Mineralization consists of chalcopyrite replacing fine-grained massive pyrite beds, but no one has performed a detailed mineralogic analysis on the LUCZ to ascertain the presence and amount of other sulphide types.

7.4.6 Lowry Middle Copper Zone

The LMSZ (Figure 7.8) reaches 80 m thick (Allard, 2021), consists of a thick breccia zone together with shale with interbeds of bedded pyrite and conglomerate and contains the LMCZ.in sulphide. Much of the breccia consists of large, rounded dolomite clasts with darker carbonaceous material, dolomitic shale clasts, and pyrite in the interstices. In places, the breccia exhibits vugs filled with bands of coarse dolospar and cored with bitumen. Peripheral to the LMCZ, the breccia shows abundant iron carbonate alteration. The LMCZ is extensively silicified and contains varying concentrations of chalcopyrite and small amounts of tennantite. No detailed mineralogical investigations have been carried out to determine the additional sulphide species that are likely present in the LMSZ. Silicification and mineralization overprint not only the dolomitic and polymictic breccia matrix but also adjacent shale, primary pyrite beds (some with net texture), conglomerate, and dolomite layers as well. SRA's geologists currently interpret the breccia a result of dissolution of a precursor dolomite unit by hydrothermal fluids and thus providing a focus for the Cu mineralizing fluids that formed the LMCZ. The LMCZ occurs 0 to 100 m above the VVFZ Figure 7.8 and formed from 85 to 125 m below the LUCZ.

Drillhole intersection spacing in the Lowry Middle Copper Zone LMCZ ranges from 40 to 100 m. 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 50 m and it progressively reduces in width and pinches out to the north, south, east, and west.

Pyrite + marcasite concentrations of typical mineralized intersections range from 9.1 to 45.2% and gangue mineral contact ranges from 38.7 to 76.0% (McArthur, 2019). Cu mineralization is predominantly chalcopyrite (8.3 to 17.0% in typical mineralized intersections) with minor tennantite (0.01 to 0.23%) observed in 30% of mineralogical samples studied to date (McArthur, 2019). Allard (2021) reports sulphide mineralization occurs dominantly as chalcopyrite and pyrite with lesser, typically fine-grained inclusions of tennantite, sphalerite, and galena. Allard also recognized abundant 1 to 10 cm veins of pyrite +/- tennantite crosscutting host lithologies. Some crystalline pyrite and chalcopyrite appear intergrown with dolomitic vug fill with in the breccia zone. The amount of primitive pyrite in the LMCZ ranges from 10.4 to 60.0% (average 49.1%) which is lower than that of the northern part of the JL UCZ (average 62.7%) indicating that the

amount of recrystallization in the LMCZ is higher than that of the northern part of the JL UCZ. From a mineralogical and Cu 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.

7.4.7 Lowry Lower Copper Zone

The Lowry Lower Copper Zone (LLCZ; Figure 7.8) shows strong similarities with JL LCZ. It occurs below the VVFZ which may truncate its southern end though the most southerly drill tests of LLCZ still located it beneath the fault zone. The LLCZ shows both primary pyrite beds and chaotic textures SRA geologists interpret as collapsed and compressed fragments of pyrite hydrothermal chimneys and crusts. Coarse grained dolomite alteration caps the sulphide mineralization. Texturally, a greater proportion of sulphide minerals are relatively coarse grained when compared with LUCZ and JL UCZ. The LLCZ is strongly silicified, and rests on a pervasively silicified footwall of densely fractured to fragmental shale.

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 to 200 m. The LLCZ is hosted by interbedded shale and conglomerate. Two bedding-subparallel zones of >1.2% Cu mineralization occur within the LLCZ: LLCZ subzone 1 and LLCZ subzone 2 Figure 7.8). 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° to 40°.

No mineralogical study has been completed for the LLCZ, but visually the mineralization in the LLCZ appears very similar to that of the JL LCZ.

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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 deposits (SSC deposits, e.g. Hayes *et al*, 2015).

The deposits feature large pyrite-rich 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 BBCP deposits are enriched in Cu-Co-Ag. Subeconomic Pb-Zn mineralization does occur outside the areas containing copper resources. Textural evidence indicates that some Cu 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.

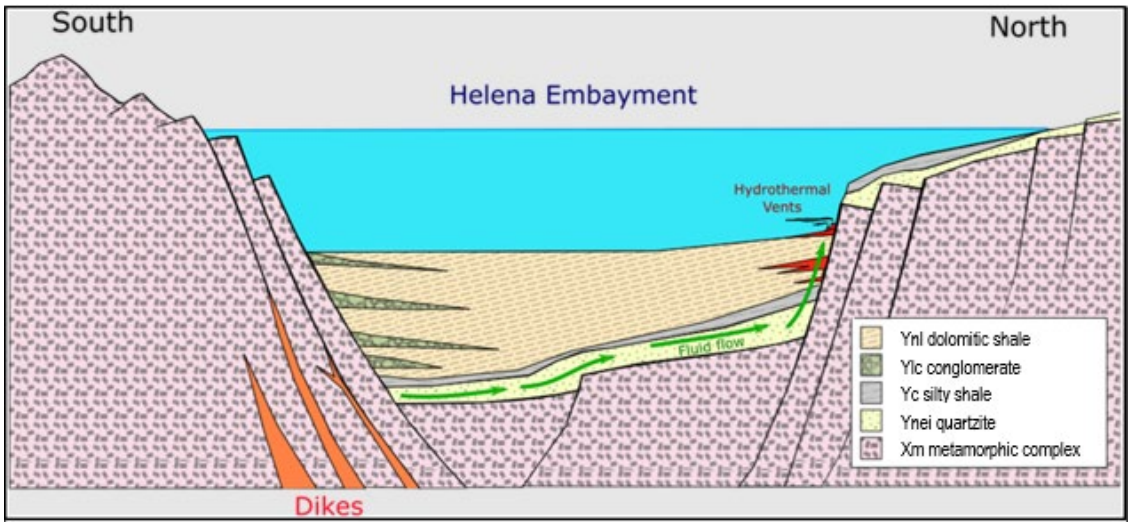
The Johnny Lee and Lowry deposits also share some features with a sub-class of SSC deposits termed reduced-facies SSC deposits: Cu-Co-(Ag) mineralization hosted by reduced, organic- and pyrite-bearing shale, silt, and carbonaceous dolomitic siltstone (Hayes *et al*, 2015). SSC deposits are epigenetic, 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 Cu, chalcocite, digenite) to more iron-rich at the top (i.e. chalcopyrite). Similar zonation is evident in the JL UCZ deposit. The association of Cu sulphide mineralization with post-lithification veins and hydraulic brecciation supports a partially epigenetic origin.

The Black Butte copper 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 Cu sulphide and associated barite. The second pulse of hydrothermal activity locally replaced the earlier pyrite with crystalline pyrite, introduced considerable Cu 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 dikes 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; White *et al*, 2014). These sulphide mounds host the copper deposits of the Black Butte Copper Project.

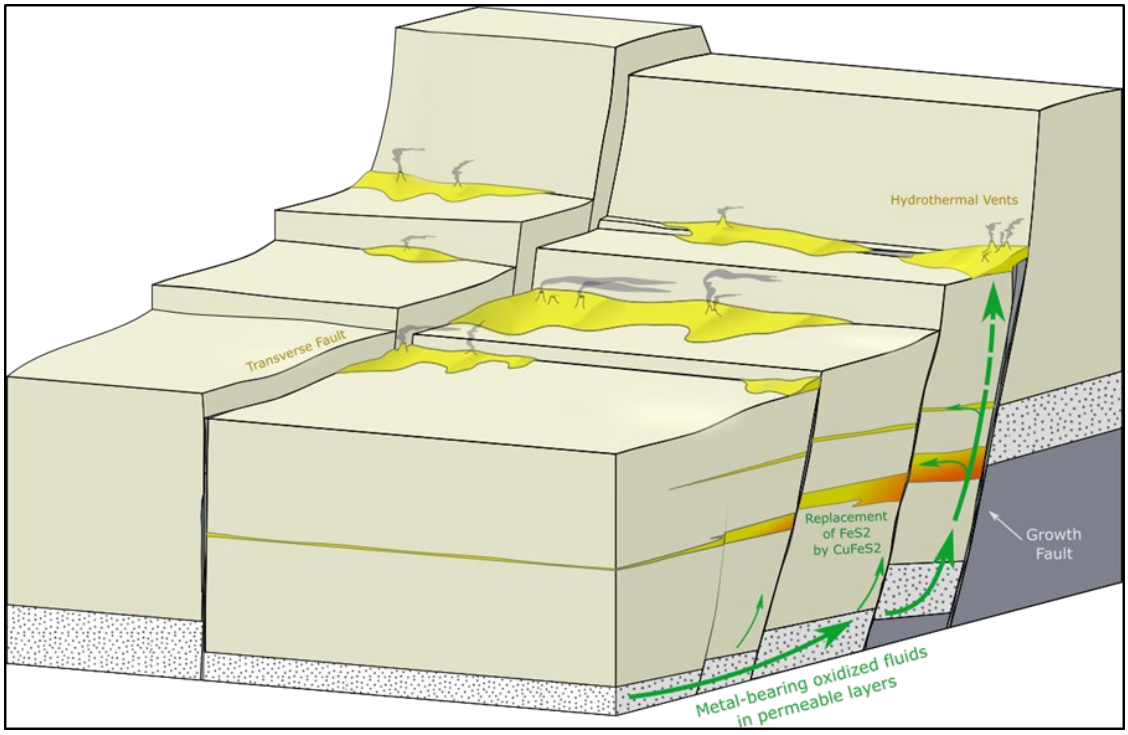


Source: SRA, 2026

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 sedimentary basin

Subsequent to formation and burial of each generation of massive bedded pyrite units, and continuing after early lithification, Cu-rich hydrothermal fluids ascended along fluid pathways along the basin margin (Figure 8.2). The origin of these Cu-mineralizing hydrothermal fluids is uncertain. White *et al* (2014) proposed that reduction of hematite in the Neihart Formation to magnetite could 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 pyrite layers formed throughout the Helena Embayment, especially along basin margin faults, the Cu mineralizing events are much more tightly 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.



Source: SRA, 2026

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

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9. EXPLORATION

9.1 Relevant Exploration Work

Homestake, CAI, UII, and BHP conducted a variety of geological, geochemical, and geophysical exploration programs between 1973 and 1995 in the Black Butte Copper Project area. This historical work included surface mapping, surface soil and rock sampling, various geophysical surveys, and core drilling. Geological 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 historical geophysical data is presently available.

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

During 2011, SRA crews collected 744 soil samples over previously un-sampled and geologically prospective areas. The sampling program was organized to provide representative samples that limit potential sampling bias. 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 501,650 East to 506,480 East and 5,176,000 North to 5,180,000 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 USZ 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 geological 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. Since 2016, SRA has conducted additional soil sampling and mapping (Figure 9.1).

9.2 Sampling Methods and Sample Quantity

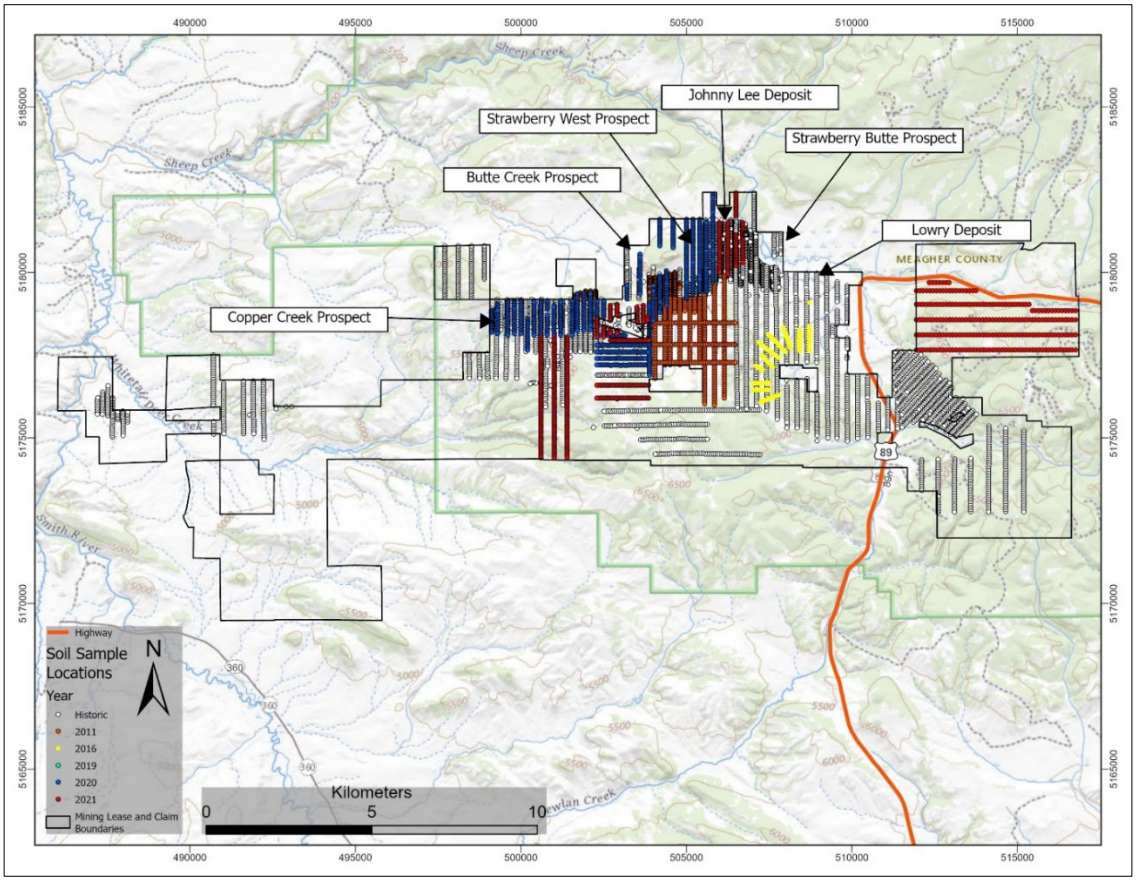
The soil sampling in 2016 consisted of 413 samples across the Newland-Chamberlain Formation contact on the east-central portion of the property (Figure 9.1). The aim of the program was to test the lower portion of Newland Formation stratigraphy for a geochemical signature of a potential extension to the JL LCZ mineralization. The sampling program was organized to provide representative samples that limit potential sampling bias. 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 µm, then digested in aqua-regia and analyzed for 35 elements using ICP-MS.

9.3 Significant Results and Interpretation

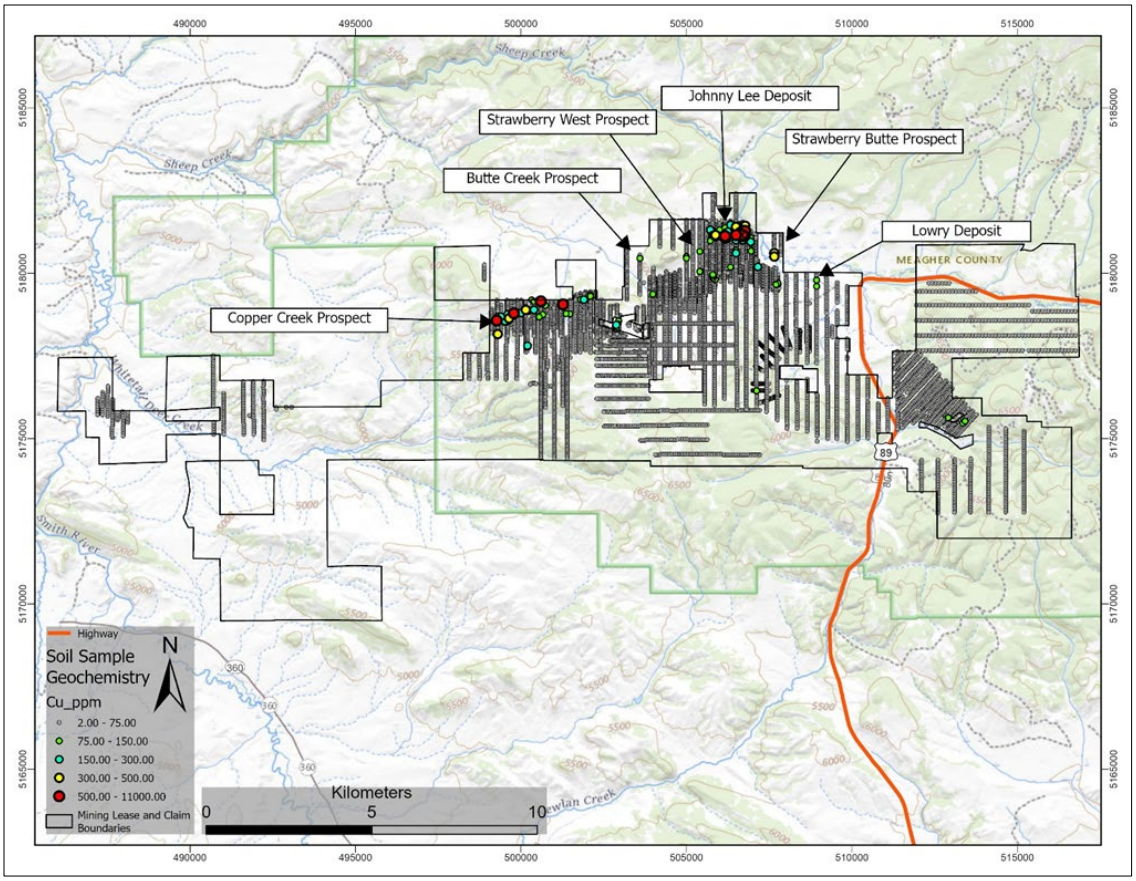
Figure 9.1 shows the location of all soil sampling campaigns over the property with associated named prospects. Figure 9.2 and Figure 9.3 show copper and zinc results, respectively, from these sampling campaigns.

The 2016 soil sampling program failed to indicate significant new anomalies. Historical 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. Additionally, soil sampling outlined zinc anomalies in the Copper Creek area.



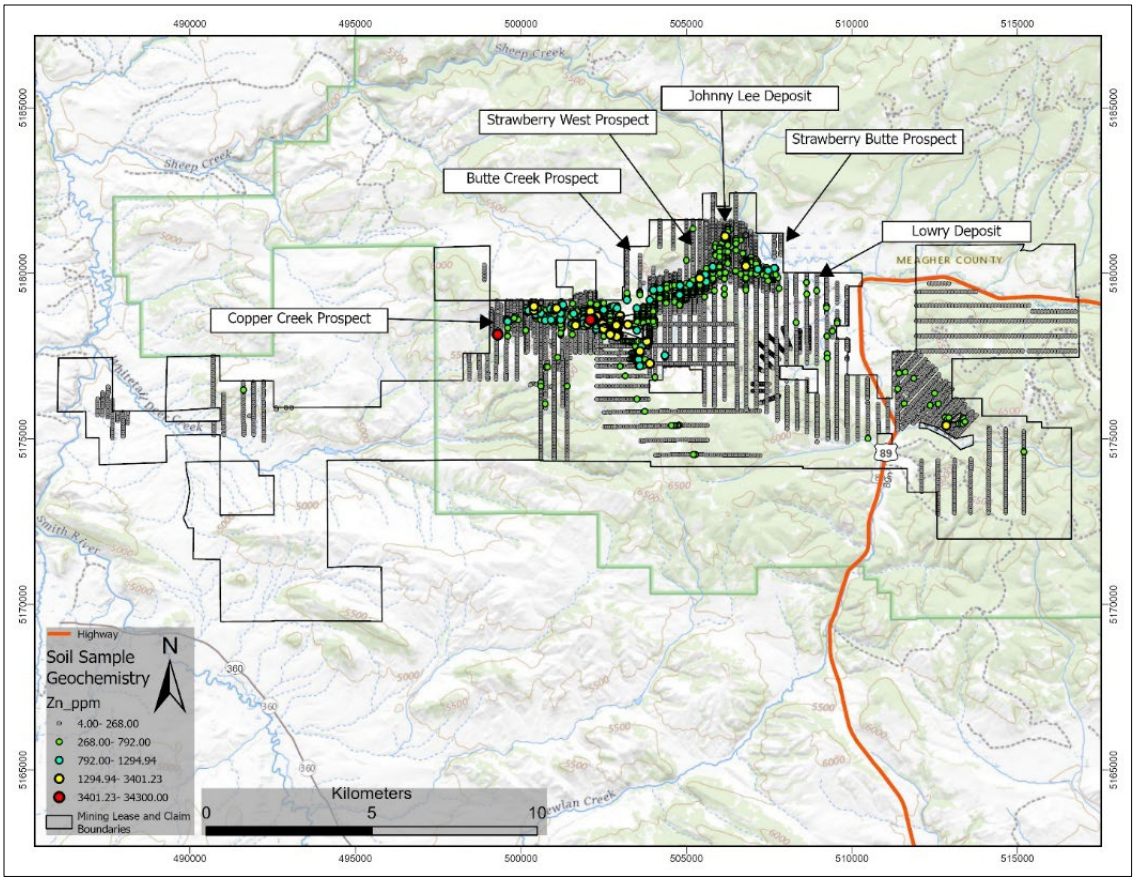
Source: SRA, 2026

Figure 9.1 Plan map showing soil sampling campaigns on the property



Source: SRA, 2026

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



Source: SRA, 2026

Figure 9.3 Plan map showing soil sampling Zn results, deposits and exploration targets

9.4 Exploration Prospects

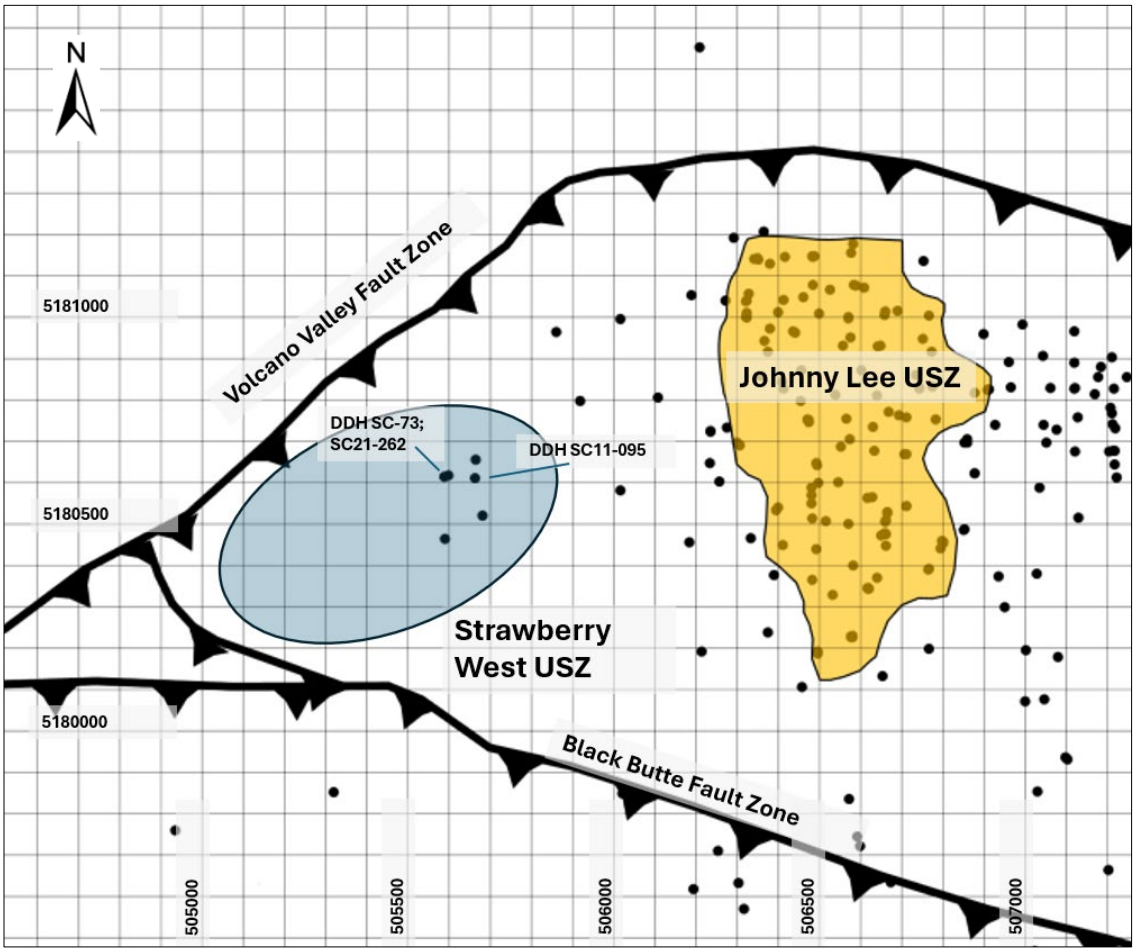
There are multiple exploration prospects in the vicinity of the Johnny Lee and Lowry deposits. There are no Mineral Resources defined on the exploration prospects. The QP for Mineral Resources has not reviewed the data associated with any prospect beyond the Johnny Lee and Lowry deposits, as these exploration prospects are not the focus of this Technical Report. The following information is provided from SRA.

9.4.1 Strawberry West Prospect

In addition to soil samples, six 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 9.4). 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.
- SC21-262: 6.8 m at 1.20% Cu from 547.80 to 554.61 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 JL UCZ but is not a continuation of this deposit, rather it represents a discrete zone of mineralization for future studies.



Source: SRA, 2026

Figure 9.4 Plan view showing the Strawberry West prospect area and the locations of the VVFZ, BBF, and Johnny Lee USZ and drillhole collar locations

9.4.2 Butte Creek Prospect

The Butte Creek area contains a 2.5 km strike length of Newland Formation with abundant sulfide zones. Nine widely spaced drillholes tests various levels of sulfide mineralized Newland Formation stratigraphy. Six drillholes tested USZ and higher sulfide zones, and these show thin zones of anomalous Cu mineralization, generally less than 0.2% over a sample width. Four of these holes continued in depth to test deeper sulfide zones including a thick (200 m) section with abundant thin zones of laminated to bedded sphalerite and

galena mineralization interlayered with shale and pyrite beds. Two additional holes collared below USZ stratigraphy, tested the Zn-bearing stratigraphy and encountered similar concentrations of Zn mineralization. SC-9 drilled the thickest composite with 24.68 m of 1.7% Zn and SC-54 drilled the highest-grade single sample with 0.3 m of 18.3% Zn. Zones of higher Zn concentrations occur scattered over as much as 550 m of drill hole SC-10. Anomalous Cu concentrations occur near the bottom of SC-9 (9.14 m of 0.3% Cu with 1.5 m of 1.0% Cu). SC-11 cut a 0.76 m zone of 1.13 % Cu with characteristics of the LSZ (dark shale with abundant dolomite alteration and coarse-grained sulfides) below a shear zone, likely a component of the VVFZ. This area has received no drilling since 1991.

9.4.3 Copper Creek Prospect

The Copper Creek Prospect has a 2,500 m strike length with a robust Cu in soil anomaly (Figure 9.2). This surface anomaly coincides with mapped gossan that SRA geologists interpret as oxidized sulfide zones of the Newland Formation. 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. SCC-44 passed through a partial USZ with 3.7 m containing 0.15% Cu, and below this encountered Cu-mineralized fragments caught up in the VVFZ with one 0.3 m interval containing 1.4% Cu. SRA geologists believe this fragment must have sourced from the LSZ. Approximately 500 m west, SC-66 intersected a 27.4 m thick USZ with 0.1% Cu, and below the VVFZ the drillhole pierced a 7.16 m thick LSZ with 380 ppm Cu (816 over 1.5 m) and Zn (0.2%). This LSZ test is 6 km west of the Johnny Lee deposit. 570 m west of SC-66, SCC-27 drilled a portion of the USZ preserved against the VVFZ. TRI's 2012 Aeroquest airborne geophysical survey identified some low resistivity anomalies that appear to correlate with high sulfide areas; these have not been drill tested. The Copper Creek area has received no drilling since 1989.

9.4.4 Newlan Creek Prospect

This 8 km by 5 km area includes outcropping LSZ and basal Newland Formation stratigraphy that produces outcrop and float of silicified conglomerate and small amounts of gossan float. Soil sampling has shown scattered low-level Zn-Pb and Cu anomalies associated with this stratigraphy. Limited drilling in the area. Five holes test the LSZ stratigraphy in the vicinity of the Hi Tariff patented claim, an occurrence of high Ag concentrations associated with the intersection of pyritic black shale interpreted as LSZ stratigraphy in contact with a clay altered Eocene felsic dike. Drill hole intersections of silicified conglomerate in three holes show anomalous but <1% concentrations of Zn and Pb. The 2012 Aeroquest airborne geophysical survey shows low resistivity anomalies downdip from portions of the LSZ outcrop belt. No drilling has taken place in the area since 1991.

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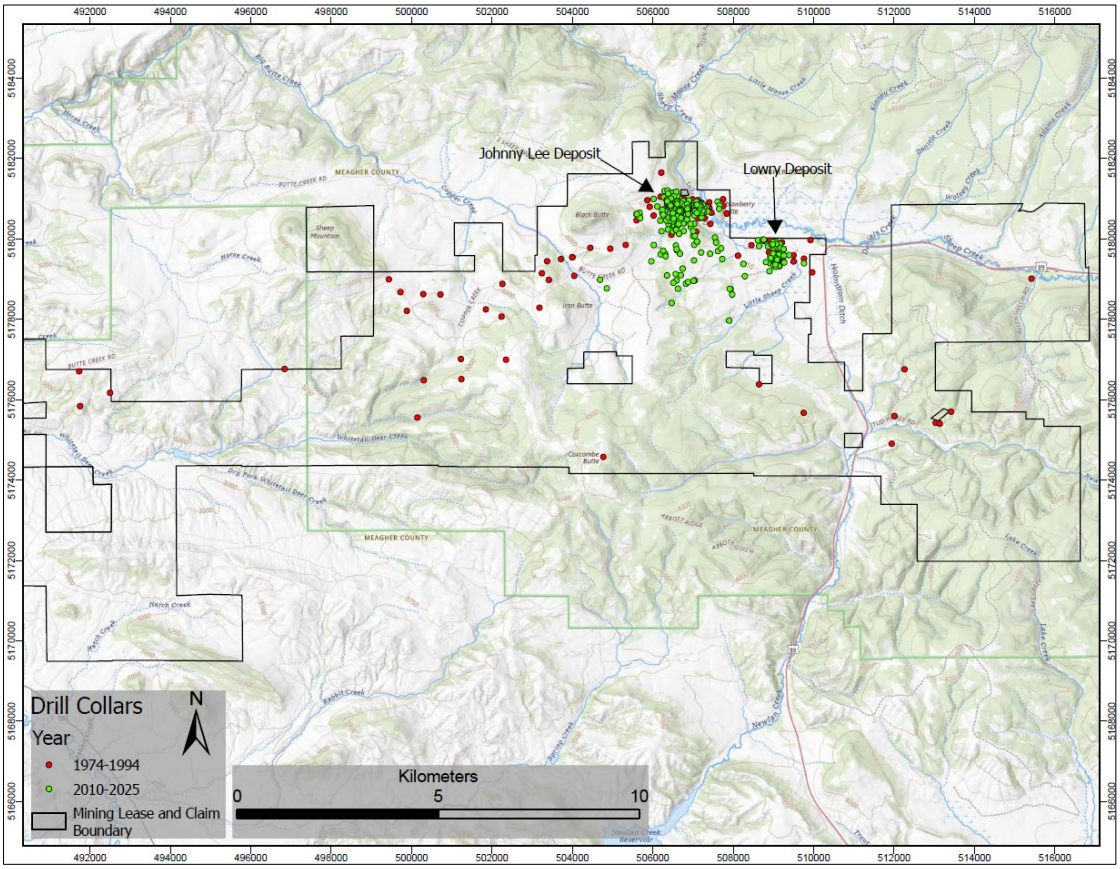
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10. DRILLING

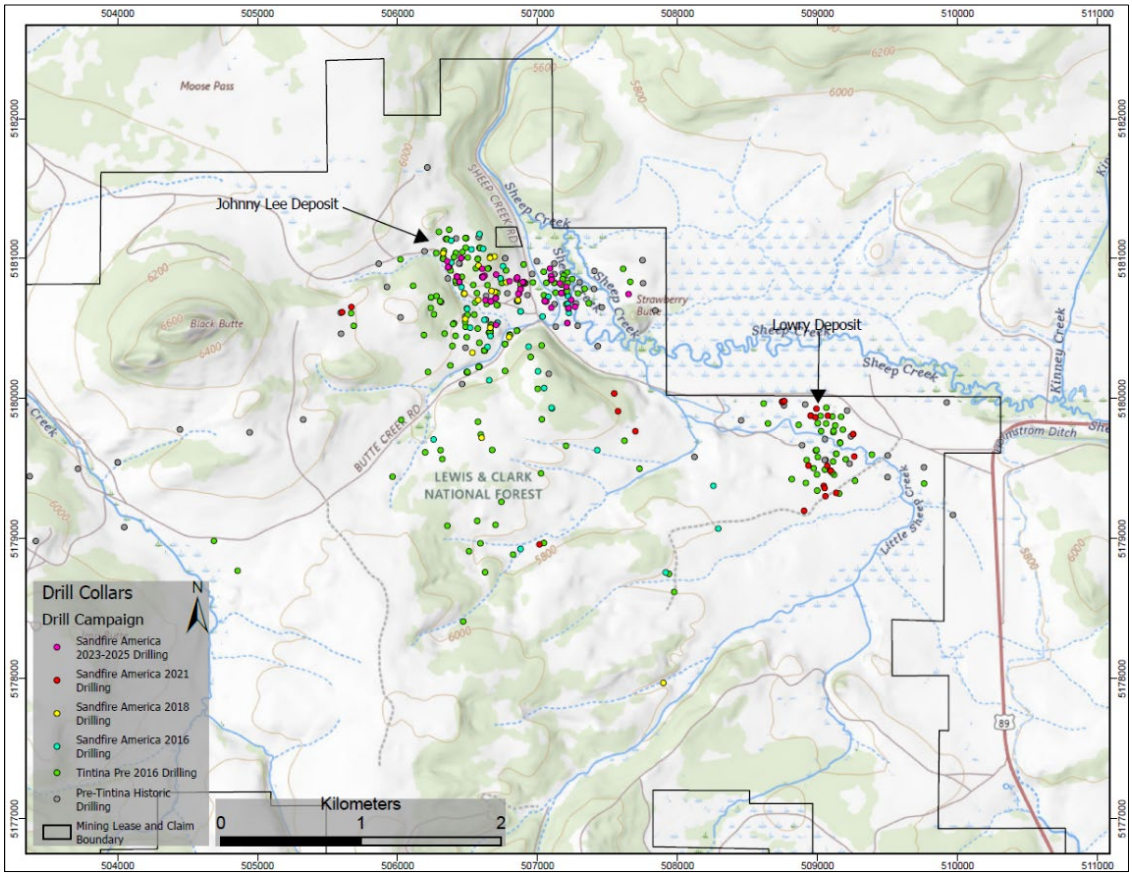
10.1 Type and Extent

The early exploration work of Homestake (1973–74), Anaconda (1977–80), and Exxon (1981–83) covered the extent of their respective portions of the district. Homestake drilled a single core hole in the Copper Creek area near the centre of the property package. Of the original 22 Anaconda drill holes, only one core hole and four RC holes fall with the current property package on the very western edge of the property. Figure 10.1 shows the pre-Tintina (red) and Tintina/SRA (green) drillhole collars. Figure 10.2 shows drillhole collars at Johnny Lee and Lowry Zones and in the areas of the proposed Johnny Lee mine facilities. Table 10.1 provides a summary of drilling by historical company.



Source: SRA, 2026

Figure 10.1 Project area showing tenement boundaries and collar locations completed from 1974 – 1990 (Pre-Tintina/SRA) and 2010 to 2025 (Tintina/SRA)



Source: SRA, 2026

Figure 10.2 Drillhole collars at Johnny Lee and Lowry Zones and in the areas of the proposed Johnny Lee mine facilities

Company	Years	No. of Holes	Core Meters Drilled	Core Diameter	RC Meters Drilled	Drill Purpose
Homestake Mining	1974	1	0	Unknown	148	Exploration
Cominco American Inc.	1978-1984	11	6381	Unknown	0	Exploration/Resource/Metallurgy
Anaconda Mining	1978	5	476		1397	Exploration
EXXON	1983	1	220	Unknown	0	Exploration
BHP-Cominco JV	1985-1988	33	12764	Unknown	166	Exploration
Cominco American Inc.	1988-1992	41	20293	HQ	0	Resource/Exploration
Tintina Montana Inc.	2010-2012	168	54433	HQ	0	Resource/Metallurgy/Geotechnical
Sandfire Resources America	2014	11	3245	HQ	0	Resource/Metallurgy
Sandfire Resources America	2015	26	999	HQ	0	Geotechnical
Sandfire Resources America	2018-2019	50	11040	PQ/HQ	0	Johnny Lee Metallurgy/Resource/Geotechnical/Sterilization
Sandfire Resources America	2021	8	5267	HQ	0	Exploration
Sandfire Resources America	2021-2022	19	8937	HQ	0	Lowry Resource/Metallurgy/Geotechnical
Sandfire Resources America	2023-2025	82	28,077	HQ	0	Johnny Lee Lower Zone expansion and infill
Total		**456**	**152,132**		**1712**	

Table 10.1 Black Butte project drilling summary

Cominco American Inc. (CAI) conducted significant exploration drilling prior to their JV with UII/BHP between 1976 and 1985. Eleven of these core holes lay in the current property package and are largely in the area around Copper Creek, and Butte Creek and some farther west.

The BHP-CAI Joint Venture drilled 33 holes on the current property package including the discovery and partial definition of the Johnny Lee Zone, as well as many exploration holes distributed across the property. The Joint Venture drilled RC Hole (SCR-28) on the property in the area of the JL UCZ. It was drilled as a twin of a core hole in the Johnny Lee Upper Zone (SCC-21). The RC twin was found not to adequately repeat the earlier core hole, so it was not utilized in the subsequent Johnny Lee Resource estimations.

Between 1988 and 1992, CAI continued resource definition in the Johnny Lee and Lowry areas as well as exploration drilling both within and beyond the current property boundaries. Forty-one of these core holes lay within the current property package.

Mantra Mining Inc. (later renamed Tintina Resources Inc. (TRI)) leased the mineral rights containing the Johnny Lee and Lowry deposits in 2010. From 2010 to 2012, TRI 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. In 2010, given the absence of assay certificates, downhole survey records, and quality assurance and quality control (QA/QC) results for the historic drill data, Tintina twinned six of the historic drillholes 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 preliminary economic assessment (PEA) completed in 2013. 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. The PEA included a Mineral Resource declaration for the Johnny Lee Upper Copper Zone (JL UCZ), the Johnny Lee Lower Copper Zone (JL LCZ), and the Lowry Middle Copper Zone (LMCZ) deposits.

Sandfire Resources America (SRA) acquired an interest in the project in 2014 and, in that year, TRI drilled 11 diamond drillholes into the JL UCZ for Mineral Resource infill, geotechnical evaluation, and metallurgical test-work. Twenty-six 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 JL UCZ recoveries that had previously not been identified. In 2017, TRI was renamed Sandfire Resources America (SRA). SRA initiated a geometallurgical study that constrained the spatial distribution of different geometallurgical characteristics of JL UCZ.

During the 2018 Phase 1 drill program, 19 diamond drillholes were completed into the JL UCZ to acquire samples for ore characterization, and 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 further constrain the geometallurgical model. During the same program, two drillholes were completed in the JL 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:

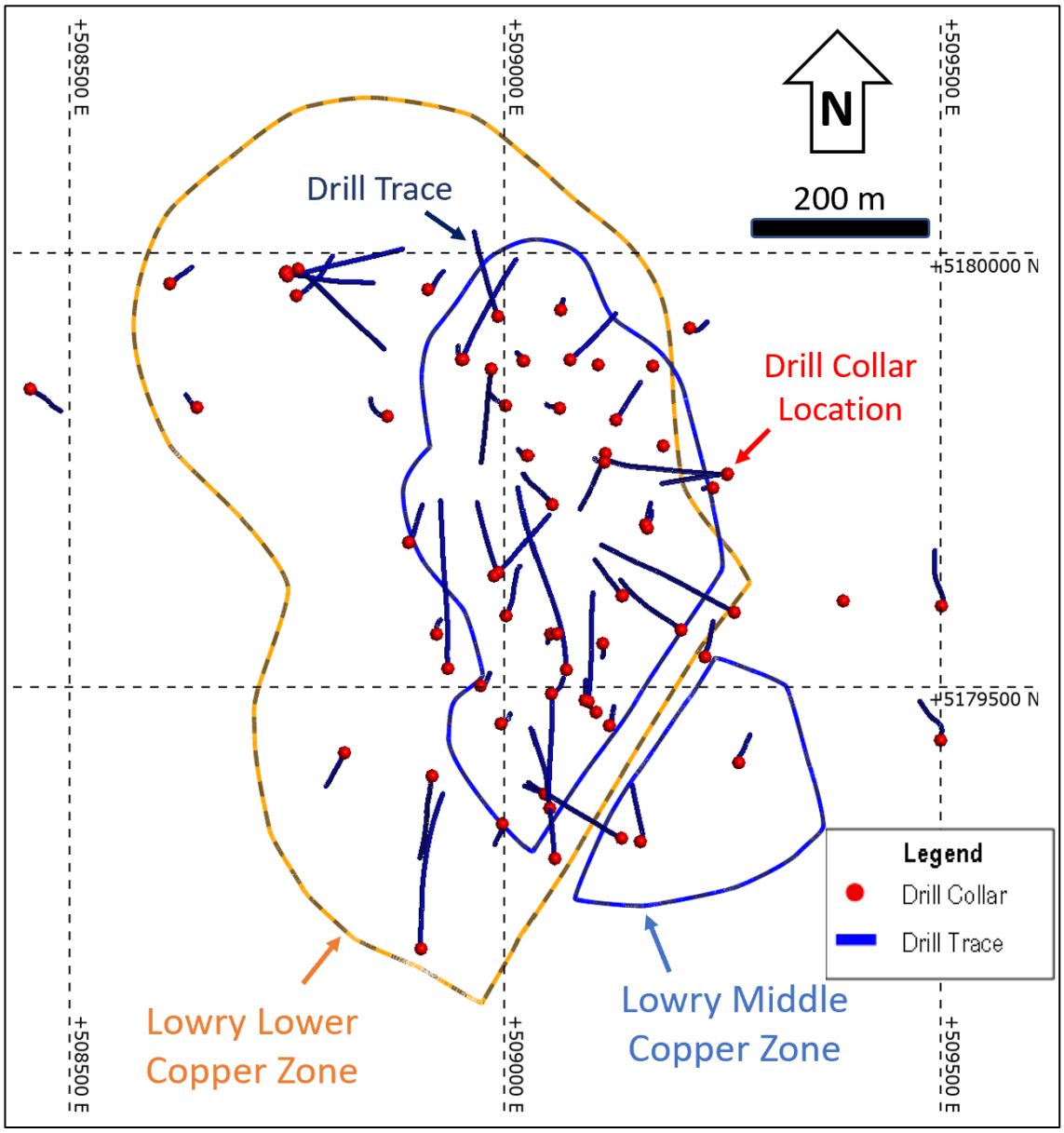
- Mineral Resource infill/geometallurgical definition drilling in critical parts of the JL UCZ and JL LCZ.
- Acquisition of geotechnical information for infrastructure, underground development and stope design.
- Hydrogeological testing of the proposed decline.
- Collection of additional metallurgical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

A total of 26 diamond drillholes were completed in the 2018/19 Phase 2 drill program for a total of 6,115 m. All drillholes in these areas are diamond drillholes that were either pre-collared using a tri-cone bit or cored from surface.

Between January and March of 2021, SRA completed 7 diamond exploration holes and from October 2021 to March of 2022 completed 19 core holes for additional definition of the Lowry Middle Copper Zone (LMCZ) and Lowry Lower Copper Zone (LLCZ). Figure 10.3 below shows the distribution of core drilling at the Lowry Deposit.

Between January 2023 and May 2025, SRA drilled 82 diamond drill holes totalling 28,077 meters in the Johnny Lee Area to expand the Johnny Lee Lower Copper Zone resource, infill drilling the previously Inferred Mineral Resource to increase confidence. Additionally, this drilling campaign provided samples for metallurgical and geotechnical test work.

Since 2014, TRI and SRA drilled at total of 196 diamond core holes over six drilling programs. The report sections below outline the drill procedures utilized in those campaigns and highlights procedural changes that were implemented as the project advanced.



Source: SRA, 2026

Figure 10.3 Lowry deposit showing drillholes and the surface projection of the Lowry Middle Copper Zone (LMCZ) and Lowry Lower Copper Zone (LLCZ)

10.2 Procedures

10.2.1 Grid

All drilling and survey activities at the Black Butte project use 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 surveyed using either a handheld Global Positioning System (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 and for subsequent programs, 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 TRI drilling programs, a Reflex EZ-Trac™ electro-magnetic 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, and all subsequent programs, a Reflex EZ-Trac™ survey instrument was used to record downhole survey data during drilling. Upon drill hole completion, a Reflex EZ-Gyro™ (north seeking gyroscope) was used to survey each hole at 3 m to 6 m intervals.

Acceptable correlations between EZ-Trac and EZ-Gyro instruments and low magnetic susceptibility readings indicate that magnetic interference of electro-magnetic survey instruments was not occurring. The 2023/2025 drilling used a Omnix38 and Omnix42 instruments. The Omnix38 was used for steeper holes while the Omnix42 was used for less steep holes.

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 Lowry Middle Sulphide Zone, the VVFZ, and within the Johnny Lee UCZ deposit near potential faults.

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 daily from the drill rig site by company field 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 mark-up.

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.

After 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 company server.

Specific gravity determinations were then made of whole-core intervals using an Archimedean (wet/dry) scale. Further description of specific gravity testing methodology is described in section 11 and 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 JL 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, colour, 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.

For the 2021/2022 drill program, core holes were logged by hand on paper sheets. This information was then entered into OCRIS™ for transfer to the database team in Perth where it was entered into the master database. Prior to transfer, electronic files were reviewed by the site team and checked for consistency errors.

For the 2023/2025 drill program, core holes were logged by hand on paper sheets. This information was then entered into MXDeposit™ for transfer to the database team in Perth where it was entered into the master database. Prior to transfer, electronic files were reviewed by the site team and checked for consistency errors.

10.2.7 Geotechnical Logging

In 2014, TR 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 Johnny Lee UCZ and JL LCZ. The geotechnical and structural data was used to inform a geotechnical evaluation of the JL UCZ and JL LCZ by Mine Design Engineering. A geotechnical engineer from Mine Design Engineering, trained TR geologists in geotechnical core logging and monitored data quality during the logging process. The logging protocols utilized in this program 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 JL 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.

Core for the 2021/2022 and 2023/2025 drill programs was oriented, where possible, by the drill crew using a Reflex ACT III tool. Core samples were matched with orientation data using a leveling jig. The bottom or top orientation is transferred to the core sample by a line scribe method. Once at the logging facility, a handheld Reflex IQ Logger is used to collect the structural orientations in the core. Down hole readings were recorded in feet and then converted to metres for 3D analysis.

10.2.8 Core Recovery

Core Recovery in the JL UCZ is considered acceptable with average recoveries greater than 90%. Very low core recoveries (<60%) in the JL UCZ generally occur in isolated individual core-runs and thus do not have a significant impact on overall grade estimation or Mineral Resources. For the 2023–2025 Johnny Upper Copper Zone drilling, recoveries in the main mineralized zones (JL UCZ HG Cu, Cu 1, and Cu 2) averaged above 92%.

Although poor core recoveries were encountered in the Volcano Valley Fault zone (VVFZ), the recoveries within the JL LCZ are considered acceptable. Historically, core recoveries in the JL LCZ have averaged above 98%. For the 2023–2025 Johnny Lee Lower Copper Zone drilling, recoveries in the main mineralized zone (JL LCZ Main) averaged above 99%.

Core recoveries from the Lowry deposit are considered acceptable with recoveries for the different mineralization zones ranging from 0 to 100%. Core recoveries from Lowry Middle Copper Zone (LMCZ) average 89.3%. Five core runs returned recoveries <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 for the LLCZ (encapsulated within the LLCZ Halo boundaries) average 86.6%.

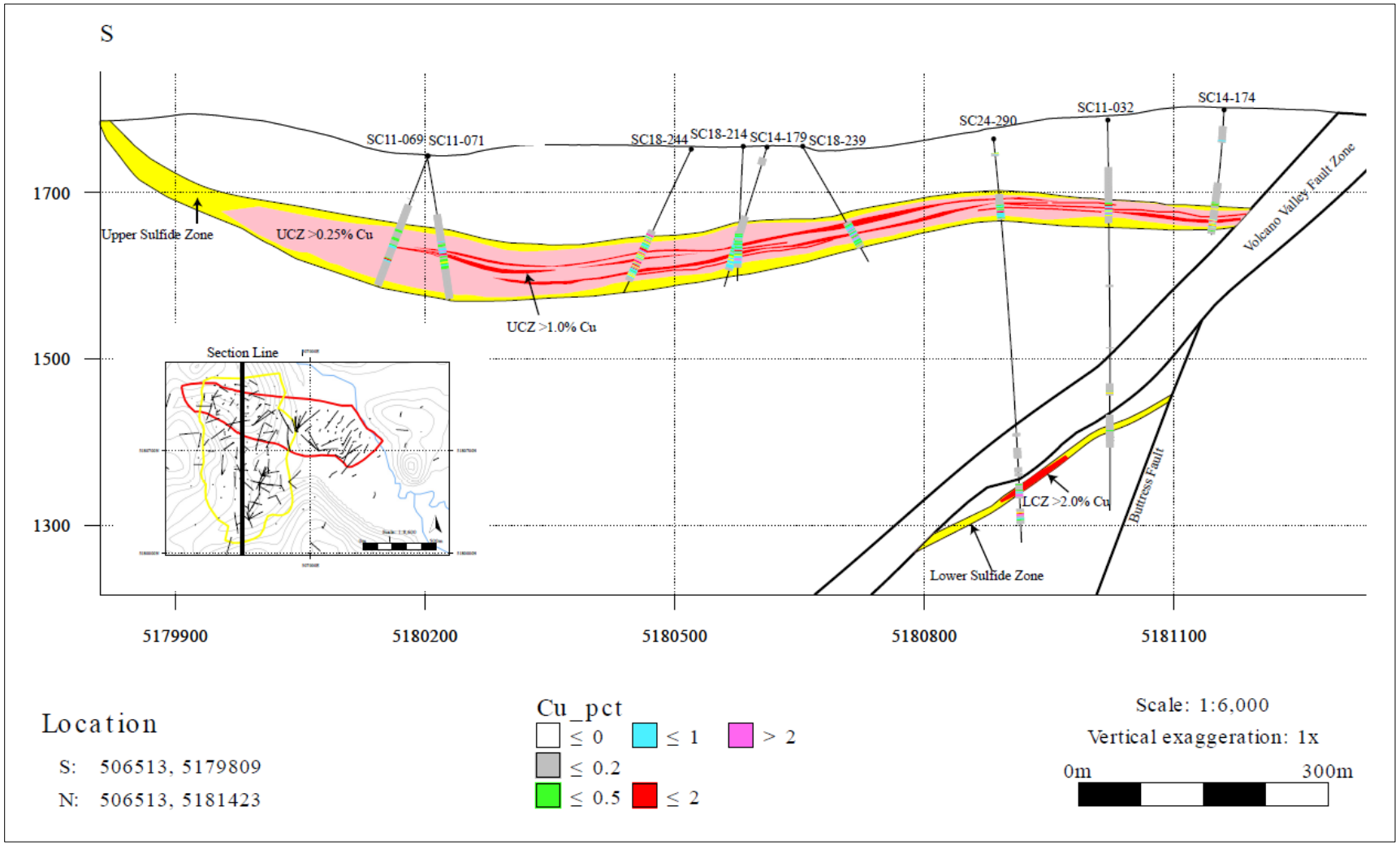
10.2.9 Drill Core Storage

In 1995, CAI donated all its drill core from the district as well as a copy of the district electronic drillhole database, including the data from the drillholes, to the University of Montana's Belt Research Center at Fort Missoula near Missoula, Montana, after they terminated their interest in the project. In 2010, Tintina obtained a copy of this data for review. 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. In 2018, the University of Montana terminated their management of the Belt Research Center core collection and gave drill core relevant to Black Butte Copper and the surrounding region to SRA. Drill core from the historical programs and recent SRA drilling are stored in a secured core shed on the Black Butte Copper site.

10.3 Interpretation and Relevant Results

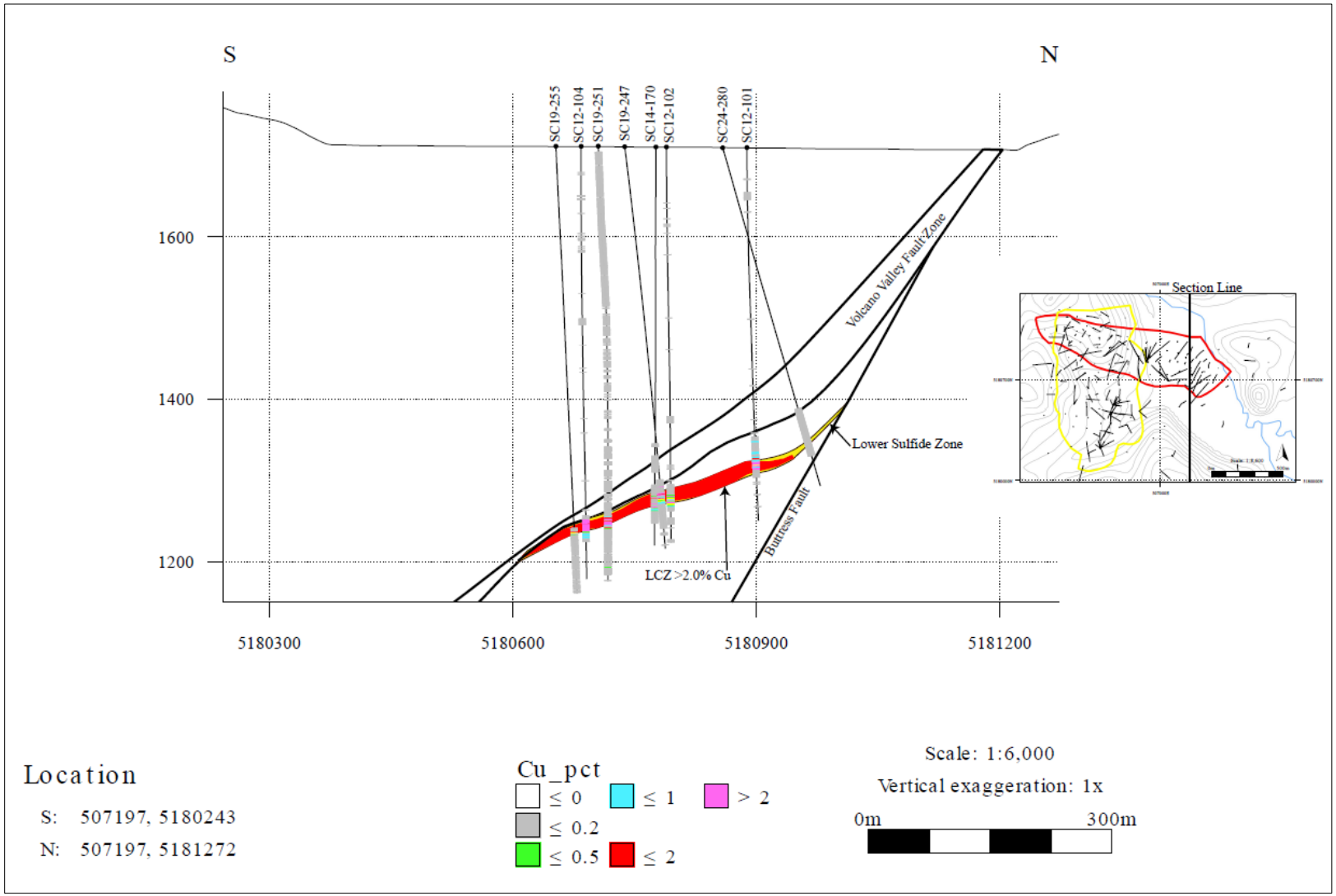
The JL UCZ (Figure 10.4) is a flat-lying to shallow dipping zone of Cu mineralization that ranges in thickness from 3 m to 42 m. Gentle folding about a W/NW trending syncline-anticline pair results in a variable dip that range from 0° to 20°. The northern edge of the JL UCZ is truncated by the hanging-wall fault of the VVFZ. The JL UCZ is transected and displaced by the N/NE striking, steeply W/NW dipping F1.

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



Source: SRA, 2026

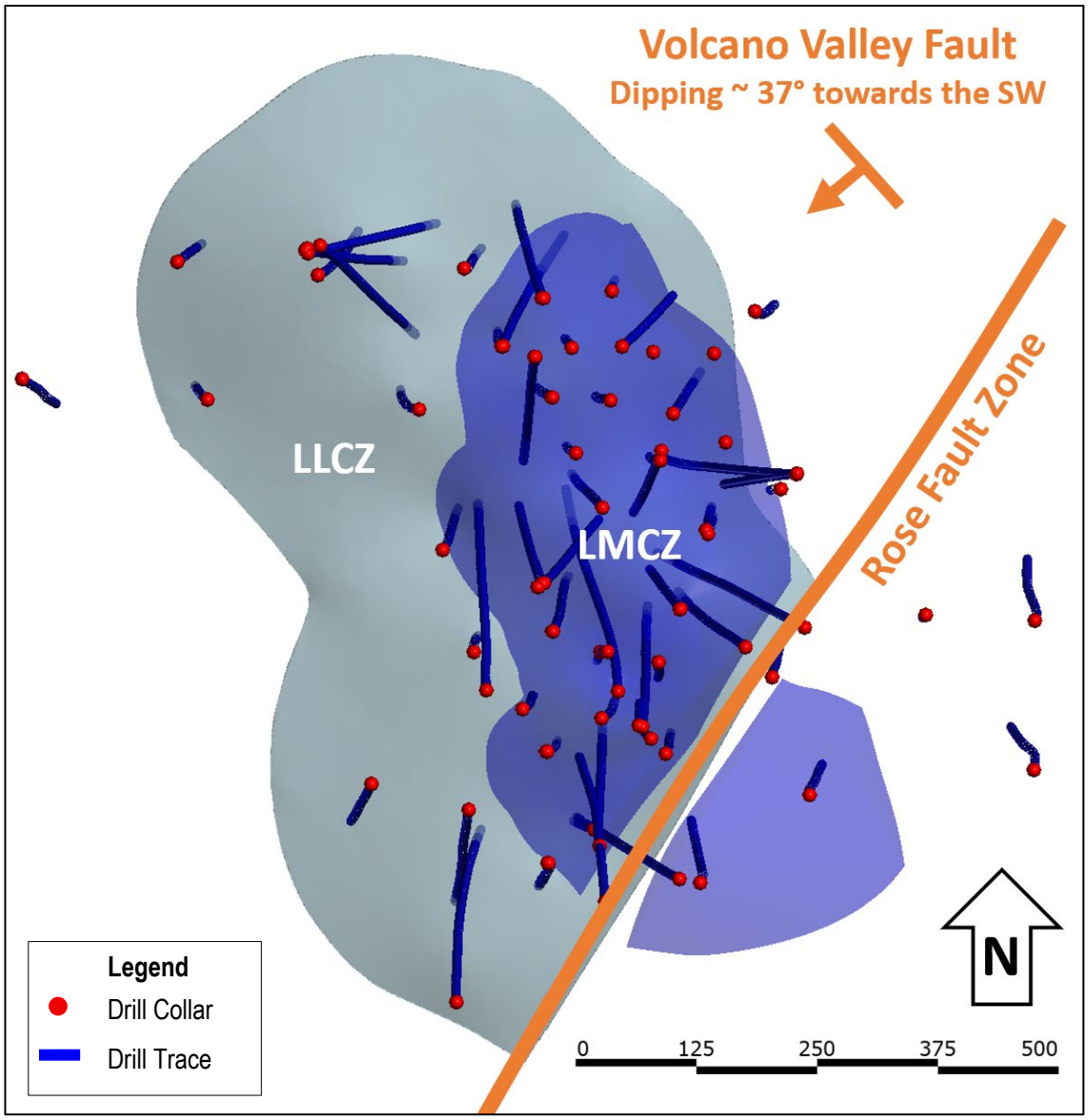
Figure 10.4 Cross-section of the Johnny Lee Deposit; view to west. See inset for line of section



Source: SRA, 2026

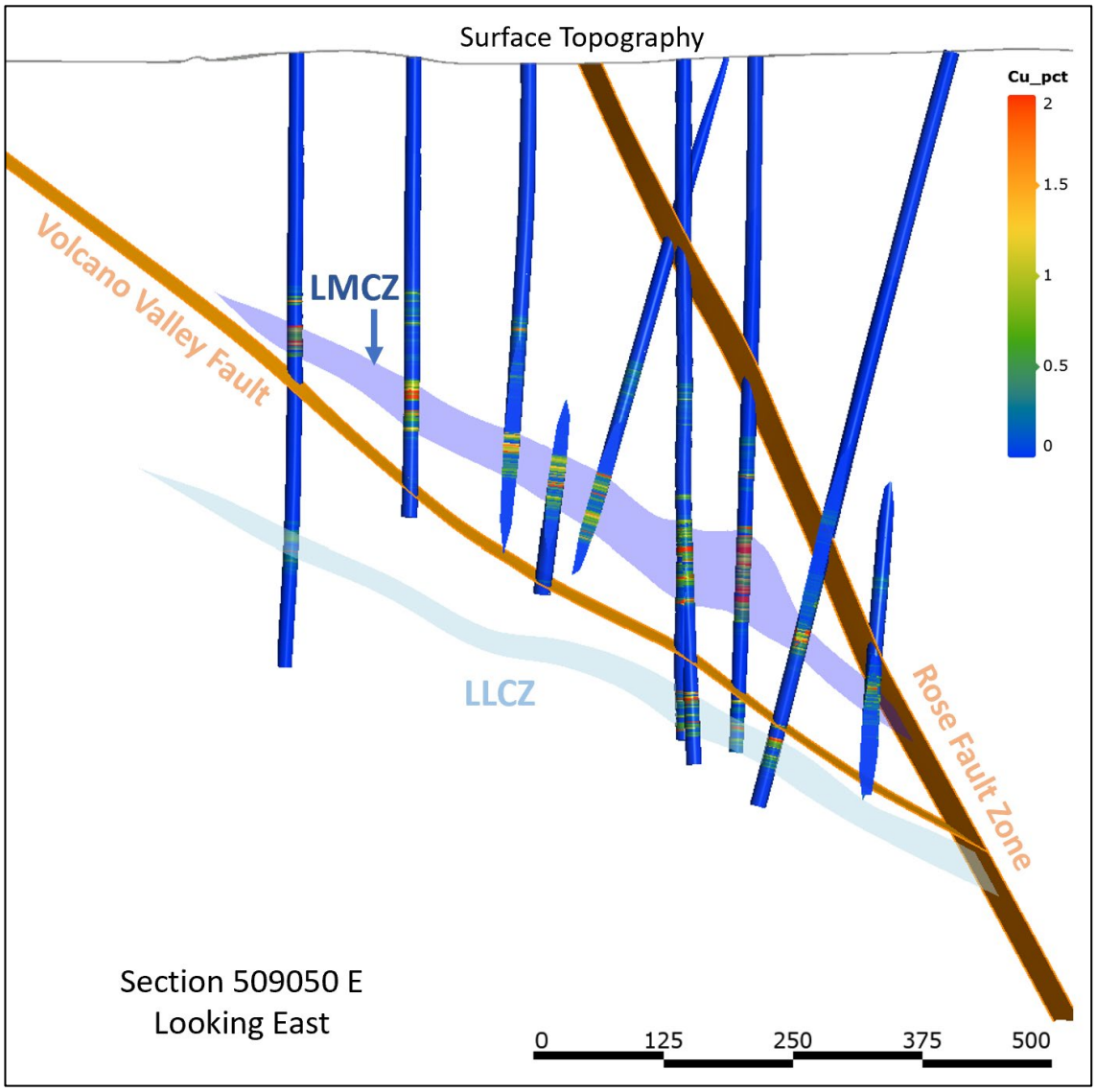
Figure 10.5 Cross section of the Johnny Lee LCZ; view to west. See inset for line of section

The Lowry LMCZ and LLCZ are shallow south dipping zones of mineralization that occur in the hanging wall of the VVFZ. Figure 10.6 and Figure 10.7 show the main Lowry mineralized zones in plan view and cross-section, respectively.



Source: SRA, 2026

Figure 10.6 Plan view of Lowry LMCZ and LLCZ



Source: SRA, 2026

Figure 10.7 Cross-section of the Lowry LMCZ and LLCZ, looking east

10.4 QP Opinion on Accuracy

Berkley Tracy, P.Geo (PGO# 3024) of SRK, acting as QP of Mineral Resources has reviewed the drilling data, procedures, and methodology, including a site visit to the Johnny Lee and Lowry project area. It is the opinion of the QP that all drilling and sampling has been performed to industry standard and the resultant geological and assay data are acceptable for use in the Mineral Resource estimation.

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11. SAMPLE PREPARATION, ANALYSIS AND SECURITY

11.1 Historical Sample Preparation

Specific records are limited for sample preparation and analytical procedures used by historical operators prior to Tintina and SRA ownership (2010). Copies of historical paper laboratory records are available for some early drillholes in the White Sulphur Springs office. Winckers *et al* (2013) reported that the laboratories used for Cominco America, Utah International and BHP samples were Silver Valley Laboratories in Kellogg, Idaho, Bondar Clegg in Vancouver, British Columbia, and an in-house BHP lab in Sunnydale, California. It was reported that "sampling was completed to the standards of the time" and that QA/QC samples (i.e., blanks, standards, duplicates) were used to track lab performance (Winckers *et al*, 2013). No known bias exists in the earlier sample grades versus later analyses that would indicate the historical laboratories were not following established preparation and analytical protocols.

For the Tintina 2010 to 2012 drilling programs, core samples were analysed by ALS Chemex Laboratory in Vancouver, British Columbia. Some samples were prepped into pulps at the ALS Chemex facility in Reno, Nevada prior to shipment to Vancouver. The ALS Chemex laboratory was certified to ISO 9001:2000 standards and operated independently from Tintina. As reported in Winckers *et al* (2013), core samples were crushed to 70% less than 2 m, then a 250-gram split was pulverized to 85% less than 75 µm. Thereafter, the splits were processed with four-acid digestion and analysed by inductively coupled plasma method for a 33-element suite (ME-ICP61a) and mineralized-material grade copper (Cu-G62). Additional details on historical sampling methodology are available in the 2013 PEA (Winckers *et al*, 2013). The QP for Mineral Resources has reviewed the previous technical documentation and is comfortable that historical security, sample preparation, and analytical protocols were consistent with generally accepted industry standards during the Tintina drilling programs.

11.2 Sample Preparation for Analysis

All Sandfire Resources America, Inc. (SRA) 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 Johnny Lee 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, and barcoded. After organization, individual samples are managed by the ALS sample management system.

The samples were dried for a minimum of eight 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 sub-sample collected for fire assay by scooping an x-pattern through the pulp pile (i.e., similar to the cone and quartering method of sub-sampling). A 25 to 50 g sub-sample was collected in the same way for acid-digest inductively coupled plasma – atomic emission spectrometry (ICP-AES). The remaining pulp material (pulp residue) was bagged and stored. Envelopes containing the acid-digest ICP-AES sub-sample were shipped to ALS Vancouver for testing, located at 2103 Dollarton Highway, North Vancouver, BC, Canada.

Whole core intervals from the 2018 Phase 1 JL 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 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 remaining material could be re-analyzed at a different laboratory (Umpire Samples).

11.3 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.

11.4 Sample Analysis

Upon completion of sample preparation at ALS Reno, a split of the analytical pulp was sent by ALS Global to ALS Vancouver for analyses. ALS Vancouver is ISO/IEC 17025 accredited for gold assay by lead collection fire assay, four acid sample digestion and multi-element analysis using 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.

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

Gold 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 (AAS) analysis. This technique allowed for Au analyses up to 10 ppm with a lower detection limit of 5 ppb. On the occasions that gold concentrations exceeded 10 ppm, repeat analyses were performed by fire assay with a gravimetric finish.

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, five 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	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

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	0.00	30

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

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%

Element	Lower Limit of Detection (ppm)	Upper Limit of Detection	Element	Lower Limit of Detection (ppm)	Upper Limit of Detection
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 over-limit umpire samples were re-analyzed for the over-limit 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 over-limit ICP-AES analyses at AAL

11.4.1 Specific Gravity Analyses

Specific gravity (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 specific gravity based on the mass 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 SGS 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. Figure 11.1 shows the SG testing apparatus used from 2018–2022. These data are 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 bulk density and quality control checking.



Figure 11.1 Specific Gravity Testing Apparatus at the Sandfire Resources America Inc Logging Facility 2018-2022

11.4.2 2023–2025 Specific Gravity Procedure

Beginning in 2023, a new more accurate SG procedure was used. Figure 11.2 shows the updated SG station at the SFA logging facility. The scale is an Ohaus Ranger 7000. This scale does not automatically calibrate and must be manually calibrated periodically using 6 Troemner ASSTM E617-18 Class 5 calibration weights. The set of weights has a Serial number of 184857 and certificate number 900894282-1 issued on 12 February 2025.



Figure 11.2 Specific Gravity Station in SFA Core Logging Facility

Scale calibration was completed about every 3 months, depending on usage. Very little analytical drift was noted during each calibration. Each weight is serialized and certified for calibration. Table 11.5. below shows the precision of each weight, as well as its density.

Weight Number	Calibrated Mass (g)	Density (g/ml)
1	2268.0097	7.85
2	2268.0107	7.85
3	2268.0347	7.85
4	2268.0067	7.85
5	2268.0017	7.85
6	2267.9997	7.85

Table 11.5 Scale Calibration Weights

Once the core is properly logged and marked for sampling, the samples are then cut as outlined in the sample preparation section and then measured for specific gravity. The half of the core not sampled and left in the box is the part that is measured for SG. This is to reduce bottlenecks at the SG station and to also cut down on cross contamination from touching the same water tank. The core is placed in a modified metal bread pan that is attached to the bottom of the scale. Since specific gravity is an intensive value independent of quantity, only a representative portion of the core is used in SG measurements. Once a subsection of the core is placed in the bread pan, a hand crank is used to bring the tank of water up to the core. Fault gouge/clayey material is not sampled via SG due to extremely inaccurate and unreliable results. The water was kept at a constant 20° using an aquatic heater. The water was changed after each use daily. The density of the water is assumed to be 0.99823 g/ml at 20°. This density was multiplied by the specific gravity to calculate the bulk density of the sample. The mass of the bread pan was considered and removed from calculation of specific gravity. This value was remeasured and updated every 3 months.

Since this is a different SG measurement method than what used previously done, a small sub section of samples was analysed at ALS using method OA-GRA08, in addition to the in-house method. Figure 11.3 below shows the comparison of the results. The results are considered acceptable.

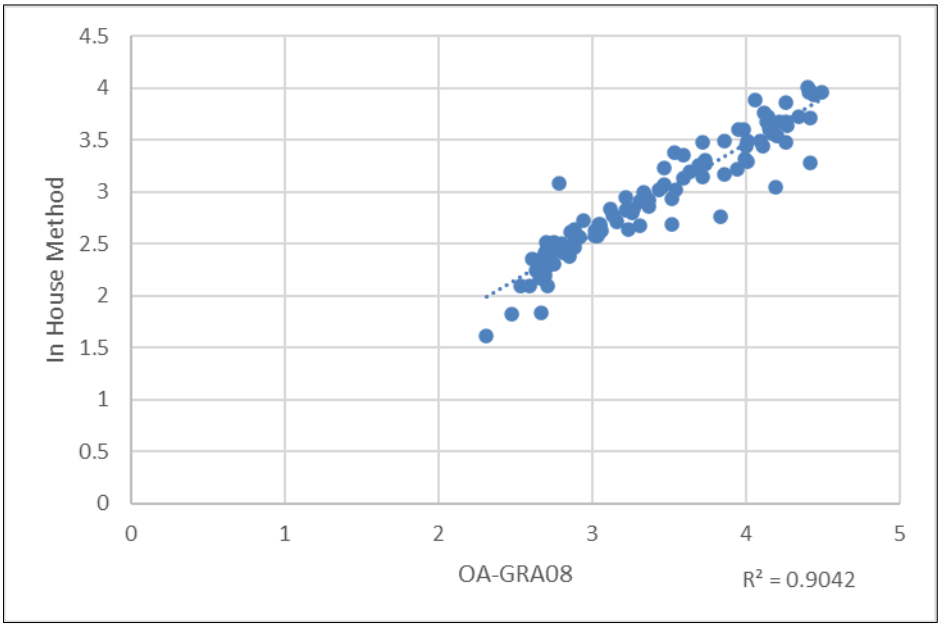


Figure 11.3 Chart comparing ALS method OA-GRA08 with new SFA SG method

11.5 Quality Assurance/Quality Control Procedures

Quality assurance and quality control procedures (QA/QC) utilized at the Black Butte project have evolved since initial investment in Tintina by Sandfire Resources NL in 2014. Minor changes were made to the QA/QC protocols at the onset of each of the drill programs that inform the Mineral Resource estimates:

- 2014 Drill Program.
- 2018 Phase 1 Drill Program.
- 2018/2019 Phase 2 Drill Program.
- 2021/2022 Drill Program.
- 2023– 2025 Drill Program.

The documentation of QA/QC procedures for Cu assays that follows is subdivided by drill program.

11.5.1 Certified Reference Materials (CRM)

11.5.1.1 2014 Drill Program

Three certified reference materials (CRMs) were purchased from WCM Minerals, 7729 Patterson Ave, British Columbia, Canada (Table 11.6) and inserted into the sample sequence at a minimum ratio of one CRM to every 20 samples. The actual CRM insertion rate during the 2014 drill program was 7%.

WCM Minerals CRM ID	Copper (%)	
	Expected Value	Standard Deviation
Cu-145	3.100	0.090
Ni-116	0.780	0.002
Pb-129	0.280	0.012

Table 11.6 Certified values for Cu in WCM Minerals CRMs used during the 2014 drill campaign

Figure 11.4 to Figure 11.6 show the CRM performance for Cu during the 2014 drill program. With the exception of Ni-116, the majority of CRMs returned analyses within acceptable limits. QA/QC failure was defined by SRA as variance beyond ± three standard deviations (±3SD) from the expected value with a warning flagged on samples beyond ± two standard deviations (±2SD). One Cu analysis of Cu-145 assayed slightly outside of the 2SD warning threshold. Ni-116 showed a negative bias with three analyses plotting outside of the 3SD failure threshold. Ni-116 is a CRM derived from pyrrhotite – pentlandite, nickel sulfide ore. It was concluded that the negative bias for Ni-116 is related to matrix and analytical technique rather than laboratory inaccuracy.

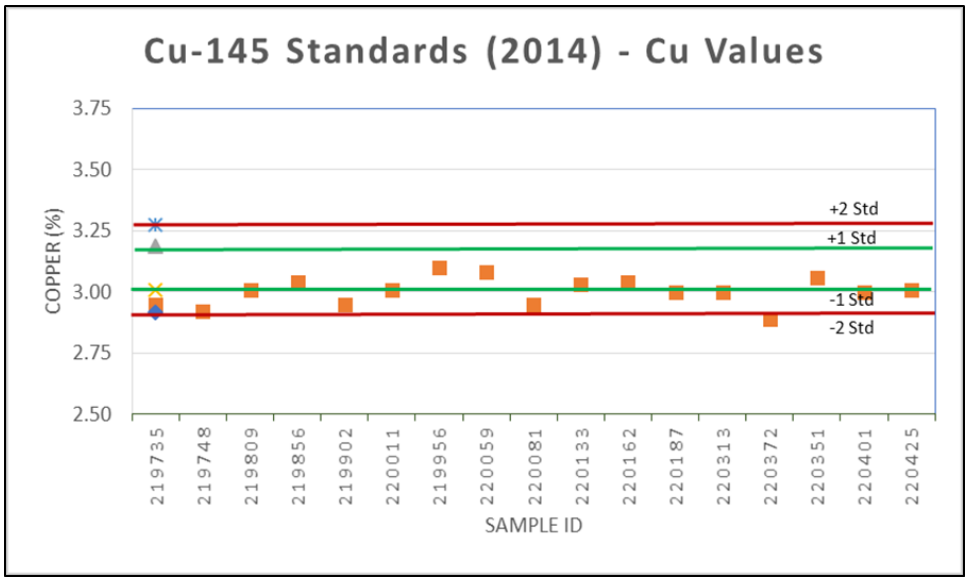


Figure 11.4 Cu-145 Cu values from the 2014 drill program

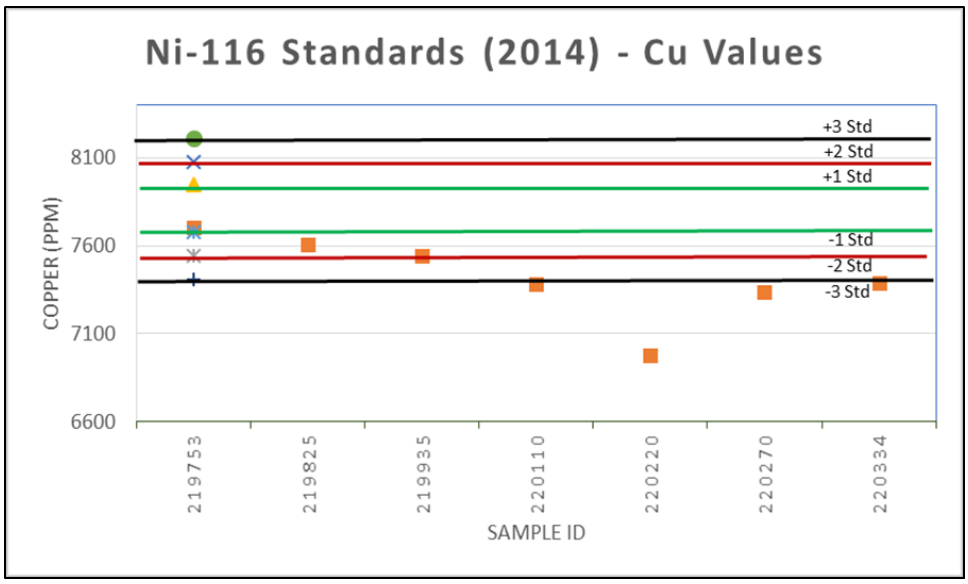


Figure 11.5 Ni-116 Cu values from the 2014 drill program

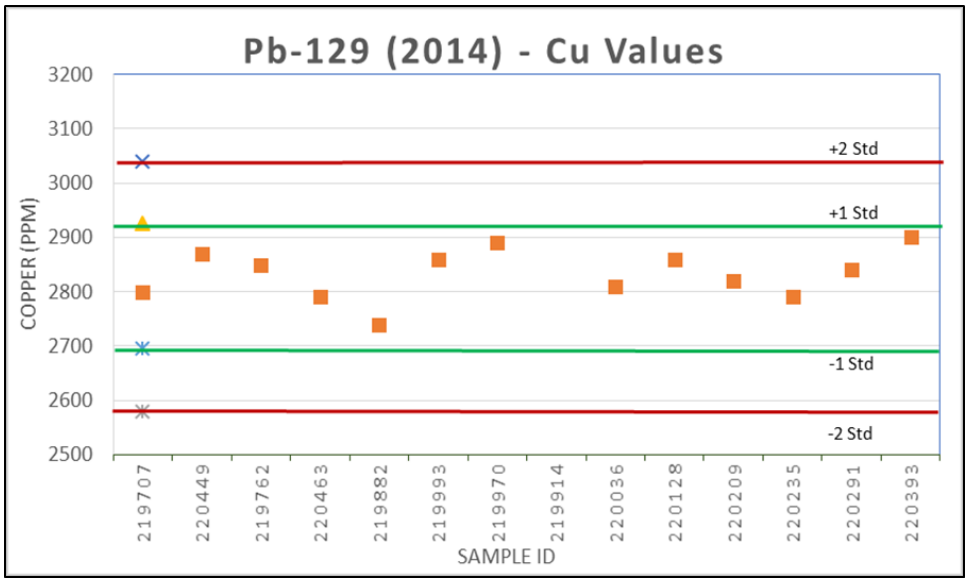


Figure 11.6 Pb-129 Cu values from 2014 drill program

11.5.1.2 2018 Phase 1 Drill Program

Four CRMs were purchased from ORE Research and Exploration (OREAS) Pty Ltd, 37A Hosie Street, Bayswater North, Victoria, Australia. Table 11.7 lists the certified copper values for these CRMs.

ORE Research CRM ID	Copper (%)	
	Expected Value	Standard Deviation
OREAS-624	3.101	0.079
OREAS-522	0.916	0.026
OREAS-134b	0.135	0.005
OREAS-136	0.031	0.001

Table 11.7 Certified values for Cu in ORE Research CRMs used during the 2018 Phase 1 drill-program

CRMs were inserted into the sample sequence at a minimum rate of two CRMs for every 20 samples. For the 2018 Phase 1 drill program actual CRM insertion rate was 14%. Figure 11.7 to Figure 11.10 show CRM copper performance during the 2018 drill program. One CRM (OREAS-522) exceeded the 2SD tolerance for a single assay, prompting reanalysis of the portion of the batch affected. The re-analyzed sample batches passed QA/QC and were incorporated into the sample database.

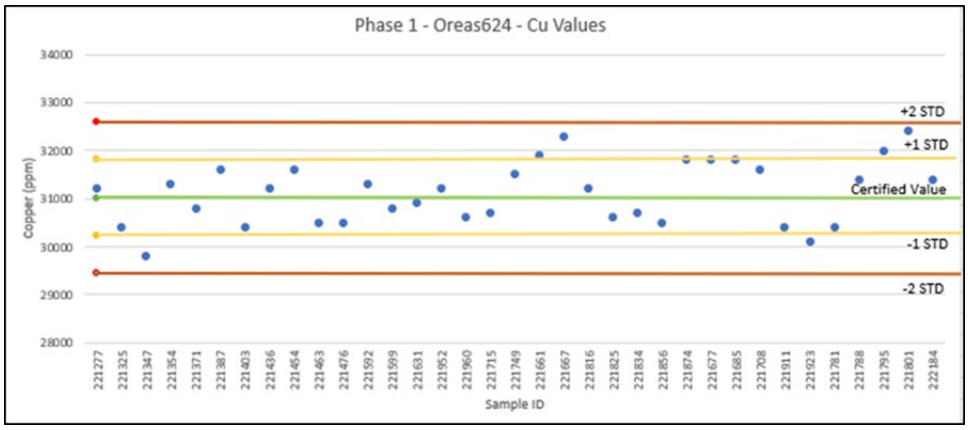


Figure 11.7 ORES-624 Cu values from the 2018 Phase 1 drill program

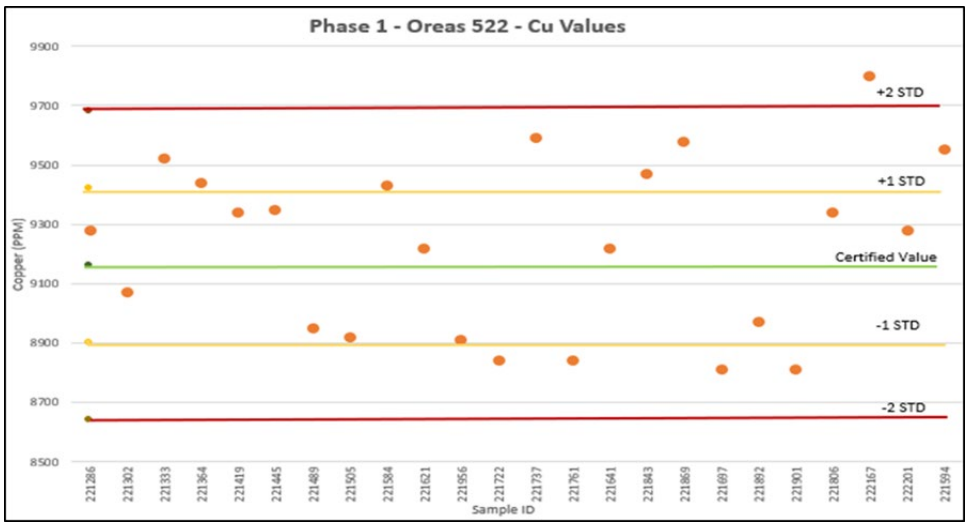


Figure 11.8 ORES-522 Cu values from the 2018 Phase 1 drill program

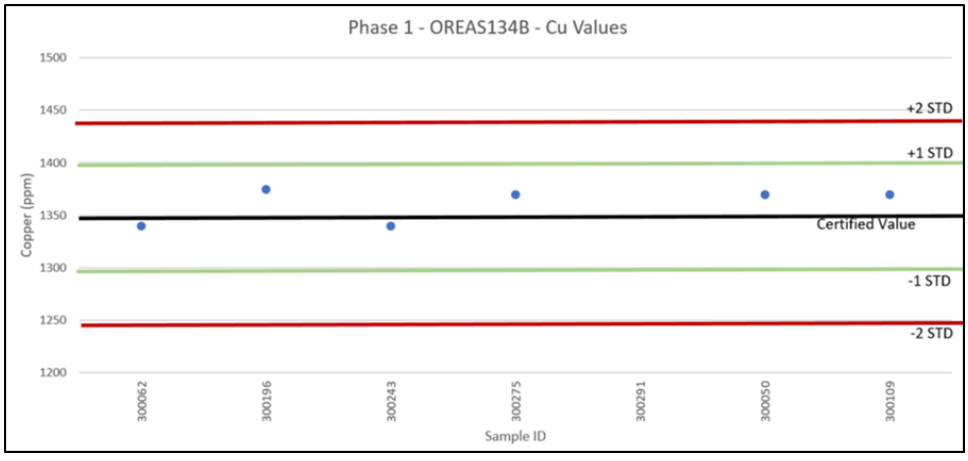


Figure 11.9 ORES-134B Cu values from the 2018 Phase 1 drill program

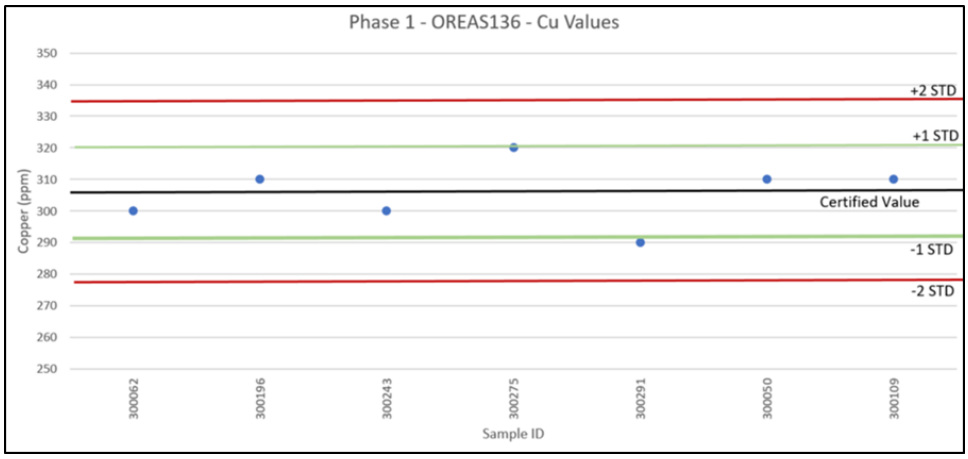


Figure 11.10 OREAS-136 Cu values from the 2018 Phase 1 drill program

11.5.1.3 2018/19 Phase 2 Drill Program

The same CRMs utilized for the 2018 Phase 1 drill program were also used during the 2018/19 Phase 2 program. Additionally, OREAS-935 was added to the CRM suite given the high Cu grades observed in the JL LCZ (Table 11.8). CRMs 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 11.11 to Figure 11.15 show CRM Cu performance for the 2018/19 Phase 2 drill program. Three OREAS-624 CRM analyses were outside of ±3SD thresholds triggering re-analyses of the sample sequence affected. All CRMs in the re-analyzed sequences passed QA/QC.

ORE Research CRM ID	Copper (%)	
	Expected Value	Standard Deviation
OREAS-935	12.550	0.388
OREAS-624	3.101	0.079
OREAS-522	0.916	0.026
OREAS-134b	0.135	0.005
OREAS-136	0.031	0.001

Table 11.8 Certified values for Cu in ORE Research CRMs used during the 2018/18 Phase 2 drill program

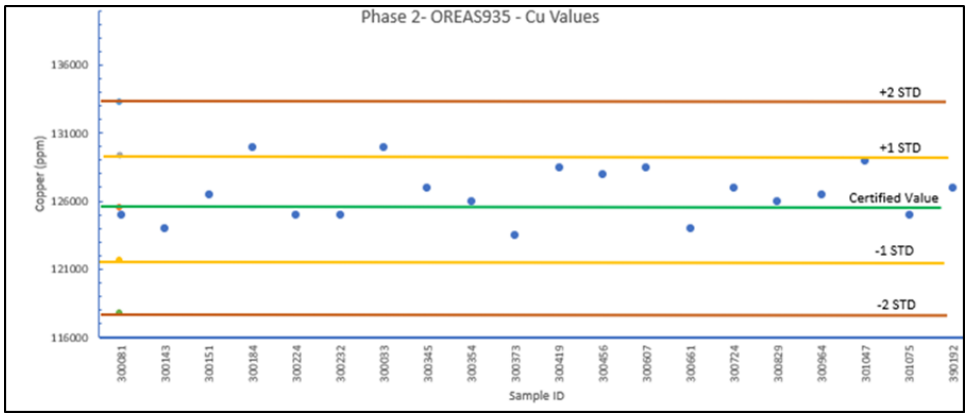


Figure 11.11 OREAS-935 Cu values from the 2018/19 Phase 2 drill program

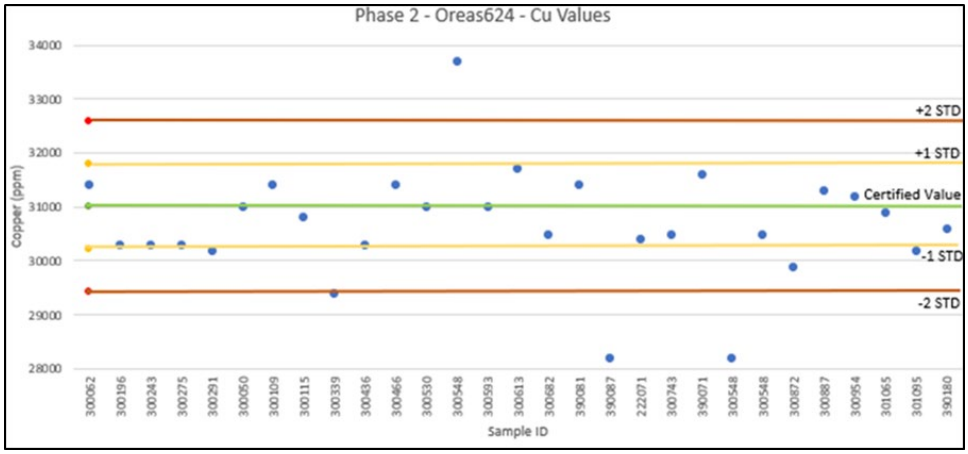


Figure 11.12 OREAS-624 Cu values from the 2018/19 Phase 2 drill program

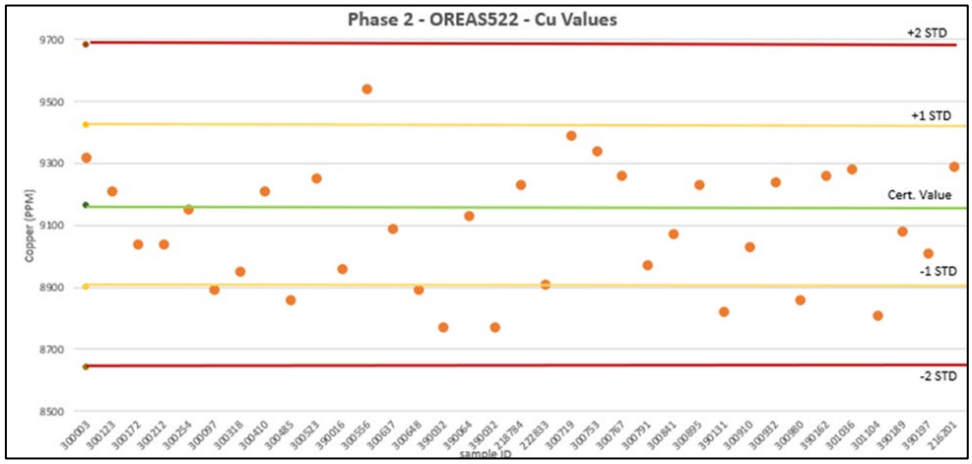


Figure 11.13 OREAS-522 Cu values from the 2018/19 Phase 2 drill program

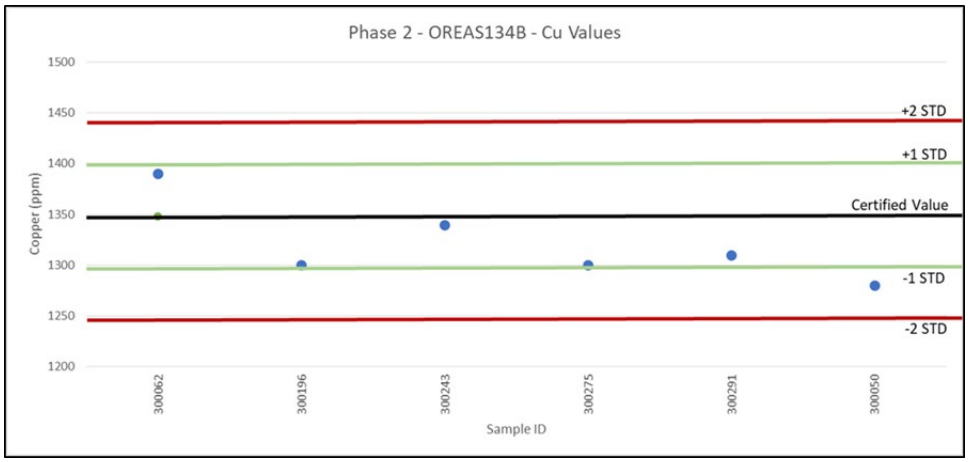


Figure 11.14 OREAS-134B Cu values from the 2018/19 Phase 2 drill program

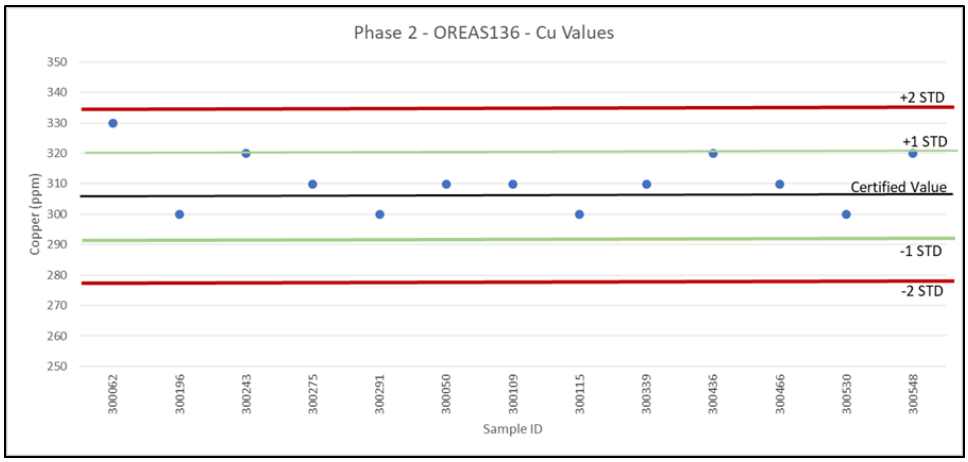


Figure 11.15 OREAS-136 Cu values from the 2018/19 Phase 2 drill program

11.5.1.4 2021/2022 Drill Program

CRMs were inserted into the sample sequence at a minimum insertion ratio of one CRM for every 20 samples. A range of low-, mid- and high-grade copper CRM values were used for this program (Table 11.9). Cu was the primary metal of interest and QA/QC performance of Ag, Pb, Zn, and As were also tracked.

ORE Research CRM ID	Copper (ppm)	
	Expected Value	Standard Deviation
OREAS-136	306	13.5
OREAS-164	22,500	800
OREAS-22e	8	0.7
OREAS-25a	34	2.3
OREAS-522	9,163	256.6

ORE Research CRM ID	Copper (ppm)	
	Expected Value	Standard Deviation
OREAS-551	5,649	174
OREAS-624	31,007	786.8
OREAS-906	3,099	102.4
OREAS-920	112	4.3
OREAS-927	10,800	240

Table 11.9 Certified values for Cu in ORE Research CRMs used during the 2021/22 drill program

Figure 11.16 to Figure 11.18 show examples of CRM Cu performance for the 2021/22 drill program. QA/QC comparisons are generated within the SRA site Access drill database. Due to up to a 4-month lag time between sample submittal and lab assay receipt for the 2021/22 drill program, QA/QC checks and requests for re-analysis of failures were made when groups of about 1,000 samples were available.

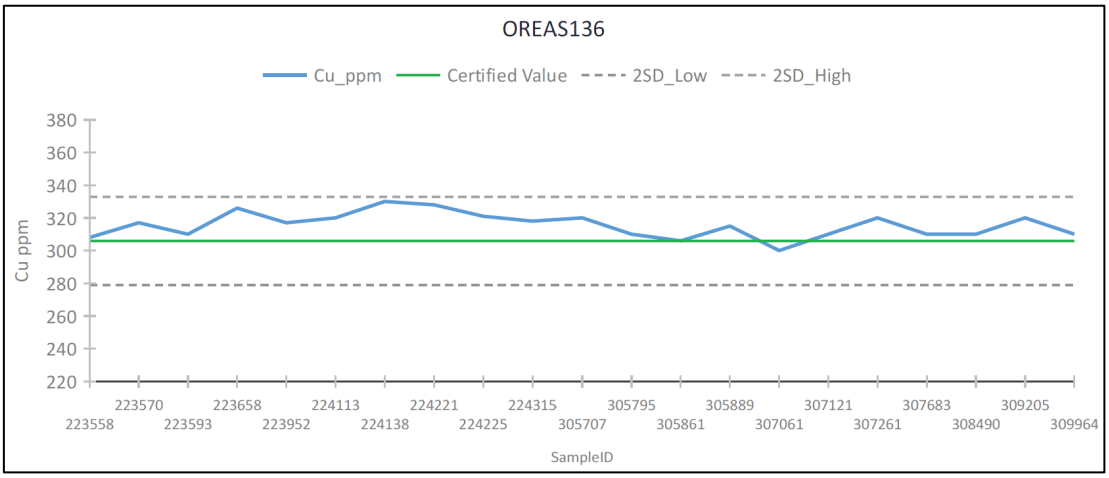


Figure 11.16 OREAS-136 Cu values from the 2021/22 drill program

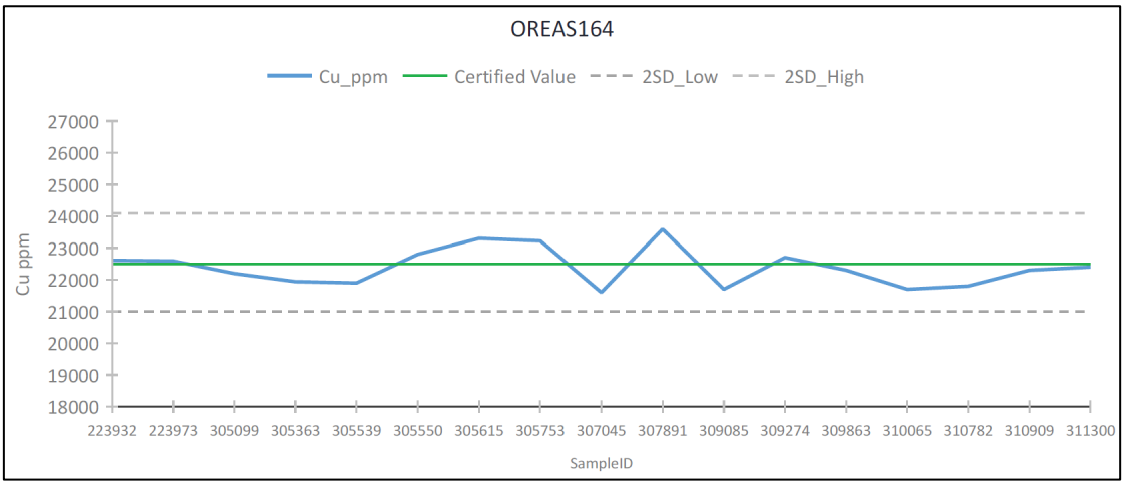


Figure 11.17 OREAS-164 Cu values from the 2021/22 drill program

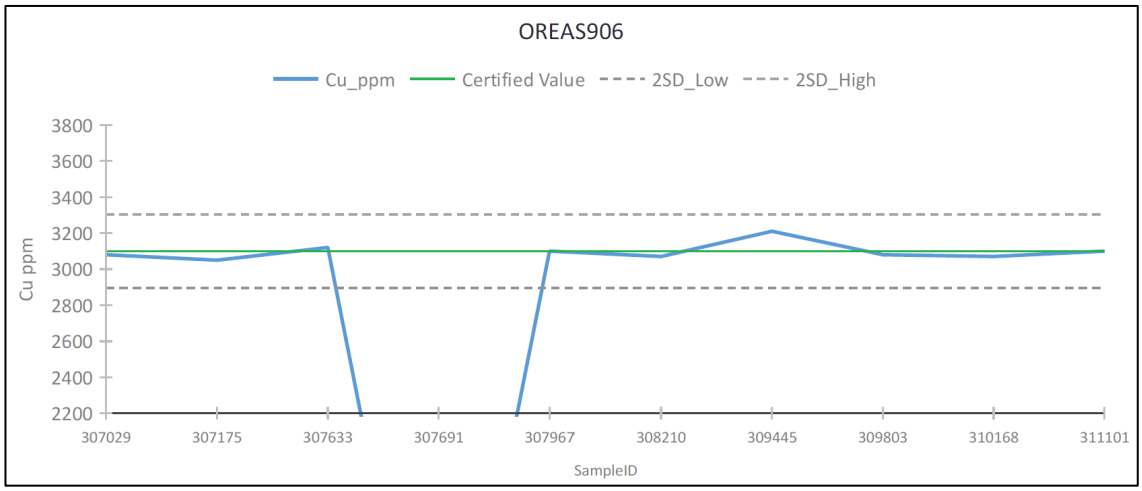


Figure 11.18 OREAS-906 Cu values from the 2021/22 drill program

11.5.1.5 2023–2025 Drill Program

CRMs were inserted into the sample sequence at a minimum insertion ratio of one CRM for every 20 samples (Table 11.10). Actual CRM insertion rate for 2023–2025 drill program was 7.1%.

Figure 11.19 to Figure 11.25 show CRM Cu performance for the 2023–2025 drill program. Two OREAS-920 and one OREAS-932 CRM analyses were outside of ±3SD thresholds triggering re-analyses of the sample sequence affected.

ORE Research CRM ID	Copper (ppm)	
	Expected Value	Standard Deviation
OREAS-165	32,000	700
OREAS-504d	11,000	240
OREAS-624	31000	790
OREAS-920	112	4
OREAS-921	274	9
OREAS-932	61,300	1810
OREAS-935	125,500	3880

Table 11.10 Certified values for Cu in ORE Research CRMs used during the 2023–2025 drill program

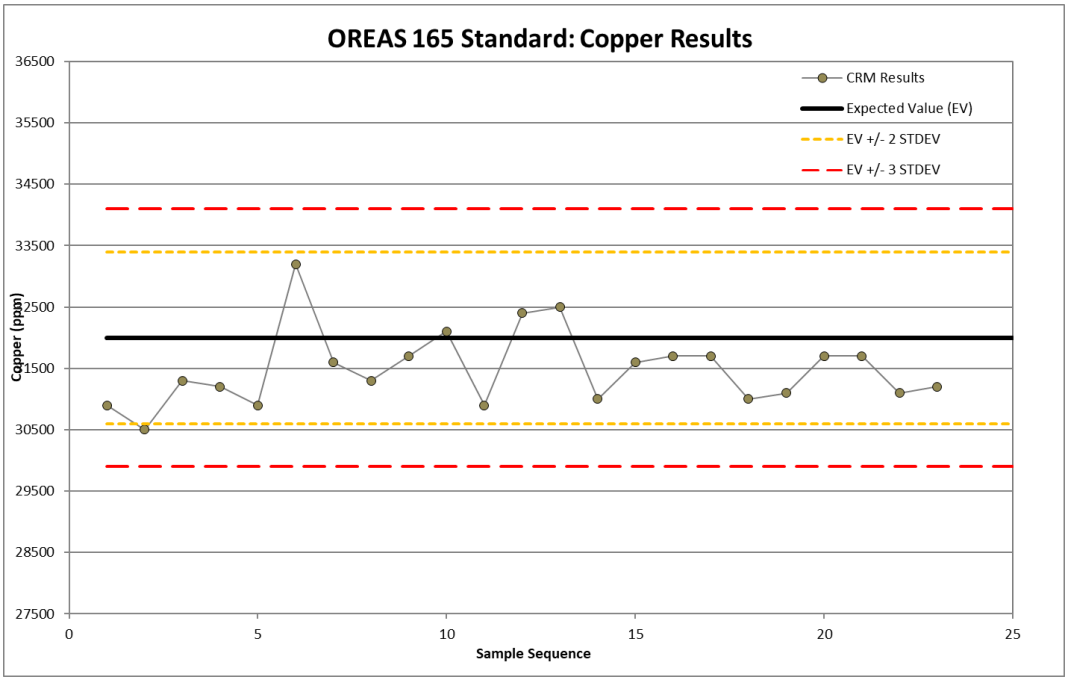


Figure 11.19 OREAS-165 Cu values from the 2023–2025 drill program

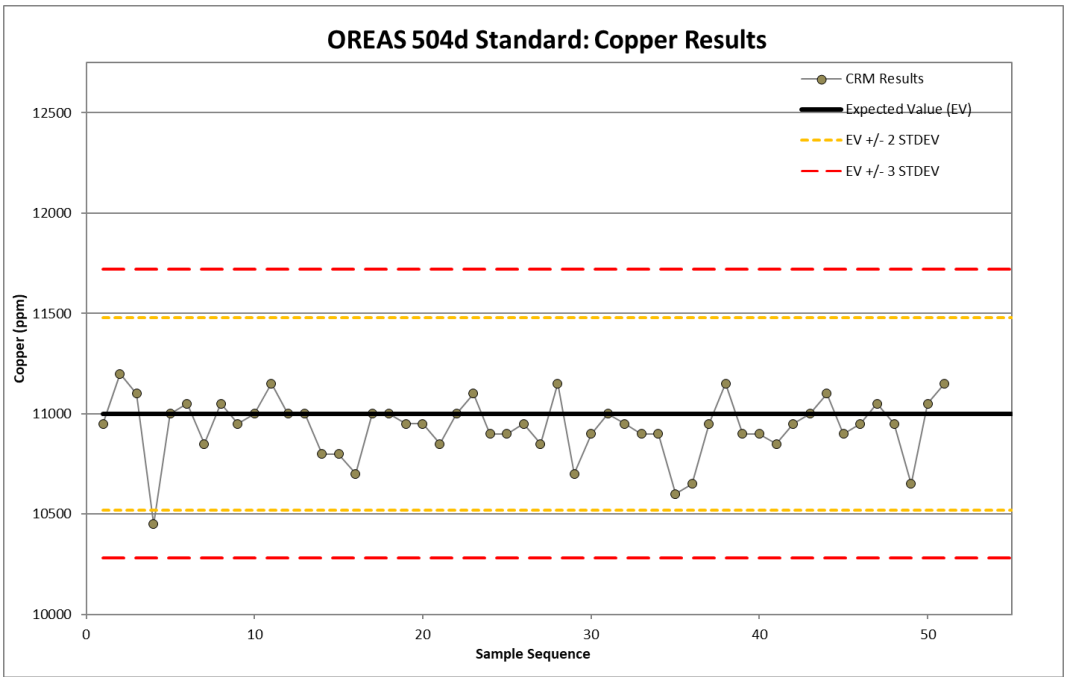


Figure 11.20 OREAS-504d Cu values from the 2023–2025 drill program

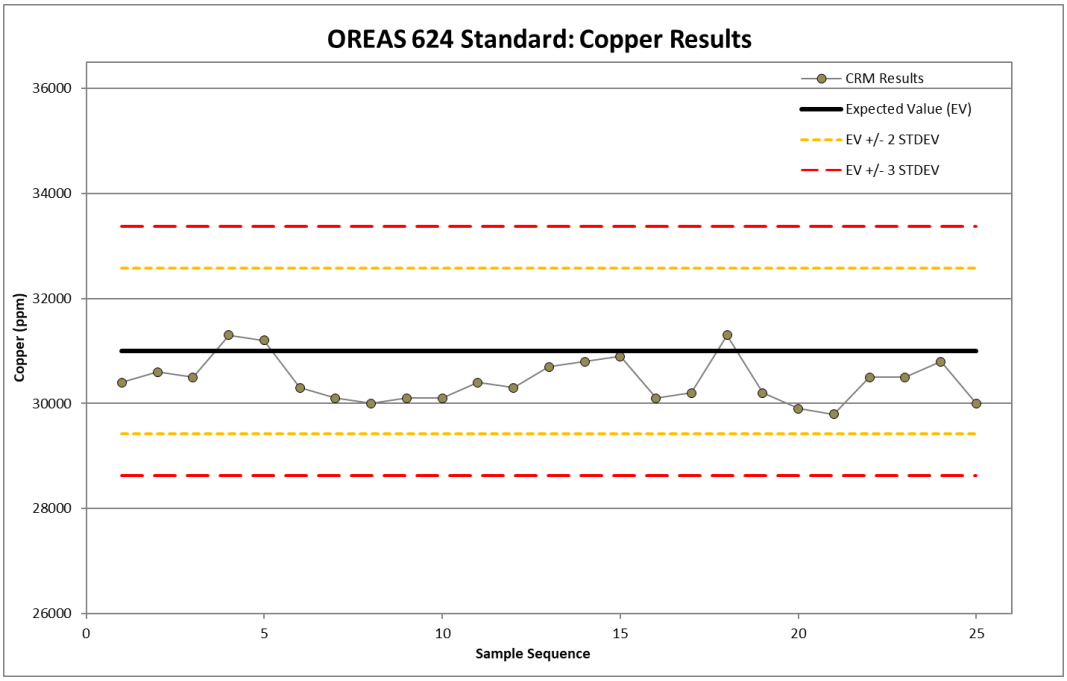


Figure 11.21 OREAS-624 Cu values from the 2023–2025 drill program

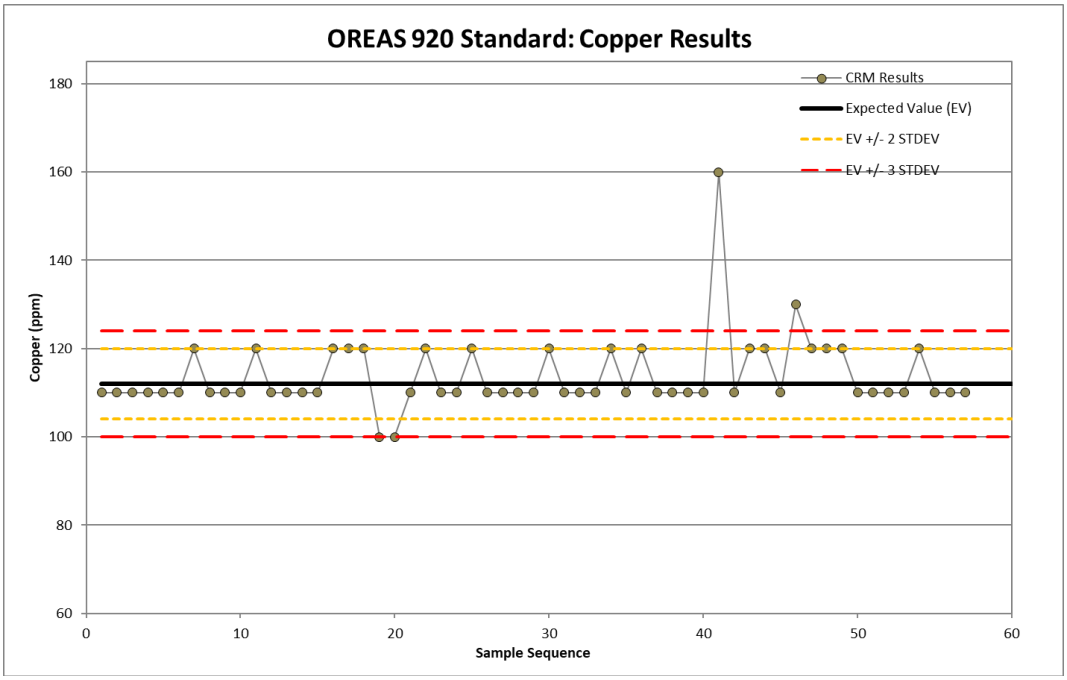


Figure 11.22 OREAS-920 Cu values from the 2023–2025 drill program

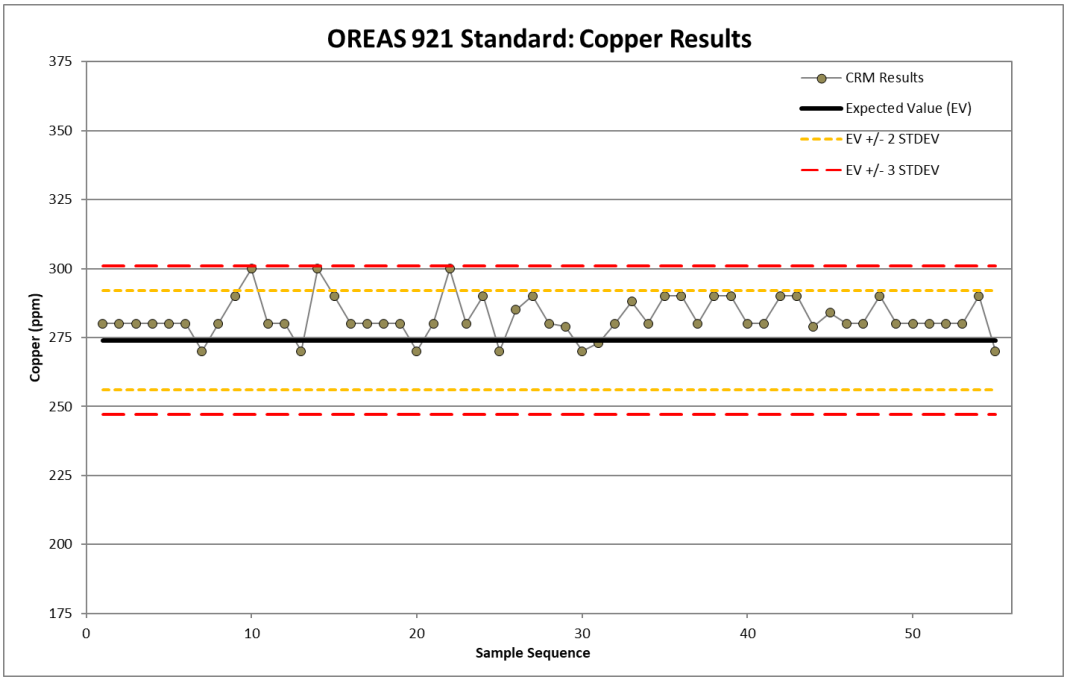


Figure 11.23 OREAS-921 Cu values from the 2023–2025 drill program

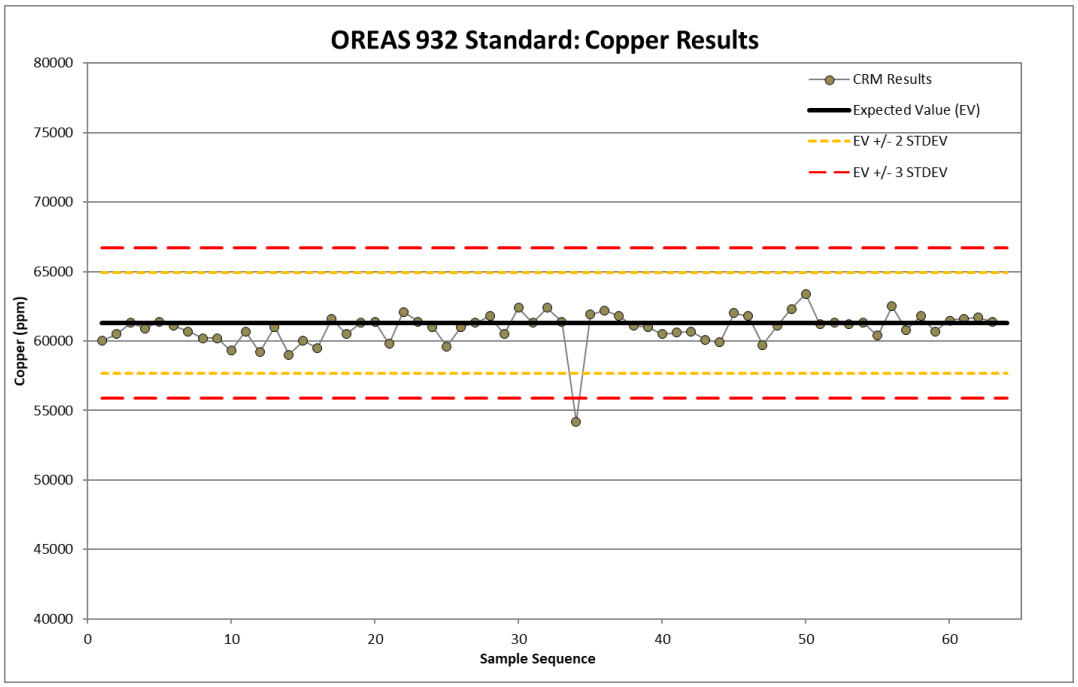


Figure 11.24 OREAS-932 Cu values from the 2023–2025 drill program

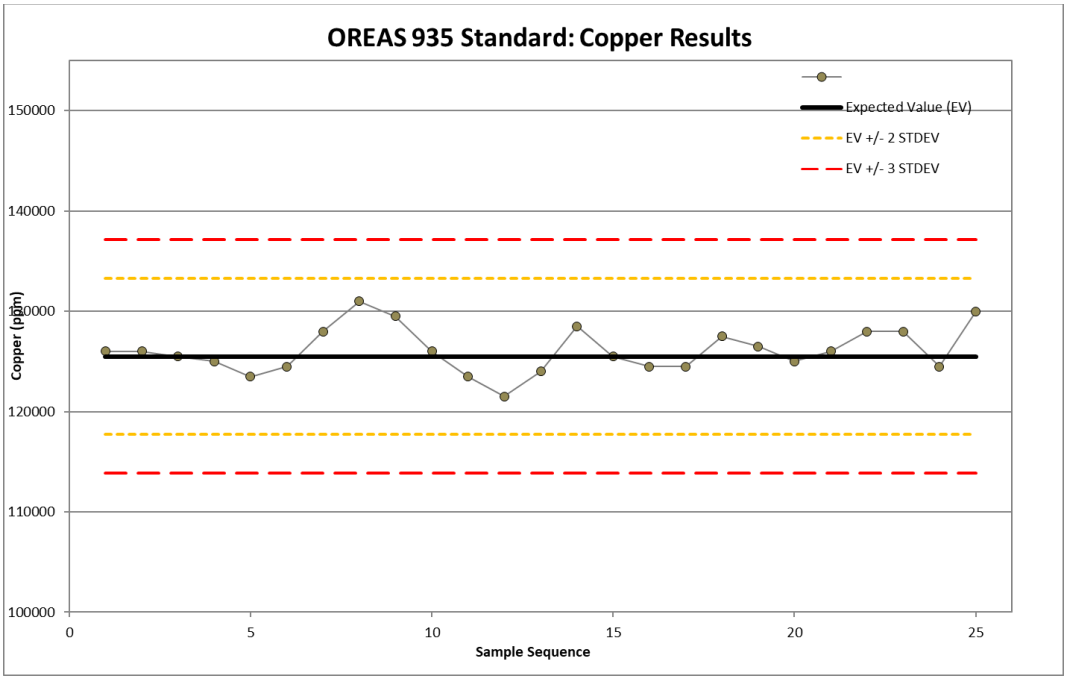


Figure 11.25 OREAS-935 Cu values from the 2023–2025 drill program

11.5.2 Blanks

11.5.2.1 2014 Drill Program

Landscaping marble chips that had been used as coarse blanks in the Tintina 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 11.26, these blanks returned poor performance during 2014, often exceeding copper detection limits by >15 times the lower laboratory detection limit. 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 copper. The use of the landscape marble was discontinued after 2014.

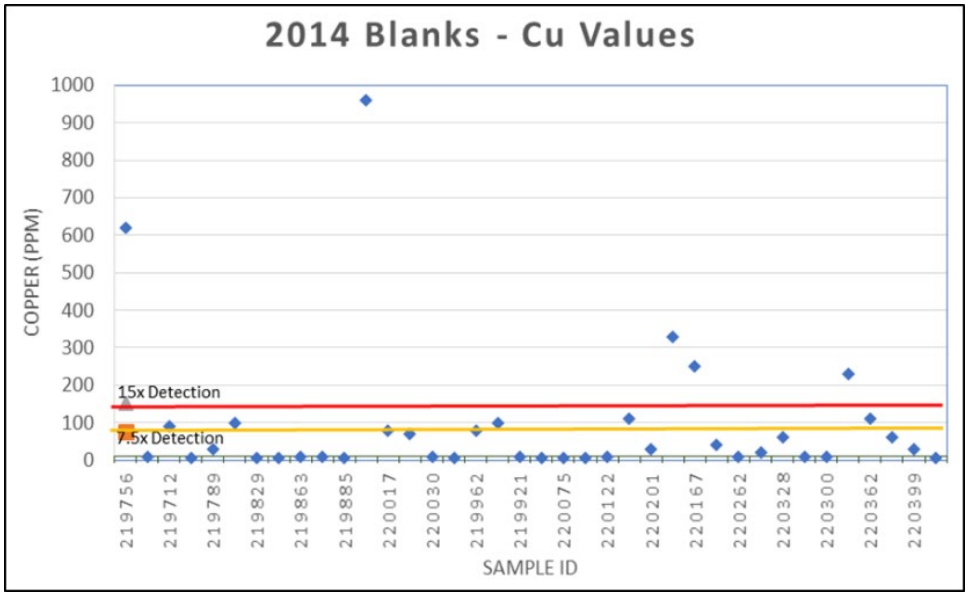


Figure 11.26 Cu results of landscape marble used as coarse blanks during the 2014 drill program

11.5.2.2 2018 Phase 1 Drill Program

With the exception of two JL 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 CRMs 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, 37A Hosie Street, Bayswater North, Victoria, Australia. OREAS-22e has a certified value of 7.97 ppm for Cu (standard deviation = 0.746 ppm). All OREAS 22e Cu values plotted below the 15 times detection threshold and the majority below the 7.5 times detection threshold (Figure 11.27).

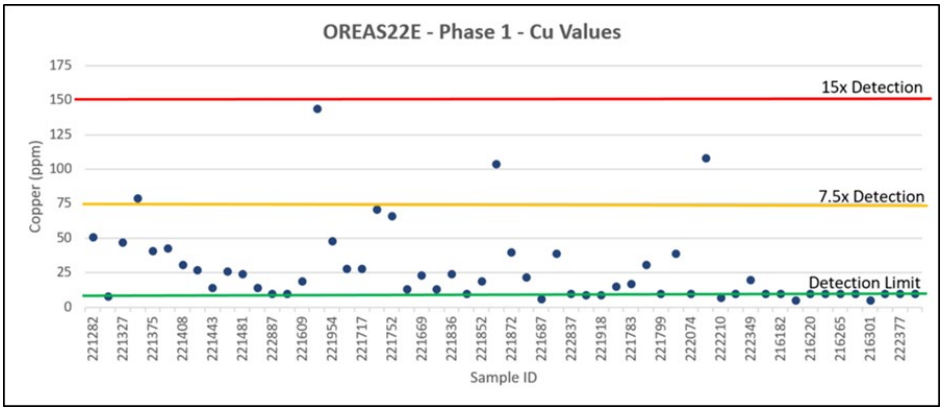


Figure 11.27 Cu results of pulp blank OREAS-22e during the 2018 Phase 1 drill program

11.5.2.3 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. 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).

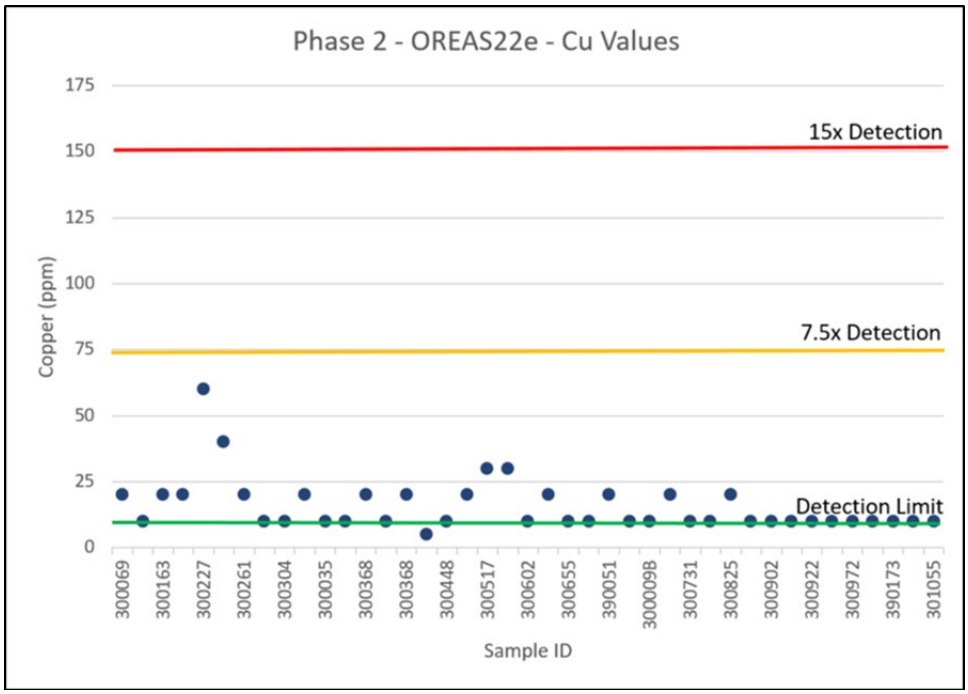


Figure 11.28 Cu results for pulp blank OREAS-22e from the 2018/19 Phase 2 drill program

Figure 11.29 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 the lower laboratory detection limit.

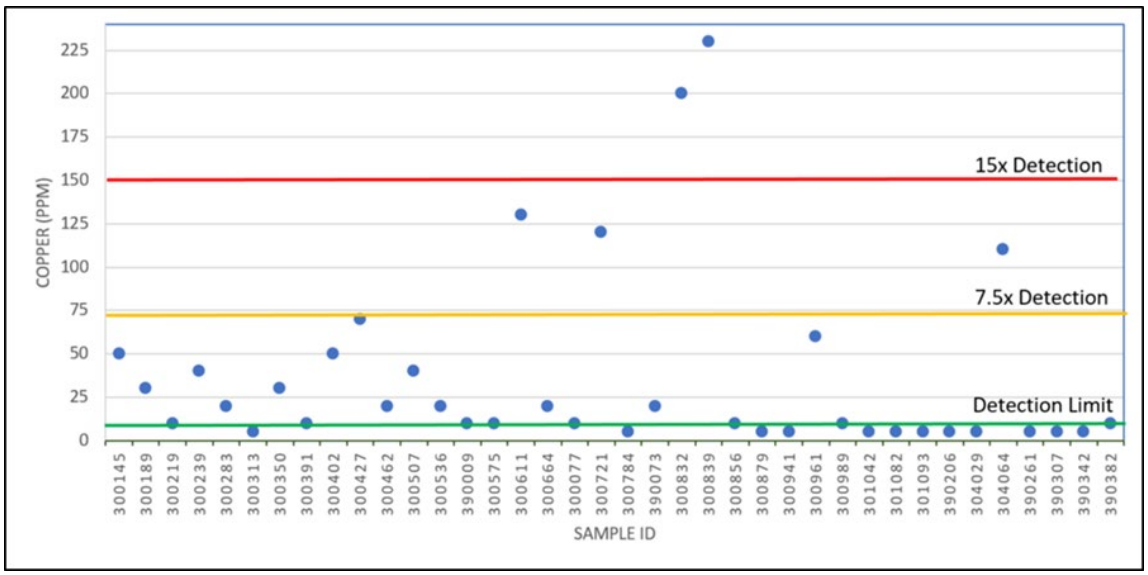


Figure 11.29 Cu results for coarse blanks for the 2018/19 Phase 2 drill program

11.5.2.4 2021/2022 Drill Program

Like the 2018/19 drill program, the 2021/22 drill program used pulp-blanks and coarse-blanks inserted into the sample sequence by SRA geologists. Coarse blanks (coarse silica blank material) were obtained in 0.5 kg packets from Analytical Solutions Ltd. Blanks were inserted at a minimum ratio of one blank every 20 samples and alternated between a coarse blank and a pulp blank.

For QA/QC review, pulp blanks (OREAS-22e and OREAS-25a) were checked in the same manner as the other CRMs by plotting the expected value against two standard deviations. Coarse blanks were plotted as a separate group, comparing assay values to threshold limit multipliers (Figure 11.30).

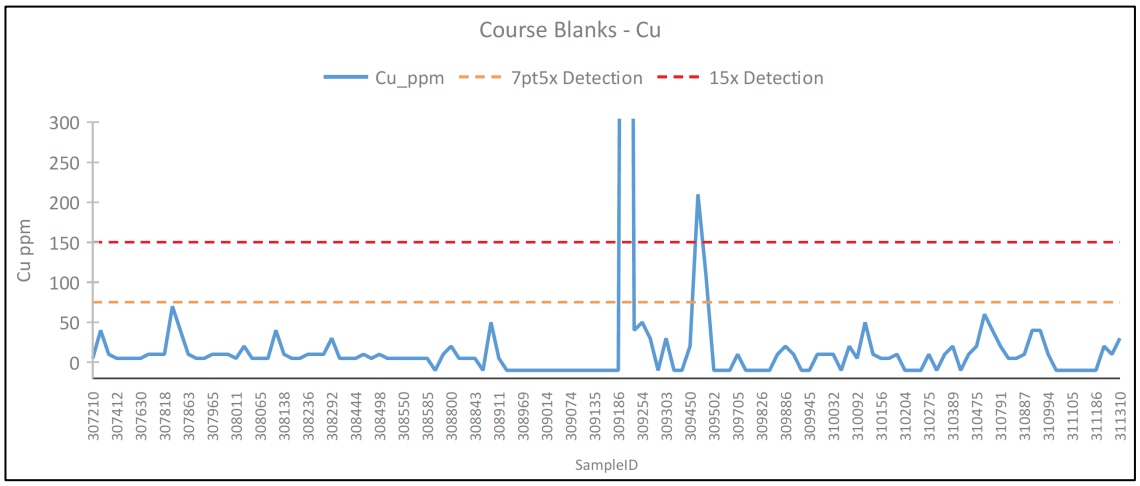


Figure 11.30 Cu results for coarse blanks for the 2021/22 drill program

11.5.2.5 2023–2025 Drill Program

The 2023–2025 drill program used pulp-blanks and coarse-blanks inserted into the sample sequence by SRA geologists, like previous campaigns. Blanks were inserted at a minimum ratio of one blank every 20 samples and alternated between a coarse blank and a pulp blank. The actual blank insertion rate was 6.3%.

Blanks were compared to five times the lower laboratory detection limit for the coarse blanks and five times the certified value for OREAS-22h (Figure 11.31 and Figure 11.32). A total of ten blanks out of 266 samples were considered to be failing and investigated by SRA. No significant laboratory contamination was determined from the control assays.

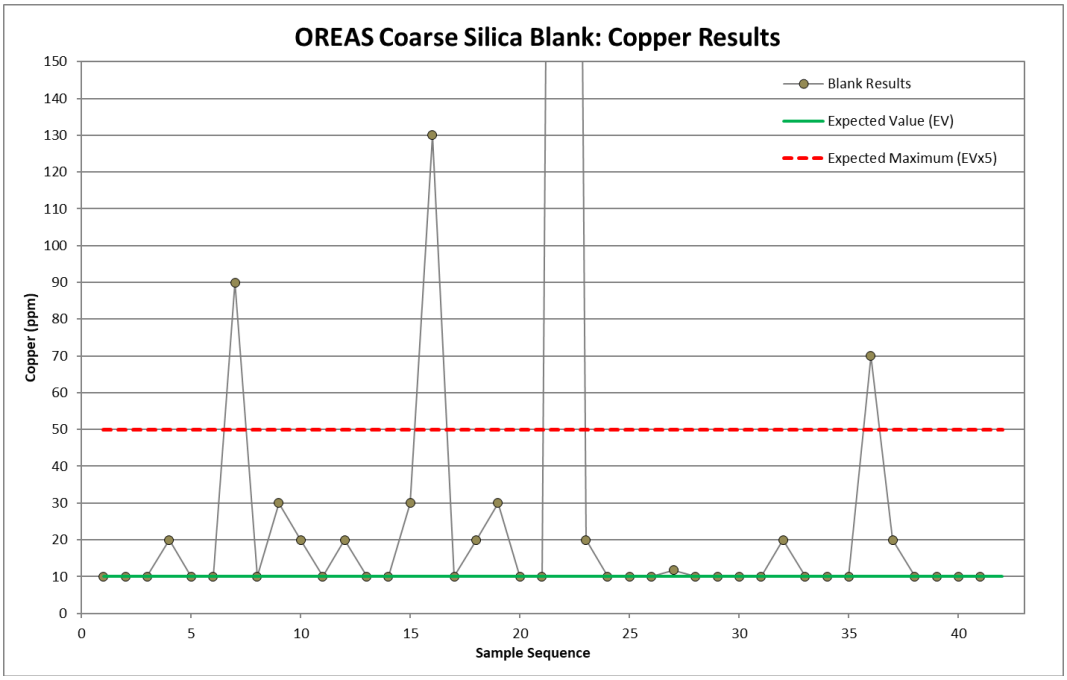


Figure 11.31 Cu results for coarse blanks for the 2023–2025 drill program

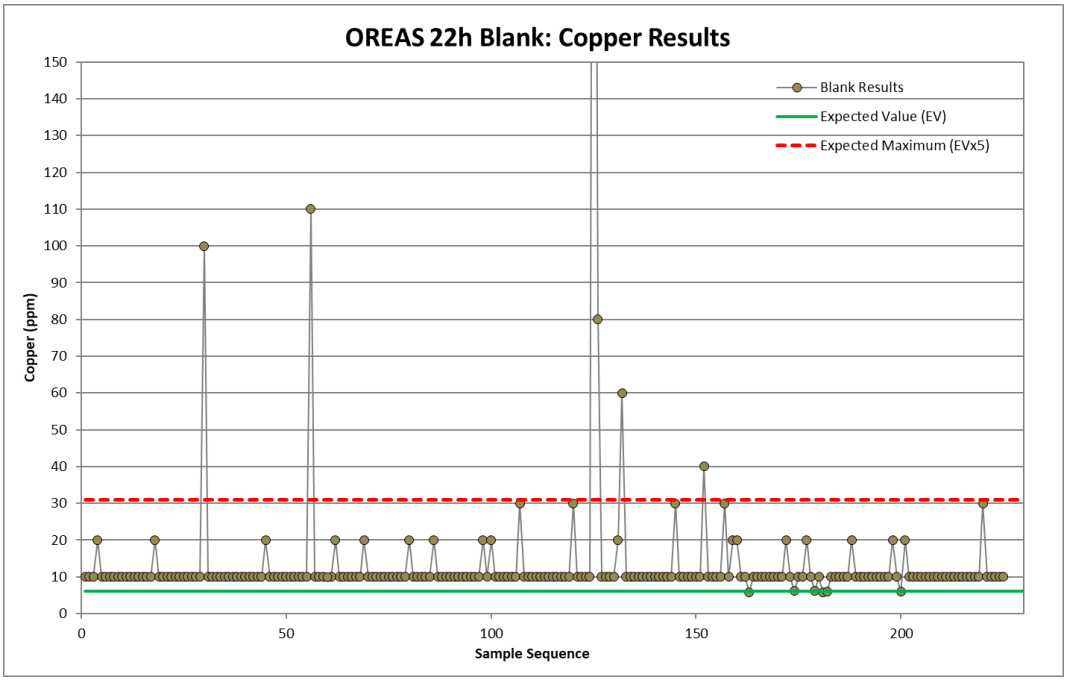


Figure 11.32 Cu results for OREAS-22h blanks over the 2023–2025 drill program

11.5.3 Duplicates

11.5.3.1 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 copper values (Figure 11.33) shows an acceptable correlation.

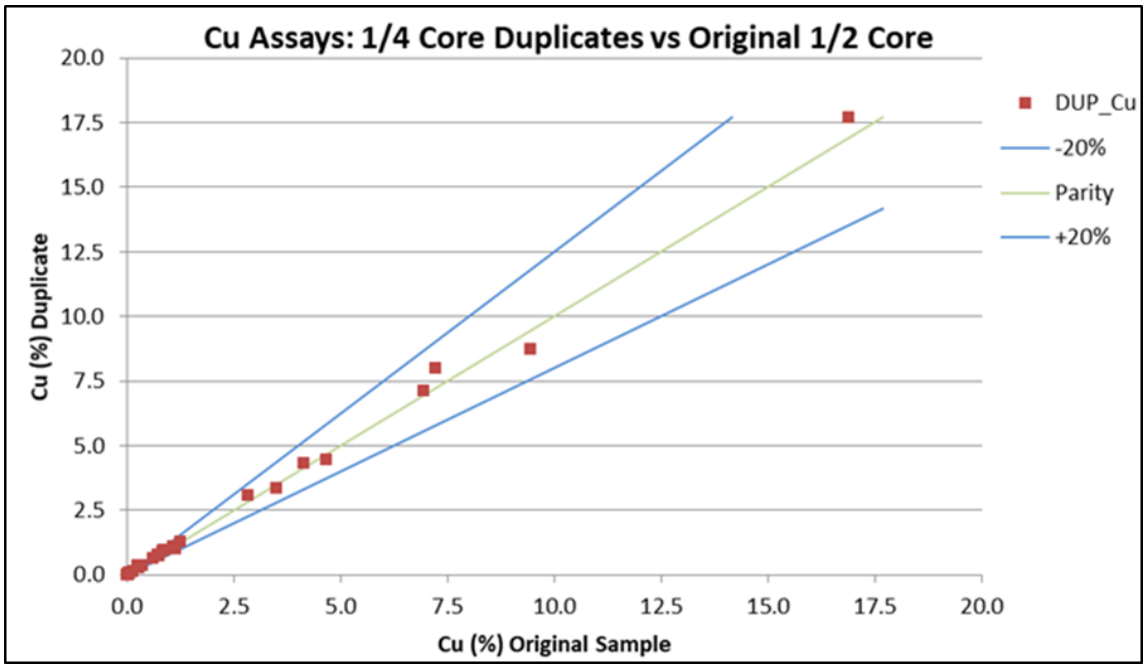


Figure 11.33 Comparison of Cu analyses from original and field duplicates from the 2014 drill program

11.5.3.2 2018 Phase 1 Drill Program

As whole PQ core from the JL 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 JL 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 shows a comparison of original versus field duplicate analyses for copper. The sample set is too small to be statistically meaningful but the correlation between available data is considered acceptable.

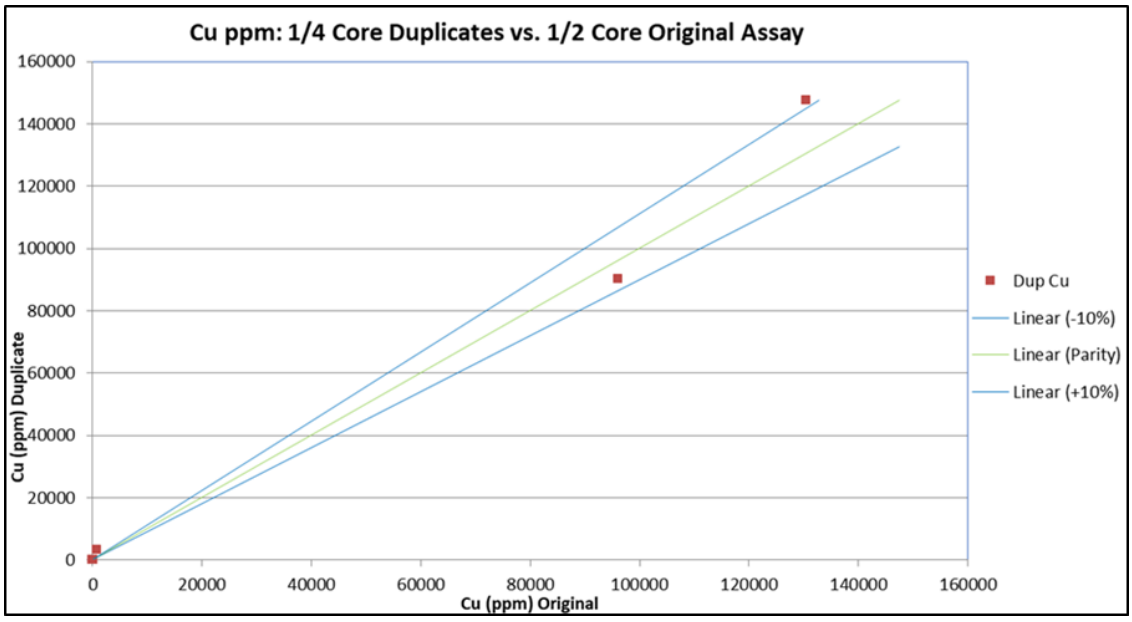


Figure 11.34 Comparison of Cu analyses from original and field duplicates from the 2018 Phase 1 drill program

Upon completion of all analyses, ALS Vancouver and ALS Reno returned the pulp residues to SRA. Thirty-nine 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 11.35. These diagrams indicate a reasonable correlation but a slight positive bias towards the ALS Vancouver results. The four CRMs 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 sample digestion method rather than laboratory error.

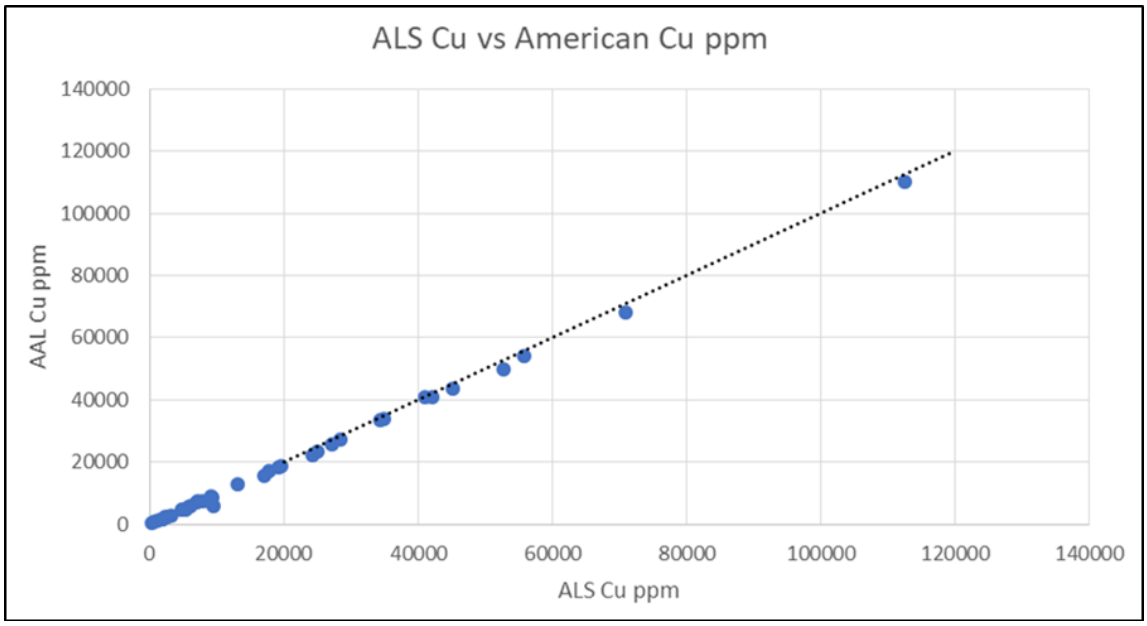


Figure 11.35 Comparison between AAL umpire and ALS original Cu assays for the 2018 Phase 1 drill program

11.5.3.3 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 analysed 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 CRMs and pulp blanks as supplied by SRA geologists and resubmitted the samples to ALS Reno for analysis.

Figure 11.36 shows the laboratory duplicate copper 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.

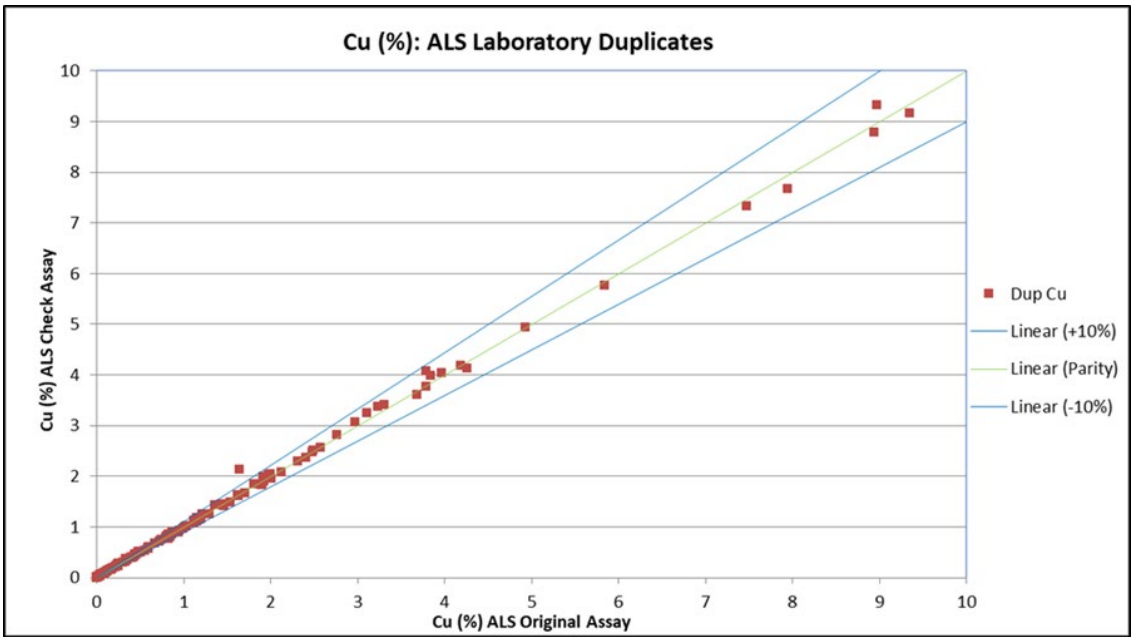


Figure 11.36 Scatter plot of laboratory duplicate Cu analyses for the 2018/19 Phase 2 drill program.

SRA geologists selected 192 pulp residues from ALS 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 third-party testing represents 13% of the total number of original analyses in this drilling campaign.

A comparison of the umpire copper results is presented in Figure 11.37. An acceptable correlation with a slight positive bias to the ALS results is evident.

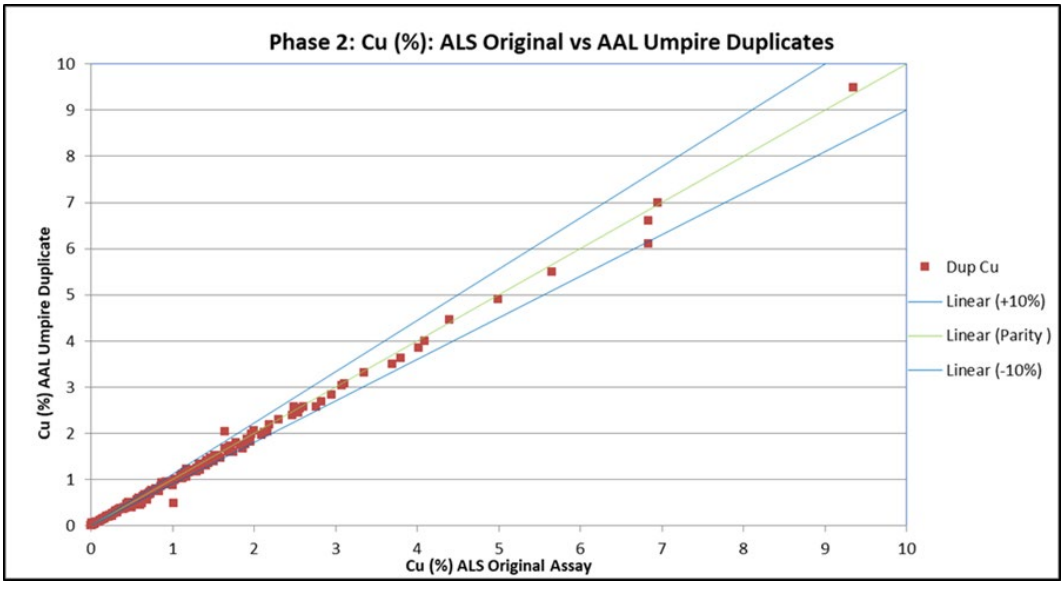


Figure 11.37 Scatter plot of umpire duplicate copper analyses for the 2018/19 Phase 2 drill program

11.5.3.4 2021/2022 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 after the other half-core sample. SRA compares field duplicates at ±10% variance among pairs, which is conservative. Due to natural variation in drill core, typical industry standard is ±30% variance among field duplicate pairs. Figure 11.38 shows the Cu field duplicate results.

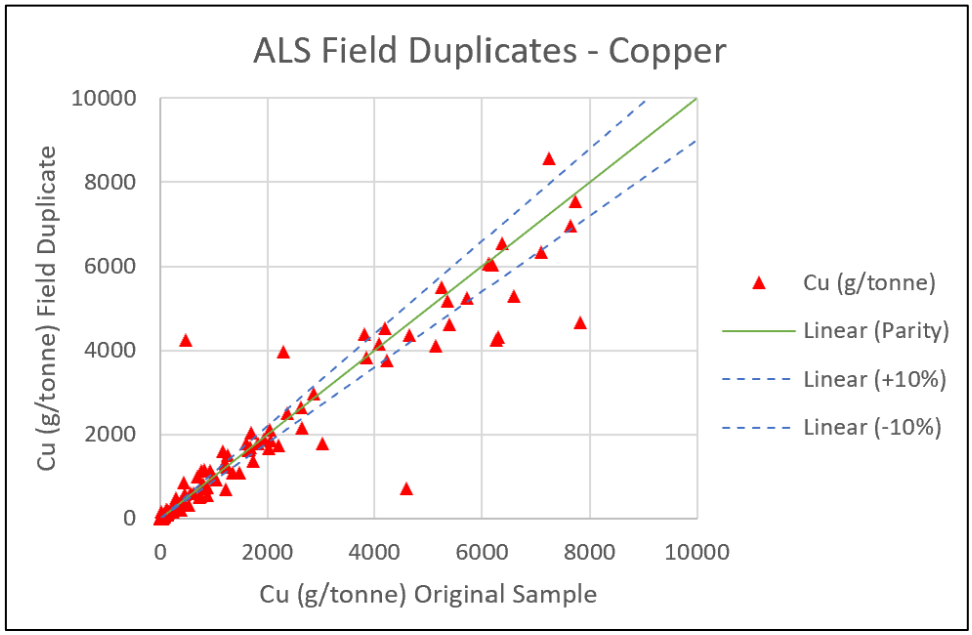


Figure 11.38 Comparison of Cu analyses from original and field duplicates for the 2021/22 drill program

11.5.3.5 2023–2025 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 after the other half-core sample. The actual field duplicate insertion rate for the 2023–2025 program was 5.6%. Pulp duplicates were analyzed at ALS, like previous programs. The actual pulp duplicate insertion rate for the 2023–2025 program wats 4.0%. Figure 11.39 and Figure 11.40 show the Cu field duplicate and pulp duplicate results, respectively. Some scatter of values is seen outside of the target tolerance bands, typically at low Cu concentrations. The field and pulp duplicate results compare favorably with no material bias.

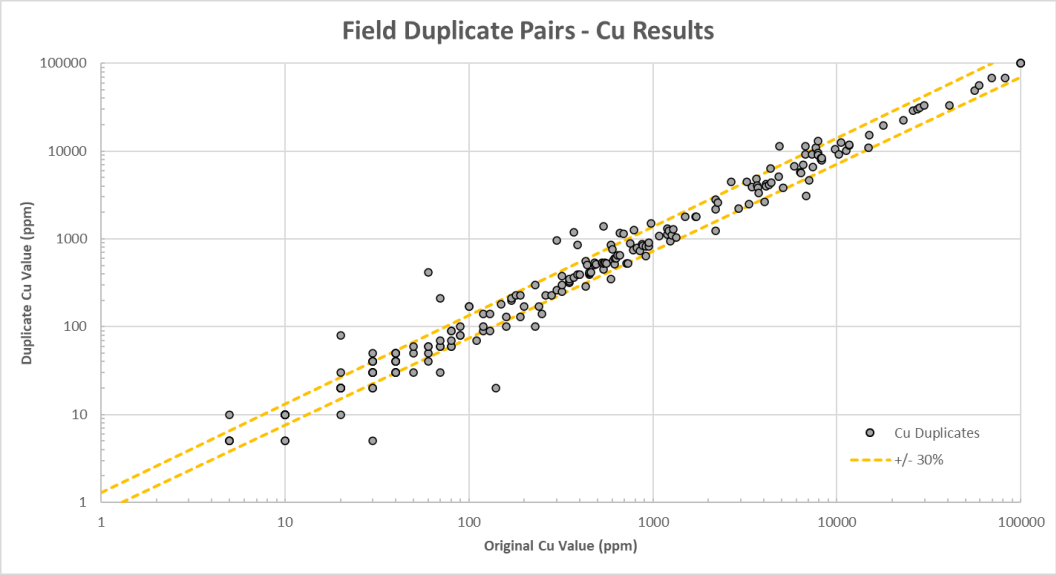


Figure 11.39 Comparison of Cu analyses from original and field duplicates for the 2023–2025 drill program

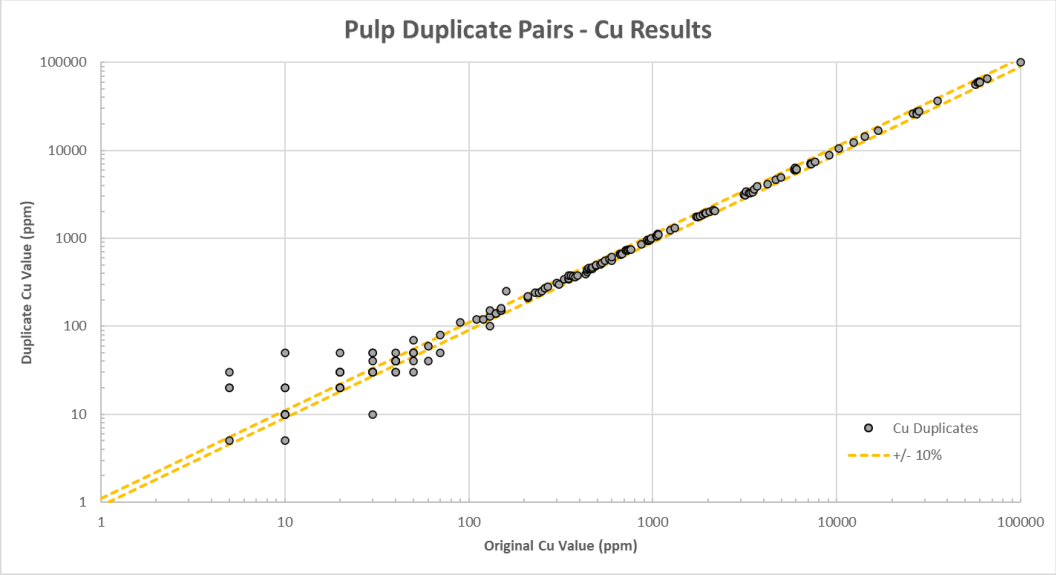


Figure 11.40 Comparison of laboratory pulp duplicate Cu analyses for the 20232025 drill program

11.5.4 Results and Actions

Since 2014, QA/QC evaluation and actions during the SRA drilling programs were largely conducted using Cu results from CRMs. Analyses of a CRM yielding a Cu analysis greater than two standard deviations from the certified value triggered a CRM failure, which is considered conservative based on standard industry practices. Generally accepted industry practice typically flags one sample with ± three standard deviations from the expected value to indicate a QA/QC failure. Generally, a threshold of ± two standard deviations is considered a warning unless multiple successive samples fail this lower threshold value.

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 Black Butte samples. All other CRMs performed adequately for this drill campaign.

During the 2018 Phase 1 and 2018/19 Phase 2 drill-programs, there were four CRM Cu failures on OREAS standards. All of these CRMs triggered re-analyses of the affected sample series with the insertion of different CRMs and pulp-blanks. The re-analysed 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.

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 copper 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. Also, 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. Re-analyses of the sample series affected by these coarse blanks was not undertaken.

QA/QC evaluation and actions during the 2021/22 drill-program were triggered by review of Cu results from CRMs and blanks. Analyses of a CRM yielding a Cu analysis greater than two standard deviations from the certified value triggered a CRM failure. More scrutiny was placed on CRMs with expected high Cu values. CRM failures for the pulp blanks were not submitted for re-assay unless a systematic failure was noted. A systematic failure was considered to include multiple failures in the same lab batch.

For 2021/22 re-assay requests, the control sample of concern along with five samples before and five samples after the control sample were run. Assay comparisons were made for the elements of interest on the 11 samples rerun. In 2022, three groups were requested for rerun by SRA with the re-assays showing close comparison to the original assay values received. No further action was requested based on the rerun comparison.

For the 2023–2025 drilling campaign, three CRM Cu failures were noted for OREAS standards using a threshold of ± three standard deviations from the certified value. Additionally, several blank failures were determined by SRA and evaluated by re-running batches of assays at ALS. SRA documented correspondence with the laboratory and evaluated requested re-assay results to determine appropriate corrective action.

11.6 Opinion on Adequacy

Berkley Tracy, P.Geo (PGO#3024) of SRK, acting as Qualified Persons (QP) for Geology and Mineral Resources, has reviewed the summary of the historical QA/QC programs, available results, and interpretation of data. Specific records are limited for sampling procedures of the pre-Tintina historical drilling programs; however, no known bias exists in the earlier sample grades compared to recent assay results. Where possible, portions of these historical data have been validated back to surviving historical records.

It is the opinion of the QP that the security, sample preparation, and analytical protocols reported for the majority of Black Butte sample data are consistent with generally accepted industry standards. Limited QA/QC sample failures and interpreted minor biases in some duplicate pairs are considered immaterial. Quality control samples, results, and interpretation are adequate for an acceptable level of confidence in the analytical data for the reporting of Mineral Resources, as per CIM (2014).

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12. DATA VERIFICATION

12.1 Site Inspections

During November 2018, SRK conducted an initial site visit and tour of the Project site, core shed, and facilities located in White Sulphur Springs, MT. SRK 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 2018 site visit and tour were hosted by SRA and included a team of multi-discipline consultants associated with the Project. SRK was given access to all aspects of the Project and the geological staff, with all questions addressed by SRA.

In September 2022, Berkley Tracy, P.Geo (PGO# 3024) of SRK, acting as Qualified Person (QP) for Mineral Resources, conducted a site visit and tour of the Project site, core facilities, and logging area. Mr. Tracy focused on data and modelling updates relating to the Lowry deposit at the time, while also assessing the Johnny Lee deposit. During the 2022 site visit, copies of drilling files containing collar coordinates, assays and downhole surveys were investigated and compared to records within the SRA modelling database. Mr. Tracy conducted an additional site inspection in June 2025 that was focused on reviewing SRA drilling, sampling, and logging procedures and verifying drillhole collar locations from the recent Johnny Lee campaign during a field visit.

From late 2018 to present, Mr. Tracy has been involved with geological modelling and resource estimation for the Johnny Lee and Lowry deposits. Currently, Mr. Tracy is acting as QP with responsibility for disclosure related to the geology and Mineral Resources of both deposits. In the opinion of the QP, 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 SRA 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.

Expedio OCRIS™ Mobile logging software was utilized by SRA geologists for the 2018–2022 drilling and MXDeposit™ logging software for was used for the 2023–2025 drilling programs. The digital drillhole logs capture data including collar, survey, lithology, mineralization, structure, sample information, specific gravity, recovery, geotechnical, and other meta-data. The logging templates were designed to match the tables of the SQL database and have library drop-down lists where only pre-defined codes can be recorded, that allows interactive, on-the-fly data validation. Once a hole has been validated, logging files are 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 PDF and text file formats. The assay data is loaded into the database by the DBA. SRA staff run QA/QC scripts to check laboratory performance and determine if assay reruns are needed.

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 quarantined until reviewed and corrected by SRA geological staff.

12.1.2 *Pre-Sandfire Resources America Johnny Lee Drill Data Verification*

In 2010, Tintina twinned five historical drillholes to confirm the geology, mineralization grade, and thicknesses recorded in the Cominco American Inc. (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 historical and Tintina drillholes, allowed Winckers *et al* (2013) to conclude that the historical data, included in the CAI database, was suitable for Mineral Resource estimation, and reporting in accordance with NI 43-101, in support of the Johnny Lee preliminary economic assessment (PEA).

As part of the 2013 PEA, Winckers *et al* (2013) checked assay certificates against database entries for 935 analyses of Tintina 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 Tintina drilling and found no errors. The QP for Mineral Resources reviewed the work performed by Winckers *et al* (2013) and determined the historical data verification is acceptable for use in the determination of Mineral Resources at the Johnny Lee deposit.

12.1.3 *Pre-Sandfire Resources America Lowry Drill Data Verification*

In 2010, Tintina twinned one historical drillhole to confirm the geology, mineralization grade, and thickness. The drill data from the historical drillhole (SC-80) and the twin hole (SC10-006) were reviewed based on historical 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. Grade and re-logging data were reviewed by the QP of Mineral Resources and determined to be acceptable for use in modelling, estimation, and reporting of Mineral Resources at the Lowry deposit.

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.

In 2020, data was selected randomly for one drillhole from each historical drill program era (SCC-39, SC-86, SC10-006, SC11-082 and SC12-155) and were validated by SRK, which represented 10% of all drillholes used to inform the previous Lowry deposit Mineral Resource estimate. Although some minor inconsistencies were noted in the historical (pre-Tintina) Cu analytical data that had been captured, these variations are not considered material. All other validation checks performed by SRK 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 represented 11.5% of the total sample intervals in the Lowry deposit during the 2020 review. 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 were unavailable for the pre-Tintina SC- and SCC- prefix drillholes. Historical 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 (up to the 2% Cu upper limit of detection) analyses by acid digest and Atomic Absorption Spectroscopy (AA) 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, sixteen 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 fourteen analyses (from a total of 188) were sourced from the high-level Cu circuit data, which resulted in an average 0.05% higher value than if low-level analyses were used consistently and is not considered a material error. Apart from inconsistency of source data, no other issues or errors were identified in the analytical data from the pre-Tintina drilling. All sample numbers from the paper records matched those in the database.

Analytical certificates from ALS (Reno or Vancouver) are available for all Tintina drilling. During 2020, 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 utilised at the time of logging (which have changed during different drill programs), the percentages of different lithotypes in each interval was recorded. This data was then compared against the current SRA database in which logging codes have been standardised. No material errors were observed.

In 2020, 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 mineralised 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 other examples, the errors 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 drillholes 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 NL personal 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: initial validation was completed upon finalization of drilling, with final validation 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, 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 three-dimensional visualization software (Micromine™) to assure that it is spatially reasonable with surrounding drillholes in terms of geology and mineralization positions.

Once all these checks were performed, the drillhole was assigned a 'PV' code (partial validated) in the Validation field of the Collar table in the database.

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 Drillhole physical data validation form

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 copper 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 (CRM), blanks, and duplicates performance was then reviewed. If assay data was considered acceptable, then the interval was assigned a 'V' code (validated) in the Validation field of the database.

12.1.6 2021–2022 SRA Lowry Drilling Data Verification

For the 2021–2022 Lowry drill program, hole collar location data was compared to the field survey log sheets. Then, drillhole data (collars, survey, geology, assay, SG) was loaded into Leapfrog Geo™ software for consistency checks. Errors detected at this stage include drill interval gaps, duplicates, invalid values, and total drill depth records. Once consistency errors were rectified, the drill data was visually checked in three-dimensions to assure spatial consistency with surrounding drillholes in terms of expected geology and mineralization positions. The visual check included review of the downhole drill trace for excessive hole deviation while drilling. Downhole survey records for suspect holes were checked against the downhole survey tool readings. Selection queries were applied to the imported drill data to highlight inconsistencies between geologic logging and assay submittals. This step assures that core intervals with visual sulphides were included in the samples sent for laboratory analysis.

Density data was checked numerically in Access and then spatially in Leapfrog Geo™. Numeric checks identify duplicates, depth overlaps, excessively low- and high-density readings. Visual checks focus on spatial relationships between density values, logged core, and the collected density distribution.

CSV files received from the ALS laboratory were loaded into Access and compared to the data export received from Perth. This method allows for a 100% assay record comparison. Minor corrections were made with respect to missing sample IDs causing missing records.

12.1.7 2023–2025 Johnny Lee Drilling Data Verification

The 2023–2025 drilling campaign followed the same data verification process and drillhole validation checklist as 2021–2022 drilling (Figure 12.1). MXDeposit™ was used by SRA to detect most errors early at the data input stage.

12.1.8 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, SG, and Structure tables as comma-separated value (.csv) files. Data associated with adjacent properties or other prospects were not provided, nor reviewed by SRK.

For the Johnny Lee and Lowry deposits, SRK was given a copy of the Leapfrog Geo™ project containing the drilling files and modelled shapes. This allowed for visual checks of the drill data, verification of value handling for missing intervals, applied corrections to the database, and statistical analysis of samples used for modelling.

As part of the 2018 QP site visit, SRK observed drilling activities, sampling, logging, cutting, and core photography activities. Each step in the observed process followed established procedures and were considered to be good industry practice in data collection and management.

In 2020, 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.

In accordance with NI 43-101 guidelines, Berkley Tracy, P.Geo (PGO# 3024), Principal Consultant, SRK Consulting (U.S.), Inc., acting as QP for Mineral Resources, visited the property in September 2022 and June 2025. During the site visits, Mr. Tracy toured the property, reviewed general operations, drilling procedures, sampling practices, and examined available drill core. In 2022, he conducted detailed data verification with random checks of hard copy drilling files and the SRA electronic drill database. During the site visits, SRK was given full access to relevant data. Site personnel were interviewed regarding procedures used to collect, record, store, and evaluate the drillhole assay data.

During the QP site visits, SRK reviewed the geology, available outcrop exposures, and the general geological understanding of the Johnny Lee and Lowry deposits. Related to use in geological modelling, the discussions between the SRA geology team and SRK focused on understanding the genesis of the deposit, the main trends of mineralization, and the role played by the lithological and structural setting. The QP for Mineral Resources considers the Black Butte project geological interpretations of mineralization continuity, controls, and host lithologies to be adequate for the project stage.

While onsite in September 2022, SRK compared a portion of the original laboratory data certificates, geological logs, and downhole deviation surveys to entries in the current SRA database. In the White Sulphur Springs office, SRK was able to source historical laboratory data on paper and in digital format. The verification data subset was chosen randomly to be representative of the entire database spatially and across multiple drilling campaign eras. In total, fifteen drillholes were audited by SRK, including eight in the Johnny Lee deposit (SCC-30, SCC-34, SC-59, SC-70, SC10-004, SC11-040, SC12-118, and SC18-239) and seven in the Lowry deposit (SCC-42, SC-80, SC-87, SC11-083, SC12-166, SC21-270, and SC22-275). The total drilling depth of the verification drillholes was 7,214.6 m, which represented 5.3% of the total meters in the database at the time.

The SRA database was compared line-by-line to the fundamental verification data for assay records and lithology logs. No errors or discrepancies were observed across 2,326 assay records and 717 lithology intervals. Slight variance in downhole deviation surveys was noted in one of the recent Lowry drillholes (SC21-270), which was a result of temporary surveys from drillhole tracking not being over-written with the final completion survey in the SRA database. Both surveys had similar hole dip and azimuth; however, SRA corrected to the final survey in the database. To confirm that this minor error was not prevalent, SRK reviewed the survey records for an additional eight drillholes from the 2021–2022 campaign. Survey records were not located for some of the historical drillholes (SCC- series) at Johnny Lee, as some of the vertical drilling was determined to not have been surveyed for downhole deviation.

No additional material discrepancies were revealed. The lack of significant errors being uncovered during data verification is uncommon. The robustness of the fundamental data compared to the current digital records provides evidence that the SRA database is maintained adequately and accurately represents the original collected sample data.

12.2 Limitations

SRK noted no data limitations were uncovered during validation. No material errors were observed during SRK's review of the database provided by SRA.

12.3 Opinion on Adequacy

The exploration and sampling protocols practised at the Black Butte Copper project are consistent with or exceed generally accepted industry standards and are deemed adequate for the project stage. It is the opinion of the QP for Mineral Resources, that data verification checks performed internally by SRA staff in combination with multiple external reviews and independent checks by SRK, have resulted in sufficient validation of the fundamental drilling database at the Black Butte Copper project. The data is acceptable and adequately reliable for use in geological modelling and the estimation of Mineral Resources.

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13. MINERAL PROCESSING AND METALLURGICAL TESTING

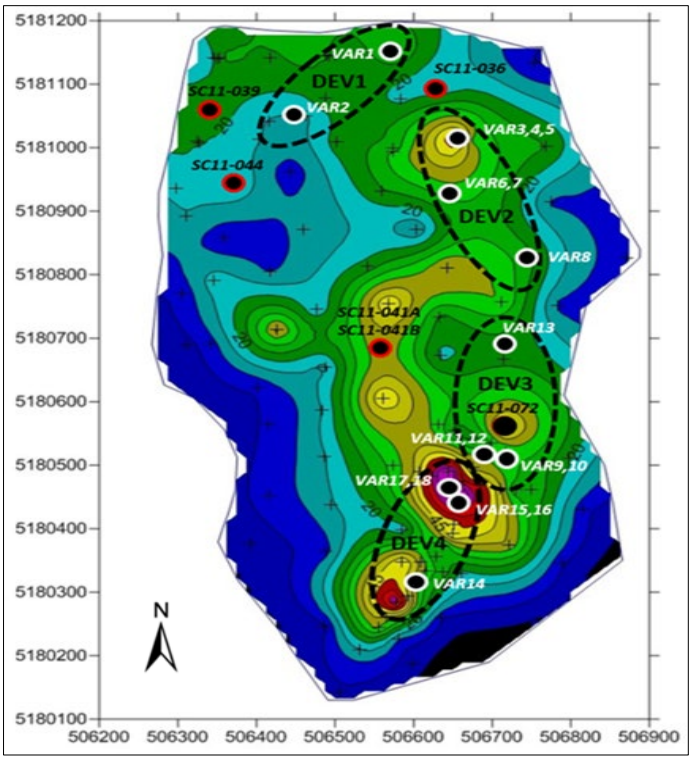
13.1 Introduction

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 Overall composite (OA) of Johnny Lee Upper Copper Zone (JL UCZ) and a master overall composite of the Johnny Lee Lower Copper Zone (JL LCZ) from the Johnny Lee deposit. Tests included optical and QEMSCAN mineralogical analysis, comminution tests, batch flotation tests and a single Locked Cycle Test (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 JL UCZ and eight variability samples from the JL LCZ. From the variability samples, four development composites were generated for the JL UCZ (Dev-Comp 1 to 4) and one development composite for the JL 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).
- 2024 PFS test program was developed to investigate the metallurgical performance of the Johnny Lee Lower Copper Zone (JL LCZ), along with blended tests of the lower zone with Johnny Lee Upper Copper Zone (JL UCZ). Composites consisting of blends of 40% JL UCZ and 60% JL LCZ were developed in conjunction with 100% of JL UCZ and JL LCZ composites as the baseline for each zone lithology. Key objectives of the test program were:
  - Assessment of the composites under the standard flotation regime conditions and reagents used in the previous test work.

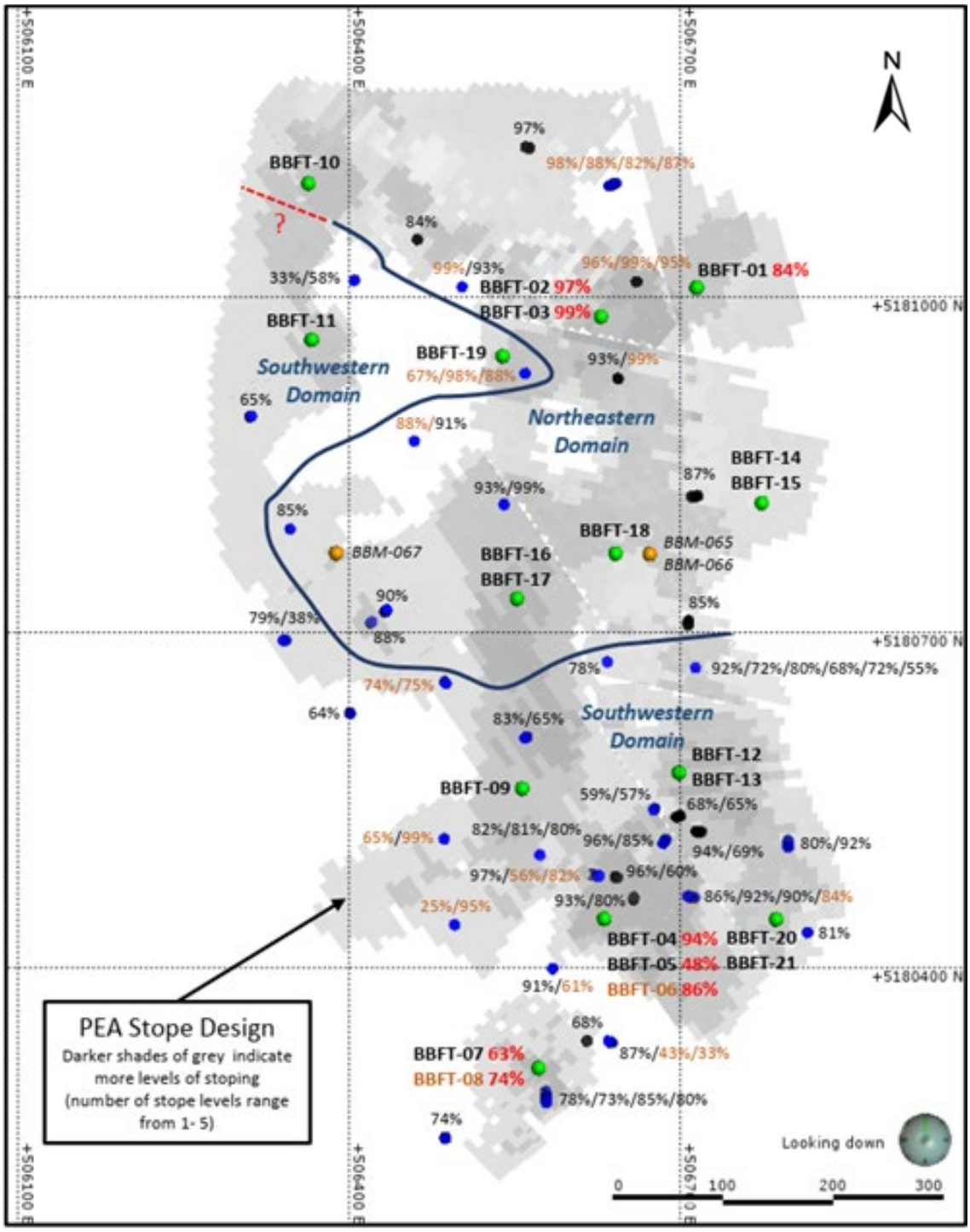
- Reassess the primary grind size and regrind grind sized used in the previous programme, on the flotation performance of the new composites.
- Assess the effects of modifying depressant dosage on the flotation performance of the constructed composites.
- Examine alternative flowsheet configurations.

- Flotation testing was conducted to a batch cleaner stage, with no locked cycle tests conducted in the new test work phase.
- 2025 PFS test program was developed to investigate the metallurgical performance of the JL LCZ through:
  - Construction of twelve composites consisting of JL UCZ and JL LCZ variability, and seven blends of JL UCZ/JL LCZ material and assess the chemical composition.
  - Assess the comminution properties of four constructed composite by performing SMC testing, Bond Abrasion Index (Ai) tests and Bond Ball Work Index (BWi)
  - Assess the flotation performance of the twelve constructed composites and seven blends of JL UCZ/JL LCZ material using the flotation conditions and reagent scheme developed in past testing programs.
  - Conduct a single locked cycle test on the blended JL UCZ/JL LCZ material



Source: SRA, 2026

Figure 13.1 PEA And PFS Sample Location Map



Source: SRA, 2026

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, 2012a and 2012b). The PEA tested five composites from the JL UCZ, one overall composite from the JL UCZ (UCZ-OA) and one overall composite from the JL LCZ (LCZ-OA). The test work included comminution testing, mineralogical studies, and LCT's. Detailed reporting of this work can be found in previous NI 43-101 study document.

13.2.1.1 Comminution Summary

Comminution tests reported a BRWi of 15 kWh/t and a BBWi of 14.8 kWh/t using a 53 µm closing screen on samples from the Jonney Lee UCZ. Jonney Lee LCZ tests reported BRWi of 12.6 kWh/t and a BBWi of 11.8 kWh/t using a 75 µm closing screen. The abrasion index was only determined for the Jonney Lee UCZ and returned an Ai figure of 0.6885 g.

13.2.1.2 Flotation Summary – Locked Cycle Tests on Upper and Lower Copper Zones

A single locked cycle test was conducted on the JL UCZ-OA composite using the developed flowsheet from the programme. Results demonstrated copper recoveries of 82.2% were achievable at an average copper 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.

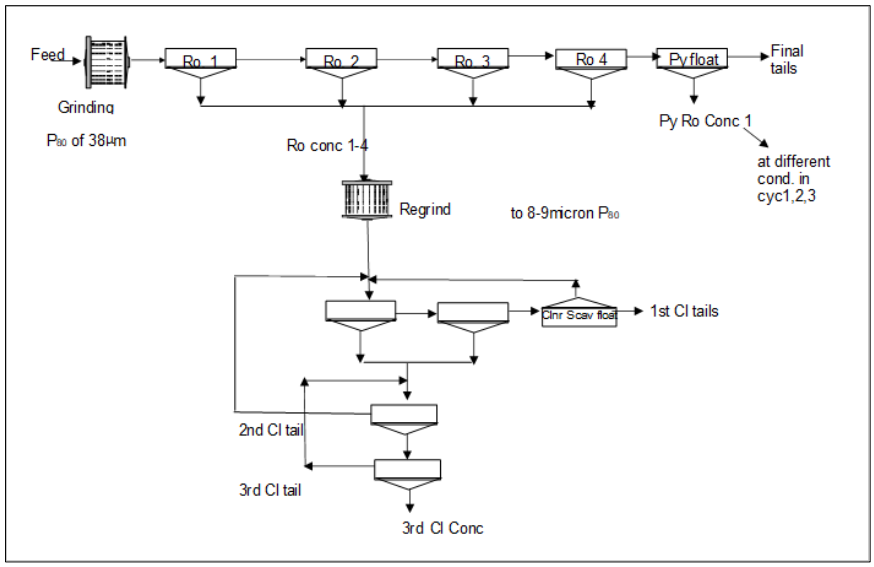


Figure 13.3 LCT PEA Flowsheet

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 flotation testing conducted on the Jonney Lee 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 JL 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 13.4 followed a similar flowsheet to the JL UCZ, with the exception being the removal of the third cleaning stage and the pyrite flotation step.

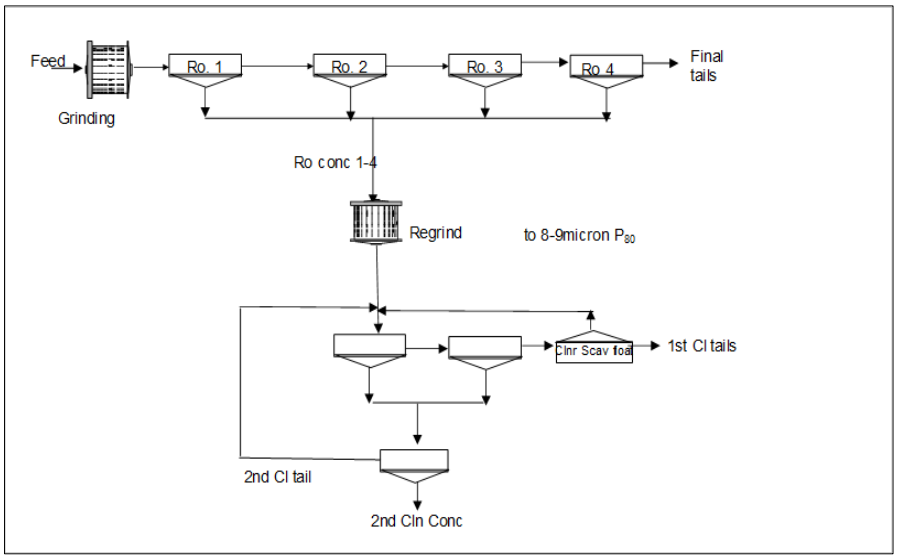


Figure 13.4 LCT Flowsheet for LCZ

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

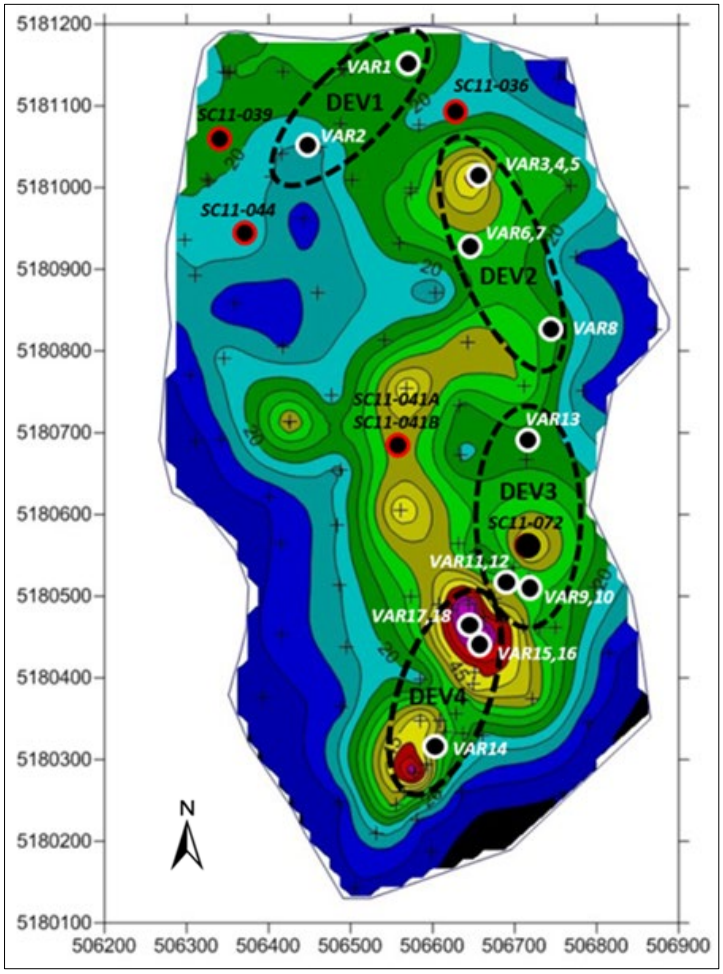
Table 13.2 LCT Results – JL LCZ Composite

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 2016 PFS study. Detailed reporting of this work can be found in previous NI 43-101 study document.

13.2.2.1 2016 Locked Cycle Tests

Five main development composites were prepared consisting of four composites for the JL UCZ and one composite for the JL LCZ. The composites examined the northern part of the JL 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 13.5 for sample locations.



Source: SRA, 2026

Figure 13.5 Development Composite Location Map

Locked cycle testing utilized a 30 µm P_{80} primary grind and a 10 µm P_{80} regrind of the rougher concentrate.

The reagent scheme was similar to the 2012 PEA test work program, and included, Lime, SIPX, Cytec 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 JL UCZ development composites used two flowsheet options. These are shown in Figure 13.6.

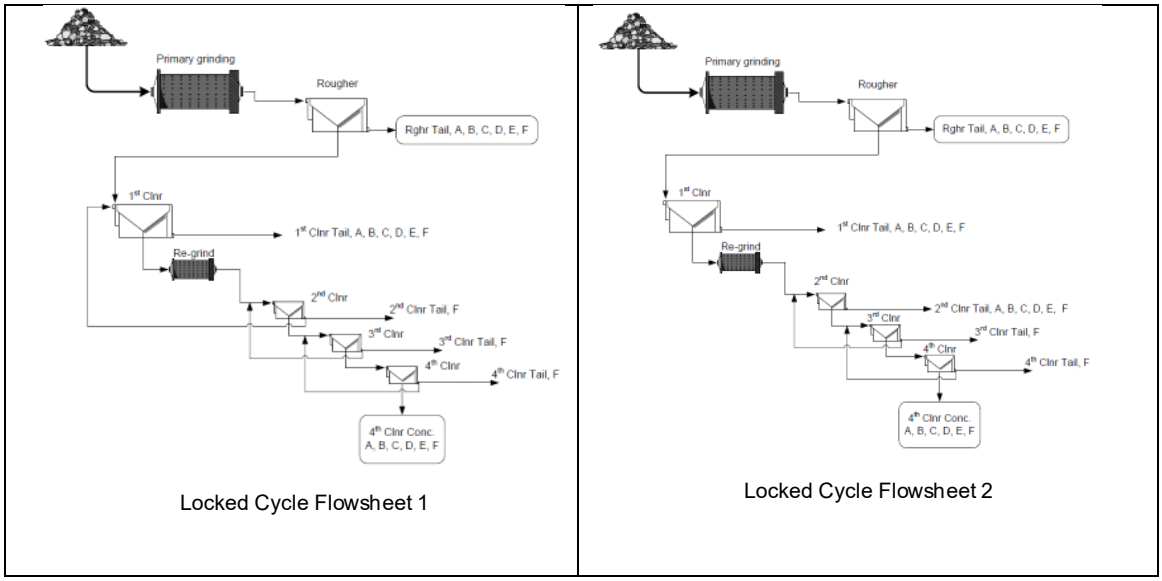


Figure 13.6 JL UCZ Development Composite LCT Flowsheets

Results from the LCT's showed considerable variation in both Cu grade and recovery to concentrate as reported in Table 13.3. 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	26.6	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	22.9	26.2	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	22.3	28.7	0.18	100	100	100	100
Dev 3-LCT3 FS2	**4th Clnr Conc.**	**6.1**	**23.1**	**32.0**	**38.8**	**1.455**	**70.3**	**8.9**	**8.3**	**49.8**
	Head (calculated)		2.01	22.0	28.5	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 13.3 Dev Comps 1 to 4 - LCT JL UCZ Summary

A single LCT was conducted on the JL 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 13.4).

Test ID	Product	Weight		Grad %, 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 13.4 JL LCZ – LCT

The primary grind utilized in the JL LCZ was coarser than the JL 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.

13.2.3 2019 Base Metal Laboratories FS Test Program – Johnny Lee Upper Copper Zone

The FS work program was undertaken at Base Metal Laboratories (BML) in Kamloops BC. The program was reported in BL02082 July 2019 and consisted of 21 variability samples taken across the JL UCZ of the deposit and one JL LCZ composite as illustrated in Figure 13.2. Detailed reporting of this work can be found in previous NI 43-101 study document.

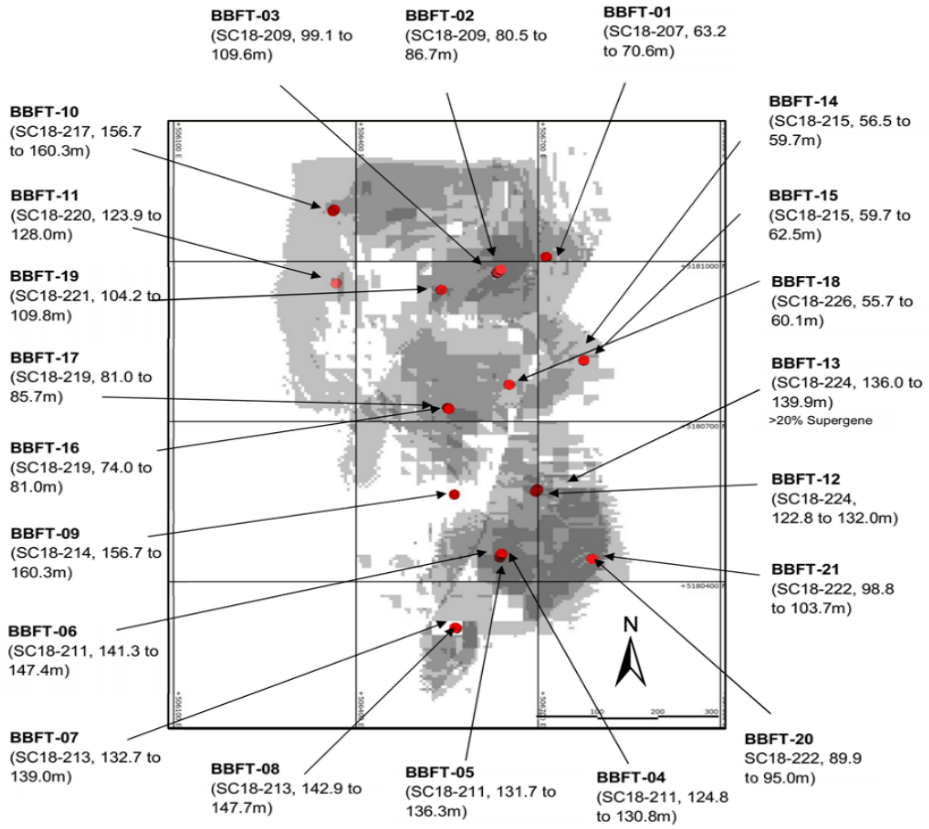
13.2.3.1 Samples

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. Table 13.5 and Figure 13.7 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%

Metallurgical Composite	MODA Recovery %	Supergene Alteration	Cu Grade	Tennantite	% Cu in Tennantite
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%
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 13.5 Metallurgical Composite Table



Source: SRA, 2026

Figure 13.7 Metallurgical Sample – Drillhole Locations

13.2.3.2 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; 2017a; and 2017b) 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 13.6 and the liberation characteristics based on a grid count and 53 µm mask are reported in Table 13.7.

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 13.6 BBFT Series Weight % Composition by Mineral Species

Metallurgical Composite	Free	Binary with					Binary	Ternary	Quat.	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 13.7 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 13.7, 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.

13.2.3.3 Locked Cycle Tests

Two sets of LCT's were performed on a subset of samples from the JL UCZ. The samples were selected by SRA technical staff and best represent the JL 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 Table 13.8. 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 13.8 LCT Summary

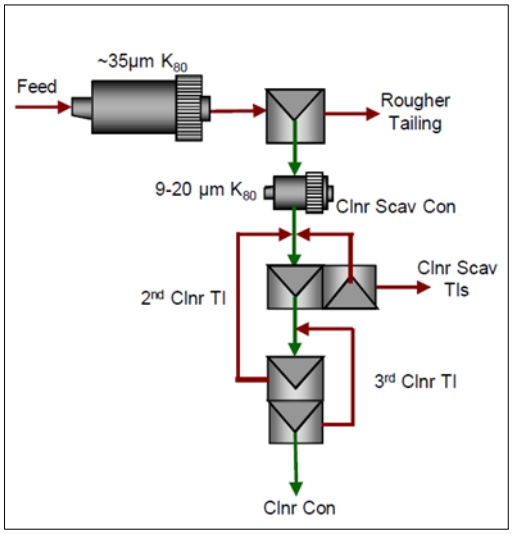
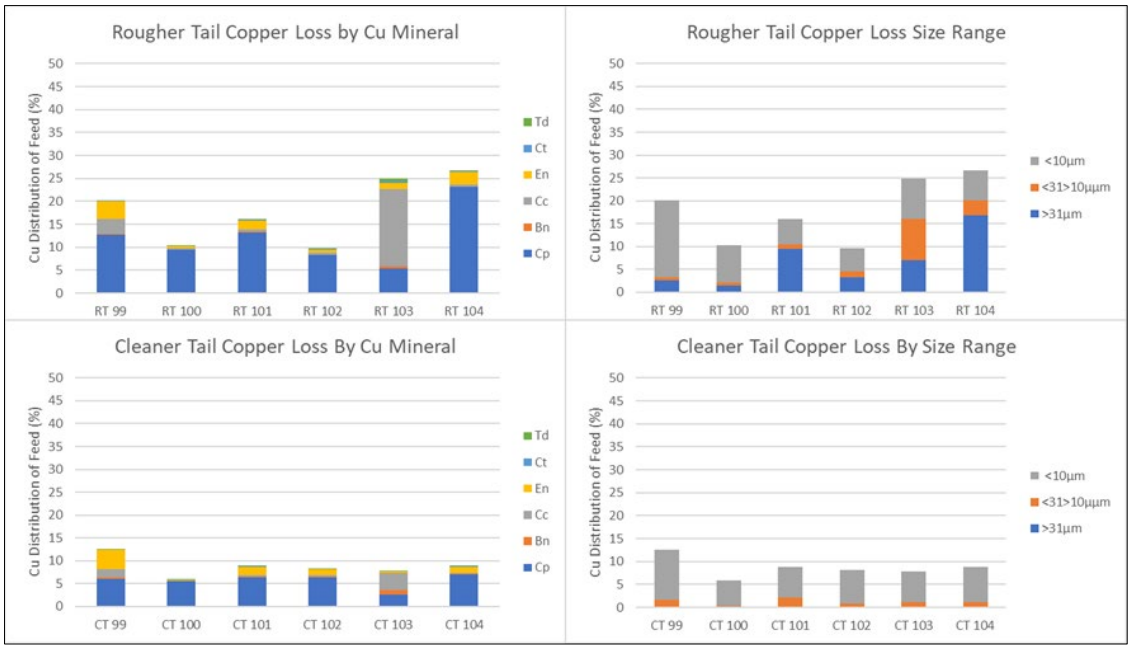


Figure 13.8 LCT's Flowsheet

13.2.3.4 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 13.9.



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

Figure 13.9 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.

13.2.3.5 Locked Cycle – Minor Element Analysis

Concentrates from the LCT's were subjected to minor element analysis (Table 13.9).

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 13.9 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.

13.2.4 2019 Base Metal Laboratories FS Test Program – Johnny Lee 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 JL 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 JL UCZ sample were conducted in this phase of the study.

13.2.4.1 FS JL Upper Copper Zone – JL Lower Copper Zone Blend Locked Cycle Test

The primary purpose of generating the JL LCZ composite was to determine the effect of blending JL UCZ and JL LCZ mineralization on Cu metallurgy. A blend of 76:24 of the JL UCZ - JL LCZ was created for additional testing. A single batch open cleaner test, followed by LCT's, were performed using the JL UCZ process flowsheet and reagents developed. (Figure 13.10).

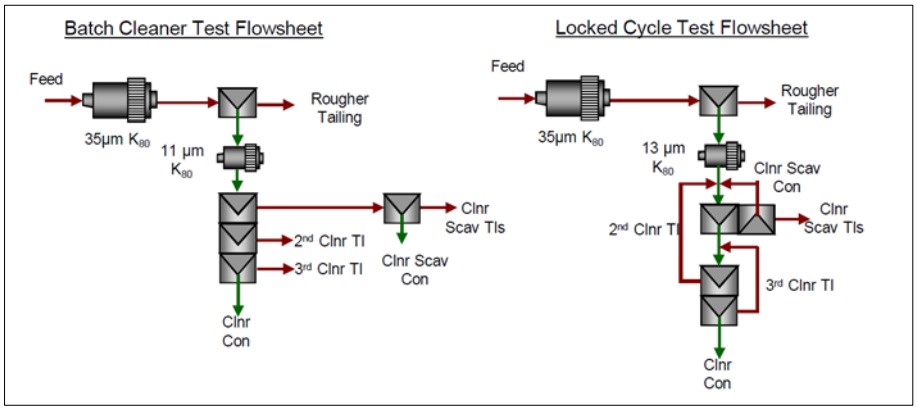


Figure 13.10 JL UCZ- JL LCZ Flotation Flowsheet Development

The performance of the composite was good and compares to the earlier test work on the JL UCZ. Overall, a 20% Cu concentrate at 70% Cu recovery was achievable in the batch open cleaner test (Figure 13.11).

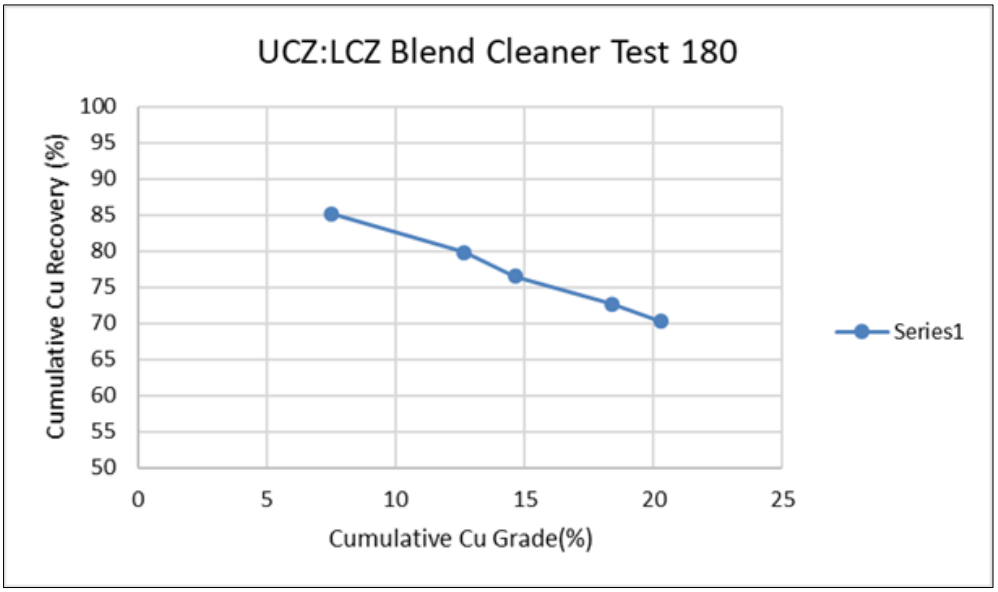


Figure 13.11 JL UCZ - JL LCZ Batch Cleaner Test (T-180)

The LCT results, given in Table 13.10, 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 13.10 LCT Performance

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

13.2.5 2022–2023 PFS Test Program – Lowry Middle Copper Zone and Lowry Lower Copper Zone

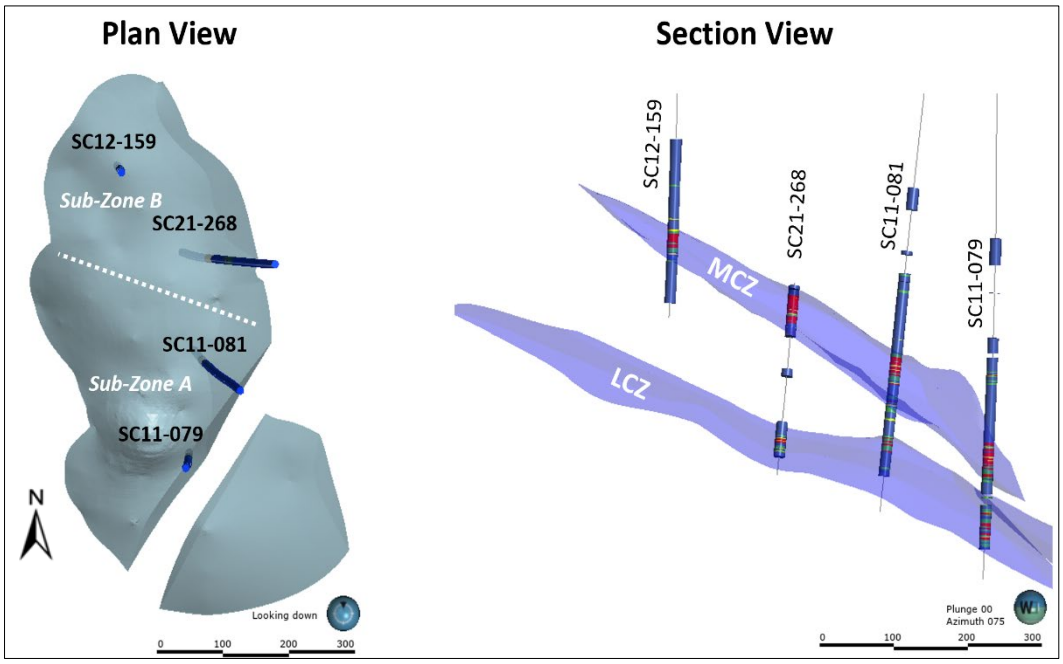
13.2.5.1 Metallurgical Samples

Approximately 201 kg of sample representing the middle and lower copper zones was shipped to Base Metals Laboratories Ltd in Canada in September 2022.

These samples constitute the two master composites for the initial test work. Table 13.11 and Figure 13.26 and Figure 13.27 detail the sample locations.

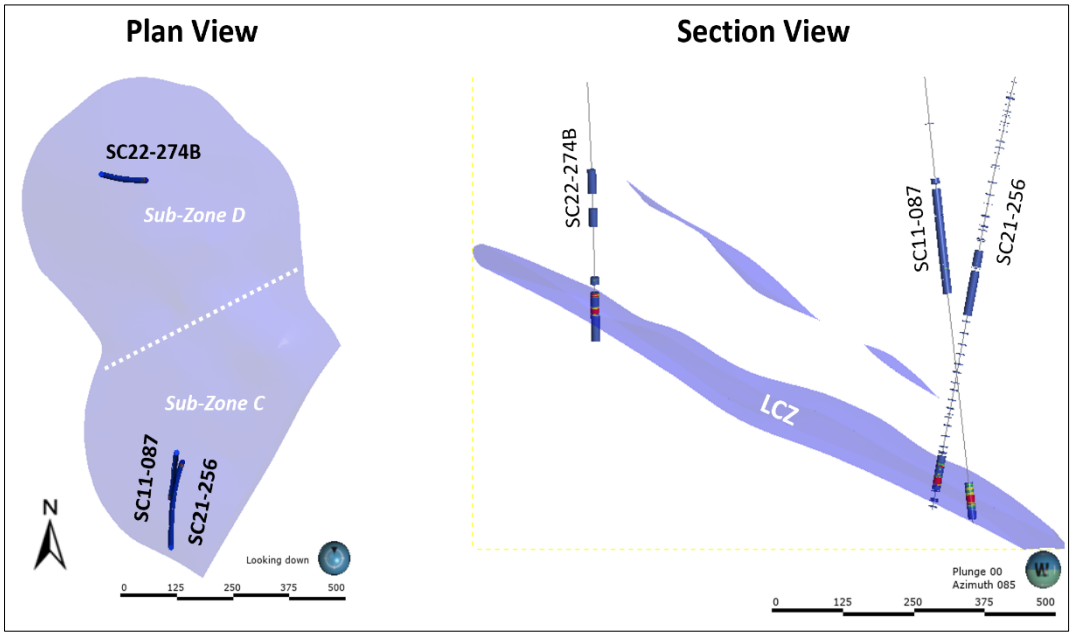
HoleID	From	To	Length	Zone	Avg_Cu%	Sub-Zone
SC11-079	557.87	577.50	19.63	LMCZ	2.22	A
SC11-081	443.66	463.58	19.92	LMCZ	1.87	A
SC12-159	292	309.7	17.70	LMCZ	1.98	B
SC21-268	377.77	400.92	23.15	LMCZ	3.47	B
		Total	**80.40**		**2.44**	
HoleID	**From**	**To**	**Length**	**Zone**	**Avg_Cu%**	**Sub-Zone**
SC11-087	809.00	823.75	14.75	LLCZ	2.67	C
SC21-256	797.40	808.7	11.30	LLCZ	3.59	C
SC22-274B	489.50	497.86	8.36	LLCZ	4.31	D
		Total	**34.41**		**3.37**	

Table 13.11 Composite Groups and Copper Percent



Source: SRA, 2026

Figure 13.12 LMCZ Sample Locations



Source: SRA, 2026

Figure 13.13 LLCZ Sample Locations

Variability composites were also constructed from discrete intervals from earlier shipment as shown in Table 13.31. Low-grade composites were prepared using the samples from the two shipments. The details of these composites are presented in Table 13.13 to Table 13.15.

Sample Name	Hole Id	From (m)	To (m)	Zone	Mass (kg)
MCZ Var 1	SC11-079	557.87	569.82	LMCZ	**14.0**
MCZ Var 2	SC11-079	569.82	577.50	LMCZ	**8.6**
MCZ Var 3	SC11-081	443.66	463.58	LMCZ	**15.0**
MCZ Var 4	SC12-159	292	309.7	LMCZ	**15.0**
MCZ Var 5	SC21-268	377.77	390.83	LMCZ	**10.0**
MCZ Var 6	SC21-268	390.83	400.92	LMCZ	**9.2**
LCZ VAR 1	SC11-087	809.00	816.25	LLCZ	**3.4**
LCZ VAR 2	SC11-087	816.25	823.75	LLCZ	**3.3**
LCZ VAR 3	SC21-256	797.40	808.7	LLCZ	**4.4**
LCZ VAR 4	SC22-274B	489.50	497.86	LLCZ	**3.5**

Table 13.12 Description of Variability Composites

Hole Id	From (m)	To (m)	Length	Mass Received (kg)	Split	Mass (kg)
SC21-263	270.51	287.3	16.79	28	1.00	28
SC21-269	576.57	597.29	20.72	35	1.00	35
SC21-267	621.49	624.19	2.7	3.6	1.00	3.6
					Total	66.6

Table 13.13 Description of LMCZ-LG1 Composite

Hole Id	From (m)	To (m)	Length	Mass Received (kg)	Split	Mass (kg)
SC22-276B	479.91	494.81	14.9	12.3	1.00	12.3
SC22-273	458.99	464.97	5.98	12.3	1.00	12.3
					Total	24.6

Table 13.14 Description of LMCZ-LG2 Composite

Hole Id	From (m)	To (m)	Length	Mass Received (kg)	Split	Mass (kg)
SC21-263	482.84	492.61	9.77	18.5	0.50	9.25
SC21-267	741.71	744.63	2.92	7	0.25	1.75
SC21-268	560.74	566.93	6.19	9.8	0.25	2.45
SC22-272	639.62	643.22	3.6	7.1	1.00	7.1
SC21-269A	459.78	469.22	9.44	7.4	1.00	7.4
					Total	27.95

Table 13.15 Description of LLCZ-LG Composite

13.2.5.2 Feed Characterization

The head analyses of the two composites are presented in Table 13.37.

The LMCZ and LLCZ composites contained 2.37% and 3.32% copper and 11.2% and 15% sulfur, respectively. The copper is predominantly present as chalcopyrite with minor levels of enargite, tennantite, bornite, chalcocite, and covellite. The non-sulfide mineral in the samples is primarily quartz.

Sample	Assays				
	% Cu	% Fe	% S	% C	% TOC
LLCZ	3.32	9.30	11.2	0.39	0.16
LMCZ	2.37	12.60	15.0	0.62	0.20

Table 13.16 Head Analyses of Composite Samples

13.2.5.3 Comminution Characteristics

The two master composites were subjected to an SAG Mill Comminution (SMC) test, a Bond's ball mill work index test at a closing screen size of 75 µm, and a Bond abrasion index determination. The results are presented in Table 13.17.

Sample	SMC A x b	Bond's BWi @ 75 microns, Kwh/tonne	Abrasion Index
LLCZ Composite	64.7	24.9	0.4
LMCZ Composite	44.7	24.2	0.8

Table 13.17 Comminution Data for the LMCZ and LLCZ Composites

13.2.5.4 Locked Cycle Tests

A single locked cycle test was conducted on each composite to establish estimates of metallurgical performance in a continuous closed-circuit process.

The test procedure is presented in Table 13.18, the flowsheet is presented in Figure 13.29, and the test results are summarized in Table 13.19.

Stage	Reagents - g/tonne					Time - minutes		Electrochemistry	
	Lime	MSP	Dextrin	3477	MIBC	Cond	Float	pH	Eh - mV
Primary Grind	1000	400	320			20		6.0	67
Aeration	930					2		9.5	-20
Rougher 1	-			40	28	1	1	9.5	3
Rougher 2	√			20	28	1	2	9.5	-19
Rougher 3	√			10	28	1	3	9.5	-16
Rougher 4	√			5	14	1	4	9.5	-13
Rougher 5	√			5	14	1	5	9.5	-13
Regrind	200	100	75			16.5		8.9	118
Cleaner 1	100			50	98	1	8	9.5	89
Cleaner 2	√			7	42	1	6	9.5	100
Cleaner 3	√			2	42	1	4	9.5	111
Cleaner Scavenger	√			10	-	1	3	9.5	101

Primary Grinding Conditions	
Mill: Media: Water Addn:	Mild Steel Mill 20kg SS rods 750mL
Regrind Conditions	
Mill: Media:	Stirred Ceramic Beads

Flotation Information	Rougher	Cleaner 1	Cleaner 2
Flotation Device:	D12	D12	D12
Cell Volume (L):	2.5	2.5	1.5
Flotation Gas:	Air	Air	Air
Impellar Speed:	1300rpm	1300rpm	1100rpm
Flotation Gas: Water Type:	Air Kamloops Tap Water		

Note: *Primary Grind: P_{80} = 62 – 83 µm*
Regrind: P_{80} = 14 – 21 µm

Table 13.18 LCT Procedure for the Composites

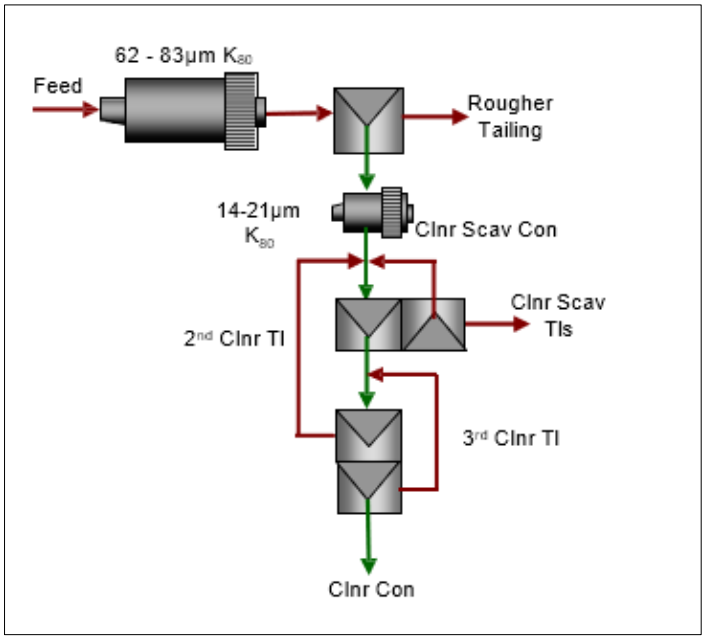


Figure 13.14 LCT Flowsheet

Product	Mass	Assay - percent or g/t				Distribution - percent			
	%	Cu	Fe	S	C	Cu	Fe	S	C
Test 13: LMCZ Cycles D+E									
Feed	100.0	2.18	12.6	14.1	0.67	100	100	100	100
Cu Con	8.5	23.3	30.8	36.8	0.41	91.1	20.9	22.3	5.2
Cu Clnr Tail	22.1	0.42	16.4	17.9	0.84	4.3	28.8	28.1	27.8
Cu Rougher Tl	69.4	0.15	9.1	10.1	0.64	4.6	50.4	49.6	67.0
Test 14: LLCZ Cycles D+E									
Feed	100.0	3.27	9.0	10.7	0.43	100	100	100	100
Cu Con	11.2	27.5	29.8	39.0	0.24	94.2	37.0	40.9	6.2
Cu Clnr Tail	18.4	0.69	19.5	24.1	0.85	3.9	39.6	41.5	36.4
Cu Rougher Tl	70.4	0.09	3.0	2.67	0.35	1.9	23.3	17.6	57.5

Table 13.19 LCT Results for LMCZ and LLCZ Composites

The tests were performed on the sample ground to a nominal P_{80} of 62 to 83 microns and a regrind of P_{80} of 14 to 21 microns. The flotation pH was maintained at 9.5 using lime and the same reagents were used in both rougher- and cleaner flotation.

Both composites performed very well as compared to the Johnny Lee samples. The final flotation copper concentrate assayed 23.3% copper and recovered 91.1% of copper for LMCZ composite. The copper recovery for LLCZ composite, assaying 27.5% copper, was 94%.

13.3 2024 BML Metallurgical Test Programme

The 2024 PFS test program was developed to investigate the metallurgical performance of the Johnny Lee Lower Copper Zone (JL LCZ), along with blended tests of the lower zone with Johnny Lee Upper Copper Zone (JL UCZ). Composites consisting of blends of 40% JL UCZ and 60% JL LCZ were developed in conjunction with 100% of JL UCZ and JL LCZ composites as the baseline for each zone lithology. Key objectives of the test program were:

- Construct composites of JL UCZ, JL LCZ and blends of JL UCZ/JL LCZ material and assess the chemical and mineralogical characteristics.
- Assess the flotation performance of the constructed composites using the flotation conditions and reagent scheme developed in past testing programs.
- Assess how primary grind sizing and regrind sizing affect the flotation performance of the constructed composites using the previously developed reagent scheme. Assess the effects of modifying depressant dosage on the flotation performance of the constructed composites.
- Examine alternative flowsheet configurations and testing procedures using the constructed composites.

13.3.1 Composite Makeup

Two zones of feed material were tested in the program, the Johnny Lee Upper Copper Zone (JL UCZ) and the Johnny Lee lower Copper Zone (JL LCZ). Blended composites were developed based on a 40% JL UCZ and 60% JL LCZ ratio as well as 100% JL UCZ and 100% JL LCZ composites to evaluate straight flotation with blended ore sources more aligned to the mining operation (Table 13.20).

Composite	JL UCZ %	JL LCZ%
Composite 1	100	
Composite 2		100
Composite 3	40	60
Composite 4	40	60
JL UCZ 1	100	
JL UCZ-2	100	

Composite	JL UCZ %	JL LCZ%
Blend A	40	60
Blend B	40	60
Blend C	40	60

Table 13.20 2024 Composite Blends

The chemical composition of the composites is report in Table 13.21.

Composite	Cu%	S%	C%	Ag%	As %	Bi%	Coi%	TOC
Composite 1	2.78	35.7	0.34	21.5	2186	289	0.10	0.23
Composite 2	8.26	33.4	1.87	7.0	1106	257	0.07	0.25
Composite 3	5.51	34.7	1.20	14.0	1405	190	0.11	0.24
Composite 4	5.98	31.7	1.38	7.0	1582	282	0.10	0.33
JL UCZ 1	1.96	39.1	0.28	24.0	1863	88	0.12	0.21
JL UCZ-2	2.14	28.6	0.55	7.0	2652	344	0.11	0.44
Blend A	6.05	33.2	1.19	12.0	1348	243	0.09	
Blend B	3.49	24.5	3.09	9.6	669	172	0.06	
Blend C	3.86	34.3	0.85	13.0	1269	218	0.10	

Table 13.21 Chemical Content Summary

The deposit is a massive sulphide deposit with sulphide minerals accounting for between 60 and 78 % of the sample mass; sulphur assays measured between 20 and 39 %. Most of this sulphur was present as pyrite/marcasite with the copper sulphides accounting for much of the remaining sulphur content. Only low levels of non-sulphide sulphur was present in the samples, with barite accounting for about 2 percent of the total sulphur in Composite 1 and JL UCZ-1.

Copper measured between 4.8 % and 8.3 % in the JL LCZ material and measured between 2.0% and 2.8 % for the JL UCZ composites. Almost all the copper was present as chalcopyrite; however, small amounts of this copper in the JL UCZ composites was measured to be within the arsenic bearing copper sulfosalts tennantite/enargite. Only traces of tennantite/enargite were measured in Composite 2 (JL LCZ).

Arsenic measured higher in the JL UCZ material compared to the JL LCZ material, with a low of about 0.01 to 0.02 % measured in LCZ-1 and LCZ-2 to a high of about 0.2 to 0.3 % measured in Composite 1 and JL UCZ-2. In the measured samples, the content of tennantite/enargite would only account for a high of 30 % (Composite 1) of the measured arsenic content; this arsenic within copper sulphide minerals would be expected be recovered as readily as copper. The remaining arsenic in the composites was not detected but could be located within the mineral structure other sulphides at contents below the QEMSCAN detection limit, potentially within the pyrite/ marcasite, and would therefore report to streams with the pyrite and marcasite. Completing laser ablation analysis on pyrite and marcasite grains could quantify arsenic present in this form.

Organic carbon measured about 0.2 to 0.4 % in the composites. The organic carbon can be problematic due to the hydrophobic nature of organic carbon and its ability to adsorb flotation reagents. This can potentially result in higher reagent consumption and/or potentially reduced reagent effectiveness. The organic carbon accounted for the majority of the overall carbon in the JL UCZ composites but only accounted for a quarter or less of the overall carbon in the JL LCZ composites. Most of the overall carbon in the JL LCZ composites was present within carbonate minerals, primarily dolomite.

Cobalt measured between 0.03 and 0.12 percent, with higher levels being measured within JL UCZ material. While cobalt sulphide minerals were measured by QEMSCAN, these accounted for only a limited percentage of the assay measured cobalt. Similarly to arsenic, this cobalt could be located within other sulphide minerals such as marcasite and/or pyrite.

Quartz was measured to be the most significant non-sulphide mineral within the composites with feldspars also measuring of note (Table 13.22).

Mineral	Comp 1	Comp 2	Comp 3	Comp 4	UCZ-1	UCZ-2
Pyrite/marcasite	61.1	50.2	59.3	50.3	73.1	50.6
Chalcopyrite	5.44	22.5	15.3	17.0	4.51	8.87
Tennantite/Enargite	0.33	<0.01	0.05	0.09	0.11	0.22
Cobalt Sulphides	0.07	0.11	0.07	0.13	0.01	0.17
Other Sulphides	0.03	0.03	0.02	0.04	0.01	0.05
Quartz	25.5	11.5	12.1	17.3	13.0	26.1
K-Feldspar	2.51	0.11	1.28	4.35	3.04	10.7
Chlorite/Clays	0.02	1.26	0.76	0.78	0.02	0.08
Dolomite	0.58	9.89	6.02	6.30	0.21	0.92
Othetr Carbonates	0.19	1.38	0.89	0.85	0.14	0.05
Barite	3.79	0.05	2.09	0.61	5.14	1.43
Apatite	0.05	2.41	1.47	1.51	0.07	0.18
Other Non-Sulphide Gangue	0.43	0.64	0.65	0.64	0.66	0.64

Table 13.22 Mineral Content Summary

13.4 Rougher Flotation Testing

Testing was completed on the composites listed in Table 13.20. Testing was conducted under the same conditions developed during the 2018 (BL282) feasibility test program.

Rougher testing used the flowsheet from the previous feasibility study as defined in Figure 13.15.

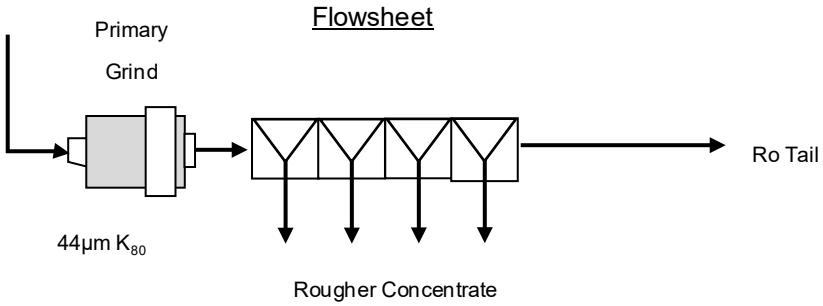


Figure 13.15 Rougher Kinetic Test Flowsheet

The circuit involves a fine primary grind and the use of two depressants monosodium phosphate (MSP) and dextrin with lime used of pH modification to pH 9.5. An aeration step preceded the rougher flotation. The dithiophosphate collector Aero 3477 was used as the copper collector and methyl isobutyl carbinol (MIBC) was used as the frother (Table 13.23).

Stage	pH	Lime g/t	MSP g/t	Dextrin g/t	3477 g/t
Primary Grind	5.0 to 7.4	1000	400	320	
Aeration	9.5	630-2960			
Rougher	9.5	160-1060			80-100

Table 13.23 Reagent setup and variance

13.4.1 Effect of Primary Grind Size

The initial rougher tests looked to re-evaluate the primary size relationship for grade and recovery and was completed on composite 1 through 4.

The grind evaluation results indicated an improvement in copper recovery with a finer primary grind. Sulphur, arsenic, silver, and bismuth all showed reduced recoveries with finer primary grind indicating the likely association of bismuth, arsenic and silver with the pyrite/marcasite. Results for Composites 1 to 4 are detailed in Table 13.24 with the grind to recovery relationship detailed in Figure 13.16.

Composite 1																		
Grind	Test	PG	Weight		Assay - percent or g/t							Distribution - percent						
time		µm K80	%	grams	Cu	Fe	S	C	Ag	As	Bi	Cu	Fe	S	C	Ag	As	Bi
18	T01	33	24.7	248.0	9.6	32.7	43.0	0.4	32.6	4589	501	92.1	32.1	31.9	25.1	36.8	56.1	45.7
16	T08	44	24.4	243.9	10.0	31.4	41.9	0.3	30.8	5075	536	91.2	30.9	29.9	27.3	34.4	56.4	46.4
14	T09	58	21.4	214.4	10.6	30.3	41.2	0.3	28.5	5052	533	86.5	26.6	26.1	16.9	28.0	51.0	40.9
12	T10	69	19.2	192.1	11.2	31.4	42.2	0.2	28.2	5389	543	82.2	24.2	23.3	16.2	25.2	48.1	37.5
11	T11	77	21.4	213.7	10.1	30.6	41.0	0.2	29.5	4859	516	80.7	27.0	26.9	18.2	29.1	50.9	41.3
4	T25 Flash	409	2.8	27.9	18.5	29.5	38.9	0.4	19.4	7524	637	19.0	3.2	3.2	2.9	2.6	10.0	6.2
16	T35 MF2	48	21.2	209.4	11.0	31.6	40.8	0.3	29.4	4893	525	88.9	26.7	26.8	20.0	30.0	53.1	43.2
Composite 2																		
Grind	Test	PG	Weight		Assay - percent or g/t							Distribution - percent						
Time		µm K80	%	grams	Cu	Fe	S	C	Ag	As	Bi	Cu	Fe	S	C	Ag	As	Bi
12	T02	38	48.3	481.4	16.0	31.9	39.9	0.9	9.6	1023	260	99.1	56.7	57.2	23.4	63.4	48.3	50.6
10	T12	40	41.8	416.2	19.2	31.8	42.1	0.6	9.4	980	237	98.6	48.2	52.3	15.1	52.8	37.0	38.8
8	T13	50	48.7	486.0	16.2	32.2	41.3	0.7	9.4	1079	251	98.5	57.8	59.2	19.0	64.1	50.3	50.4
7	T14	57	48.7	485.4	16.3	32.4	42.6	0.6	9.2	906	226	98.3	57.4	60.2	17.4	59.4	44.1	46.3
6	T15	58	52.2	517.8	15.1	32.3	42.5	0.6	8.9	950	254	98.6	63.0	67.2	17.0	63.6	48.7	52.9
5	T24	84	56.9	565.9	14.0	29.9	41.2	0.9	9.5	1099	269	98.1	67.6	73.8	26.7	70.0	57.3	61.2
2	T26 Flash	351	6.5	64.8	24.2	31.7	40.5	0.2	8.2	560	147	19.9	7.7	7.6	0.5	7.8	3.5	4.3
11	T36 MF2	32	44.7	441.8	17.9	33.1	40.7	0.5	8.7	880	222	98.2	54.1	56.9	13.0	54.1	40.3	43.8
Composite 3																		
Grind	Test	PG	Weight		Assay - percent or g/t							Distribution - percent						
Time		µm K80	%	grams	Cu	Fe	S	C	Ag	As	Bi	Cu	Fe	S	C	Ag	As	Bi
12	T03	38	39.8	397.4	14.2	32.8	41.6	0.6	16.3	1615	214	97.8	46.7	46.6	18.7	46.5	47.5	44.9
10	T16	47	38.0	378.0	14.5	32.1	41.9	0.4	17.8	1548	200	95.8	43.9	45.1	14.5	43.7	43.2	41.7
8	T17	67	42.0	418.7	12.9	32.1	43.0	0.5	19.5	1443	192	95.6	49.4	50.3	17.2	52.1	46.3	45.9
7	T18	83	43.6	434.6	12.4	32.1	42.8	0.5	15.8	1426	194	94.9	51.4	52.4	18.5	46.6	46.1	46.6
6	T19	86	43.7	435.7	12.4	31.9	42.1	0.5	23.0	1690	240	94.6	51.3	53.9	17.1	56.7	50.9	52.0
1.5	T27 Flash	729	3.8	37.4	22.8	30.6	38.5	0.3	11.9	1332	132	14.9	4.3	4.2	0.9	3.1	3.6	2.8
11	T38 MF2	40	38.594	382.2	14.2	33.9	42.7	0.4	17.7	1535	195	95.4	45.9	47.9	12.0	48.1	46.0	43.3
Composite 4																		
Grind	Test	PG	Weight		Assay - percent or g/t							Distribution - percent						
Time		µm K80	%	grams	Cu	Fe	S	C	Ag	As	Bi	Cu	Fe	S	C	Ag	As	Bi
12	T04	38	39.3	391.0	14.5	31.0	39.0	0.7	14.3	1958	342	98.0	48.6	48.7	20.5	60.7	48.0	46.9
10.0	T20	44	36.5	363.9	16.2	30.4	39.4	0.6	11.4	2678	461	98.2	44.6	46.4	16.8	70.0	75.4	73.1
8.0	T21	57	37.3	371.2	15.7	31.7	42.0	0.5	10.5	2193	416	97.7	47.4	48.8	12.4	69.0	74.0	73.1
7	T22	68	37.3	371.5	15.7	32.3	41.1	0.5	11.2	2302	416	96.9	48.6	48.2	14.0	73.6	78.1	77.1
6	T23	90	37.8	375.5	15.2	31.2	41.2	0.5	11.0	2348	418	94.2	47.8	51.0	15.7	51.0	48.8	50.0
1.5	T28 Flash	825	8.13	80.00	23.8	30.2	38.5	0.4	9.5	1718	259	31.0	10.3	10.2	2.6	9.7	8.5	7.6
11	T38 MF2	39	32.622	325.6	18.7	32.4	40.9	0.4	9.1	1780	324	96.9	41.1	42.9	10.9	39.9	38.0	39.1

Table 13.24 Grind Size Rougher Tests

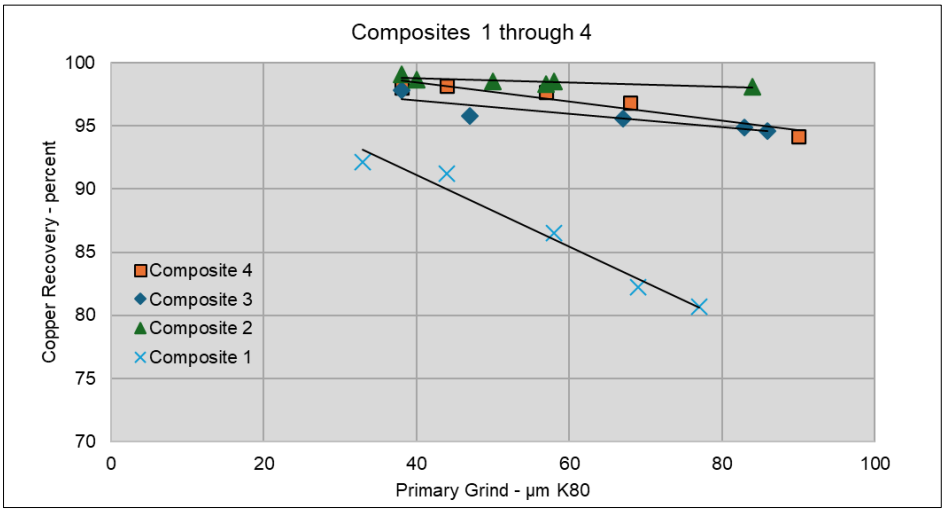


Figure 13.16 Copper recovery vs Grind Size for 4 composites

The performance of each test is illustrated in Figure 13.17 taken from the BML report.

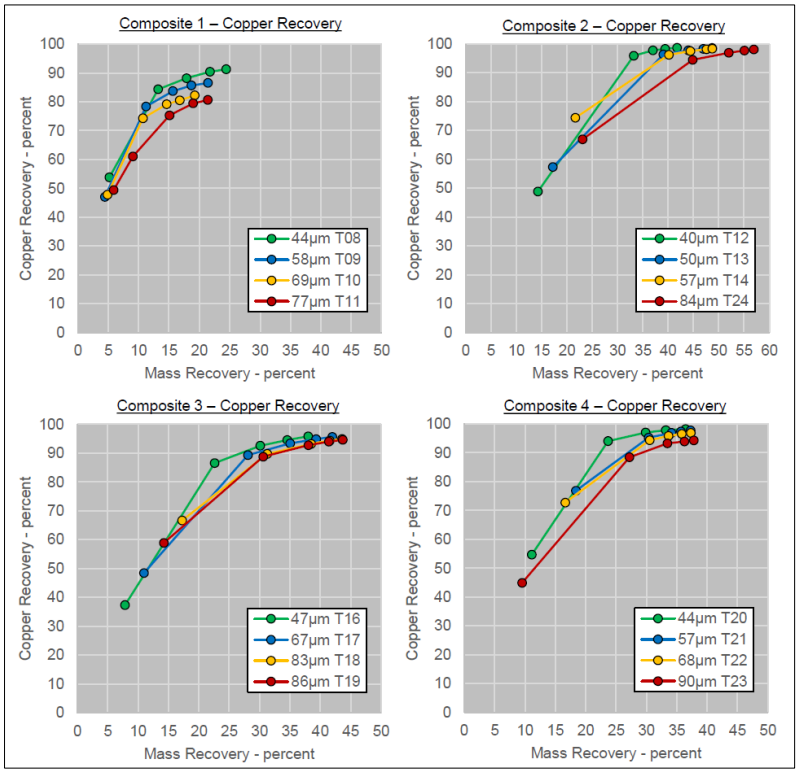


Figure 13.17 Grind Size Optimisation – Rougher Stage

13.4.2 MF2 Rougher Testing – Alternative Flowsheet

As part of the current PFS update several alternative flotation flowsheet options were examined. The flowsheet targeted a coarse primary grind P_{80} 250 µm with a primary copper rougher stage followed by regrinding of the rougher tailings to the previous primary grind target P_{80} 39 µm. The objective of this flowsheet was to reduce the amount of mass requiring the fine primary grind which was used in the standard flowsheet. The total rougher residence time was 15 minutes for the standard flowsheet compared to 14 minutes for the combined coarse and fine roughers using the MF2 flowsheet. The flowsheet is detailed in Figure 13.18 and the reagent additions are shown in Table 13.27.

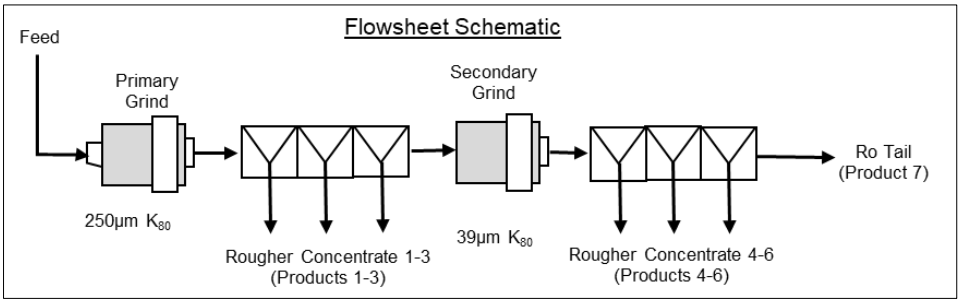


Figure 13.18 MF2 Flowsheet Option

Results for the MF2 flowsheet on each composite are detailed in Table 13.25 and Table 13.26.

Comp	Test	Product	Weight	Assay - percent or g/t							Distribution - percent						
			%	Cu	Fe	S	C	Ag	As	Bi	Cu	Fe	S	C	Ag	As	Bi
1	R-35	Products 1 to 3	5.8	16.7	31.4	40.8	0.2	23.4	7298	649	37.0	7.2	7.4	4.1	6.6	21.7	14.6
		Products 1 to 6	21.2	11.0	31.6	40.8	0.3	29.4	4893	525	88.9	26.7	26.8	20.0	30.0	53.1	43.2
2	R-36	Products 1 to 3	23.4	20.5	32.9	41.0	0.3	7.5	596	149	59.1	28.2	30.1	4.5	24.3	14.3	15.4
		Products 1 to 6	44.7	17.9	33.1	40.7	0.5	8.7	880	222	98.2	54.1	56.9	13.0	54.1	40.3	43.8
3	R-37	Products 1 to 3	13.2	17.4	33.8	43.1	0.2	13.4	1240	142	39.9	15.7	16.5	2.7	12.4	12.7	10.8
		Products 1 to 6	38.6	14.2	33.9	42.7	0.4	17.7	1535	195	95.4	45.9	47.9	12.0	48.1	46.0	43.3
4	R-38	Products 1 to 3	10.9	22.7	32.1	40.0	0.3	8.2	1633	278	39.4	13.6	14.0	2.4	12.1	11.7	11.2
		Products 1 to 6	32.6	18.7	32.4	40.9	0.4	9.1	1780	324	96.9	41.1	42.9	10.9	39.9	38.0	39.1

Table 13.25 Coarse Flotation (1-3) and Regrind Rougher Flotation (1-6)

Comp	Test	Product	Weight	Assay - percent or g/t							Distribution - percent						
			%	Cu	Fe	S	C	Ag	As	Bi	Cu	Fe	S	C	Ag	As	Bi
1	R-35	Products 1 to 3	5.8	16.7	31.4	40.8	0.2	23.4	7298	649	37.0	7.2	7.4	4.1	6.6	21.7	14.6
		Products 4 to 6	15.4	8.8	31.7	40.8	0.3	31.6	3986	479	51.8	19.4	19.5	15.9	23.5	31.4	28.6
2	R-36	Products 1 to 3	23.4	20.5	32.9	41.0	0.3	7.5	596	149	59.1	28.2	30.1	4.5	24.3	14.3	15.4
		Products 4 to 6	21.2	15.0	33.4	40.3	0.7	10.1	1194	303	39.2	25.9	26.8	8.6	29.7	26.0	28.4
3	R-37	Products 1 to 3	13.2	17.4	33.8	43.1	0.2	13.4	1240	142	39.9	15.7	16.5	2.7	12.4	12.7	10.8
		Products 4 to 6	25.4	12.6	33.9	42.5	0.4	20.0	1689	223	55.5	30.2	31.4	9.2	35.7	33.3	32.5
4	R-38	Products 1 to 3	10.9	22.7	32.1	40.0	0.3	8.2	1633	278	39.4	13.6	14.0	2.4	12.1	11.7	11.2
		Products 4 to 6	21.7	16.6	32.5	41.3	0.5	9.5	1853	346	57.5	27.5	28.9	8.5	27.8	26.4	27.8

Table 13.26 Stage 1 Rougher (1-3) and Stage 2 Rougher (4-6)

Noting that Composite 1 is 100% JL UCZ and that composite 2 was 100% JL LCZ, it is interesting to note that Composite 2 100% JL LCZ returned 59% copper recovery with 20.5% copper rougher grade, while Composite1 100% JL UCZ produced 27% Cu recovery at 16.7% Cu rougher grade. This indicated that the lower copper zone mineral grain size is more open to a coarser primary flotation response. The blended Composites 3 and 4 behaved similar to each other with increase mass pull and grade likely to be associated with the 60% blend of JL LCZ ore.

Following both the coarse and fine rougher stages, the average combined rougher recoveries using the MF2 flowsheet were about 1 percent lower in both mass and copper recoveries at an average total mass recovery of about 34 percent and an average total copper recovery of about 95 percent. Results for the MFS floats are presented in Figure 13.19.

There was a reduced carbon recovery recorded using the MF2 flowsheet compared to the standard flowsheet. This may have been a result of the overall increase in depressant dosage added because of the additional grinding stage. The same dosage of depressant was added to the primary grind, and an additional dosage was added to the secondary grind.

Reagent additions for the MF2 flowsheet – Average to test work

Stage	pH	Lime g/t	MSP g/t	Dextrin g/t	3477 g/t
Primary Grind	4.9 to 6.1	1000	400	320	
Aeration	9.5	1410-2960			
Roughers 1-3	9.5	290-540			50-60
Secondary Grind	8.4-9.3	500	200	150	
Rougher 4-6	9.5	290-490			45-60

Table 13.27 MF2 Reagent Scheme

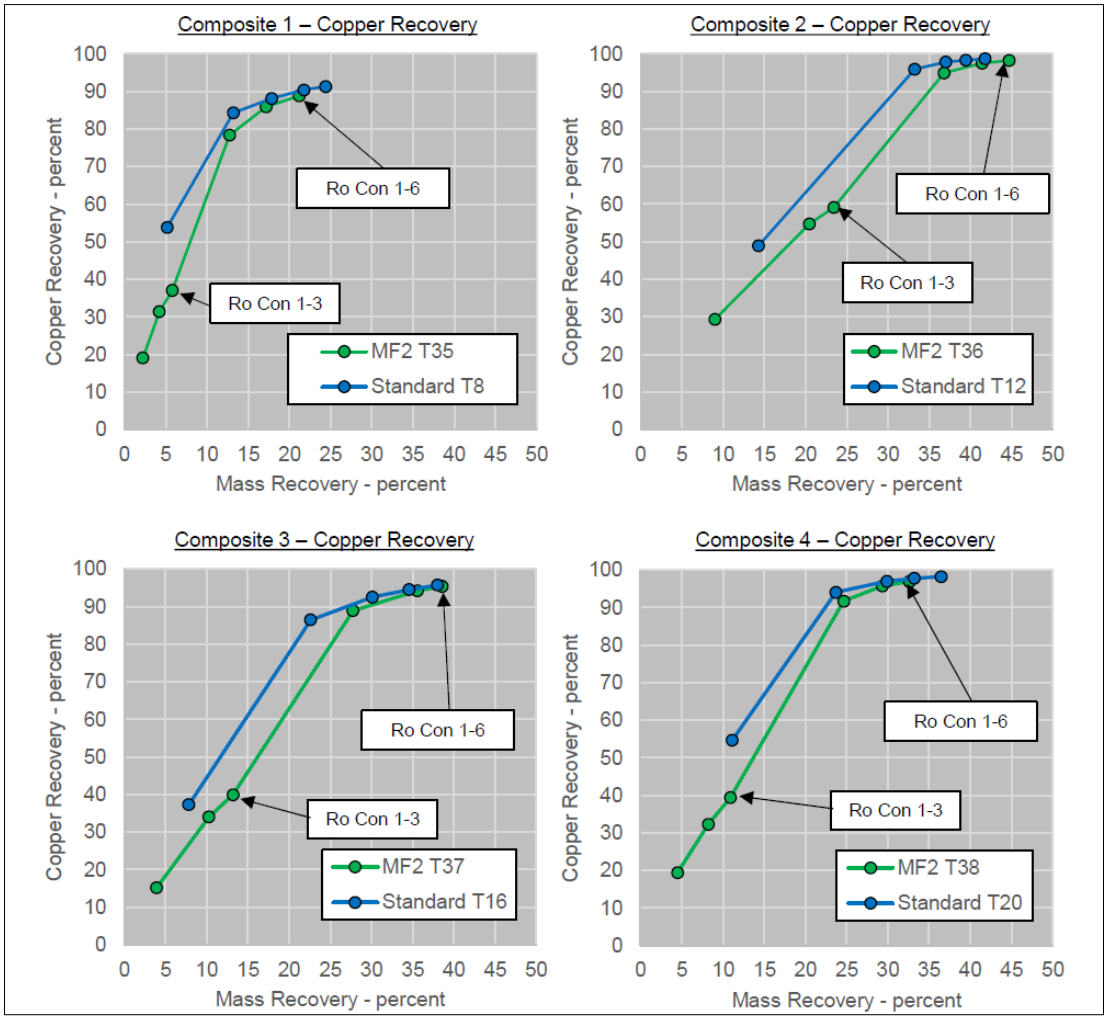


Figure 13.19 MF2 Copper Recovery Results

13.4.3 Flash Rougher Flotation – Alternative Flowsheet

A test on each of the composites 1 through 4 was completed to assess the performance of a flash flotation stage. A short primary grind was utilised in the tests which ranged significantly from P_{80} of 359 µm to 825 µm across the composites. The same depressant dosages were utilized in the primary grinds; the flotation pH target was the same at 9.5, and a 20 g/tonne 3477 dosage was added as collector. The flowsheet is shown in Figure 13.20 and the results are summarised in Table 13.28.

Between 15 and 31 % of the copper was recovered in this stage at mass recoveries of 3 to 8 %, and at copper grades of between 18.5 and 24.2 %. The highest copper recovery of 31 % was recorded from Composite 4 at the 825 µm P_{80} primary grind sizing at a flash concentrate grade of about 23.8 % copper.

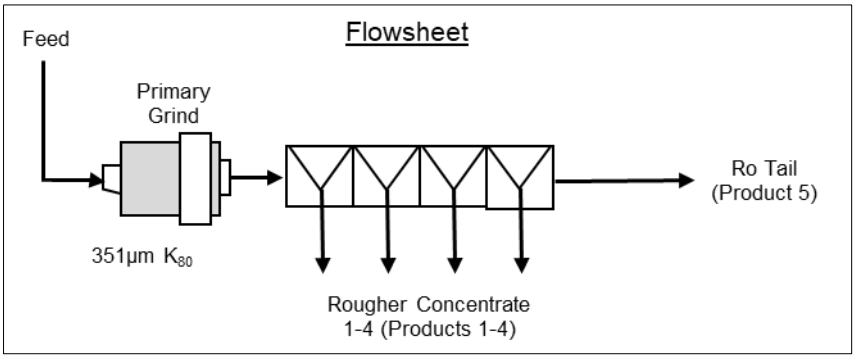


Figure 13.20 Flash Rougher Flowsheet

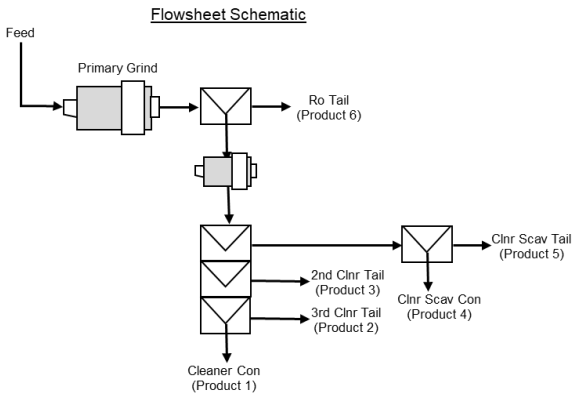
Comp	Test	PG	Weight		Assay - percent or g/t						Distribution - percent					
		µm K80	%	grams	Cu	Fe	S	Ag	As	Bi	Cu	Fe	S	Ag	As	Bi
1	T25 Flash	409	2.8	27.9	18.5	29.5	38.9	19.4	7524	637	19.0	3.2	3.2	2.6	10.0	6.2
2	T26 Flash	351	6.5	64.8	24.2	31.7	40.5	8.2	560	147	19.9	7.7	7.6	7.8	3.5	4.3
3	T27 Flash	729	3.8	37.4	22.8	30.6	38.5	11.9	1332	132	14.9	4.3	4.2	3.1	3.6	2.8
4	T28 Flash	825	8.13	80.00	23.8	30.2	38.5	9.5	1718	259	31.0	10.3	10.2	9.7	8.5	7.6

Table 13.28 Flash Rougher Results

13.5 Cleaner Circuit

13.5.1 Conventional Circuit (2020 FS Basis)

Initial cleaner testing followed the same principle developed in the previous feasibility study with the rougher concentrate regrind to a low target of 10-15 µm P_{80}. This flowsheet included three stages of direct cleaning with one stage of cleaner scavenger. The flowsheet and the reagent conditions are shown in Figure 13.21. The test results are summarised in Table 13.29.



Comp	Test	Conditions
Comp 3	Cl-29	Repeat T03 targeting 18µm regrind 14mm K80
Comp 3	Cl-30	Repeat T03 targeting 25µm regrind 18mm K80
Comp 4	Cl-32	Repeat T07 targeting 18µm regrind 15mm K80
Comp 4	Cl-33	Repeat T07 targeting 25µm regrind 21mm K80
Comp 1	Cl-39	Repeat T01 with half MSP and Dextrin 11µm K80
Comp 2	Cl-40	Repeat T02 with half MSP and Dextrin 12.5mm K80
Comp 3	Cl-41	Repeat T03 with half MSP and Dextrin 11 mm K80
Comp 4	Cl-42	Repeat T04 with half MSP and Dextrin 12 mm K80

Figure 13.21 Flowsheet and reagent condition setup

Comp	Test	Product	Weight	Assay - percent or g/t							Distribution - percent						
			%	Cu	Fe	S	C	Ag	As	Bi	Cu	Fe	S	C	Ag	As	Bi
		Concs 1 to 4	20.0	27.3	32.6	39.1	0.1	13.5	931	138	93.7	23.5	22.5	2.3	19.6	15.2	17.1
Comp 3	CI-29	Rougher Conc	40.5	14.0	33.1	39.8	0.6	17.6	1445	201	97.7	48.3	46.5	20.8	51.7	47.8	50.4
		Concs 1 to 4	28.0	20.14	33.6	41.2	0.3	15.9	1501	189	96.5	32.4	33.6	7.7	34.0	33.1	32.1
Comp 3	CI-30	Rougher Conc	43.7	13.11	33.0	39.5	0.7	16.2	1600	208	98.3	49.8	50.4	25.0	54.3	55.2	55.2
		Concs 1 to 4	21.7	26.79	31.6	37.8	0.3	7.8	1264	240	94.9	25.7	26.9	4.3	25.9	19.3	21.5
Comp 4	CI-32	Rougher Conc	31.9	18.82	30.7	36.2	0.7	8.1	1665	294	98.0	36.8	37.9	17.1	39.6	37.4	38.7
		Concs 1 to 4	22.4	26.07	32.4	40.1	0.2	8.3	1458	271	96.7	27.2	28.9	3.7	28.4	23.4	26.3
Comp 4	CI-33	Rougher Conc	33.9	17.54	31.8	38.4	0.7	8.5	1708	299	98.4	40.2	41.8	17.9	43.7	41.5	44.0
		Concs 1 to 4	11.8	17.83	32.5	43.5	0.3	23.6	3670	420	82.2	15.3	15.7	10.4	13.9	24.1	20.8
Comp 1	CI-39	Rougher Conc	24.6	9.24	31.3	41.8	0.3	23.3	3727	392	88.8	30.8	31.3	27.3	28.6	50.9	40.6
		Concs 1 to 4	25.5	28.77	30.1	38.1	0.3	6.4	395	108	93.1	28.3	29.2	3.7	26.3	11.8	13.7
Comp 2	CI-40	Rougher Conc	41.8	18.61	31.2	39.3	0.7	7.0	769	178	98.8	48.0	49.3	16.9	47.4	37.5	37.3
		Concs 1 to 7	15.1	31.25	30.5	39.5	0.1	9.2	423	73	83.0	16.2	16.7	1.1	11.3	5.8	7.6
Comp 3	CI-41	Rougher Conc	36.8	14.97	31.6	40.7	0.6	14.6	1424	182	96.7	40.9	42.0	18.3	43.7	47.1	45.6
		Concs 1 to 8	17.6	30.77	29.7	38.3	0.2	7.7	704	149	88.5	20.7	21.3	2.4	20.8	9.5	11.6
Comp 4	CI-42	Rougher Conc	42.3	14.18	28.9	37.8	0.8	8.4	1665	284	98.1	48.7	50.4	27.0	54.1	53.8	53.2

Table 13.29 Cleaner Flotation Test Results

As with rougher testing, monosodium phosphate (MSP) and dextrin were added as depressants in both the primary grind and regrind, with the dithiophosphate collector Aero 3477 used as the collector. The rougher and cleaner circuits are both completed at pH 9.5. In this program, testing was completed varying regrind and depressant dosages; two coarser regrind targets were tested using Composites 3 and 4; and a 50 % lower depressant dosage was tested using Composites 1 through 4.

Tests using the lower depressant dosage measured poorer copper recoveries (ref Figure 13.22) compared to tests with the baseline dosage; this was more apparent for the samples containing JL LCZ material. If the depressant acts by blocking the organic carbon or other absorbative non-sulphide gangue, then the reduced depressant dosage could be reducing collector effectiveness by allowing the problematic material to adsorb the reagent.

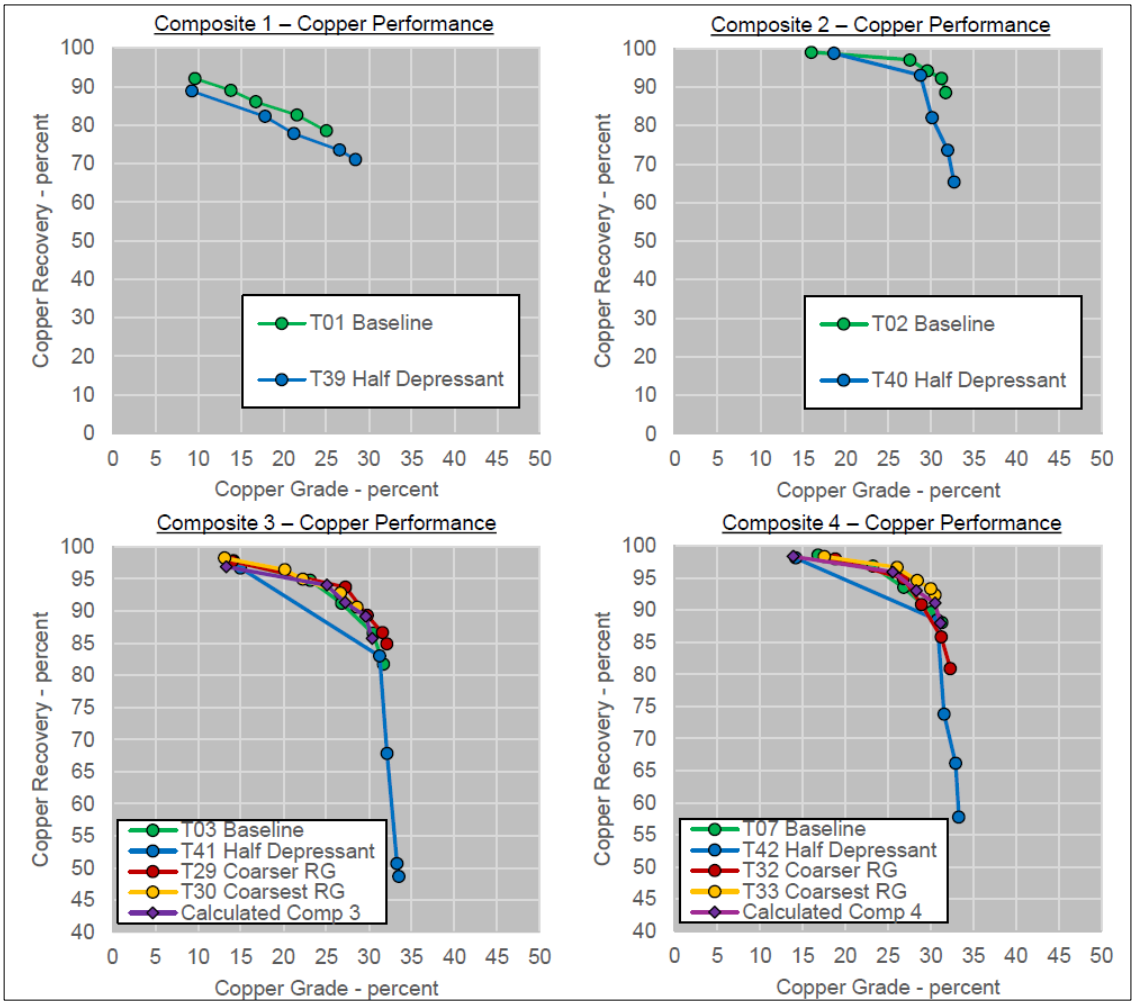


Figure 13.22 Reduced Depressant Results

13.5.2 Alternative Cleaner Circuit Flowsheet

An alternative flowsheet design (Figure 13.23) was investigated using Blends A, B and C looking to move the Jameson Cell from post regrind stage to a pre-regrind option. The pre-cleaner option would target the development of a coarser high-grade concentrate with the cell tails being subsequently reground as per the current flowsheet design. The flowsheet is detailed in Figure 13.23. Following rougher flotation and prior to regrinding, a high-grade 1st cleaner concentrate was recovered and cleaned one subsequent time; the 1st and 2nd high grade cleaner tailings were reground and cleaned through a three-stage low grade dilution cleaner circuit.

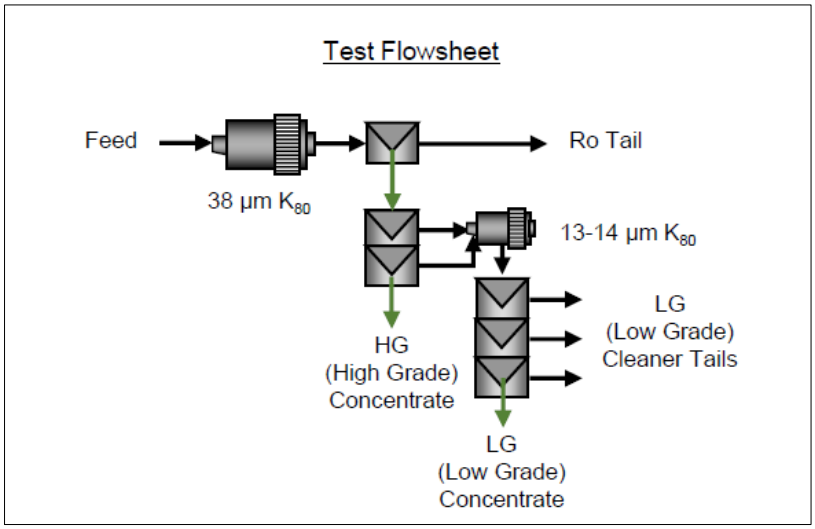


Figure 13.23 Alternative Cleaner Flowsheet

BL1694-43 Blend A (40% UCZ / 60% LCZ)

Cumulative Balance

Product	Weight	Assay - percent or g/t							Distribution - percent						
	%	Cu	Fe	S	C	Ag	As	Bi	Cu	Fe	S	C	Ag	As	Bi
HG Cln Conc	20.5	24.9	31.6	40.0	0.2	12.0	1680	235	88.4	24.5	24.9	3.0	20.5	25.4	19.7
HG Cln Conc + LG Conc	27.2	20.4	33.2	42.5	0.2	15.0	1943	276	96.0	34.1	35.1	4.9	33.9	39.0	30.7
HG+LG+3 Cln Tail (CT)	28.3	19.6	33.2	42.5	0.3	15.2	1925	277	96.2	35.5	36.6	6.0	35.8	40.2	32.1
HG+LG+3CT+2CT	30.7	18.21	33.1	42.4	0.3	15.5	1946	286	96.7	38.4	39.6	8.3	39.5	44.1	35.9
HG+LG+3CT+2CT+1CT	41.9	13.52	31.5	39.8	0.6	14.8	1843	290	97.9	49.9	50.6	21.4	51.6	56.9	49.6
Rougher Tail	58.1	0.21	22.8	28.0	1.7	10.0	1006	212	2.1	50.1	49.4	78.6	48.4	43.1	50.4
Recalc. Feed	100.0	5.78	26.5	32.9	1.3	12.0	1356	245	100	100	100	100	100	100	100

Table 13.30 Alt Cleaner Flowsheet Blend A test

High grade concentrate (HG Cln Conc, Table 13.30) produced a 24.9% Cu concentrate grade at 88% Cu recovery. Combined with the 3rd cleaner concentrate from the reground tails stream achieved 96% Cu recovery with dilution of the combined concentrate back to 20.4%. Arsenic has remained low <2000 ppm with a combined recovery of 39% As. Overall the change to the flowsheet to include the coarse pre-cleaner stage looks beneficial on Blend A.

BL1694-44 Blend B (40% UCZ / 60% LCZ)

Cumulative Balance

Product	Weight	Assay - percent or g/t							Distribution - percent						
	%	Cu	Fe	S	C	Ag	As	Bi	Cu	Fe	S	C	Ag	As	Bi
HG Cln Conc	16.9	21.2	32.0	41.5	0.3	15.0	2085	251	91.3	26.6	28.4	1.4	24.0	35.8	24.3
HG Cln Conc + LG Conc	21.5	17.6	33.2	43.5	0.3	16.9	2142	272	95.9	35.0	37.7	2.1	34.4	46.6	33.3
HG+LG+3 Cln Tail (CT)	22.4	16.9	33.3	43.7	0.3	17.0	2134	275	96.2	36.5	39.5	2.3	36.1	48.4	35.1
HG+LG+3CT+2CT	24.3	15.61	32.7	42.6	0.5	16.9	2067	273	96.7	39.0	41.9	3.9	38.9	51.0	38.0
HG+LG+3CT+2CT+1CT	34.3	11.17	29.3	36.9	1.2	15.4	1754	257	97.7	49.4	51.2	13.3	50.2	61.1	50.4
Rougher Tail	65.7	0.14	15.7	18.4	4.1	8.0	585	132	2.3	50.6	48.8	86.7	49.8	38.9	49.6
Recalc. Feed	100.0	3.93	20.4	24.7	3.1	10.6	987	175	100	100	100	100	100	100	100

Table 13.31 Alt Cleaner Flowsheet Blend B test

The Blend B sample (Table 13.31) also responded well to the alternative flowsheet option with a combined recovery of 95.9%, concentrate grade suffered with the low-grade concentrate diluting the grade from 21.2 to 17.6% Cu and increasing the arsenic grade to 2142 ppm As.

BL1694-45 Blend C (40 % UCZ / 60 % LCZ)

Cumulative Balance

Product	Weight	Assay - percent or g/t							Distribution - percent						
	%	Cu	Fe	S	C	Ag	As	Bi	Cu	Fe	S	C	Ag	As	Bi
HG Cln Conc	16.1	22.8	31.6	41.0	0.3	15.0	2219	288	85.1	19.2	18.8	4.1	17.2	27.4	20.9
HG Cln Conc + LG Conc	21.8	18.6	32.8	43.6	0.3	17.6	2244	301	94.0	27.0	27.1	6.1	27.3	37.6	29.6
HG+LG+3 Cln Tail (CT)	22.6	18.0	32.9	43.7	0.3	17.8	2229	301	94.4	28.2	28.3	6.8	28.6	38.8	30.8
HG+LG+3CT+2CT	24.5	16.73	32.7	43.4	0.4	17.8	2190	301	95.1	30.3	30.3	8.8	31.0	41.2	33.2
HG+LG+3CT+2CT+1CT	37.1	11.24	31.4	40.9	0.6	17.5	1959	292	96.9	44.1	43.4	23.2	46.3	55.9	48.9
Rougher Tail	62.9	0.21	23.5	31.5	1.3	12.0	912	180	3.1	55.9	56.6	76.8	53.7	44.1	51.1
Recalc. Feed	100.0	4.31	26.4	35.0	1.0	14.1	1301	222	100	100	100	100	100	100	100

Table 13.32 Alt Cleaner Flowsheet Blend C test

Blend C results (Table 13.32) showed a good grade on the high-grade concentrate at lower 85% recovery, however combined copper recovery was still high at 94% Cu. Again, grade is the issue with lower combined final concentrate of 18.6% Cu.

The LG (low grade concentrates measured about 4 to 7 % copper and contained 5 to 9 % of the feed copper. Combined, the two concentrates ranged from about 18 to 20 % copper. The LG concentrates were highly diluted by pyrite, which comprised over three quarters of the low-grade concentrate mass; while the HG concentrates were about a quarter to one third pyrite. In contrast, concentrates produced from the blend Composites 3 and 4 using the baseline conditions were less than one fifth pyrite, based on mineral estimates from sulphur and copper assays.

13.5.3 Jameson Cell Testing

Bench scale dilution cleaner tests were completed as per the Glencore Jameson cell procedure to illustrate the inclusion of a Jameson cell as a cleaner scalper. A primary grind to the target size of 38 µm was applied. A straight kinetic rougher flotation stage was used to develop the required concentrate for cleaning. No regrind stage was used and the flotation was conducted as a dilution cleaner with first stage concentrate feeding the second stage, with second stage feed the third stage. Two tests were performed: one each with Composite 3 and Composite 4. The test results are summarized in Table 13.33 and Table 13.34.

Product	Weight		Assay - percent or g/t							Distribution - percent						
	%	grams	Cu	Fe	S	C	Ag	As	Bi	Cu	Fe	S	C	Ag	As	Bi
Cu Con 3A	8.9	88.7	30.1	30.9	40.1	0.1	8.0	588	98	46.1	9.6	10.3	0.6	5.6	4.2	5.3
Cu Con 3A+3B	15.4	152.7	29.1	31.4	39.8	0.1	8.8	709	111	76.9	16.8	17.6	1.3	10.7	8.8	10.3
Cu Con 3A+3B+3C	18.6	185.2	27.5	32.0	40.5	0.2	10.8	979	138	88.1	20.8	21.6	2.4	15.9	14.7	15.7
Cu Con 3A+3B+3C+3D	23.3	231.4	23.6	33.1	41.8	0.2	13.8	1330	173	94.4	26.9	27.9	4.2	25.5	24.9	24.5
Cu 2nd Cln Conc	25.1	249.8	22.1	33.5	42.2	0.3	14.5	1382	178	95.3	29.4	30.4	5.3	28.9	27.9	27.2
Cu 1st Cln Conc	29.8	296.4	18.9	34.1	42.6	0.3	15.5	1464	186	96.7	35.5	36.4	8.0	36.6	35.1	33.8
Cu Rgh Conc	41.4	411.6	13.8	33.1	40.0	0.6	15.5	1553	202	98.1	47.7	47.6	20.3	50.9	51.7	51.0
Rougher Tail	58.6	582.2	0.19	25.6	31.2	1.7	10.6	1026	137	1.9	52.3	52.4	79.7	49.1	48.3	49.0
Recalc. Feed	100.0	993.8	5.82	28.7	34.9	1.2	12.6	1244	164	100	100	100	100	100	100	100

Table 13.33 Test 31 - Composite 3

Product	Weight		Assay - percent or g/t							Distribution - percent						
	%	grams	Cu	Fe	S	C	Ag	As	Bi	Cu	Fe	S	C	Ag	As	Bi
Cu Con 3A	10.2	101.9	30.3	32.1	38.6	0.1	6.0	764	178	51.6	12.5	12.7	1.0	9.6	5.6	7.8
Cu Con 3A+3B	17.9	178.4	28.3	32.5	38.7	0.2	6.9	1074	226	84.4	22.2	22.3	2.3	19.3	13.8	17.4
Cu Con 3A+3B+3C	21.6	215.5	25.9	33.2	39.4	0.2	7.5	1310	261	93.3	27.4	27.4	3.9	25.6	20.3	24.3
Cu Con 3A+3B+3C+3D	23.7	236.4	24.3	33.6	40.0	0.3	7.9	1436	277	95.9	30.4	30.4	5.4	29.4	24.5	28.3
Cu 2nd Cln Conc	24.9	248.1	23.4	33.6	40.2	0.4	8.1	1529	286	96.9	32.0	32.1	6.8	31.8	27.3	30.6
Cu 1st Cln Conc	27.6	275.0	21.3	33.8	40.4	0.5	8.4	1642	291	97.7	35.6	35.8	9.9	36.4	32.5	34.6
Cu Rgh Conc	32.7	325.5	18.1	33.0	39.4	0.7	8.4	1772	300	98.3	41.2	41.3	16.3	43.0	41.6	42.1
Rougher Tail	67.3	670.3	0.15	22.9	27.2	1.6	5.4	1209	200	1.7	58.8	58.7	83.7	57.0	58.4	57.9
Recalc. Feed	100.0	995.8	6.01	26.2	31.2	1.3	6.4	1393	233	100	100	100	100	100	100	100

Table 13.34 Test 34 -Composite 4

Copper recovery based on the coarse Jameson cell concentrate indicates 94 to 96% of copper recovery is achievable with a higher grade concentrate 23.6 to 24.3% copper reporting in the two test runs. The result helps to validate the use of the Jameson cell as a pre-cleaner on the High Grade option with the tails to be treated by regrinding to further increase mineral liberation. In both Jameson cell tests, arsenic remained low at <1,500 ppm in final concentrate.

13.6 2025 Metallurgical Test Program

Variation composites were shipped to Base Metal Labs (BML) for additional metallurgical testing to verify the previous study outcomes. The test program, BL16941, was conducted to determine the metallurgical response of the Johnny Lee Lower Copper Zone (JL LCZ), and blended JL LCZ and Johnny Lee Upper Copper Zone (JL UCZ) material. This current program has undertaken additional testing including variability testing and other testing to support the updated pre-feasibility study.

There were several objectives for this program:

- Construct twelve composites consisting of JL UCZ and JL LCZ variability, and seven blends of JL UCZ/JL LCZ material and assess the chemical composition.

- Assess the comminution properties of four constructed composite by performing SMC testing, Bond Abrasion Index (Ai) tests and Bond Ball Work Index (BWI).
- Assess the flotation performance of the twelve constructed composites and seven blends of JL UCZ/JL LCZ material using the flotation conditions and reagent scheme developed in past testing programs.
- Conduct a single locked cycle test on the blended JL UCZ/JL LCZ material.

13.6.1 Composite Head Chemical Analysis

The chemical contents of each of the variability samples, along with blends of 100%JL LCZ and 100%JL UCZ and comminution composites, were analyzed using standard analytical techniques. Chemical head assay data is summarized in Table 13.35.

Copper measured between 2.11 % and 2.89 % in the JL UCZ material, 1.88 % and 12.40 % in the JL LCZ materials, 0.81 % and 3.15 % in comminution composites, 6.3 % in 100 % blended JL LCZ material and 2.96 % in 100% blended JL UCZ material. Sulphur content ranged between 11.28 % and 38 %.

Sequential copper assays were also conducted to provide an indication of the copper mineral species. CuOX is copper present in weak acid dissociable copper minerals, typically oxides, while CuCN is copper dissociable in cyanide, consisting of secondary copper minerals and bornite. Low CuOX was measured for most samples indicating most copper is present as sulphides. CuCN copper was slightly higher, indicating a small portion of the copper as secondary copper minerals. UCZ Var 2 was slightly anomalous, with a higher portion of the copper measured as both CuOX and CuCN and may pose challenges in flotation.

Total Organic Carbon (TOC) measured between 0.08 % and 1.90 %across the samples, with the highest value in LCZ VAR 5 and COMM 2. TOC can interfere with flotation due to its natural hydrophobicity, and adsorption of reagents.

Arsenic and antimony measured highest in the UCZ VAR 1, measured at 3485 and 74.1 g/tonne, respectively.

Comp ID	Cu%	Co ppm	Fe%	Au g/t	Ag g/t	As g/t	Bi g/t	Sb g/t	S %	C %	TOC %	CuOX %	CuCN %
UCZ VAR1	2.89	1795	27.2	<0.05	14.15	3485	272.5	74.1	32.3	0.20	0.19	0.12	1.05
UCZ VAR 2	2.11	730	28.4	<0.04	9.00	709	80.2	34.4	33.1	0.30	0.21	0.11	0.45
UCZ VAR 3	2.20	75	18.9	<0.05	17.60	203	9.3	20.3	26.9	0.22	0.19	0.40	1.65
LCZ VAR 1	5.51	725	32.6	0.51	8.05	406	161.5	19.1	36.4	2.04	0.14	0.15	0.80
LCZ VAR 2	4.64	535	31.0	0.33	5.55	234	116.0	10.6	34.4	2.82	1.29	0.13	0.42
LCZ VAR 3	1.88	585	29.1	1.00	5.50	626	120.5	14.1	33.3	1.59	0.45	0.06	0.24
LCZ VAR 4	5.51	730	28.0	0.28	9.40	616	149.6	17.1	32.5	1.64	0.91	0.14	1.00
LCZ VAR 5	8.00	630	34.6	0.52	8.25	293	1958	15.0	38.4	2.42	1.88	0.22	1.05
LCZ VAR 6	4.84	490	27.9	0.11	7.00	475	200.4	14.2	31.5	1.73	0.17	0.12	0.18
LCZ VAR 7	6.50	160	19.8	0.51	2.40	266	121.0	10.8	22.3	0.31	0.08	0.09	0.19

Comp ID	Cu%	Co ppm	Fe%	Au g/t	Ag g/t	As g/t	Bi g/t	Sb g/t	S %	C %	TOC %	CuOX %	CuCN %
LCZ VAR 8	4.84	205	20.8	0.11	6.80	248	152.2	8.4	21.5	4.04	0.27	0.09	0.11
LCZ VAR 9	12.40	1565	20.1	0.11	4.00	2197	352.0	12.8	22.4	0.22	0.11	0.13	0.29
100% JL LCZ	6.30	420	25.0	0.43	6.20	443	166.2	19.0	29.5	2.19	0.44	0.13	0.43
100% JL UCZ	2.96	1005	22.1	,0.01	12.20	1885	144.2	45.0	31.3	0.33	0.21	0.11	0.33
COMM 1	2.75	450	24.6	0.48	6.40	431	139.6	22.6	26.5	03.26	0.38	0.12	0.15
COMM 2	1.64	370	21.8	0.42	6.40	419	156.0	17.8	24.4	2.44	1.90	0.08	0.14
COMM 3	3.15	160	21.1	0.32	2.00	283	72.8	16.8	23.7	0.35	0.11	0.09	0.32
COMM 4	0.81	145	11.9	0.06	3.20	169	82.4	11.0	11.3	4.31	0.17	0.02	0.05

Table 13.35 Head Assays

13.6.2 2025 Comminution Testing

SMC and JK parameters were determined from the current comminution test samples COMM 1 through 4. All 4 samples were taken from JL LCZ core with the aim of testing variability across the deposit. The summary is detailed in Table 13.36 and Table 13.37.

Sample	Dwi	Mia	Mic	A	b	SG	ta	SCSE
	Kmh/m3	kWh/t	kWh/t					kWh/t
COMM 1	5.58	11.8	4.2	62.9	1.1	3.86	0.46	8.02
COMM 2	5.85	14.4	5.3	73.5	0.76	3.28	0.44	9.27
COMM 3	6.66	10.7	5.6	77.5	0.70	3.6	0.39	9.27
COMM 4	5.00	9.5	4.9	68.8	0.89	3.06	0.52	8.72

Table 13.36 SMC Test results

Sample	Bond Ball Mill Work Index					
	CSS	F80	P80	Gpr	Bwi	Ai
	µm	µm	µm		kWh/t	
COMM 1	75	2312	55	1.07	15.1	0.13
COMM 2	75	2204	56	0.97	16.6	0.21
COMM 3	75	2387	59	1.00	16.5	0.57
COMM 4	75	2422	57	1.12	14.8	0.13

Table 13.37 Bond Mill Work Index

The SMC Axb values report in Table 13.36 indicated moderate to moderately soft material for SAG mill impact breakage.

The Bond ball mill work index (BWI) tests were conducted on the samples with a closing screen size of 75 µm, for a P_{80} ~58 µm. The generated work index, reported in Table 13.37, of 14.8 to 16.6 kWh/t are considered moderate.

13.6.3 *2025 Cleaner Tests on Variability Composites*

A single cleaner test on each variability composite and blended material, two optimization tests on selected variability sample, and single locked cycle test on 50%JL LCZ / 50%JL UCZ blend were conducted. All testing was completed using the test parameters developed during the BL1694 test program. The flowsheet included a HG circuit to recover final concentrate grade material prior to regrinding, and subsequent LG circuit processing. All testing was completed using the test parameters developed during the BL1694 test program. The flowsheet included a HG circuit to recover final concentrate grade material prior to regrinding, and subsequent LG circuit processing and is shown in Figure 13.24. Cleaner Variability flotation tests results are summarised in Table 13.38.

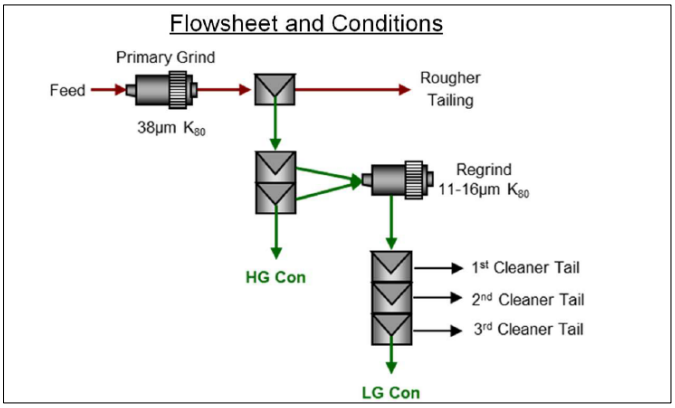


Figure 13.24 Cleaner Test Flowsheet

Test Id	Sample ID	Regrind	Weight	Assay % or g/t			Distribution - %		
		P80 µm	%	Cu	Fe	S	Cu	Fe	S
1	UCZ VAR 1	11	12.0	18.9	32.5	40.1	80.8	15.4	15.5
2	UCZ VAR 2	13	9.4	17.8	31.9	38.8	83.7	11.2	11.5
3	UCZ VAR 3	16	22.7	6.1	39.8	40.0	49.4	45.7	36.3
13	UCZ VAR 3	12	14.5	6.5	28.7	62.3	45.1	28.3	36.3
4	LCZ VAR 1	20	28.4	18.1	38.4	42.2	96.4	31.8	33.5
14	LCZ VAR 1	11	16.9	30.9	29.4	35.1	94.2	18.4	17.5
5	LCZ VAR 2	14	13.5	33.6	31.8	36.1	92.3	13.6	14.4
6	LCZ VAR 3	16	6.9	23.9	34.6	39.2	94.5	7.6	8.3
7	LCZ VAR 4	22	19.0	27.8	33.8	38.0	96.0	21.3	22.4
8	LCZ VAR 5	14	22.6	31.6	33.0	37.1	90.0	21.8	22.3
9	LCZ CAR 6	15	19.8	25.1	38.8	38.4	95.0	25.5	25.0

Test Id	Sample ID	Regrind	Weight	Assay % or g/t			Distribution - %		
		P80 µm	%	Cu	Fe	S	Cu	Fe	S
10	LCZ VAR 7	23	24.2	27.3	35.9	35.5	95.8	39.5	39.5
11	LCZ VAR 8	13	17.3	28.7	37.9	37.6	96.8	29.3	31.1
12	LCZ VAR 9	13	41.4	33.1	34.5	34.6	93.5	64.2	64.4

Table 13.38 Cleaner Tests – Variability Sample

The JL LCZ variability samples resulted in higher copper recoveries and concentrate grades compared to JL UCZ (Figure 13.25) composites which has followed previous flotation testing assessment. Combined HG and LG concentrate copper recovery measured between 90 and 97.4 % in JL LCZ samples with an average copper grade of 28 %, and between 83.7 and 59.4 % in JL UCZ with an average copper grade of 14.3 %.

The poor flotation response recorded for UCZ VAR 3 (T3), was reassess with additional dosage of depressants, MSP and dextrin while reducing 3477 collector dosage. This modified test (T13) resulted in no significant change in performance compared to initial test, (T3), and still resulted in low concentrate grade of 6.5% copper and low recovery measuring 45 % copper.

The poor flotation performance of UCZ VAR 3 may be attributed to severe copper sulphide locking within pyrite, as shown in Figure 13.25, as well as the presence of weak acid dissociable copper and cyanide soluble copper, and indication of copper oxide minerals and secondary copper minerals.

In LCZ VAR 1, a reduction in regrind size and collector dosage (3477) significantly improved concentrate grade. The combined HG and LG concentrate grade increased from 18.1 percent in (T4) to 30.9 percent in (T14), indicating the benefit of reagent and grind optimization.

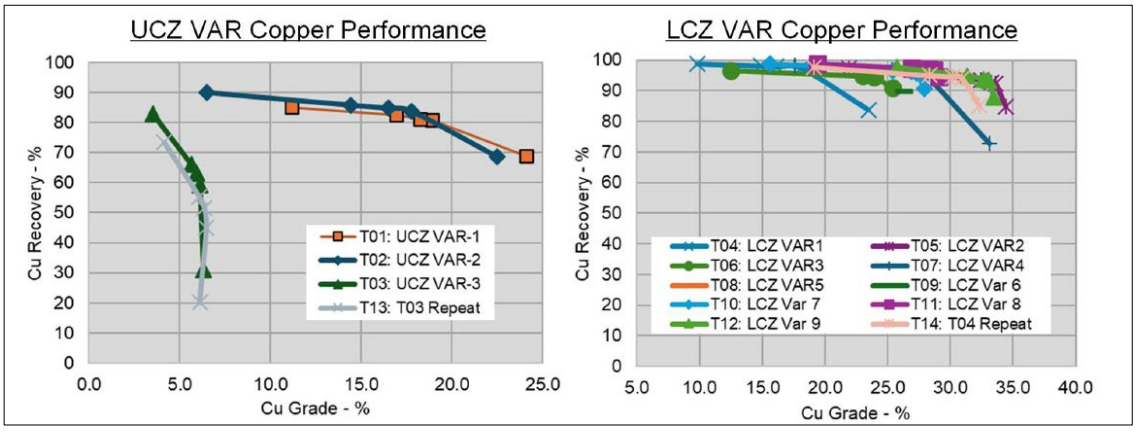


Figure 13.25 Copper Recovery for Variability Samples (JL UCZ and JL LCZ)

A series of blended cleaner testing was also conducted with the results recorded in Table 13.39 and shown in Figure 13.26.

Test Id	Sample ID	Regrind	Weight	Assay % or g/t			Distribution - %		
		P80 µm	%	Cu	Fe	S	Cu	Fe	S
15	100% JL LCZ	16	16.9	28.6	31.9	36.5	94.3	21.8	21.7
18	80% JL LCZ / 20% JL UCZ	11	19.3	28.9	30.5	35.6	96.5	22.3	23.2
Calc			15.6	27.6	31.8	36.8	92.1	19.9	19.8
19	60% JL LCZ / 40% JL UCZ	11	16.9	27.2	30.7	35.2	94.0	20.0	20.0
Calc			14.3	26.5	31.7	37.1	89.5	19.0	18.0
17	50% JL LCZ / 50% JL UCZ	13	15.1	25.6	31.9	37.3	83.3	18.3	18.7
Calc			13.7	25.8	31.7	37.3	88.0	17.1	17.1
20	40% JL LCZ / 60% JL UCZ	12	15.8	24.2	31.5	36.9	91.2	18.7	19.0
Calc			13.0	25.1	31.6	37.5	86.3	16.2	16.2
21	20% JL LCZ / 80% JL UCZ	14	13.0	25.3	30.2	37.4	88.7	15.2	15.7
Calc			11.7	23.4	31.4	37.9	82.2	14.4	14.5
16	LCZ VAR 5	12	10.5	21.3	31.2	38.5	76.9	12.6	12.9

Table 13.39 Combined HG and LG Concentrates on Blend

A decreasing copper performance was observed with increasing JL UCZ material proportion in the blends (as expected, Figure 13.26). Copper recovery measured the highest in 100% JL LCZ at 94.3 % at a combined copper concentrate grade of 28.6 % while 100% JL UCZ measured the lowest at 76.9 % recovery and combined copper concentrate grade of 21.3 %. The measured performance of the blend composites was generally in line with the calculated results from the individual samples used to make the blends, indicating no poisoning effect from the JL UCZ material on the JL LCZ material.

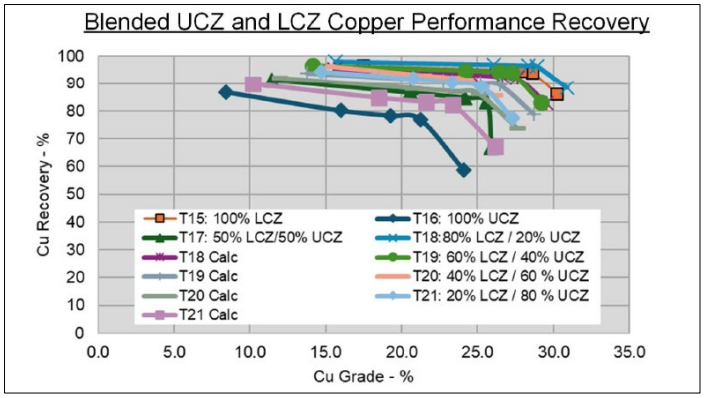


Figure 13.26 Copper Recovery for various blends

13.6.4 2025 Locked Cycle Test

A single locked cycle test was completed using 50%JL LCZ / 50%JL UCZ blend and following the established cleaner test conditions. A summary of conditions is presented in Figure 13.27 and the results are summarised in Table 13.40. The locked cycle testing achieved a copper recovery of 86.6% compared

to 81.3 % in open circuit test, producing a final concentrate with 24.5% Cu in HG concentrate and 13.7% Cu in LG concentrate. The combined copper concentrate grade was 22.8% Cu.

Test Id	Product	Weight	Assay % or g/t			Distribution - %		
		%	Cu	Fe	S	Cu	Fe	S
LCT-22	Feed	100	4.43	22.9	31.1	100	100	100
	HG Conc	14.5	24.5	27.7	36.6	80	17.5	17.0
	LG Conc	2.8	13.7	28.9	42.6	8.62	3.51	3.73
	LG 1CT	20.1	0.94	22.7	33.9	4.26	21.6	21.9
	Ro Tails	62.7	0.50	21.0	28.4	7.08	57.4	57.3

Table 13.40 LCT Summary

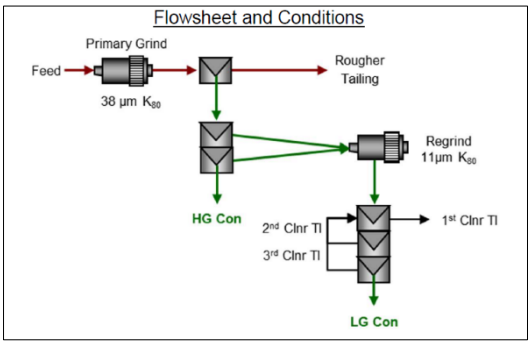


Figure 13.27 LCT Flowsheet

13.7 Conclusions

The following conclusions are drawn from the metallurgical testing:

- The current PFS metallurgical test program has been completed on material from the Black Butte project. This test program utilized test conditions developed in previous feasibility study and applied them to a deeper zone and to blends of the shallower and deeper zone material. Alternative flowsheet configurations were also tested.
- The deposit is a massive sulphide with over half the measured mass being sulphide, primarily pyrite. The copper was present primarily as chalcopyrite with minor amounts measured as present in the arsenic bearing copper sulphide mineral tennantite in the JL UCZ material, but not for the JL LCZ material. Arsenic in the feed was not accounted for by the tennantite measured by the QEMSCAN Bulk Mineral Analyses (BMA); past microprobe analyses measured a sub-microscopic content of arsenic, along with other elements (Co, Bi, and others), within the pyrite. Arsenic not measured by QEMSCAN might be attributable to being sub-microscopic within pyrite and therefore would be expected to report to the same streams as pyrite in flotation.

- The JL UCZ material, with higher measured arsenic within copper sulphides, would see a greater recovery of arsenic to the copper concentrate (as was previously noted in the feasibility study testing); this is what was measured in testing. Further assessment of the sub-microscopic content of arsenic within the pyrite, along with its variability, and the measurable amount within tennantite, could allow for greater predictability of arsenic content in concentrates based on feed measurements.
- In general, the JL LCZ material measured higher flotation recoveries of copper compared to the JL UCZ material. Using the JL UCZ (Composite 1), the standard baseline conditions produced a copper concentrate measuring 25 % copper at an open circuit copper recovery of about 79 %; for the JL LCZ (Composite 2). Using the same conditions, a copper concentrate measuring about 32 % copper was produced at an open circuit copper recovery of about 89 %.
- Optimization of the flowsheet and reagent additions were examined. In general the alternative flowsheet option describe in section 13.5.3 maintained the plant primary fine grind at P80 of 38 µm and allowed for the Jameson cell to be moved from post regrind to before and provide a flotation stage at the coarser concentrate grind size. Tailings from the HG concentrate stage would then be reground for the 3 stages of cleaning producing a LG concentrate to be blended with the high grade. The separation of the circuit for the plant design is important as it will not overgrind free floating and clean chalcopyrite and will also recover the tennantite regardless. This leaves the lower grade binary sulphides to be ground to P80 10-14 µm for further liberation and flotation to better eject the pyrite and the arsenic not associated with tennantite.

The viability testing conducted in 2025 indicated:

- Comminution tests including SMC, Bond Ball Mill Work Index, and Abrasion test were completed on four comminution composites representing COMM 1 through 4. The calculated Axb values ranged from 54.3 to 68.9 while the Bond ball mill work index values ranged between 14.8 and 16.6 kWh/tonne, an abrasion Index ranged from 0.13 to 0.57. These results indicate that the samples are moderately soft in Sag Milling, and moderately hard in ball milling.
- Flotation results indicated that JL LCZ samples achieved higher copper recovery and concentrate grades compared to JL UCZ samples. The flotation circuit testing included a high grade (HG) circuit to recover final concentrate grade material before regrinding and further processing through the low grade (LG) cleaner circuit. Combined HG and LG copper concentrate recovery ranged from 90 to 97.4 % in JL LCZ with an average copper grade of 28 %, while recovery in JL UCZ ranged from 59.4 to 83.7 % with an average copper grade of 14.3 %.

- A general trend of declining copper recovery and concentrate grade was measured with increasing JL UCZ proportions in blend composites. However, there was no clear effect of poisoning on the blended samples; although there was some variation between the calculated results based on the arithmetic average of the blend samples and the measured performance. The overall trend was generally in line with the expected calculated performance. Hence the JL UCZ composite did not negatively impact performance of the JL LCZ composite.
- A single locked cycle test on a 50%/50% JL LCZ/JL UCZ blend achieved a copper recovery of 86.6 % at a combined HG and LG concentrate grade of 22.8 % copper.

13.7.1 Effects on plant design

The plant design has been changed following the current PFS test work program. The key changes are:

- Jameson cell size has been increased and the cell has been moved ahead of the regrind circuit.
- Regrind will now treat Jameson cell tails rather than rougher concentrate plus cleaner scavenger concentrate.
- The cleaner scavenger concentrate will report to the regrind mill feed sump rather than the regrind mill cyclone feed sump. As the material has already been classified once there is a potential to again bypass the mill, this was avoided by directing the flow to the mill without the classification stage.
- A review of the flotation times and plant scale up factor required the addition of additional cells in the cleaner circuit.

13.8 Geometallurgy

13.8.1 Johnny Lee Upper Copper Zone

The JL 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, JL 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 JL 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 was developed which allows estimation of variability test cleaner Cu recovery using the mineragraphy derived metrics and is as follows.

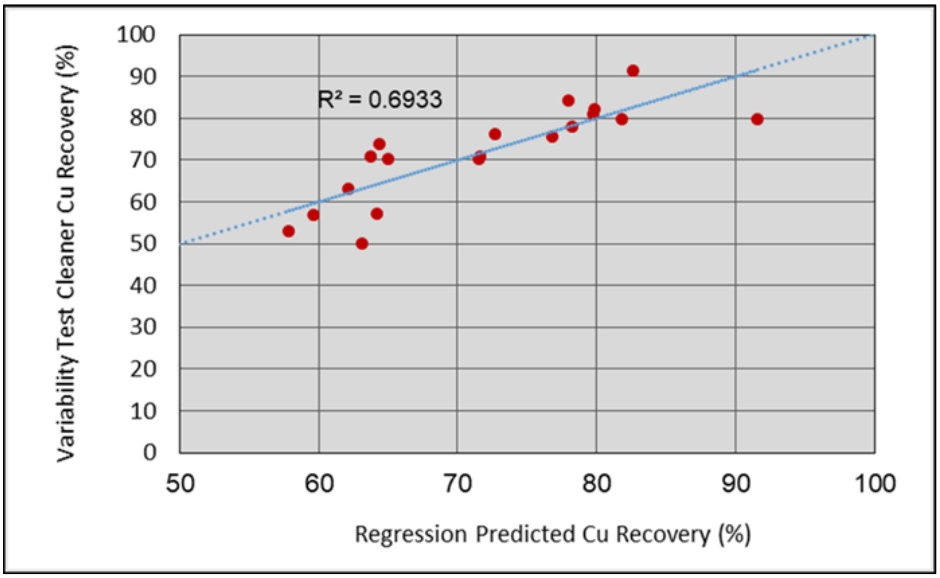
Variability Test Cu Cleaner Recovery = 94.144 + (0.10615*(A+B)) + (-0.28667*(C+D)) + (-0.26708*E)

- 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 non-chalcocite bearing JL UCZ composites, spaced throughout the JL 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 (Figure 13.30 to Figure 13.33). The geological model boundaries (wireframes) used to constrain this model are identical to those of the JL 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

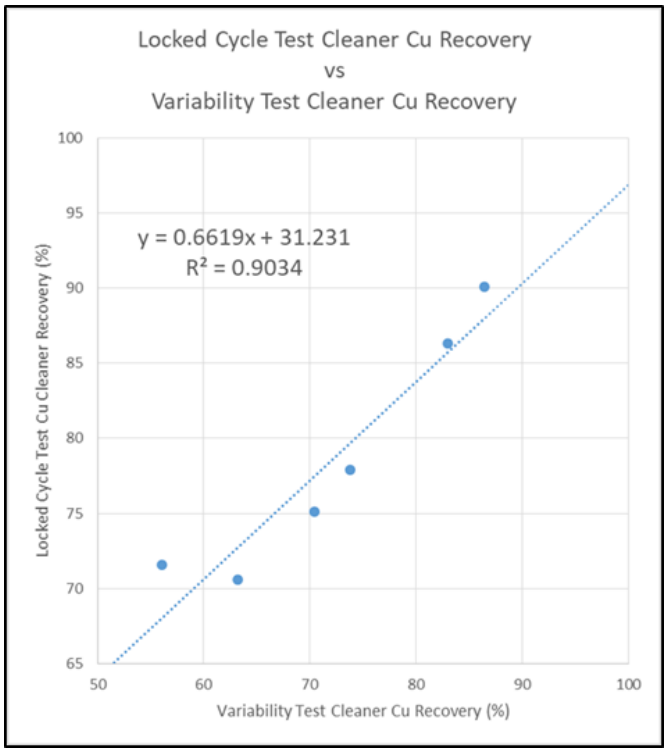
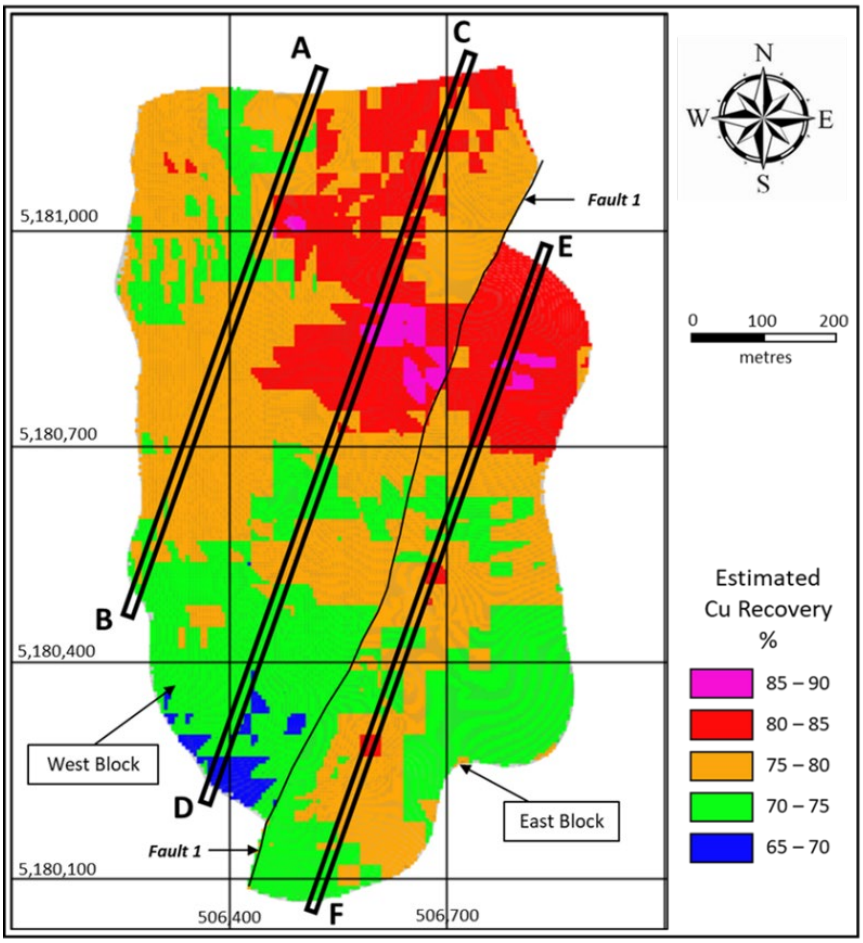
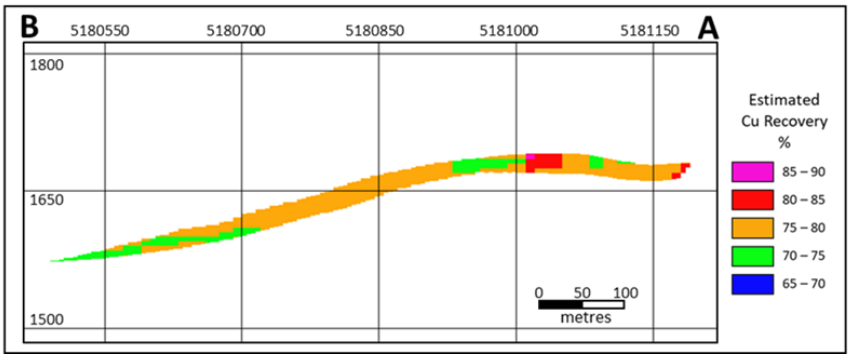


Figure 13.29 LCT Cleaner Recovery vs Variability Test Cleaner Recovery



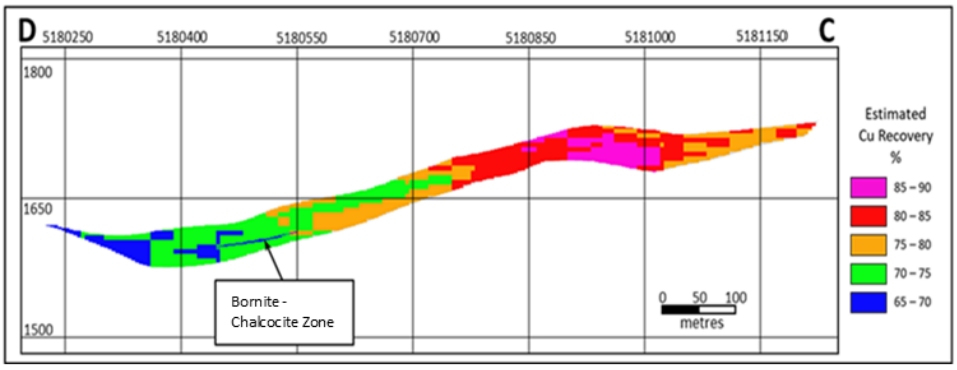
Source: SRA, 2026

Figure 13.30 Geometallurgy Block Model



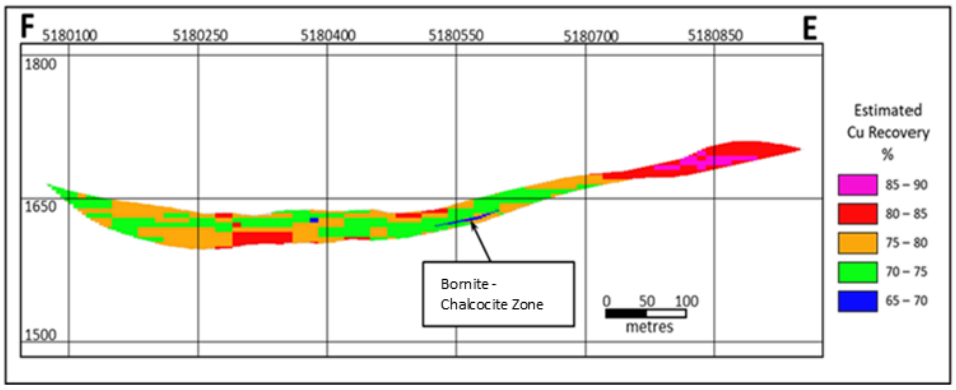
Source: SRA, 2026

Figure 13.31 Cross-section A to B



Source: SRA, 2026

Figure 13.32 Cross-section C to D



Source: SRA, 2026

Figure 13.33 Cross-section E to F

With the exception of a small portion of the NE corner of the JL UCZ the surficial regolith profile does not intersect the JL UCZ. The portion of the JL UCZ that is surficially oxidised has been excluded from the UCZ Mineral Resource estimate. An area dominated by bornite-chalcocite occurs near the intersection of a bedding sub-parallel brittle-ductile shear zone. Here, the mineral assemblage consists of chalcocite, bornite, and minor covellite.

Two open-circuit variability metallurgical tests were performed on composites of >20% bornite-chalcocite assemblage material (BBFT06 and BBFT13). The two composites returned similar Cu recoveries that were both markedly lower than those of the surrounding chalcopyrite-dominated 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% bornite-chalcocite zones in the geometallurgical Cu recovery model.

With not additional variability testwork conducted during the PFS on the JL UCZ, the same approach was used to calculate the metallurgical recovery for JL UCZ as was previously reported. The recovery model was applied to each mining block to calculate a predicted recovered coper metal value.

13.8.2 Johnny Lee Lower Copper Zone

Mineralogical investigation of the JL JL LCZ composites has shown copper sulfide 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 JL JL LCZ using the flow sheet that was developed for the JL UCZ. The variability testwork conducted during the PFS showed a range of recoveries between 94.4% to 97.7% (for the 9 variability samples tested and targeting 23% Cu in concentrate). The arithmetic average of the variability samples tested was 96.2%. Allowing for a 1% Cu recovery deduction due to ideal laboratory testing conditions, a global 95.2% Cu recovery has been used for the JL JL LCZ.

13.8.3 Lowry Middle Copper Zone

Prior to the 2022 and 2023 metallurgical test campaigns, limited metallurgical testing had historically been undertaken on the Lowry Middle Copper Zone (LMCZ) deposit. A single batch cleaner test on a LMCZ ore composite, undertaken for the 2013 PEA, recovered 84% of the Cu to a concentrate grading 24.9%.

10 LMCZ drillhole composites were submitted for systematic mineralogical analyses (McArthur, 2019). Using the regression derived relationship between mineralogy derived copper sulfide 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 LMCZ geometallurgical model.

13.9 Concentrate Specification

The process design criteria and recovery analysis used a copper concentrate grade of 22.5% Cu. This was determined from a combination of LCT and variability tests.

Analyses of the copper concentrates from variability testing of JL UCZ composites have reported potentially deleterious levels of arsenic. Various arsenic containing minerals have been identified through mineralogical analysis, with only some associated with copper reporting to concentrate. All samples tested from the JL LCZ showed low arsenic in copper concentrate. Arsenic is forecast to range between 1,900 to 7,520 ppm in copper concentrate over the life of the project, with lower levels forecast when JL UCZ and JL LCZ are blended (with the higher percentages of JL LCZ in feed yielding the lowest arsenic in concentrate levels.

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14. MINERAL RESOURCE ESTIMATES

The Mineral Resources for the Black Butte Copper Project are comprised of two discrete deposits: the Johnny Lee and the Lowry copper deposits. Resource estimation for the Johnny Lee deposit (in 2025) and the Lowry deposit (in 2023) was completed using Leapfrog Geo™ and Leapfrog Edge™ software for domaining, data analysis, variography, block modelling, grade estimation, model validation, and reporting of Mineral Resources. This work was performed under the guidance of the SRK QP for Mineral Resources.

From late 2018 to present, the current QP for Mineral Resources has been involved with geological modelling and resource estimation for both deposits at the Black Butte Copper Project. The Mineral Resource modelling and estimation for the Johnny Lee and Lowry deposits were reviewed and accepted by Berkley Tracy, P.Geo (PGO #3024) of SRK Consulting (U.S.), Inc., acting as QP for Mineral Resources for the Black Butte Copper Project.

14.1 Drillhole Database

The Black Butte Copper drill data were exported from the Sandfire Resource Ltd Structured Query Language (SQL) database, managed in Perth Australia, to a local MS Access drill database project. Locally on the site in Montana, an MS Access database was used to format drill data, check drilling records, and track lab sample QA/QC. Drill data was exported as comma-separated value (CSV) files (e.g. collar, survey, assay, density) for modelling in Leapfrog Geo software, currently utilizing version.

A total of 305 drillholes have been used to inform the 2025 Johnny Lee geology model and Mineral Resource estimate including historical and recent drilling between 2010 and present (Table 14.1). The data cutoff date for the Johnny Lee Upper Copper Zone (JL UCZ) is 10 June 2025. The data cutoff date for the Johnny Lee Lower Copper Zone (JL LCZ) is 18 June 2025.

A total of 73 drillholes have been used for the 2023 Lowry geology model and Mineral Resource estimate (Table 14.2). No additional drilling has been completed in the Lowry area since the 2023 model.

No new drilling or any testing was completed at the Black Butte Copper Project after the data cut-off dates that materially affects the Mineral Resource estimate, reported effective November 5, 2025.

Descriptions of drilling procedures, sample preparation, sample analysis, and QA/QC are 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	Total Meters
Historical (1986 to 1991)	45	18,692.4
SRA 2010	5	1,509.7
SRA 2011	75	18,792.3

Drilling Campaign	No. of Holes	Total Meters
SRA 2012	52	13,834.4
SRA 2014	11	3,245.1
SRA 2018/2019	46	9,497.8
SRA 2021	5	2,994.7
SRA 2024/2025	66	26,857.2
Total	**305**	**95,423.5**

Table 14.1 Drillholes informing the 2025 Johnny Lee Mineral Resource Estimate by drilling campaign

Drilling Campaign	No. of Holes	Total Meters
Historic Drilling	16	9,751.8
SRA 2010	1	579.1
SRA 2011	18	11,628.5
SRA 2012	17	8,088.8
2013 through 2020	0	0.0
SRA 2021	9	5,052.8
SRA 2022	12	5,485.5
Total	**73**	**40,586.5**

Table 14.2 Drillholes informing the 2023 Lowry Mineral Resource Estimate by drilling campaign

14.1.1 Geometallurgical data

For the Johnny Lee deposit, a total of 125 composites, covering the entire Upper Copper Zone (JL UCZ), have been investigated using systematic particle mineralogy (McArthur, 2017a, 2017b; 2018; 2019). Mineralogical observations from these composites allowed the >20% Chalcocite/Bornite zone to be modelled. In 2019, mineralogically determined copper sulphide liberation estimates were used, along with the metallurgical test data from variability and Locked Cycle Test (LCT) work to develop a regression-based relationship between liberation and expected Cu recovery.

After removal of the >20% Chalcocite/Bornite composites and compositing six pairs of narrow-interval composites (that occur adjacent to each other in the same drillholes), 113 mineralogy composites are available for Cu recovery estimation at Johnny Lee. Using the regression-based relationship, fresh rock JL UCZ Cu recoveries range from 68.2% to 87.9% with an average recovery of 77.4%. The 2019 JL UCZ geometallurgical recovery model was extended by SRA into the 2025 block model using a nearest neighbour (NN) interpolation of the 2019 block centroids. For the JL Lower Copper Zone (JL LCZ), a single Cu recovery of 95.2% was coded into the 2025 block model via post-estimation scripting.

Lowry mineralized intercepts from seven drillholes of the 2021–2022 drilling campaign were used to generate a composite within the Lowry Middle Copper Zone (LMCZ) and a separate composite within the Lowry Lower Copper Zone (LLCZ). Selected samples were sent to the Base Metal Labs in Kamloops Canada for rock characterization and recovery testing. The LMCZ composite sample generated a 91% copper recovery with the LLCZ yielding a 94% recovery. For Lowry, historical metallurgical recovery was assigned previously using the mean recovery of 86% Cu from prior testwork across the deposit. Based on the updated testwork, increased recovery values were used in the current calculation of NSR cut-off values.

14.2 Geologic Model

A 3-D lithostratigraphic and structural framework model of the Johnny Lee deposit was developed in 2019 and updated in 2025. Within the model framework, resource wireframes were constructed for the JL UCZ and JL LCZ using Seequent's Leapfrog Geo™ software version 2024.1.3.

A similar process was used to generate a structural model and stratigraphic units within the Lowry deposit. Wireframes were constructed for the LMCZ and LLCZ using Seequent's Leapfrog Geo™ software version 2021.2.5 software during 2022 and 2023.

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 how the orebodies geologically 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 mineralization show different densities, a framework model is used to assign densities to different lithotypes such that more accurate development and haulage metrics can be estimated.

3-D lithostratigraphic and structural framework models have been developed for the Johnny Lee and Lowry areas using Leapfrog Geo™. These models encompass all areas of the proposed mine development outlined in this report.

14.2.2 Johnny Lee Upper Copper Zone (JL UCZ)

For modelling the JL UCZ, Cu analyses were used as the principal wireframe constraint followed by consideration of 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 JL UCZ. In 2019, multiple Cu values were trialled based on natural breaks in slope or data, with a 1.2% Cu target ultimately providing appropriate mineralization breaks and continuity of mineralization within

the JL 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 defining the JL UCZ modelling was selected at 1.2% Cu.

The mineralized zones were modelled using the raw analytical data at a 1.2% Cu boundary target grade. To create realistic geological wireframes using this approach, it was essential to incorporate some intervals of <1.2% Cu mineralization within the wireframe. Strict rules regarding inclusion of certain maximum widths of internal low-grade material have not been applied during wireframe generation but rather, geologically realistic solids have been modelled that incorporate variable amounts of <1.2% Cu mineralization. The main mineralized high-grade zone is JL UCZ_HG Cu with surrounding lower grade zones of JL UCZ Cu1 and Cu2.

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 Basis Function (RBF) implicit interpolation to create a vein solid by linking hanging wall (HW) and footwall (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 manually-generated polylines (e.g. strings and points) to refine the Vein solids. The >1.2% Cu mineralized zones within the JL UCZ are contained within an envelope of >0.40% Cu Halo and surrounding by outer anomalous sulphide mineralization zones. Modelling of these solids was completed in Leapfrog Geo™.

For historical drilling, certain vertical drillholes did not receive downhole surveys. The earlier campaigns only fully surveyed batches of drillholes. The historical drillholes with downhole surveys were evaluated for drift from the vertical planned orientation. For the JL UCZ, it was determined that any historical drillholes within 5 m of surveyed modern drillholes would be omitted due to lower confidence in spatial precision. The following drillholes were ignored in modeling and estimation within the JL UCZ: SCC-19, SCC-23, SCC-34, and SCC-40.

Both host lithostratigraphy and mineralization are truncated in the north by the Volcano Valley HW Fault (Figure 14.1 and Figure 14.2). A 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 surfaces and solids used to directly constrain the Mineral Resource estimate for the JL UCZ are listed in Table 14.3.

14.2.3 Johnny Lee Lower Copper Zone (JL LCZ)

The JL LCZ, although having more visible copper sulphides than the JL 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 the JL UCZ was adopted for modelling of the JL LCZ. Geological continuity of the JL LCZ was best achieved using an approximate cut-off of >2% Cu.

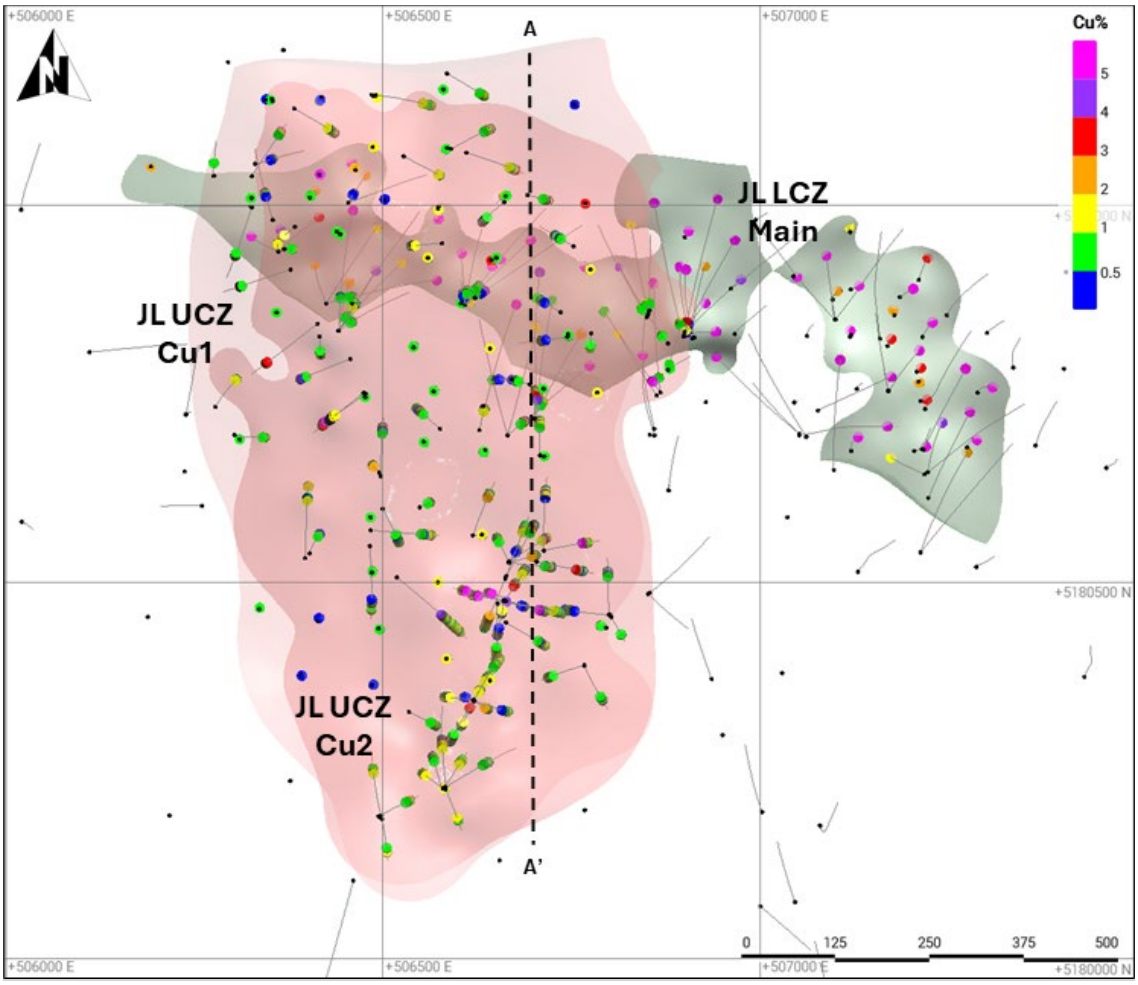
The >2% Cu mineralization shell was used to define Veins in Leapfrog Geo™ software using Interval Selection. Data-defined trends were utilized to create pinch-outs at Vein terminations. Limited use was made of polylines, in areas of lower drill-density, to ensure Veins followed trends consistent with that of the lithostratigraphic host units.

The >2% Cu mineralized zones formed a nearly continuous Main Zone with a smaller, separate Upper Zone. All Lenses are predominantly enclosed within a bedded pyrite, massive sulphide unit referred to as the Johnny Lee Massive Sulphide halo unit (JL LCZ MASU). The halo unit and adjacent clastic sediment lithofacies were modelled as part of the 3-D lithostratigraphic model. The background geological unit is the Neihart Formation.

The JL LCZ units are truncated in the north at the Buttress Fault and in the south by the Volcano Valley FW Fault (Figure 14.2). The 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.

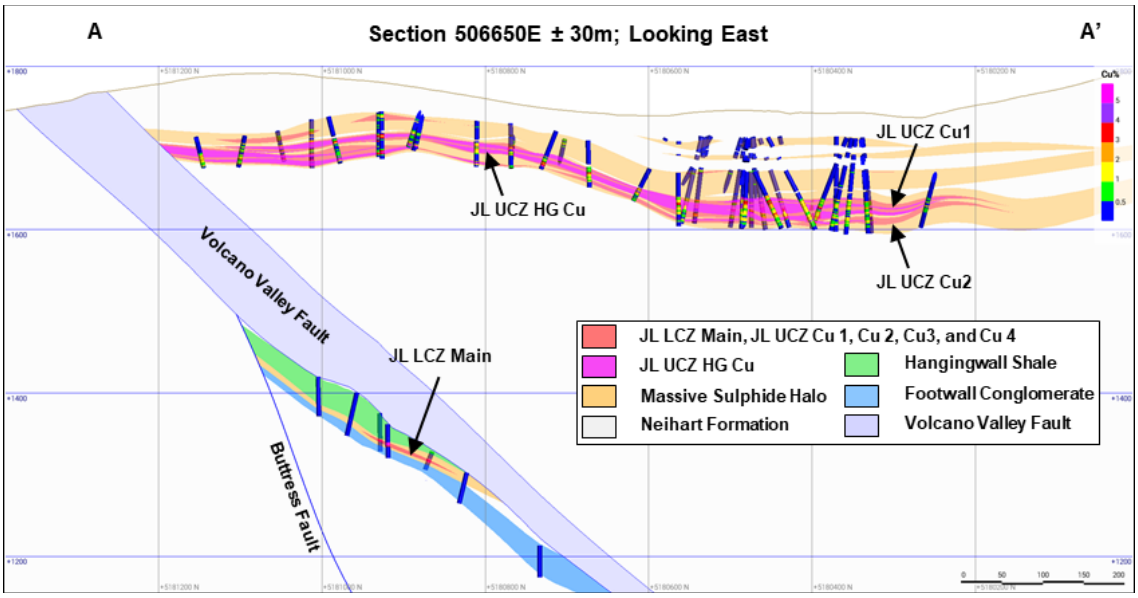
Similar to the JL UCZ, certain vertical drillholes in the JL LCZ did not receive downhole surveys. Based on analysis of historical drilling with downhole surveys, it was determined that any historical drillholes within 20 m of surveyed modern drillholes would be omitted in the JL LCZ data due to lower confidence in spatial precision. These deeper drillholes have the potential for more deviation from vertical than the shallower drilling in the JL UCZ. The following drillholes were ignored in modeling and estimation within the JL LCZ: SCC-34, and SCC-46.

Both host lithostratigraphy and mineralization are truncated in the north by the Volcano Valley HW Fault (Figure 14.1 and Figure 14.2). The surfaces and solids used to directly constrain the Mineral Resource estimate for the JL LCZ are listed in Table 14.3. Table 14.4 demonstrates the close comparison between volume of key mineralized wireframes and the block volumes.



Source: SRA, 2026

Figure 14.1 Plan view of the JL UCZ and JL LCZ Wireframes and Mineralized Composites



Source: SRA, 2026

Figure 14.2 Cross-section of the JL UCZ Halo and Vein models

Wireframe Name	Estimation Domain	Wireframe Type	Description
JL UCZ_HG CU Zone	JL UCZ_HG Cu	Solid	High grade Cu shell
PY01_CU01	JL UCZ_Cu 1	Solid	Low grade Cu shell
PY01_CU02	JL UCZ_Cu 2	Solid	Low grade Cu shell
PY01_CU03	JL UCZ_Cu 3	Solid	Low grade Cu shell
PY02_CU01	JL UCZ_Cu 4	Solid	Low grade Cu shell
PY01	JL UCZ_Sulf 1	Solid	Sulphide halo
PY02	JL UCZ_Sulf 2	Solid	Sulphide halo
PY03	JL UCZ_Sulf 3	Solid	Sulphide halo
PY04	JL UCZ_Sulf 4	Solid	Sulphide halo
PY05	JL UCZ_Sulf 5	Solid	Sulphide halo
1_VVF HW 25	VVF HW	Surface	Volcano Valley Fault hangingwall
2_VVF FW 25	VVF FW	Surface	Volcano Valley Fault footwall
JL LCZ_main	JL LCZ_main	Solid	High grade Cu shell - main
JL LCZ_upper	JL LCZ_upper	Solid	High grade Cu shell - upper
JL LCZ_MASU Halo	JL LCZ_masu halo	Solid	Low grade Cu Massive Sulphide (MASU) halo
JL LCZ_FW Congl	JL LCZ_fw congl	Solid	Footwall conglomerate unit
JL LCZ_HW Shale	JL LCZ_hw shale	Solid	Hanging wall shale unit

Table 14.3 Wireframes constraining the JL UCZ and JL LCZ Mineral Resource estimate

Domain	Wireframe Volume (m^3)	Octree Block Volume (m^3)	Percent Difference (%)
JL UCZ HG Cu	3,142,600	3,144,719	0.07%
JL UCZ Cu 1	2,464,800	2,463,527	-0.05%
JL UCZ Cu 2	2,431,200	2,430,234	-0.04%
JL LCZ Main	733,340	733,151	-0.03%
JL LCZ Upper	95,892	95,636	-0.27%
JL LCZ MASU	3,175,900	3,175,789	0.00%

Table 14.4 Comparison of Mineralized Wireframes Volumes to Block Volumes

14.2.4 Lowry Upper Copper Zone (LUCZ)

The LUCZ is composed of four thinly bedded units spaced 5 to 20 m apart. Three of the beds are positioned west of the Rose fault and one bed is located on the east side of the fault. Drill intercepts for these modelled shapes are relatively low-grade copper compared to the LMCZ and LLCZ. The selection of drill intercepts for wireframe construction relied upon the orientation of the geological bedding in the LMCZ to determine likely bedding angles in the LUCZ. All wireframing for Lowry was constructed in Leapfrog™ Geo.

14.2.5 Lowry Middle Copper Zone (LMCZ)

Two distinct zones of economic Cu mineralization, termed the Lowry Middle Cu Zone (LMCZ) and Lowry Lower Copper Zone (LLCZ) are hosted by a sub-parallel, iron bearing sediment with interbedded shale and conglomerate units. Drillhole intersection spacing in the Lowry deposit's LMCZ ranges from 10 to 100 m. Drillhole spacing targeted an average of 50 m within the mineralized zone.

The LMCZ is hosted by broad zones of dolomitic breccia in interbedded pyrite layers, conglomerate, carbonaceous shale, dolomitic shale and dolomite. The host unit that contains most of the mineralization in the LMCZ has been modelled as a single wireframe solid. Within this solid, there are elevated Cu grades modelled as internal wireframes for domaining during block estimation. Higher Cu grades are located on the eastern side of the shape, adjacent to the Rose fault.

Cu mineralization within the LMCZ ranges in grade and continuity. Changes in grade can occur over short distances, both vertically and horizontally. This change is evident when plotting the grade profile of Cu down the drillhole in mineralized zones. For modelling, a threshold of 1.2% Cu was targeted to define the Lowry domains. The host unit and mineralization are truncated against the Rose fault zone. Figure 14.3 shows the solids that form the LMCZ model in plan view.

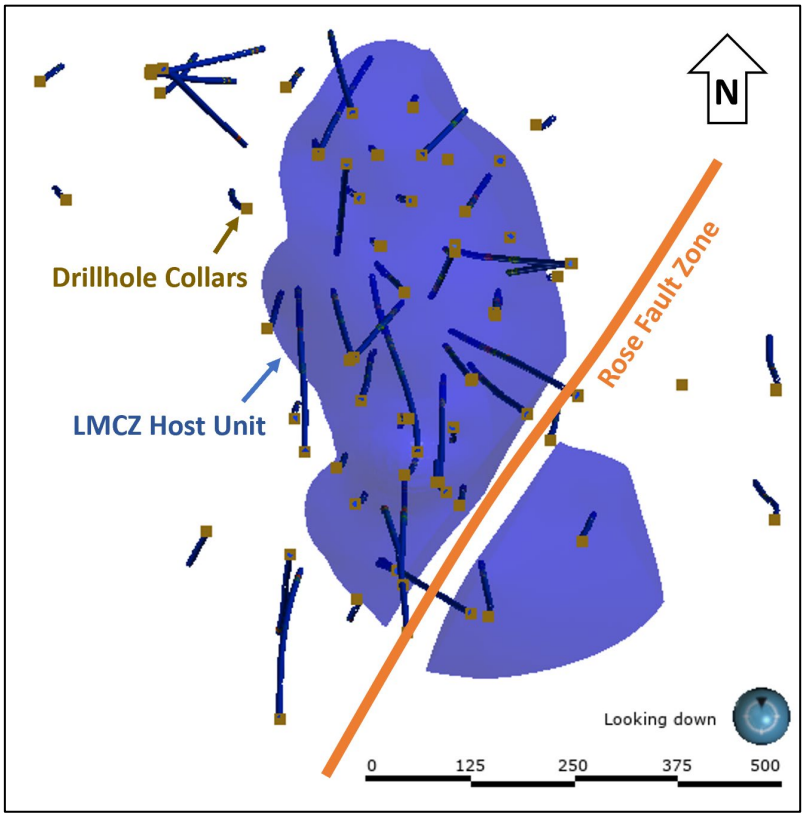


Figure 14.3 LMCZ model in Plan view

14.2.6 Lowry Lower Copper Zone (LLCZ)

A dozen drillholes that intersected the LMCZ were stopped-short of the deeper LLCZ. Consequently, drillhole spacing in the LLCZ is wider than the LMCZ, ranging from 50 to 200 m. The LLCZ occurs below the Volcano Valley Hanging Wall (VVHW) with the mineralized base following the Volcano Valley Footwall Zone (VVFZ).

The LLCZ is hosted by interbedded shale, conglomerate and pyrite beds. The host unit that contains much of the mineralization in the LLCZ has been modelled as a single solid containing a range of high and low Cu grades. A 1.2% Cu threshold was used for modelling the LLCZ domain.

Figure 14.4 shows the LLCZ relative to the upper LMCZ model in plan view. Figure 14.5 is a cross-section view portraying the relationship of the Lowry mineralization domains to the Volcano Valley and Rose faults.

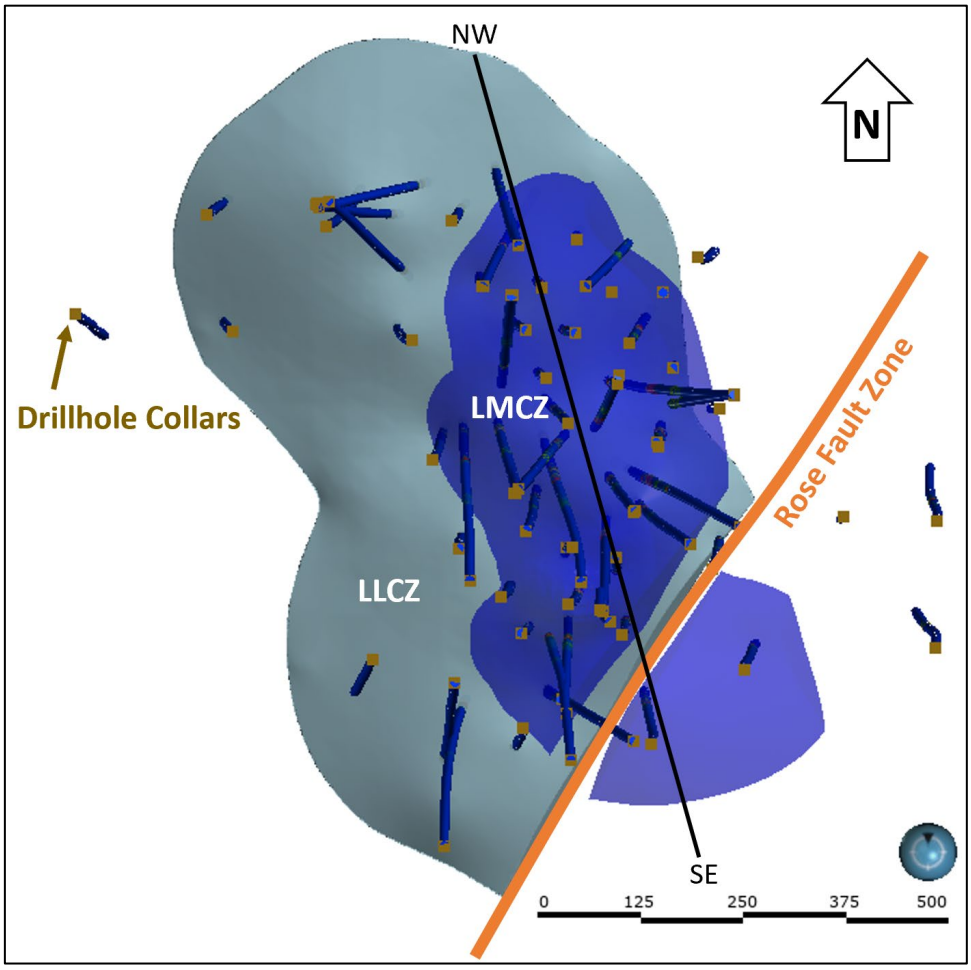


Figure 14.4 Plan View of the LLCZ below the LMCZ

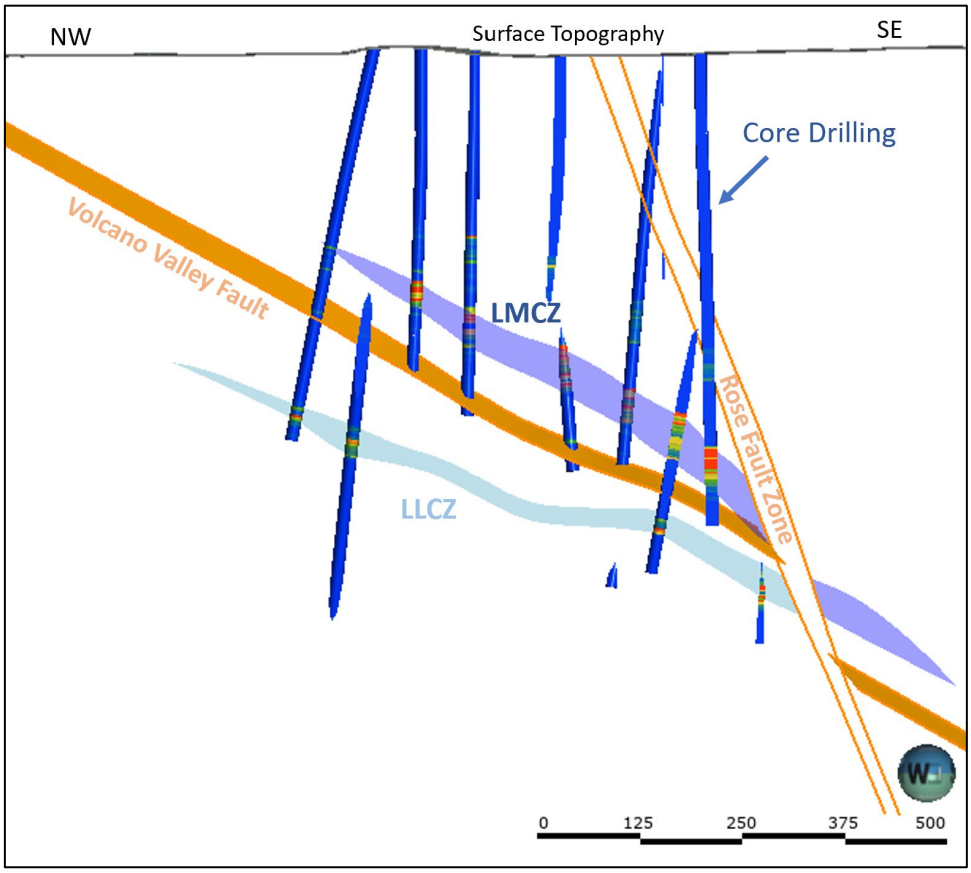


Figure 14.5 Cross-section of the LMCZ & LLCZ along line NW-SE (Figure 14.4) looking northeast

14.3 Density

The bulk density at the Black Butte Copper project is defined using the measured specific gravity (SG) data obtained from drill core on each deposit. The term density is used in this Technical Report to convey bulk density and is synonymous with SG.

14.3.1 Johnny Lee Upper Copper Zone and Lower Copper Zone

SG displays variability within both strongly and weakly Cu mineralized material within the JL UCZ. In particular, SG of the potentially economic mineralization in the northern part of the JL UCZ was generally lower (by approximately 0.8) than that of the southern part of the JL UCZ. In contrast to the JL UCZ, the JL LCZ has relatively consistent SG values

Figure 14.6 and Figure 14.7 show material types in diamond drill core in the northern and southern parts of the JL 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 JL USZ, the ferruginous unit that hosts the JL UCZ mineralization.

In the northern part of the JL USZ, the zones of massive sulphide and pyrite beds were defined for 2018 logging purposes as >50% sulphide over a >30 cm width. These zones are characterized by an increased proportion of narrow shale and siltstone intercalations when compared to the southern part of the JL USZ.

Due to the variable nature of SG in the JL UCZ, it was determined that an appropriate approach was to estimate density into the primary wireframes. This allows for a robust geologic framework and a sufficient density dataset for estimation of the main units. The density estimate was conducted via inverse distance weighting power of two (IDW2) interpolation into the following JL UCZ units: HG Cu, Cu 1, Cu, 2, and Sulf 1 through Sulf 5. An IDW2 interpolation was used to estimate density in the following JL LCZ units: Main, Upper, MASU Halo, FW Congl, and HW Shale. For zones with insufficient data and areas of domains without estimated values, mean values for the individual domains were assigned by block scripting (Table 14.5).

Wireframe Name	Estimation Domain	Density
JL UCZ HG CU Zone	JL UCZ HG Cu	3.86
PY01_CU01	JL UCZ Cu 1	3.70
PY01_CU02	JL UCZ Cu 2	3.90
PY01_CU03	JL UCZ Cu 3	3.85
PY02_CU01	JL UCZ Cu 4	3.85
PY01	JL UCZ Sulf 1	3.59
PY02	JL UCZ Sulf 2	3.62
PY03	JL UCZ Sulf 3	3.15
PY04	JL UCZ Sulf 4	3.15
PY05	JL UCZ Sulf 5	3.15
Volcano Valley Fault	VVF	2.89
JL LCZ main	JL LCZ main	3.48
JL LCZ upper	JL LCZ upper	3.48
JL LCZ MASU Halo	JL LCZ masu halo	3.48
JL LCZ FW Congl	JL LCZ fw congl	2.93
JL LCZ HW Shale	JL LCZ hw shale	2.69

Table 14.5 Summary of assigned Density values for the JL UCZ and JL LCZ Mineral Resource estimateframe Name

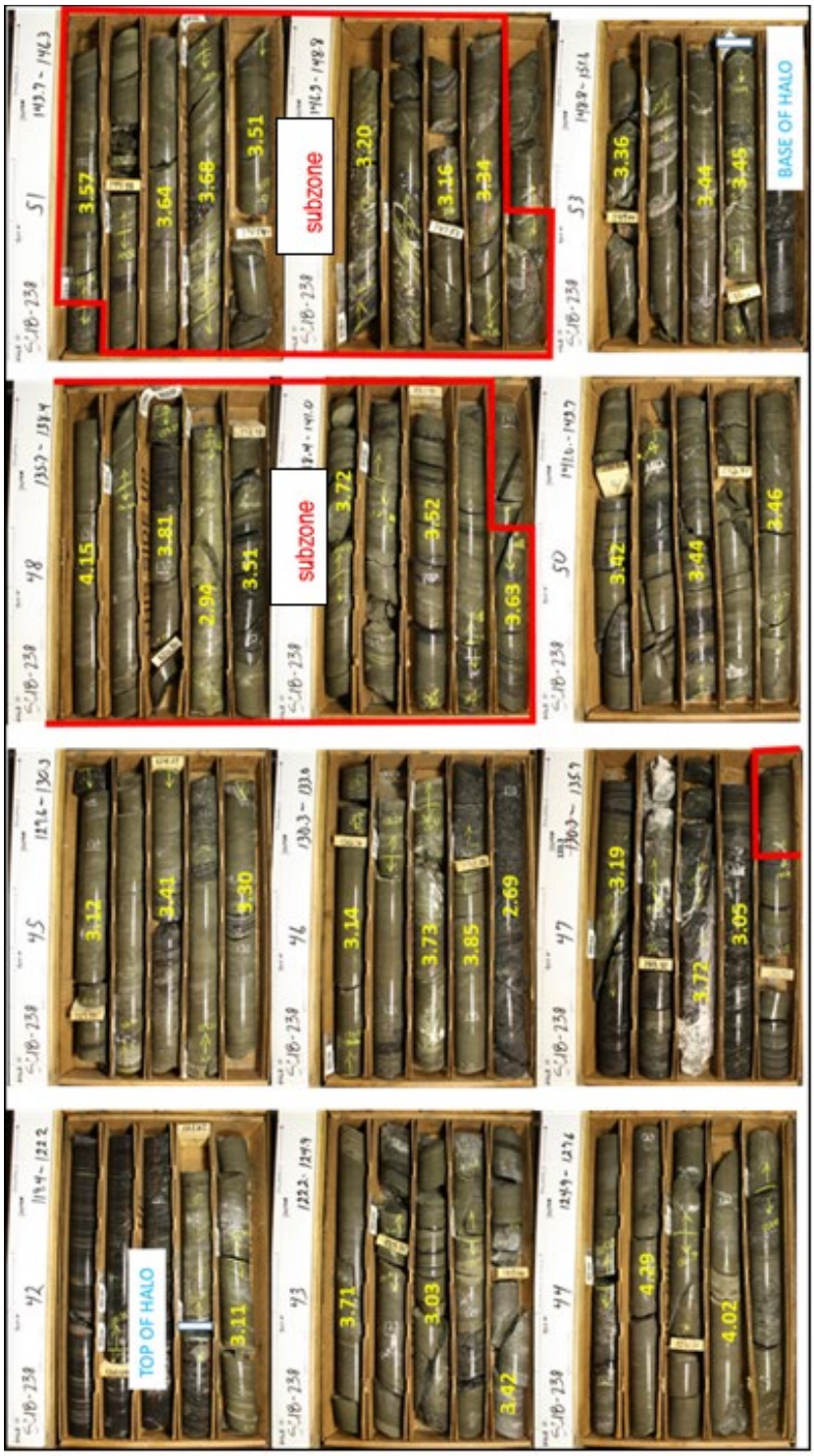


Figure 14.6 Core photographs for SC18-238 showing top and base of Halo and >1.2% Cu Subzone



Figure 14.7 Core photographs for SC18-245 showing top and base of Halo and >1.2% Cu Subzone

14.3.2 Lowry Middle Copper Zone and Lower Copper Zone

SG measurements were collected on core sample segments and for sample intervals matching the assay length. The segments used for SG measurements were primarily gathered between 2010 to 2015. Measuring SG for entire sample intervals started in 2018 and continued through the 2022 drill program. SG measurements were primarily taken within pyrite and chalcopyrite zones where copper mineralization occurs.

The host units for the LMCZ and LLCZ consist of variable proportions of interlayered massive sulphide and pyrite beds shale, conglomerate, and breccia showing locally variable SG. Density within the LMCZ and LLCZ was estimated from available data within the modelled sulphide units.

Due to the different SG measurement types, density sample data was separated into two groups. Measurements for short segments (~10 cm) were denoted in the database as "POINT" with the full-length (~1.0 m) method called "INTERVAL" as described in Chapter 11. To account for the different sample collection lengths, interval values were composited by 2.5 m, then appended to the point data for statistical evaluation and block estimation.

The SG data values were estimated into the block model using the sulphide wireframes as domains. If a block within the sulphide wireframe did not obtain an estimated value due to search neighborhood restrictions, an SG value was assigned based on the mean SG for that sulphide wireframe. An SG value of 3.14 was used for blocks external to all sulphide shapes. These steps were accomplished using a block calculation script within Leapfrog Edge™ software. All scripts are retained within the Leapfrog project file, which allows tracking of assigned values to modelled shapes.

14.4 Compositing, Summary Statistics, and Capping

14.4.1 Johnny Lee Compositing

Historical sample intervals range in length from 0.1 to 6.6 m with a mean of 1.0 m. Since 2018, sample intervals have been restricted to a range between 0.3 and 1.3 m. The overall sample length distribution presented in Figure 14.8 for the JL UCZ and Figure 14.9 for the JL LCZ. For the JL UCZ, data are filtered for inclusion within estimation domains HG Cu, Cu 1, and Cu 2, which are the main mineralized zones. For the JL LCZ, data are filtered for inclusion within estimation domains JL LCZ Main and JL LCZ Upper.

After reviewing compositing, it was determined that 1.5 m was the most appropriate compositing length considering the data and geological models. Using Leapfrog Geo™, data were composited targeting 1.5 m length with downhole intervals constrained by domain boundaries and residual lengths less than 0.75 m distributed equally to the composite group. Additionally, 3 m composites were run for block height comparisons during estimation validation. Summary composite length histograms are presented in Figure 14.10 by estimation domain. In each histogram, the small intervals represent residual lengths cut by domain boundaries.

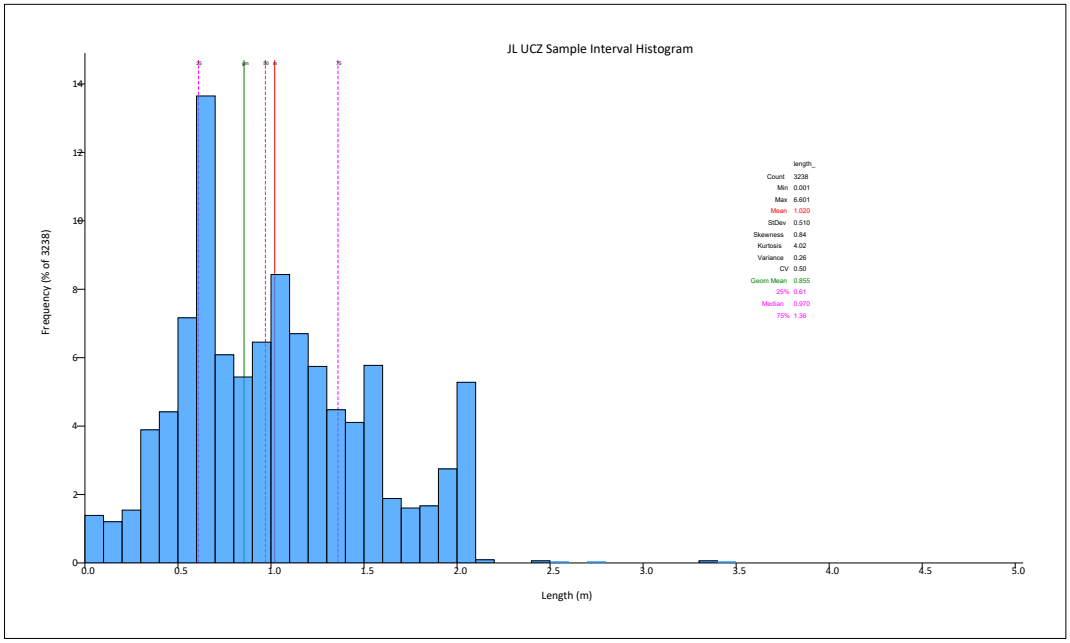


Figure 14.8 Drilling Sample Lengths in the JL UCZ

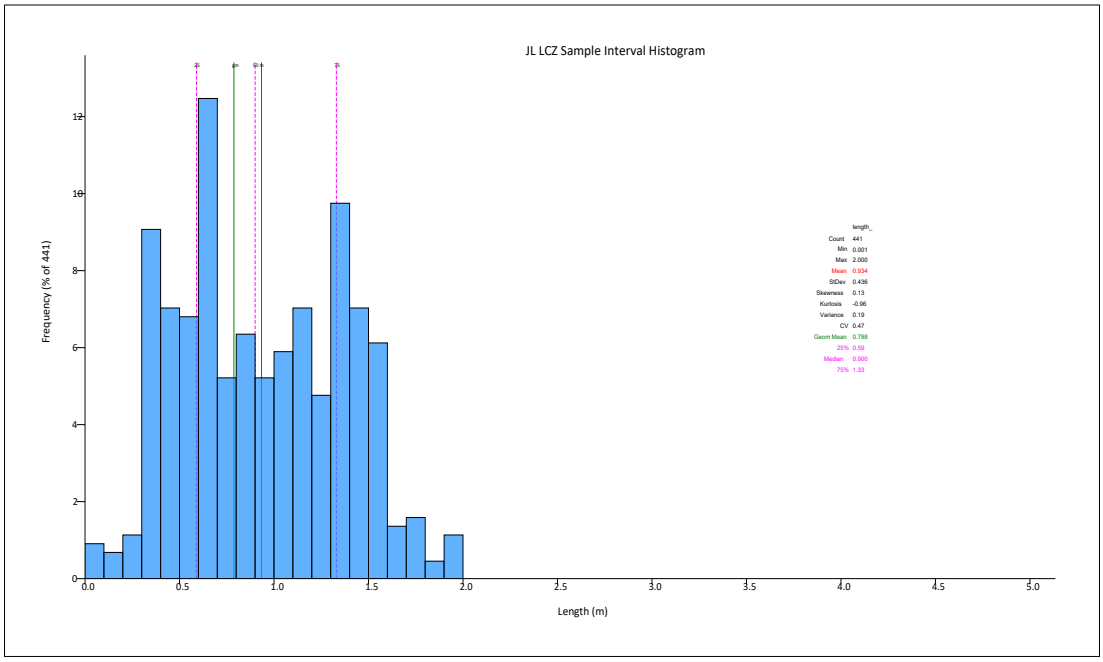


Figure 14.9 Drilling Sample Lengths in the JL LCZ

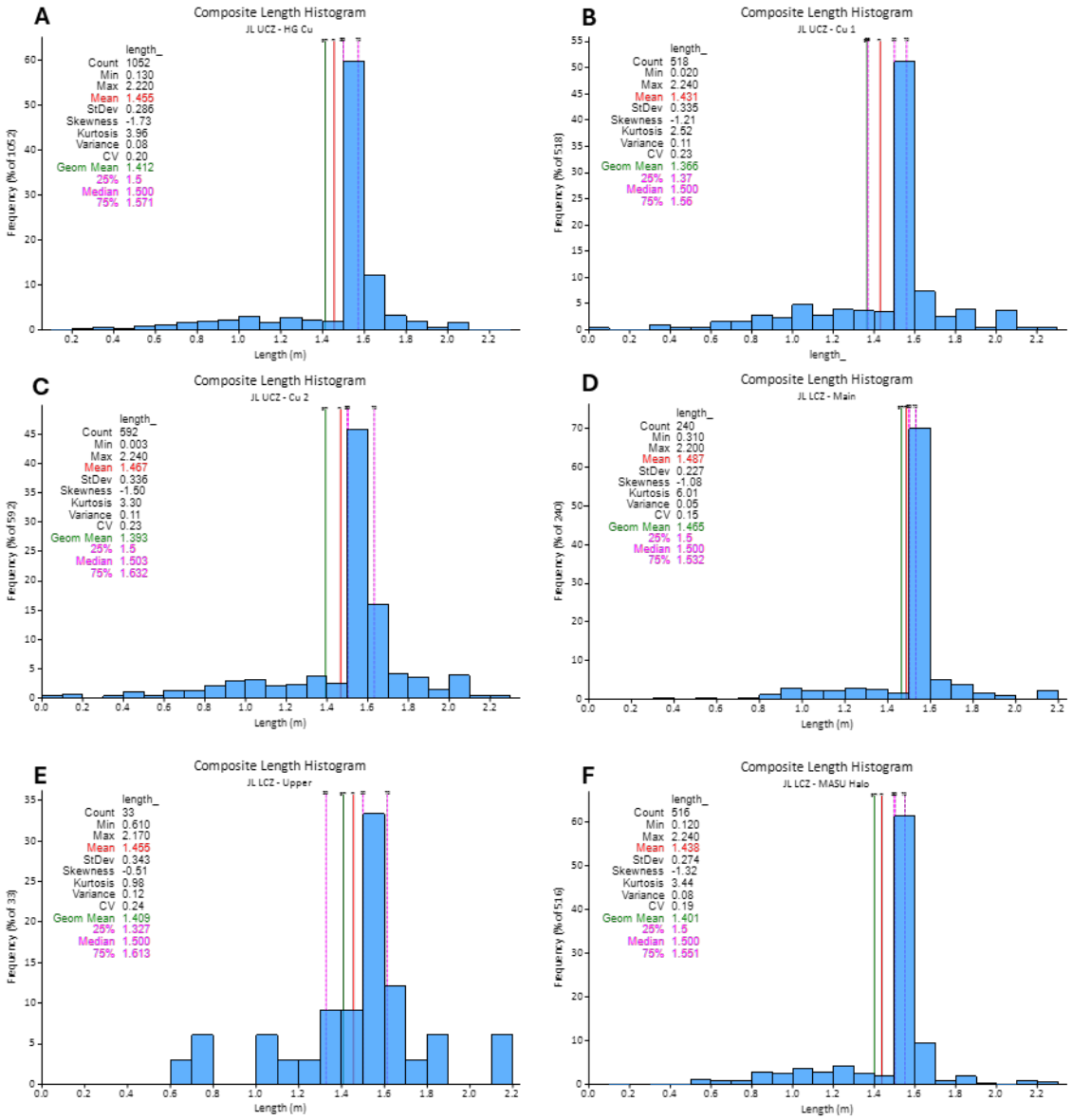


Figure 14.10 Composite Lengths for each of the Johnny Lee Cu Domains

In Figure 14.10, the histograms represented are: A = JL UCZ HG Cu, B = JL UCZ Cu 1, C = JL UCZ Cu 2, D = JL LCZ Main, E = JL LCZ Upper and F = JL LCZ MASU Halo.

14.4.2 Lowry Compositing

Original sample lengths were collected in groups of mainly 1.5 m or 2.0 m intervals. There was a total of 10,924 samples from the Lowry deposit with a Cu assay value. Sample length is determined by the logging geologist according to variances seen in the core, such as changes in lithology or alteration. Additionally, visual indication of Cu grade is estimated by the abundance of chalcopyrite. Many high-grade (>1.2% Cu)

zones are sampled on intervals less than 2 m in length based on sampling related to visual estimates during logging.

Raw drilling data was analyzed with a histogram of sample lengths presented in Figure 14.11 The mean length is 1.54 m with a significant percentage of data collected at 1.5 m and 2.0 m intervals represented by the two spikes in the histogram.

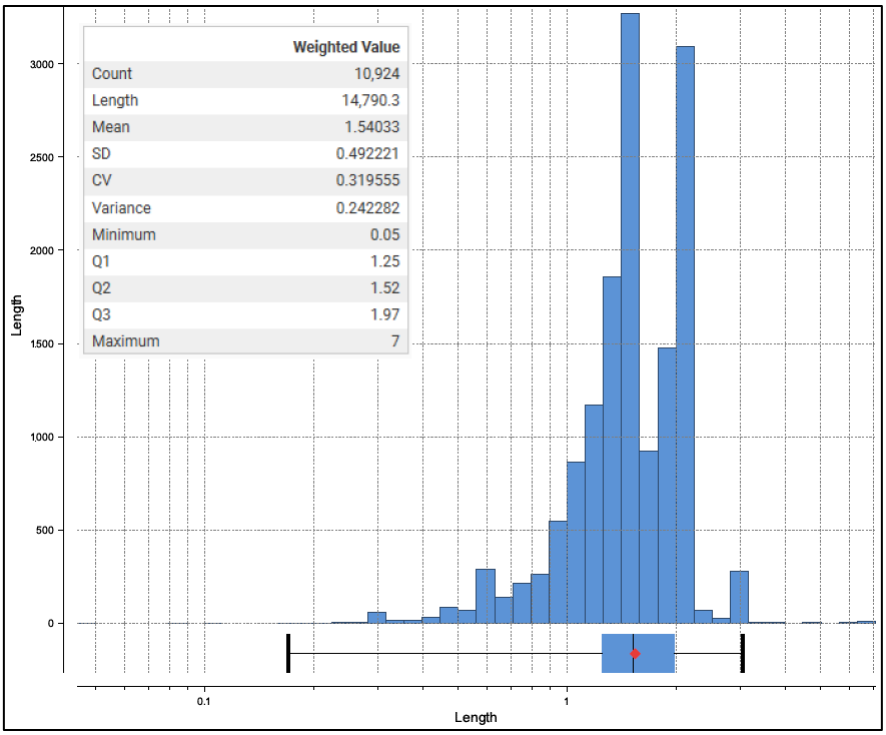


Figure 14.11 Histogram of Sample Lengths in the Lowry deposit

The relation of Cu grades to sample length was investigated using a scatter plot of the two variables. Linear regression shows a slight negative relationship indicating no material bias between Cu grade and the sampled length.

Lowry mineralized material is planned to be fed to the nearby Johnny Lee plant using either drift and fill or cut and fill mining methods, depending upon orebody height. The selected composite was 2.5 m which is appropriate for the selective mining unit (SMU) for the anticipated mining method. The 2.5 m composite is half the 5 x 5-meter decline ramp and stope drive dimensions.

Compositing was performed using Seequent's Leapfrog Geo™ v2021.2.5 software using "Subset of Codes". This option breaks composite intervals at modelled wireframe boundaries and allows the combining of composite "tails" to intervals within the run group. Intervals less than 1.0 m were merged with the previous interval. The 2.5 m compositing parameters resulted in 1,096 Cu composites within the modelled LMCZ and

LLCZ zones having a mean length of 2.460 m. As part of the Leapfrog Geo sample import routine, drill core not sampled (and consequently not assayed) is replaced with a null 0.0001% Cu value. This allows for the creation of waste composites along missing drill intervals during composite creation.

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 14.6 and Table 14.7). All domains show consistently low variance and Coefficient of Variation (CV). This analysis demonstrates that a stationarity assumption is reasonable for the style of deposit and that linear estimation is appropriate for Cu and SG using the modelled domains. Histograms for Cu are provided in Figure 14.12.

Cu %	JL UCZ HG Cu	JL UCZ Cu 1	JL UCZ Cu2	JL LCZ Main	JL LCZ Upper	JL LCZ MASU
Composites	1052	518	592	240	33	516
Average SG	2.53	0.69	0.66	7.07	3.95	0.24
Min SG	0.001	0.026	0.001	0.050	0.051	0.001
Max SG	16.4	8.41	1.49	20.51	12.86	6.10
Variance	2.72	0.24	0.06	25.18	14.47	0.23
Std. Deviation	1.649	0.488	0.236	5.018	3.804	0.476
CV	0.65	0.70	0.36	0.71	0.96	2.02

Table 14.6 Pre-Capping Composite Statistics by Johnny Lee Domain for Cu

Specific Gravity	JL UCZ HG Cu	JL UCZ Cu 1	JL UCZ Cu2	JL LCZ MASU	JL LCZ FW Congl	JL LCZ HW Shale
Composites	496	166	204	212	179	189
Average SG	3.86	3.70	3.90	3.48	2.69	2.93
Min SG	2.77	2.24	2.68	2.63	2.18	2.01
Max SG	4.69	4.48	4.54	4.40	4.02	4.20
Variance	0.14	0.19	0.20	0.22	0.03	0.13
Std. Deviation	0.37	0.43	0.45	0.47	0.18	0.36
CV	0.10	0.12	0.12	0.13	0.07	0.12

Table 14.7 Pre-Capping Composite Statistics by Johnny Lee Domain for SG

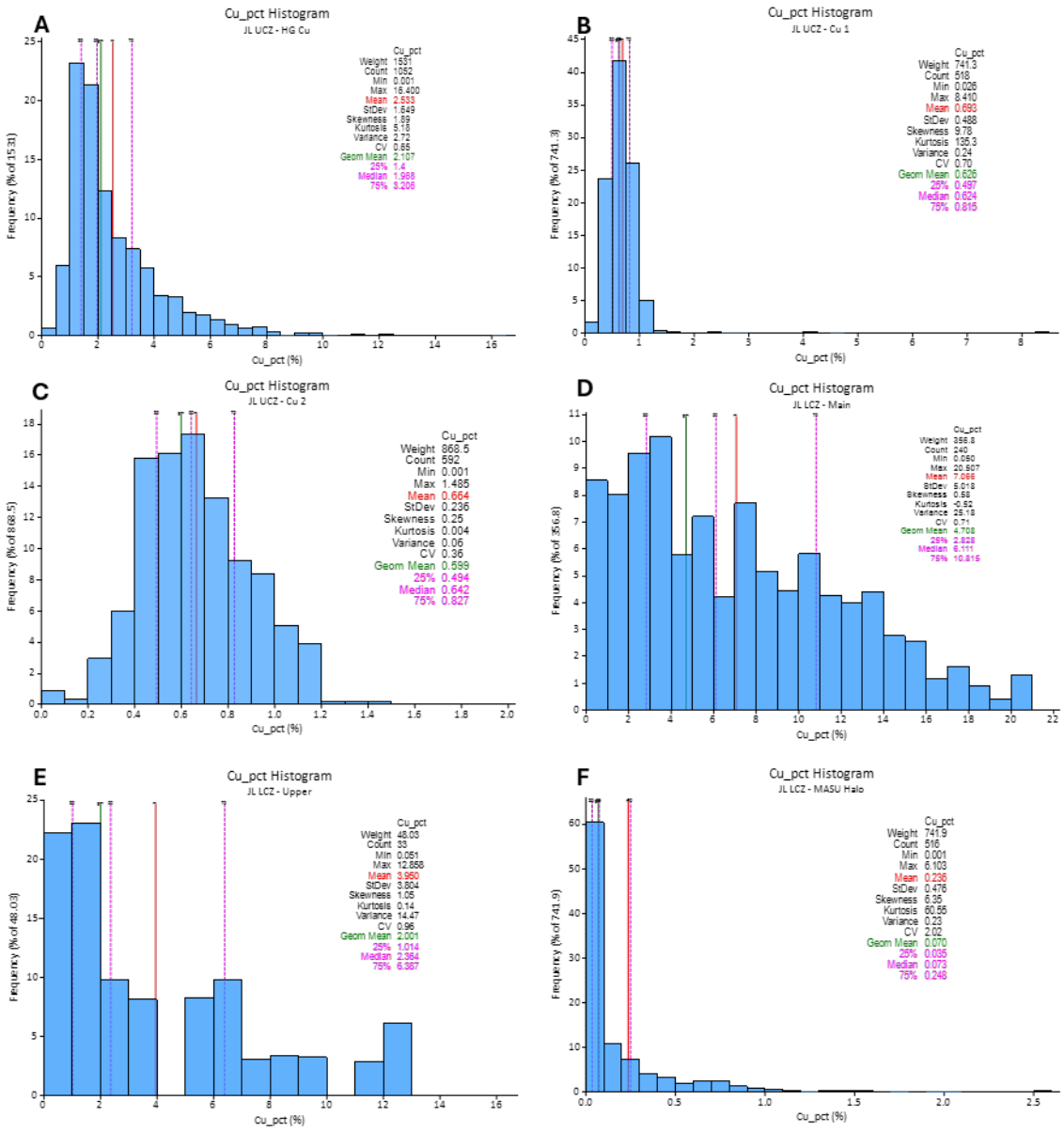


Figure 14.12 Cu Histogram Distributions by Johnny Lee Domain

In Figure 14.12, the histograms represented are: A = JL UCZ HG Cu, B = JL UCZ Cu 1, C = JL UCZ Cu 2, D = JL LCZ Main, E = JL LCZ Upper and F = JL LCZ MASU Halo.

14.4.4 Lowry Summary Statistics

Exploratory Data Analysis (EDA) was conducted on composited data for Ag, As, Ba, Co, Cu, Fe, Pb, Zn and SG by domain with Cu being the key economic variable of interest. A series of statistical and graphical outputs were generated including descriptive univariate statistics, bivariate statistics, histograms, and box and whisker plots. Summary descriptive statistics for key variables by domain are provided in Table 14.8.

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 are observed through difficulty obtaining a robust modelled variogram for spatial continuity and a reduction in mean grade from composites to estimated blocks. Statistics for modelled shapes and domains were generated using the Leapfrog Geo software and are provided as Figure 14.13 through Figure 14.15.

Name	Count	Mean	SD	CV	Variance	Minimum	Median	Maximum
Ag_ppm	1096	9.40	7.35	0.78	54.06	0.05	8.26	43.00
As_ppm	1096	942.07	1067.28	1.13	1139094	0.1	656.6	9570.0
Ba_ppm	1057	4940	20870	4	435577381	10.0	356.3	248061.6
Co_ppm	1096	521.48	547.89	1.05	300182	5	380	5250
Cu_pct	1096	1.14	1.35	1.18	1.82	0.0001	0.6765	12.0775
Fe_pct	1095	11.37	7.01	0.62	49.15	0.75	10.53	36.47
Pb_ppm	1096	645.25	587.47	0.91	345121	5.0	531.2	5020.0
Zn_ppm	1096	223.83	474.92	2.12	225553	5.0	91.8	5390.0

Table 14.8 Summary Statistics for Composited Data within the LMCZ and LLCZ

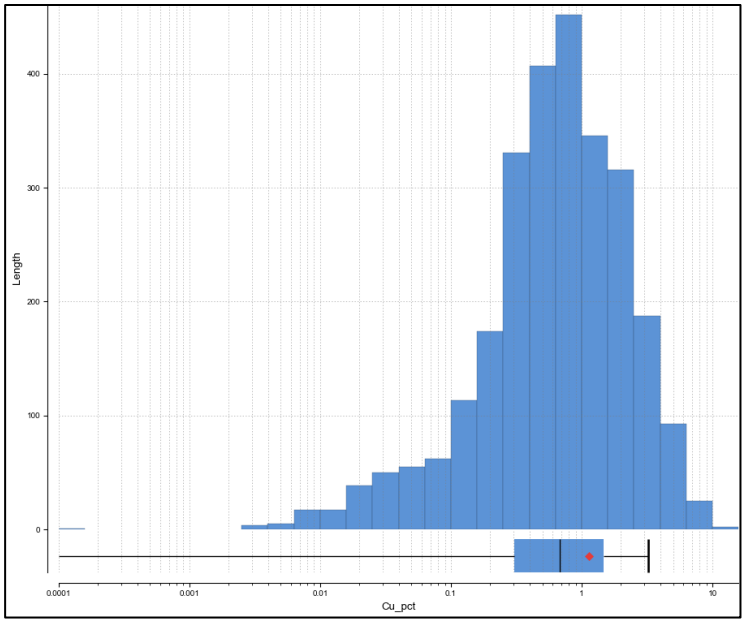


Figure 14.13 Histogram of Composited Cu grades within the LMCZ and LLCZ

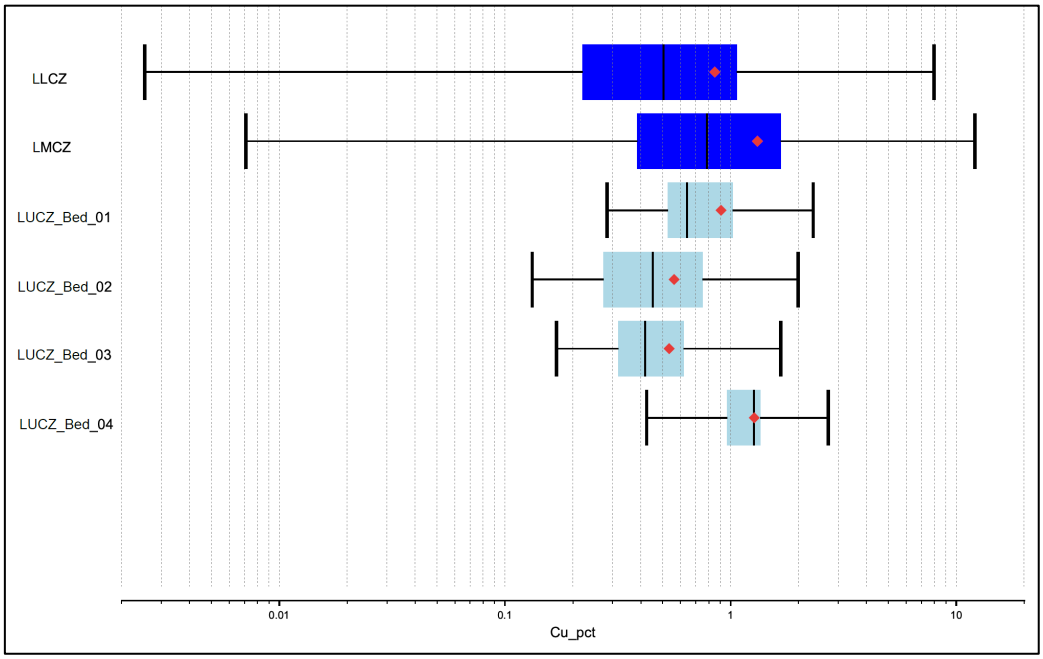


Figure 14.14 Box and Whisker Plot of Cu grades (log scale) within the Lowry grade shapes

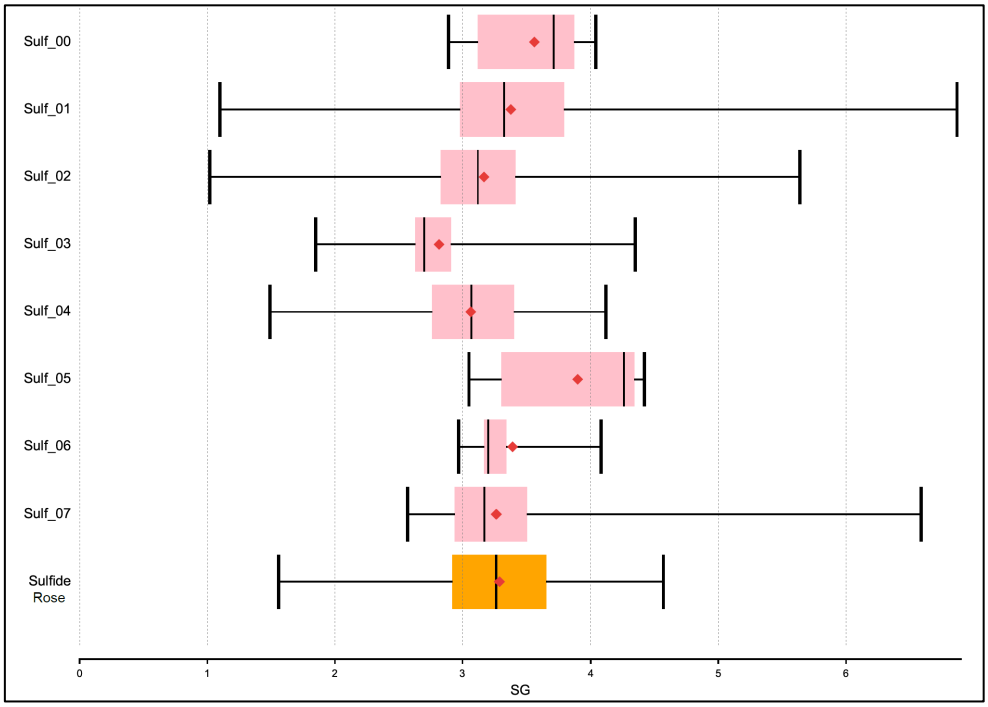


Figure 14.15 Box and Whisker Plot of SG by Sulphide Domain in the Lowry deposit

Bivariate correlations were calculated for key variables with summary values presented in Table 14.9. The highest directly correlated elements are Co-As, likely from these elements being contained within pyrite. Additionally, Ag-Pb, and Cu-As-Co also likely associated with pyrite and marcasite.

	Ag_ppm	As_ppm	Ba_ppm	Ca_pct	Co_ppm	Cu_pct	Fe_pct	Pb_ppm	Sr_ppm	Zn_ppm
Ag_ppm	1									
As_ppm	0.53	1								
Ba_ppm	0.03	-0.08	1							
Ca_pct	-0.19	-0.17	0.00	1						
Co_ppm	0.62	0.87	-0.10	-0.23	1					
Cu_pct	0.49	0.55	-0.09	-0.10	0.50	1				
Fe_pct	0.72	0.34	0.12	-0.08	0.42	0.28	1			
Pb_ppm	0.69	0.39	0.01	-0.14	0.37	0.13	0.69	1		
Sr_ppm	-0.17	-0.07	0.11	0.03	-0.14	-0.12	-0.14	-0.13	1	
Zn_ppm	0.12	0.38	0.00	-0.06	0.07	0.10	0.08	0.41	0.05	1
					Correlation Scale					
	0.10	0.20	0.30	0.40	0.50	0.60	0.70	0.80	0.90	1.00
	No Correlation								High Correlation	

Table 14.9 Correlation Coefficients for Key Elements in the Lowry Cu Zones

14.4.5 Johnny Lee Outliers and Capping

After compositing, Cu values were capped for the JL UCZ HG Cu, JL UCZ Cu 1 and JL LCZ MASU domains based on disintegration analysis of Cu log probability plots. No capping or cutting of data was applied to the other Johnny Lee domains, after reviewing high-grade intercepts in 3D. The capping was selected to minimize the effect on estimation of isolated high-grade outliers, without materially reducing metal or modifying a material proportion of data. Capped values by domain are presented in Table 14.10.

Domain	Cu Grade Cap (%)	# Composites Capped
JL UCZ HG Cu	10.1	4
JL UCZ Cu 1	4.51	2
JL UCZ Cu 2	none	0
JL LCZ Main	none	0
JL LCZ Upper	none	0
JL LCZ MASU	3.11	2

Table 14.10 Summary of Cu (%) Grade Capping at Johnny Lee

14.4.6 Lowry Outlier and Capping

Composited values were analyzed for high-yield outliers that could 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 by variable. Due to the difference in Cu grade between zones, cap analysis was run on samples separately in the LMCZ and LLCZ. In all cases,

a capped value is assigned at the limits but does not exclude those samples. Capped values by domain are presented in Table 14.11.

Domain	Cu Grade Cap (%)	# Composites Capped
Sulphide 1-7	1.00	3
Beds 1-6	2.00	4
LMCZ	8.00	4
LLCZ	4.50	5

Table 14.11 **Summary of Cu (%) Grade Capping at Lowry**

Overall, capping reduced the highest yield outlier data while not materially affecting the population of data. Capping results in an improved composited database appropriate for estimation. It is the opinion of the QP for Mineral Resources that capping has reduced the effect of high-yield outlier values appropriately, while not materially reducing overall metal content.

14.5 Variogram Analysis and Modelling

14.5.1 Johnny Lee Variography

Spatial analysis for the Johnny Lee deposit was conducted on the key economic variable Cu by domain using variography. The modelled variograms were generated using Seequent's Leapfrog Geo™ v2024.1.3 software with the Edge extension. Variography was not calculated independently for secondary variables.

Variogram maps were generated for Cu to determine dominant orientations of continuity within each domain. Outputs of the variogram map analysis show the primary direction of spatial continuity aligns with the major geological direction of the mineralized body, typically down dip along each mineralized bed. Additionally, for estimation, variable orientation searches were utilized based on the estimation domains.

Variograms for the JL UCZ and JL LCZ for Cu are summarized in Table 14.12.

Domain	Rotation			Nugget	Structure 1 (m)				Structure 1 (m)			
	Dip	Dip Az	Pitch		Sill	Major	Semi Major	Minor	Sill	Major	Semi Major	Minor
JL UCZ HG Cu	12	222	67	0.1	0.72	40	40	3	0.18	150	100	20
JL UCZ Cu 1	12	222	68	0.1	0.42	91	90	2.5	0.48	150	100	10
JL UCZ Cu 2	5	217	66	0.1	0.75	30	30	5	0.15	150	100	10
JL UCZ Cu 3	5	320	155	0.001	0.55	25	25	5	0.45	75	75	10
JL UCZ Cu 4	5	158	110	0.1	0.48	50	25	3	0.42	100	75	7
JL UCZ Sulf 1	7.5	270	24	0.01	0.42	60	60	5	0.57	275	200	10
JL UCZ Sulf 2	7.5	270	67	0.01	0.55	60	60	5	0.44	250	250	15
JL UCZ Sulf 3	6.5	265	158	0.05	0.51	80	25	5	0.44	150	75	10

Domain	Rotation			Nugget	Structure 1 (m)				Structure 1 (m)			
	Dip	Dip Az	Pitch		Sill	Major	Semi Major	Minor	Sill	Major	Semi Major	Minor
JL UCZ Sulf 4	6.5	265	158	0.05	0.51	80	25	5	0.44	150	75	10
JL UCZ Sulf 5	5	270	12	0.05	0.52	150	125	5	0.43	200	175	10
JL LCZ Main	25	170	67	0.1	0.45	50	50	5	0.45	120	90	10
JL LCZ Upper	30	190	90	0.05	0.475	25	25	3	0.475	75	75	7
JL LCZ MASU	22	175	112	0.1	0.8	78	77.5	4	0.1	125	125	10

Table 14.12 Summary of Cu Variogram Parameters at Johnny Lee

14.5.2 Lowry Variography

Spatial analysis for the Lowry deposit was conducted on the key economic variable Cu by domain using variography. Modelled variograms were generated using Seequent's Leapfrog Geo™ v2021.2.5 software with the Edge extension. Variography was not calculated for secondary variables.

Variogram maps were generated for Cu to determine dominant orientations of continuity within each domain. Outputs of the variogram map analysis show the primary direction of spatial continuity aligns with the major geological direction of the mineralized body, typically down dip along each mineralized bed.

In some cases, no robust variogram could be modelled due to poor continuity and/or low data population for certain domains. Widening the search angle provided a better fit, which was applied to variography analysis, when required. Spatial continuity was evaluated only for Cu by domain as the primary economic variable of interest. Experimental variograms were calculated with acceptable structure in the LMCZ and LLCZ mineralized domains.

Variography for Lowry is summarized in Table 14.13.

Domain	Rotation			Nugget	Structure 1 (m)				Structure 1 (m)			
	Dip	Dip Az	Pitch		Sill	Major	Semi Major	Minor	Sill	Major	Semi Major	Minor
Bed_01	23	183	54	0.20	0.60	120	40	10	0.20	150	75	20
Bed_02	24	168	132	0.20	0.35	160	90	10	0.45	200	120	20
Bed_03	24	168	105	0.20	0.60	80	60	5	0.20	125	80	15
Bed_04	28	177	80	0.20	0.60	150	100	10	0.20	200	150	20
LLCZ_02	30	210	76	0.20	0.40	100	60	5	0.40	200	120	15
LLCZ_03	25	199	114	0.20	0.40	150	100	5	0.40	200	130	15
LLCZ_04	22	210	82	0.20	0.40	75	50	5	0.40	100	75	15

Domain	Rotation			Nugget	Structure 1 (m)				Structure 1 (m)			
	Dip	Dip Az	Pitch		Sill	Major	Semi Major	Minor	Sill	Major	Semi Major	Minor
LLCZ_07	17	243	55	0.20	0.40	100	75	5	0.40	150	150	15
LLCZ_West	30	210	76	0.20	0.40	150	130	5	0.40	250	200	15
LMCZ_01	32	182	102	0.10	0.20	100	80	5	0.70	150	100	15
LMCZ_06	31	194	100	0.10	0.70	100	75	10	0.20	150	100	20
LMCZ_South	28	188	110	0.10	0.60	150	75	10	0.30	200	100	20
LMCZ_West	32	182	102	0.10	0.20	100	80	5	0.70	230	100	15
Sulf_00	28	148	165	0.10	0.70	150	100	10	0.20	200	150	20
Sulf_01	26	147	95	0.10	0.60	125	125	10	0.30	200	150	20
Sulf_02	32	182	95	0.10	0.60	125	125	10	0.30	200	150	20
Sulf_03	28	206	170	0.10	0.70	150	125	15	0.20	200	150	30
Sulf_Rose	70	133	134	0.10	0.60	225	100	50	0.30	250	150	75
Sulf_South	34	167	165	0.10	0.70	300	200	30	0.20	350	250	60
Sulf_West	30	180	46	0.10	0.70	150	100	15	0.20	200	150	25

Table 14.13 Summary of Cu Variogram Parameters at Lowry

14.6 Block Model Parameters

14.6.1 Johnny Lee deposit

The resource block models for the JL UCZ and JL LCZ were combined into a single combined octree sub-blocked model and estimated using Leapfrog Geo™ version 2024.1.3. The block models are defined in the NAD83/UTM Zone 12 North coordinate system. The JL UCZ and JL LCZ are separate deposits and were treated and estimated independently. Although the JL UCZ and JL LCZ are contained within one resource block model, the Upper and Lower zones are typically exported into separate models with smaller file sizes for mine planning. A summary of the model geometry is presented in Table 14.14.

Johnny Lee UCZ and Johnny Lee LCZ Model Geometry			
	Northing	Easting	Elevation
Minimum Coordinates	506,000	5,180,000	795
Maximum Coordinates	507,500	5,181,500	1,875
Boundary Size	1,500	1,500	1,080
Size in Blocks	60	60	360
Parent Block Size	25 m	25 m	3 m
Minimum Sub-cell	1.5625 m	1.5625 m	1.5 m

Table 14.14 Johnny Lee Combined Block Model Geometry

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 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 Seequent's Leapfrog Geo™ v2021.2.5 software with the Edge extension. The extents were selected to encompass the Lowry drilling and the primary mineralized zones. The northern boundary of the model is truncated near a current SRA property boundary. The model was created in Universal Transverse Mercator (UTM) coordinate system, Zone 12N. A summary of the model geometry is presented in Table 14.15.

Lowry Model Geometry			
	Northing	Easting	Elevation
Minimum Coordinates	506,900	5,177,947	795
Maximum Coordinates	508,400	5,178,987	1,800
Boundary Size	1,500	1,040	1,005
Size in Blocks	75	52	201
Parent Block Size	20 m	20 m	5 m
Minimum Sub-cell	2.5 m	2.5 m	2.5 m

Table 14.15 Lowry Block Model Geometry

The 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 to provide an accurate volumetric representation of the modelled wireframes and match the proposed SMU, based on the proposed mine fleet for Johnny Lee.

It is the opinion of the QP for Mineral Resources that the Lowry deposit block model provides an acceptable representation of interpreted 3-D wireframes domaining the primary mineralized zones. Further, the parent block size was selected to allow for appropriate estimation of 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 domains were estimated using Ordinary Kriging (OK) with variable orientation surfaces defining the local search orientation. Variography and search neighborhoods were defined by the geometry of the orebody. The estimation was performed using a multi-pass search with a tighter initial pass that utilized closer spaced data. In areas where there was limited data that did not meet the minimum criteria for estimation and areas with no drilling, a scripted waste value (0.0005% Cu) was assigned to the block.

The first pass search is restricted to approximately 90% of the variogram range based on review of variograms for each domain, which estimates the majority of blocks. The second pass search is at the sill of the variography or twice the initial search distances to estimate remaining peripheral blocks that typically contain reduced data support. Due to undulations in the mineralization geometry for the JL UCZ and JL LCZ, dynamic anisotropy is employed to control searches and improve the local estimate.

The search parameters for Cu are provided in Table 14.16.

Domain	Pass 1						Pass 2					
	Ellipse Range			Samples			Ellipse Range			Samples		
	Major	S-Major	Minor	Min	Max	Per Hole	Major	S-Major	Minor	Min	Max	Per Hole
JL UCZ HG Gu	100	75	15	5	10	2	200	150	30	3	10	N/A
JL UCZ Cu 1	100	75	10	5	10	2	200	150	20	3	10	N/A
JL UCZ Cu 2	100	75	10	5	10	2	200	150	20	3	10	N/A
JL UCZ Cu 3	50	50	10	5	10	2	100	100	20	3	10	N/A
JL UCZ Cu 4	75	50	10	5	10	2	150	100	20	3	10	N/A
JL UCZ Sulf 1	200	150	10	5	10	2	275	200	20	3	10	N/A
JL UCZ Sulf 2	200	200	15	5	10	2	250	250	30	3	10	N/A
JL UCZ Sulf 3	100	50	10	5	10	2	150	75	20	3	10	N/A
JL UCZ Sulf 4	100	50	10	5	10	2	150	75	20	3	10	N/A
JL UCZ Sulf 5	150	125	10	5	10	2	200	175	20	3	10	N/A
JL LCZ Main	75	75	25	4	6	2	100	100	50	1	6	N/A
JL LCZ Upper	50	50	25	4	6	2	75	75	50	1	6	N/A
JL LCZ MASU	75	75	25	4	6	2	125	125	50	1	6	N/A

Table 14.16 Search parameters for Cu by Domain in the Johnny Lee deposit

SG estimation was conducted via IDW2 interpolation with separate 1.5-m composites broken on domains in the JL UCZ and JL LCZ. A mean value based on composite data within each domain was applied to unestimated blocks.

The search parameters for bulk density estimation are provided in Table 14.17.

Domain	Pass 1						Pass 2					
	Ellipse Range			Samples			Ellipse Range			Samples		
	Major	S-Major	Minor	Min	Max	Per Hole	Major	S-Major	Minor	Min	Max	Per Hole
JL UCZ HG Cu	100	75	15	5	10	2	200	150	30	3	10	N/A
JL UCZ Cu 1	100	75	10	5	10	2	200	150	20	3	10	N/A
JL UCZ Cu 2	100	75	10	5	10	2	200	150	20	3	10	N/A
JL UCZ Sulf 1	200	150	10	5	10	2	275	200	20	3	10	N/A
JL UCZ Sulf 2	200	200	15	5	10	2	250	250	30	3	10	N/A
JL UCZ Sulf 3	100	50	10	5	10	2	150	75	20	3	10	N/A
JL UCZ Sulf 4	100	50	10	5	10	2	150	75	20	3	10	N/A
JL UCZ Sulf 5	150	125	10	5	10	2	200	175	20	3	10	N/A
JL LCZ MASU	75	75	25	4	6	2	125	125	50	1	6	N/A
JL LCZ FW Congl	75	75	25	4	6	2	125	125	50	1	6	N/A
JL LCZ HW Shale	75	75	25	4	6	2	125	125	50	1	6	N/A

Table 14.17 **Search parameters for SG by Domain in the Johnny Lee deposit**

The estimation of recovery for the JL UCZ via kriging was not suitable due to insufficient data support. In order to convert this data into the 2019 3-D geometallurgical model, various interpolation protocols were trialled, including: Radial Base Function isotropic, Radial Base Function anisotropic, IDW2 and 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). For the JL UCZ, the estimation of recovery in concentrate was extended from the 2019 geometallurgical recovery model developed by SRA into the 2025 block model using a NN interpolation of the 2019 block centroids.

For the JL LCZ, recovery is assigned a blanket value of 95.2%. It was determined that estimation of block recovery was not required due to the low variability of JL LCZ recovery testing and consistency in this variable across the JL LCZ.

14.7.2 Lowry deposit

Resource estimation was performed for the Lowry deposit by SRA staff using Seequent's Leapfrog Geo™ v2021.2.5 software with the Edge extension. The focus of estimation was on Cu as the key economic variable of interest. OK was selected as the preferred method for block estimation 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. To estimate remaining blocks, IDW3 was selected as the preferred method for moderately to poorly informed blocks. IDW3 provides a reasonable representation of quality variables while not overly smoothing or biasing. The search pass number is stored in the "Cu_Pass" variable to aid in assessing block confidence and resource classification. In areas where there was limited data that did not meet the minimum criteria for estimation and areas with no drilling, a scripted waste value (0.0005% Cu) was assigned to the block. The search parameters for Cu are provided in Table 14.18.

Domain	Search Orientation			Pass 1						Pass 2					
				Ellipse Range			Samples			Ellipse Range			Samples		
	Dip	Dip Az	Pitch	Major	S-Major	Minor	Min	Max	Per Hole	Major	S-Major	Minor	Min	Max	Per Hole
Bed_01	23	183	54	150	75	20	3	10	2						
Bed_02	24	168	132	200	120	20	3	10	2						
Bed_03	24	168	105	125	80	15	3	10	2						
Bed_04	28	177	80	105	70	10	3	10	2	200	150	20	2	10	2
LCZ_02	30	210	76	200	120	15	3	10	2						
LCZ_03	25	199	114	200	130	15	3	10	2						
LCZ_04	22	210	82	100	75	15	3	10	2						
LCZ_07	17	243	55	150	150	15	3	10	2						
LCZ_West	Variable Orientation			130	110	10	3	10	2	250	200	15	2	10	2
MCZ_01	32	182	102	150	100	15	3	10	2						
MCZ_06	31	194	100	150	100	20	3	10	2						
MCZ_South	28	188	110	115	55	10	3	10	2	200	100	20	2	10	2
MCZ_West	Variable Orientation			150	70	10	3	10	2	230	100	15	2	10	2
Sulf_00	28	148	165	200	150	20	3	10	2						
Sulf_01	26	147	95	200	150	20	3	10	2						
Sulf_02	32	182	95	200	150	20	3	10	2						
Sulf_03	28	206	170	200	150	30	3	10	2						
Sulf_Rose	70	133	134	250	150	75	3	10	2						
Sulf_South	34	167	165	210	140	25	3	10	2	350	250	60	2	10	2
Sulf_West	30	180	46	200	150	25	3	10	2						

Table 14.18 **Search Parameters for Cu by Domain in the Lowry deposit**

SG was estimated in the block model using an IDW3 estimator with varying search parameters by domain (Table 14.19). A mean value based on composite data within each domain was applied to unestimated blocks. SG domains used modelled sulphide shapes within the Lowry deposit.

Domain	Search Orientation			Ellipse Range			Samples		
	Dip	Dip Az	Pitch	Major	S-Major	Minor	Min	Max	Per Hole
Sulf_00	27	150	162	180	100	30	2	16	2
Sulf_01	27	151	136	180	140	40	4	16	4
Sulf_02	32	182	130	160	100	30	4	16	4
Sulf_03	30	210	68	200	160	40	4	16	4
Sulf_04	38	170	120	220	160	40	4	16	2
Sulf_05	32	174	120	300	200	60	2	16	2
Sulf_06	31	178	116	300	200	80	2	16	2
Sulf_07	28	170	96	160	120	30	2	16	2
Sulf_Rose	72	132	128	180	120	30	4	16	4

Table 14.19 ***Search Parameters for SG by Domain in the Lowry deposit***

14.8 Model Validation

14.8.1 Johnny Lee deposit

14.8.1.1 Visual Validation

Cross-sections through the JL UCZ and JL LCZ were reviewed for visual validation of block model grades. Examples of these cross-sections are presented below in Figure 14.16 to Figure 14.19 that illustrate the block model Cu grades and SG values provide an acceptable representation of the composite data.

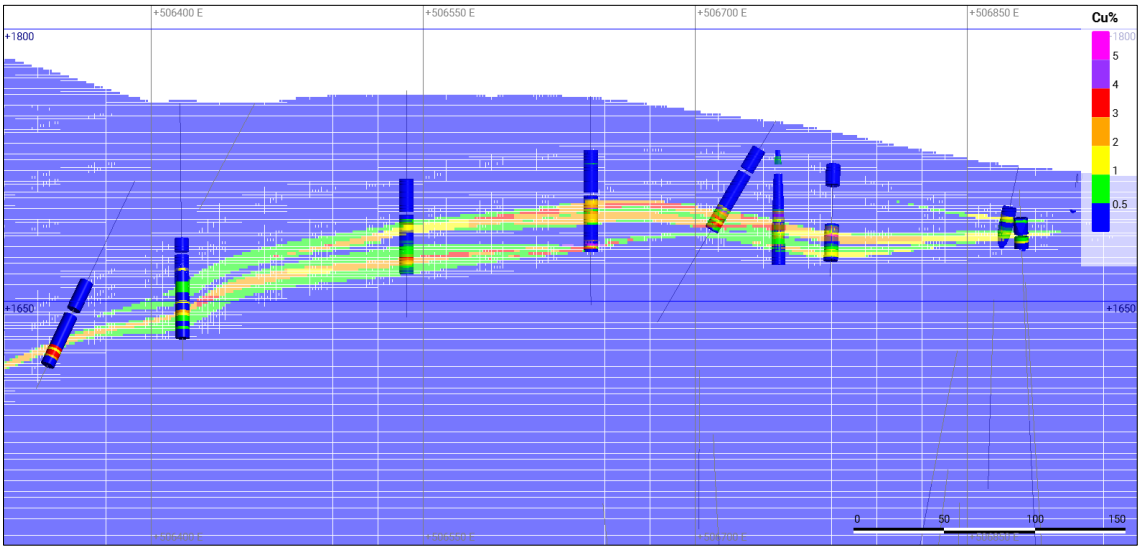


Figure 14.16 JL UCZ Section N5180805 (±17.5 m) Looking North at Cu Block Model versus Cu Composites

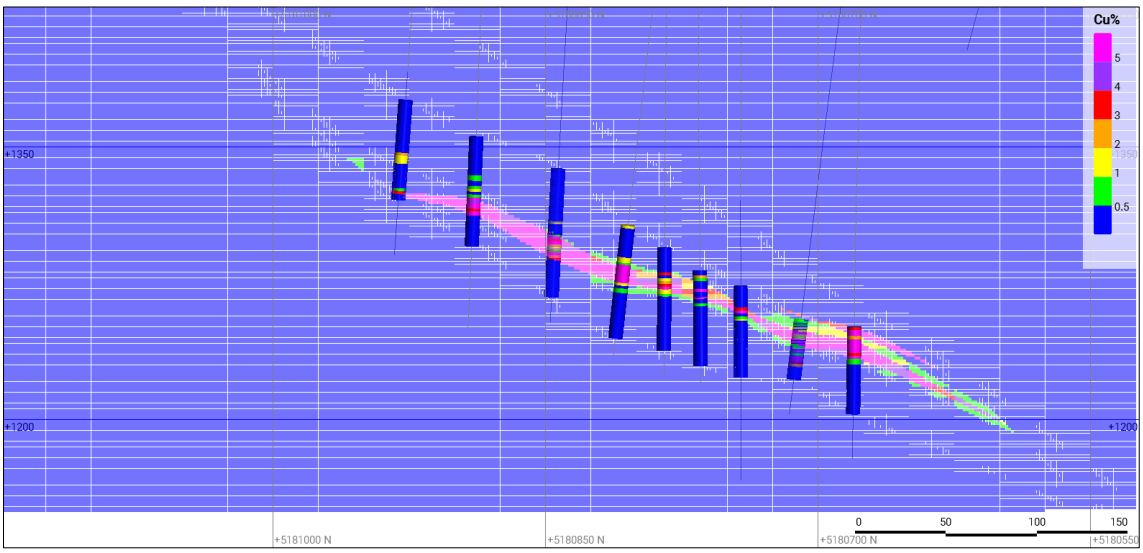


Figure 14.17 JL LCZ Section E507220 (±22.5 m) Looking East at Cu Block Model versus Cu Composites

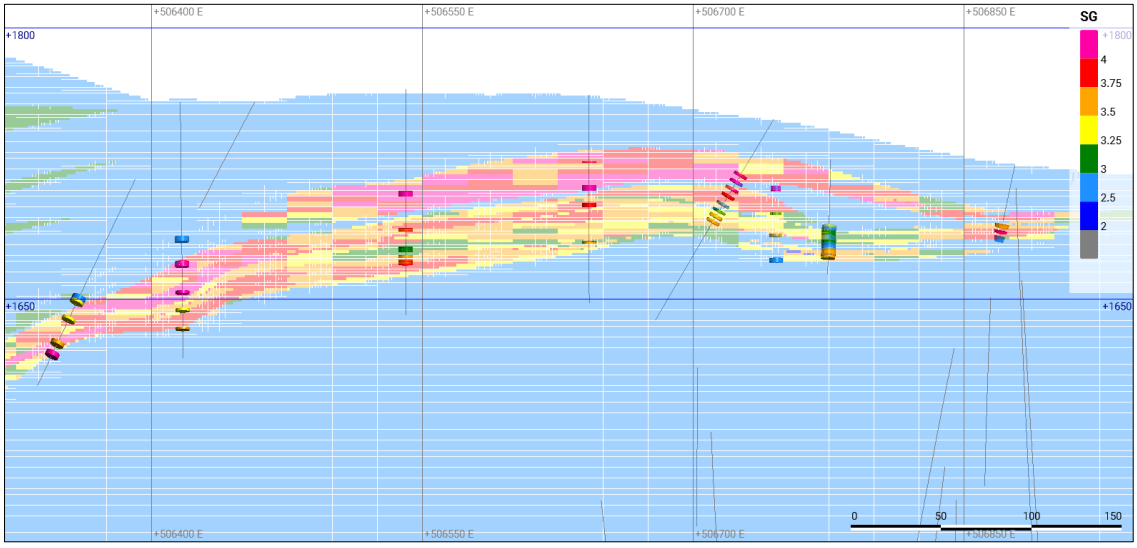


Figure 14.18 JL UCZ Section N5180805 (±17.5 m) Looking North at SG Block Model versus SG Composites

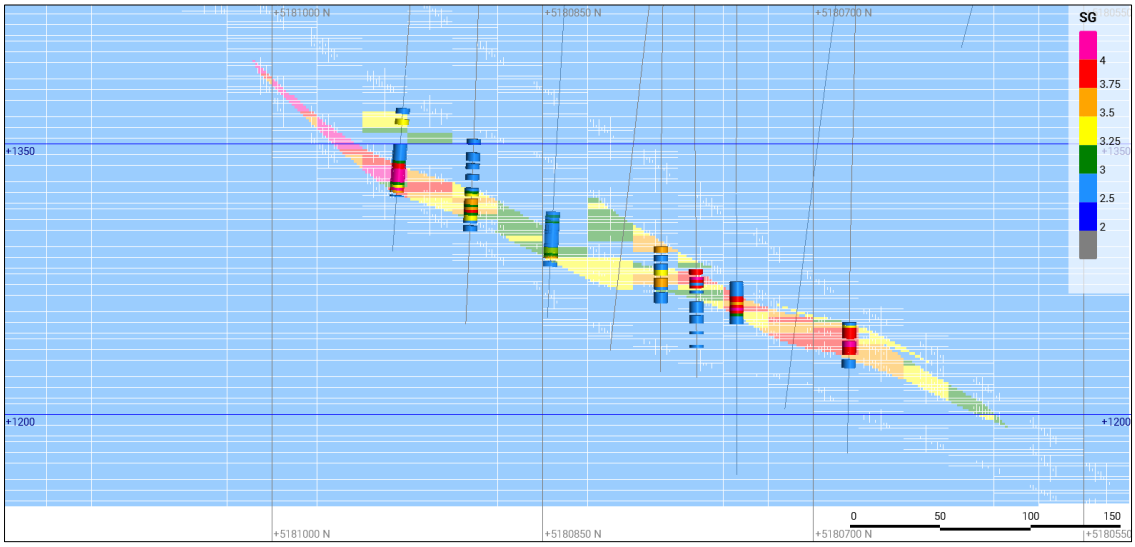


Figure 14.19 JL LCZ Section E507220 (±22.5 m) Looking East at SG Block Model versus SG Composites

14.8.1.2 Statistical Validation

Composites and block grades were compared to determine differences in the descriptive statistics of the data by domain through the estimation process (Figure 14.24 and Figure 14.20). The statistical comparisons did not identify obvious biases or significant changes in mean values due to the volume-variance effect.

Domain	Mean Composite Grade, Capped (Cu%)	Mean Block Grade (Cu %)	Mean Grade IDW (Cu %)	Mean Grade NN (Cu %)	Percent Difference Blocks to Comps (%)	Percent Difference Blocks to NN Estimate (%)
JL UCZ HG Cu	2.53	2.47	2.49	2.43	-2.4%	1.7%
JL UCZ Cu 1	0.69	0.66	0.66	0.66	-4.9%	0.3%
JL UCZ Cu 2	0.66	0.65	0.65	0.65	-1.6%	-0.2%
JL LCZ Main	7.07	6.32	6.64	6.95	-11.9%	-10.0%
JL LCZ Upper	3.95	4.24	4.43	3.88	6.8%	8.3%
JL LCZ MASU	0.23	0.21	0.21	0.21	-9.3%	2.6%

Table 14.20 Summary Statistics for Cu Composite Grade (Capped) and Block Grade by Domain

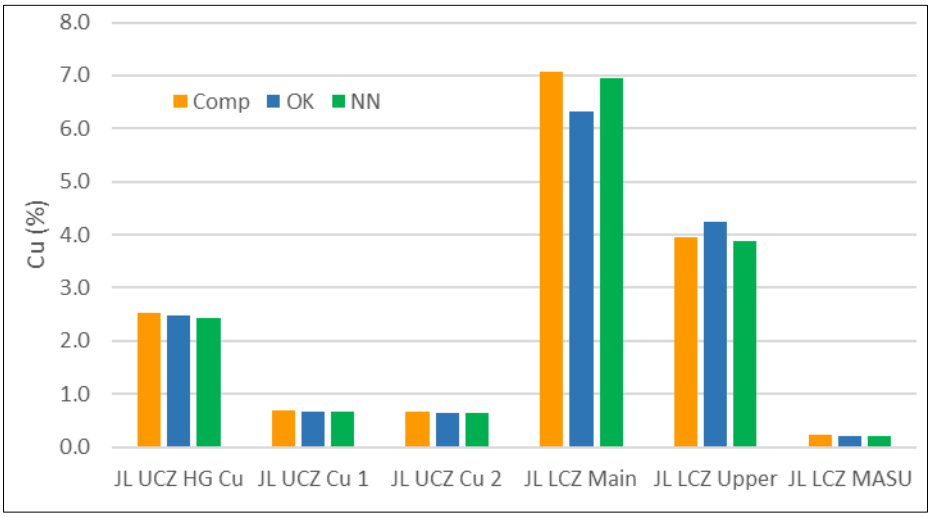


Figure 14.20 Comparison of JL Mean Cu Grades for Composites (orange), OK Blocks (blue), and NN Blocks (Green)

14.8.1.3 Swath Plots

Swath plots were calculated in the X, Y, and Z directions to compare estimation values across the block model. This provides insight into potential biases in the estimation methodology. Swath plots examples are presented in Figure 14.21 and Figure 14.22. In each plot, the red line is the OK estimate, the green lines are the NN estimate (with both 1.5 m and 3.0 m composites), the blue line is the IDW2 estimate, and the red bars are the block volume. It is the opinion of the QP for Mineral Resources that the Lowry model validates well against informing data and is suitable for use in the determination of Mineral Resources as per CIM guidelines and standards.

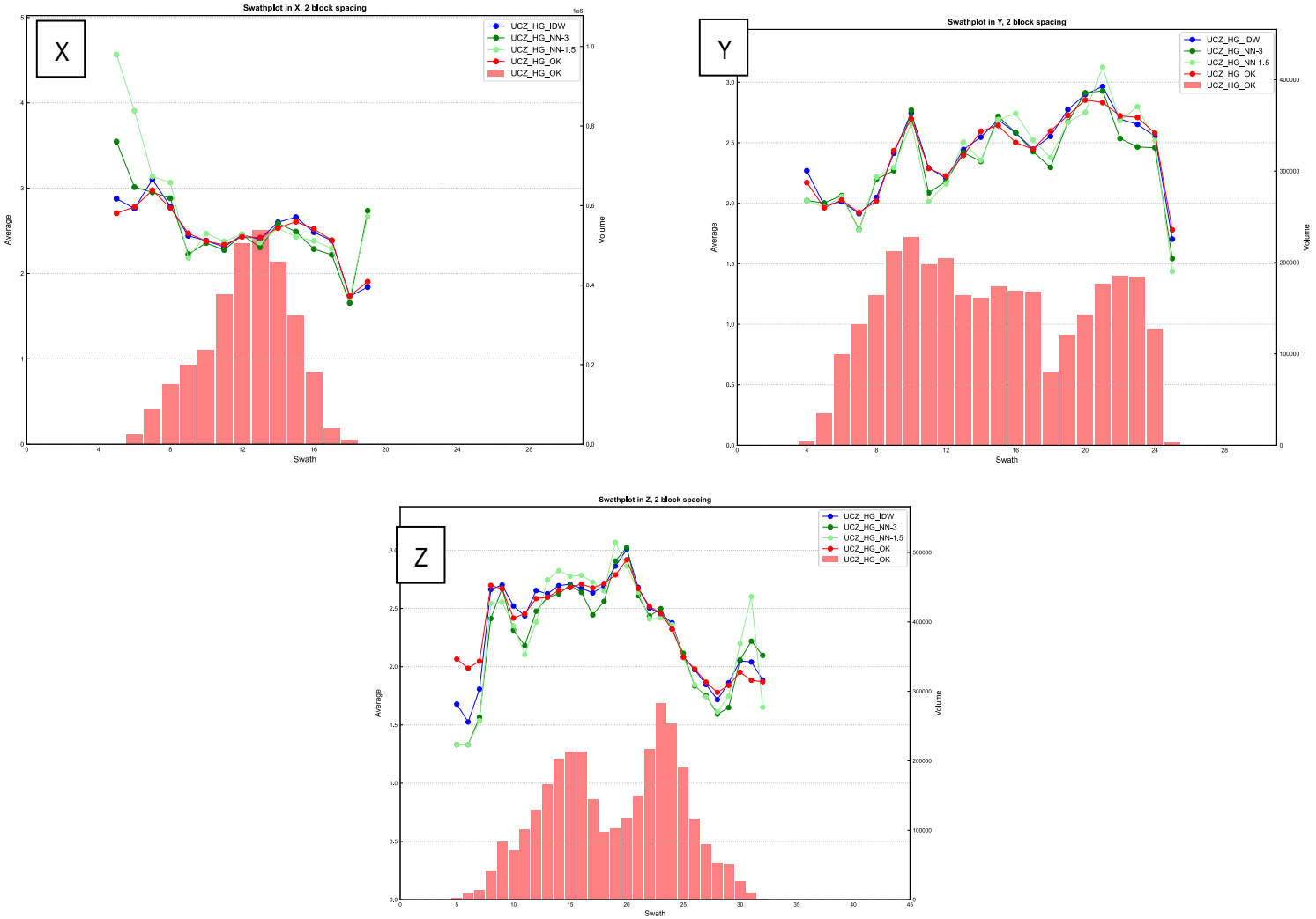


Figure 14.21 JL UCZ HG Cu Swath Plots

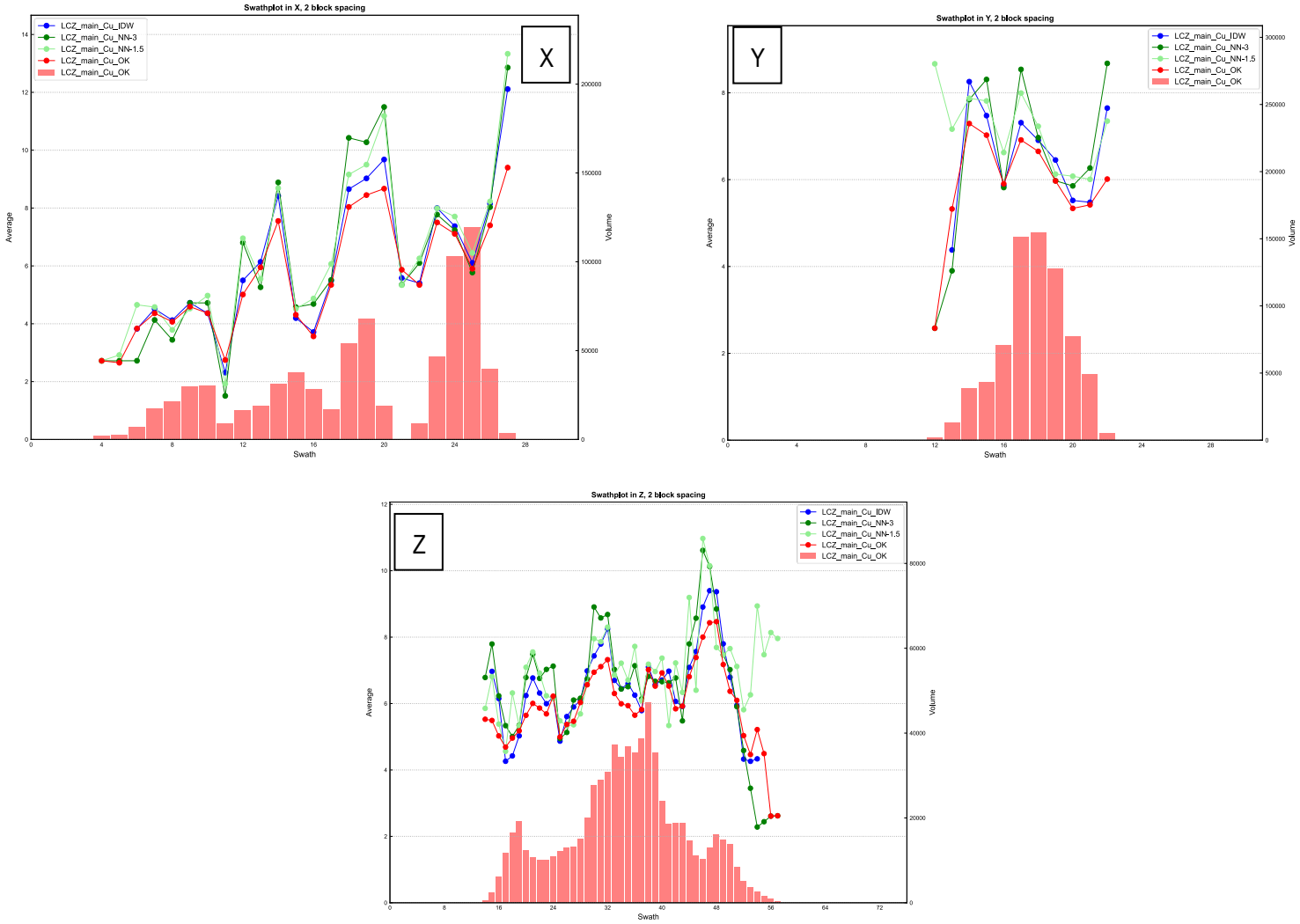


Figure 14.22 JL LCZ Main Swath Plots

14.8.2 Lowry deposit

The 2023 Lowry block model was validated using multiple techniques, including visual validation of block grades against composited drilling data, statistical comparison of composites and block grades through swath plots, and mean comparison by domain.

14.8.2.1 Visual Validation

Three-dimensional and cross-sectional visual comparisons were made between estimated block grades and drilling grades. Visual validation was performed using Seequent's Leapfrog Geo™ v2021.2.5 software. Estimated variables were compared with blocks showing an acceptable representation of drilling grades. An example of a visual validation cross-section is provided in Figure 14.23.

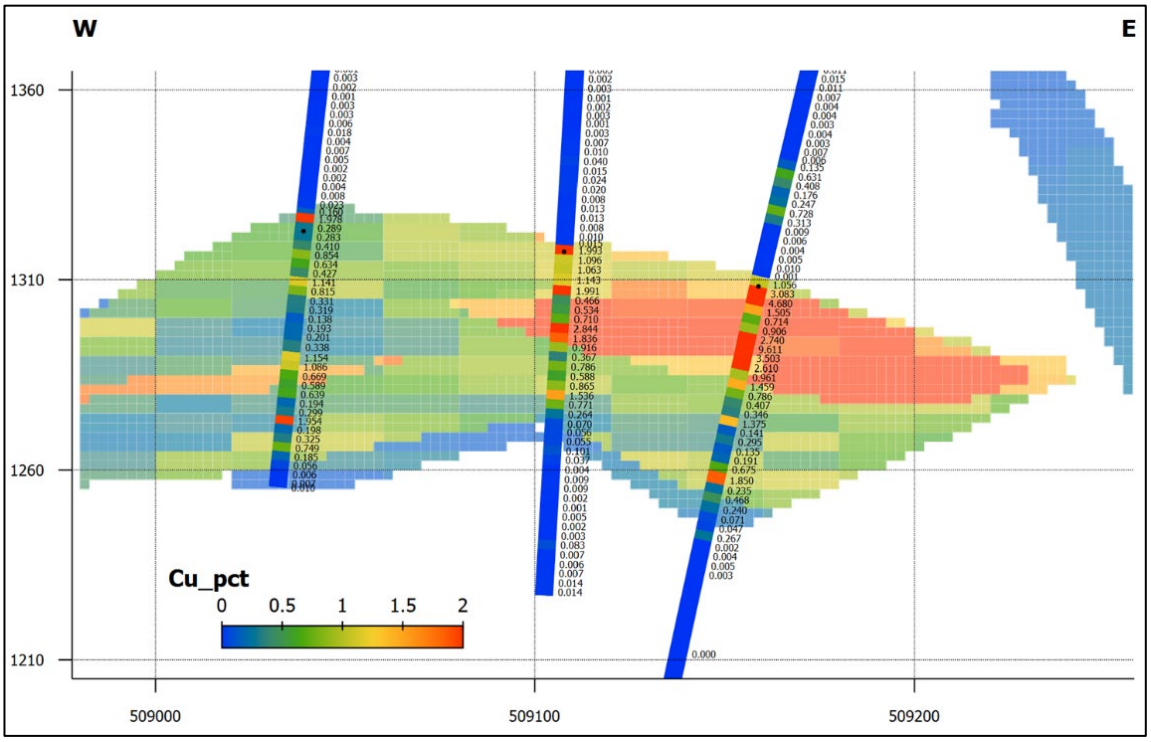


Figure 14.23 Cross-section N5179640 Comparing Cu Composites to Block Grades

14.8.2.2 Statistical Validation

Composites and block grades were compared to determine differences in the descriptive statistics of the data by domain through the estimation process (Figure 14.24 and Figure 14.25). The statistical comparisons did not identify obvious biases or significant changes in mean values due to any material volume-variance effects.

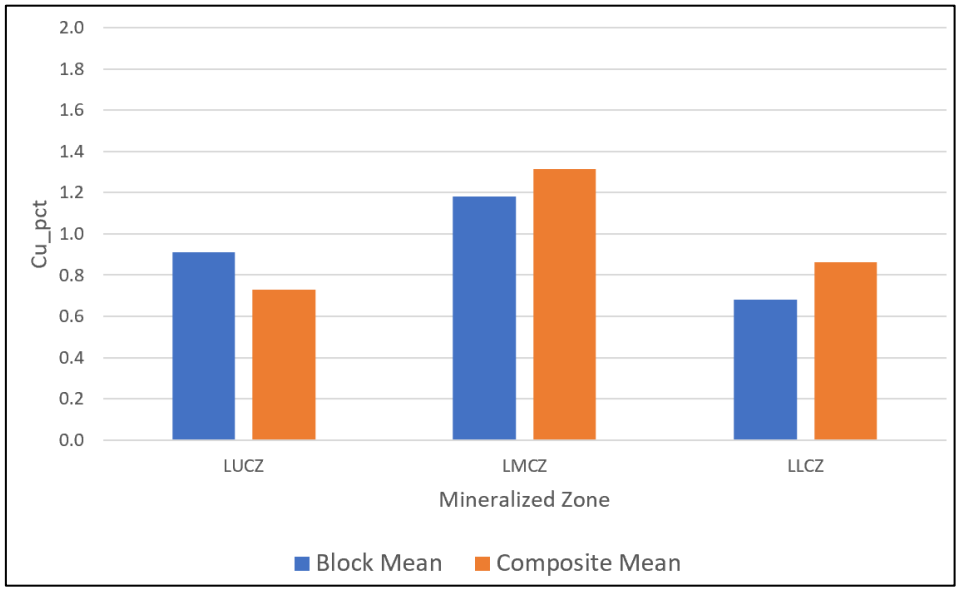


Figure 14.24 Comparison of Lowry Mean Cu Grades by Mineralized Zone for Composites (orange) and Blocks (blue)

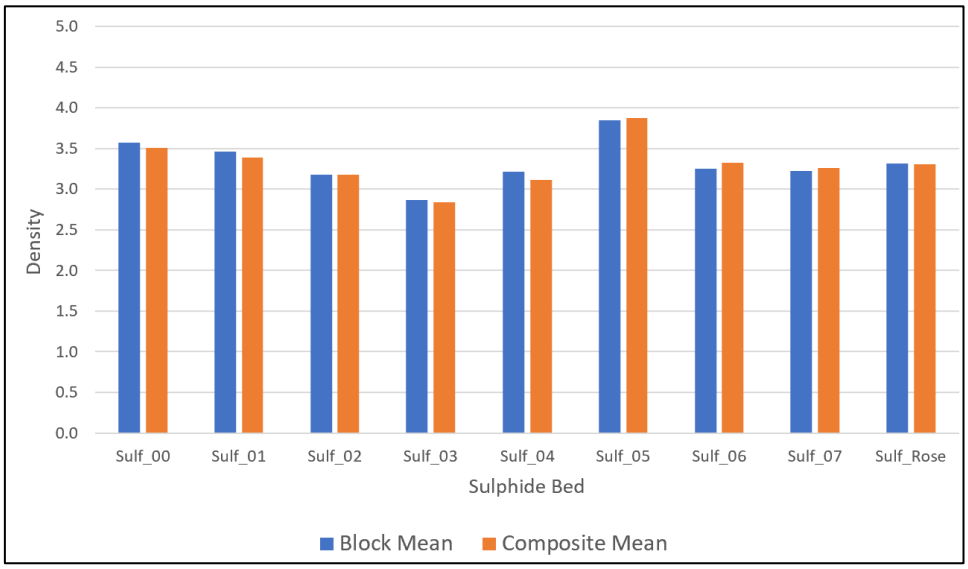


Figure 14.25 Comparison of Lowry Mean SG between Composites (orange) and Blocks (blue)

14.8.2.3 Swath Plots

Swath plots were calculated in the X, Y, and Z directions to compare estimation values across the block model. This provides insight into potential biases in the estimation methodology. Swath plots are presented in Figure 14.26. In each plot, the blue line is the OK estimate, the green line is the NN estimate, the red line is the IDW3 estimate, and the green bars are the block volume. It is the opinion of the QP for Mineral Resources that the Lowry model validates well against informing data and is suitable for use in the determination of Mineral Resources as per CIM guidelines and standards.

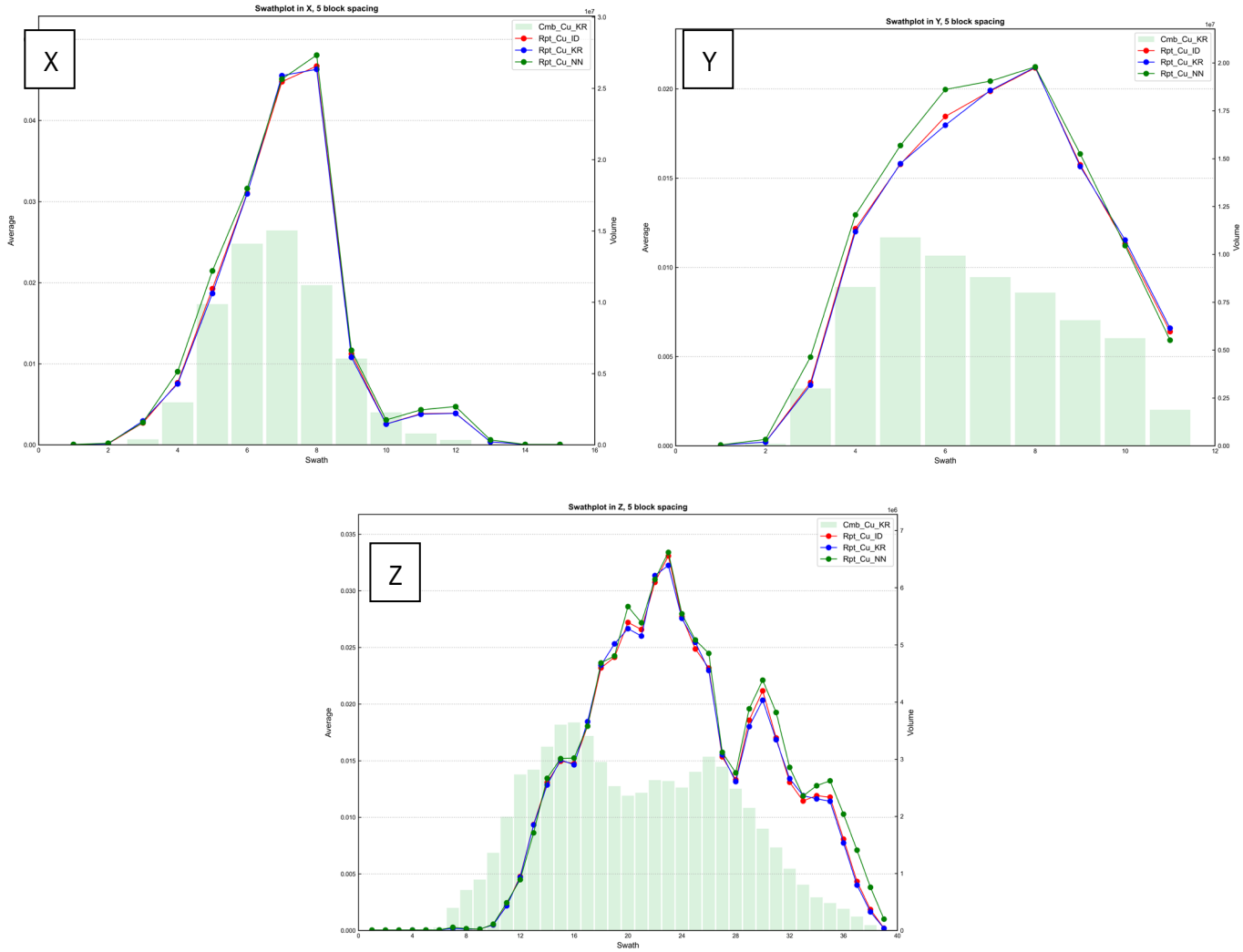


Figure 14.26 Lowry Swath Plots

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 JL UCZ and JL 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. Certain secondary variables were estimated into blocks for mine planning purposes, but are not reported as Mineral Resources and should not be considered to have the same confidence as the key economic variable of Cu. Figure 14.27 and Figure 14.28 show the classification results for the LMCZ and LLCZ, respectively.

Measured Mineral Resource classification is assigned to blocks based upon high confidence in geology, Cu grade, SG continuity, acceptable QA/QC of fundamental data, increased confidence in estimation quality based on greater than 5 drillholes, kriging efficiency values greater than 0.5, drill spacing less than 50 m, and reasonable recovery data. Measured blocks in the JL UCZ (HG Cu, Cu 1, and Cu 2 zones) and JL LCZ (Main zone) are concentrated in zones of tight drill spacing.

Indicated Mineral Resource classification is assigned to blocks based on moderate to high confidence in geology, Cu grade, SG continuity, acceptable QA/QC of fundamental data, moderate confidence in estimation quality based on greater than 5 drillholes, drill spacing of approximately 100 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, SG continuity, acceptable QA/QC, lower confidence in estimation quality based on kriging efficiency, and drill spacing greater than 50 m with a lower 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 JL UCZ Sulf 1, JL LCZ Upper, and JL LCZ MASU Halo zones have been classified as Inferred Mineral Resources due to lower confidence grade continuity. Thin portions at the edges of the wireframe with grade-thickness less than the approximate cut-off grade were excluded. The McGuire parcel, JL UCZ Sulf 2 through Sulf 4, and JL UCZ Cu 3 and Cu 4 were excluded from resources and coded as exploration potential.

It is the opinion of the QP for Mineral Resources that the classification for the Johnny Lee deposit Mineral Resource is reasonable for the type of mineralization and deposit.

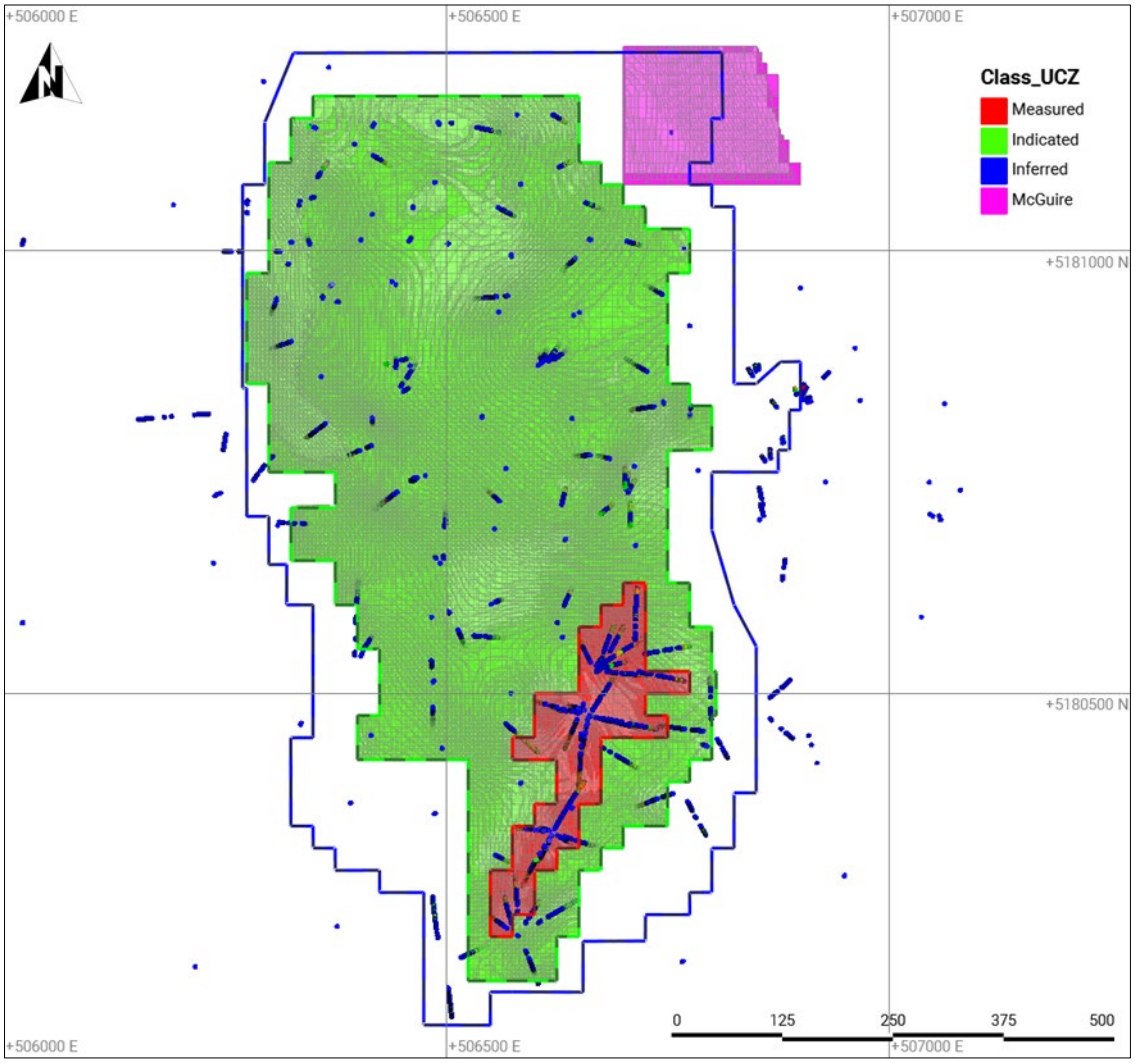


Figure 14.27 Plan View of the JL UCZ showing Mineral Resource Classification

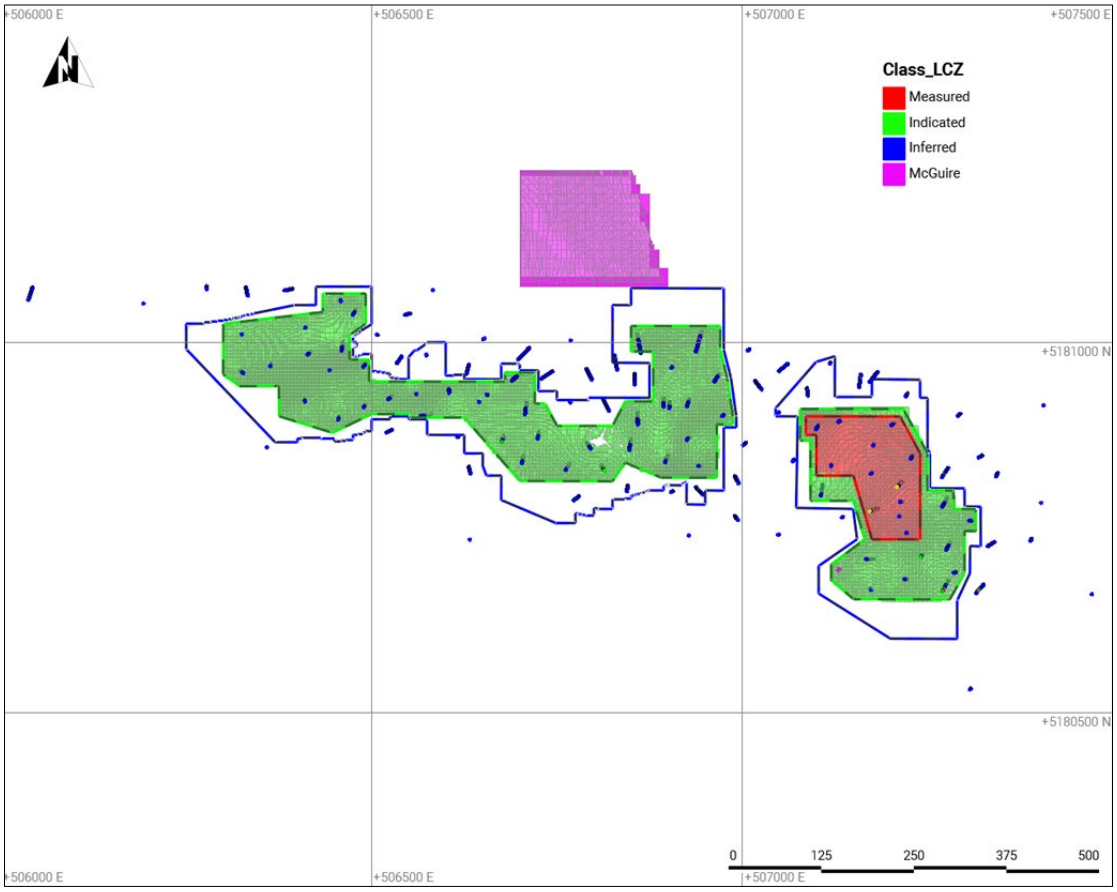


Figure 14.28 Plan View of the JL LCZ showing Mineral Resource Classification

14.9.2 Lowry Deposit Resource 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, slope or regression, average distance to samples, and estimation run pass). Blocks within the LUCZ, LMCZ and LLCZ have been categorized consistent with NI 43-101 and the CIM definitions and guidelines (CIM, 2014).

A combination of block scripting and manual smoothing with polylines was used to apply the appropriate block classification of Mineral Resource categories. Certain secondary variables were estimated into blocks for mine planning purposes but are not reported as Mineral Resources and should not be considered to have the same confidence as the key economic variable of Cu. Figure 14.29 and Figure 14.30 show the classification results for the LMCZ and LLCZ, respectively.

Measured Mineral Resource classification is assigned to blocks based upon high confidence in geology, Cu grade continuity, acceptable QA/QC of fundamental data, confidence in estimation quality based on slope of regression or kriging efficiency values greater than 0.75, average spacing less than 50 m, with at least eight samples used in the block estimate. Measured blocks in the LMCZ and LLCZ are in zones of tight drill spacing.

Indicated Mineral Resource classification is assigned to blocks based on moderate to high confidence in geology, Cu grade continuity, acceptable QA/QC of fundamental data, moderate confidence in estimation quality based on kriging efficiency between 0.35 and 0.75, average sample spacing less than 100 m, with at least 5 samples used in the block estimate. Indicated blocks constitute most of the Mineral Resources.

Inferred Mineral Resource classification is assigned to blocks based on moderate confidence in geology, limited Cu grade continuity, acceptable QA/QC, lower confidence in estimation quality based on kriging efficiency, and average sample spacing greater than 100 m. Inferred blocks represent mineralized material within a modelled volume with limited data supporting the continuity. The modelled volume has been defined by drilling and structural boundaries in most directions. The LUCZ has been classified as Inferred Mineral Resources due to the thin beds with erratic grade and short-range continuity. The McGuire parcel was excluded from resources.

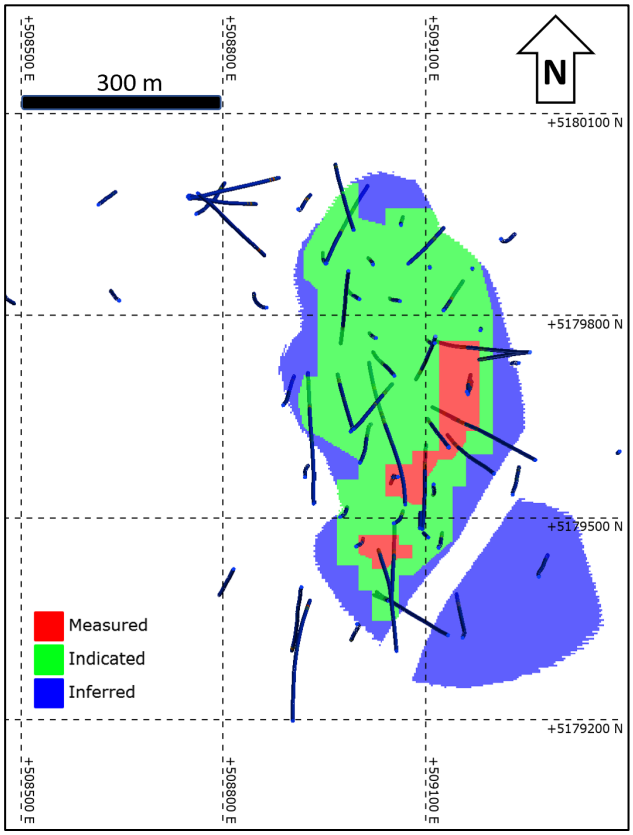


Figure 14.29 Plan View of the LMCZ showing Mineral Resource Classification

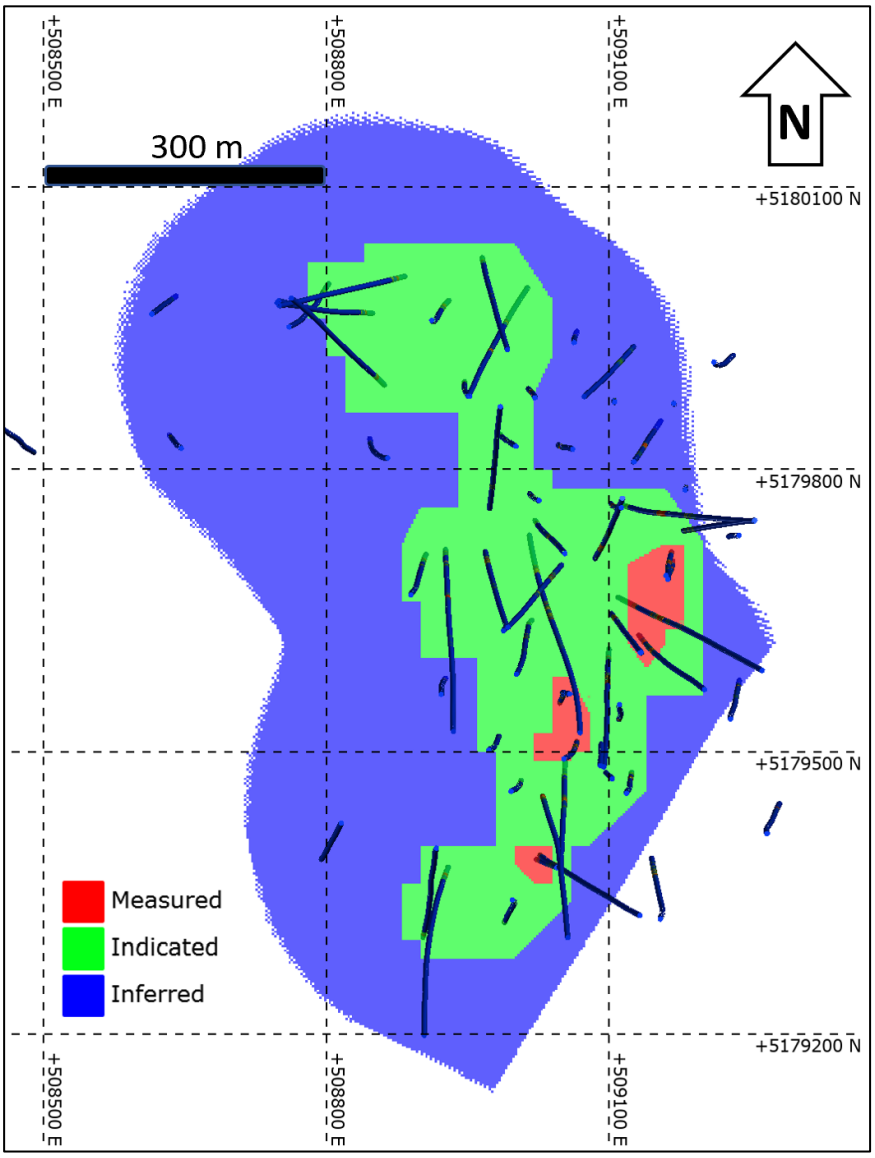


Figure 14.30 Plan View of the LLCZ showing Mineral Resource Classification

14.10 Demonstration of Potential for Economic Extraction

As per CIM definitions of Mineral Resources, material must demonstrate reasonable prospects for eventual economic extraction (RPEEE). For the Black Butte Copper Project, CoGs have been applied to each deposit to satisfy this implication. Additionally, grade-thickness values were considered to eliminate areas that lacked mineable widths of mineralization.

14.10.1 Johnny Lee and Lowry Cut-Off Value

SRK applied a Net Smelter Return (NSR) Cut-Off Value (CoV) based on the scoping study cost estimates that were used as the basis of the PFS mine design and schedule. The CoV accounts for operational costs based on the assumed mining method proposed, assumed mill processing costs, calculated metallurgical recovery of Cu, and market-driven Cu pricing. The following technical and economic parameters are assumed and accounted for in the determination of the NSR CoV:

- Metallurgical recoveries for Cu are estimated as a block model variable for the JL UCZ and assigned at the mean of 95.2% Cu recovery for the JL LCZ. Metallurgical recovery for Cu has been assigned to the Lowry deposits: 91% for both LUCZ and LMCZ and 94% for LLCZ.
- Mill Processing Costs are based on dry concentrate and include costs for treatment (US$79.00 per dry material tonne (dmt) concentrate), refining (US$0.079 per pound of payable Cu), freight (US$167.00 per dmt concentrate), and a 2% NSR royalty.
- A long-term Cu price of US$4.70/lb (US$10,360/t) is based on current market consensus forecast for upside Cu pricing.

Using these metrics, a CoV of US$45.00/t NSR, which is approximately equal to 0.52% recoverable Cu, is used to report the Johnny Lee Mineral Resources. For the Lowry deposit, Mineral Resources are reported using an economic NSR cut-off value of US$110.00/t, approximately equivalent to 1.28% recoverable Cu.

The previous Johnny Lee deposit modeling and estimation was completed in 2019. For this report, the Johnny Lee deposit was updated to reflect current geological understanding with an estimation effective date of 5 November 2025.

The Lowry deposit modeling and estimation was completed on 10 February 2023. For the purposes of this Technical Report, the only work completed subsequent to that date was the calculation of NSR values in the Lowry block model as of 5 November 2025. No additional drilling or metallurgical testwork was conducted at Lowry since the previous 2023 estimate. No other modifications, updates, or changes were made between the previous 2023 Lowry block model and the current 2025 model that materially affect the resource estimate.

14.11 Mineral Resource Statement

Mineral Resources for the Johnny Lee and Lowry Deposits have been calculated and reported using an economic CoV, as estimated in the resource block model. The Mineral Resource statements are supported by drilling, analyses, geological modelling, and metallurgical studies that determined updated recoveries. The deposits have been classified into Measure, Indicated, and Inferred Mineral resources, as per CIM definitions.

Mineral Resource summaries for the Johnny Lee deposit and the Lowry deposit are presented in Table 14.21 and Table 14.22, respectively.

Mineral Resource Category	Quantity (Mt)	Cu (%)	Total Metal (kt)
Measured + Indicated (M+I) Mineral Resources			
JL UCZ			
Measured	2.3	2.0	46
Indicated	13.8	1.9	261
Sub-Total M+I	**16.1**	**1.9**	**307**
JL LCZ			
Measured	0.5	6.5	34
Indicated	2.3	5.1	121
Sub-Total M+I	**2.9**	**5.4**	**154**
Combined JL UCZ + JL LCZ			
Measured	2.8	2.8	80
Indicated	16.1	2.4	382
Total M+I	**18.9**	**2.4**	**462**
Inferred Mineral Resources			
JL UCZ	2.6	1.5	38
JL LCZ	0.8	3.2	26
Total Inferred	**3.4**	**1.9**	**64**

Source: SRK, 2025

Note:

- *The effective date of this Mineral Resource is 5 November 2025.*
- *The Mineral Resources in this estimate were prepared in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Standards on Mineral Resources and Reserves, Definitions and Guidelines (CIM, 2014) prepared by the CIM Standing Committee on Reserve Definitions and adopted by CIM Council.*
- *Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. Inferred Mineral Resources have a high degree of uncertainty as to their economic and technical feasibility. There is no certainty that all or any part of an Inferred Mineral Resources will be converted to Measured or Indicated Mineral Resources in the future.*
- *To demonstrate reasonable prospects for eventual economic extraction, Mineral Resources are reported using an economic NSR cut-off value of US$45.00/t, which is approximately equivalent to 0.52% recoverable Cu, based on long-term Cu price assumptions of US$4.70/lb and variable recovery and mill processing cost assumptions.*
- *Variable metallurgical recovery has been estimated in the JL UCZ on a block basis with a consistent 95.2% Cu recovery applied to the JL LCZ.*
- *There are no known legal, political, environmental, or other risks that could materially affect the potential future development of the Mineral Resources. All Mineral Resources are located within land currently under control or lease to Sandfire Resources America, Inc.*
- *All quantities are rounded to the appropriate number of significant figures to reflect the relative accuracy of the estimate; consequently, sums may not add up due to rounding. Cu assay values were capped where appropriate, and*
- *The Mineral Resources for the Johnny Lee Upper Copper Zone (JL UCZ) and Johnny Lee Lower Copper Zone (JL LCZ) were reviewed and approved by Berkley Tracy, P.Geo (PGO#3024) of SRK Consulting (U.S.), Inc., a Qualified Person as defined by CIM.*

Table 14.21 Black Butte Copper Project Mineral Resource Estimate for the Johnny Lee Deposit at NSR Cut-Off Value of US$45.00/t, as of November 5, 2025 – SRK Consulting (U.S.), Inc.

Mineral Resource Category	Quantity (Mt)	Cu (%)	Total Metal (kt)
Measured + Indicated (M+I) Mineral Resources			
Lowry MCZ			
Measured	1.2	2.6	30
Indicated	3.6	2.4	86
Sub-Total M+I	**4.8**	**2.4**	**116**
Lowry LCZ			
Measured	0.1	1.8	1
Indicated	1.7	2.2	37
Sub-Total M+I	**1.8**	**2.1**	**38**
Combined Lowry UCZ + MCZ + LCZ			
Measured	1.2	2.5	31
Indicated	5.3	2.3	123
Total M&I	**6.6**	**2.4**	**154**
Inferred Mineral Resources			
Lowry UCZ	1.3	2.3	30
Lowry MCZ	0.9	2.1	18
Lowry LCZ	0.6	1.7	10
Total Inferred	**2.8**	**2.1**	**58**

Source: SRK, 2025

Note:

- *The effective date of this Mineral Resource is 5 November 2025.*
- *The Mineral Resources in this estimate were prepared in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Standards on Mineral Resources and Reserves, Definitions and Guidelines (CIM, 2014) prepared by the CIM Standing Committee on Reserve Definitions and adopted by CIM Council.*
- *Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. Inferred Mineral Resources have a high degree of uncertainty as to their economic and technical feasibility. There is no certainty that all or any part of an Inferred Mineral Resources will be converted to Measured or Indicated Mineral Resources in the future.*
- *To demonstrate reasonable prospects for eventual economic extraction, Mineral Resources are reported using an economic NSR cut-off value of US$110.00/t, which is approximately equivalent to 1.28% recoverable Cu, based on long-term Cu price assumptions of US$4.70/lb and variable recovery and mill processing cost assumptions.*
- *Metallurgical recovery for Cu has been assigned to the Lowry deposits: 91% for both LUCZ and LMCZ and 94% for LLCZ.*
- *There are no known legal, political, environmental, or other risks that could materially affect the potential future development of the Mineral Resources. All Mineral Resources are located within land currently under control or lease to Sandfire Resources America, Inc.*
- *All quantities are rounded to the appropriate number of significant figures to reflect the relative accuracy of the estimate; consequently, sums may not add up due to rounding. Cu assay values were capped where appropriate, and*
- *The Mineral Resources for the Lowry Upper Copper Zone (LUCZ), Lowry Middle Copper Zone (LMCZ), and Lowry Lower Copper Zone (LLCZ) were reviewed and approved by Berkley Tracy, P.Geo (PGO#3024) of SRK Consulting (U.S.), Inc., a Qualified Person as defined by CIM.*

Table 14.22 Black Butte Copper Project Mineral Resource Estimate for the Lowry deposit at NSR Cut-Off Value of US$110.00/t, as of November 5, 2025 – SRK Consulting (U.S.), Inc.

14.12 Mineral Resource Sensitivity

14.12.1 Johnny Lee deposit

A series of grade-tonnage curves were calculated to assess the sensitivity of Measured and Indicated Mineral Resources to changes in CoV. Summary tabulated Mineral Resource sensitivity for Measured and Indicated classified material is presented in Table 14.23. Figure 14.31 and Figure 14.32 show grade-tonnage curves for the JL UCZ and JL LCZ, respectively. In the JL UCZ, only HG Cu, Cu 1, and Cu 2 report material Measured and Indicated resources. In the JL LCZ, only the Main zone is classified with Measured and Indicated resources. Inferred material is not included in the graphs or table below. The Johnny Lee NSR cut-off value of US$45.00/t is highlighted in bold.

JL Upper Copper Zone		
NSR Cutoff ($/t)	Cu (%)	Tonnage
15	1.5	24.3
45	**1.9**	**16.1**
60	2.5	11.0
75	2.5	10.6
90	2.5	10.3
105	2.6	9.6
120	2.7	8.6
135	2.8	7.4
150	3.0	6.2
165	3.1	5.1
180	3.3	3.9
195	3.5	2.7
210	3.7	1.9
225	3.8	1.3

JL Lower Copper Zone		
NSR Cutoff ($/t)	Cu (%)	Tonnage
15	3.6	4.4
45	**5.4**	**2.9**
60	5.8	2.6
75	6.1	2.5
90	6.3	2.4
105	6.4	2.4
120	6.4	2.3
135	6.4	2.3
150	6.4	2.3
165	6.4	2.3
180	6.5	2.3
195	6.5	2.3
210	6.6	2.2
225	6.6	2.2

Table 14.23 Johnny Lee Mineral Resource Sensitivity by Zone – Measured and Indicated

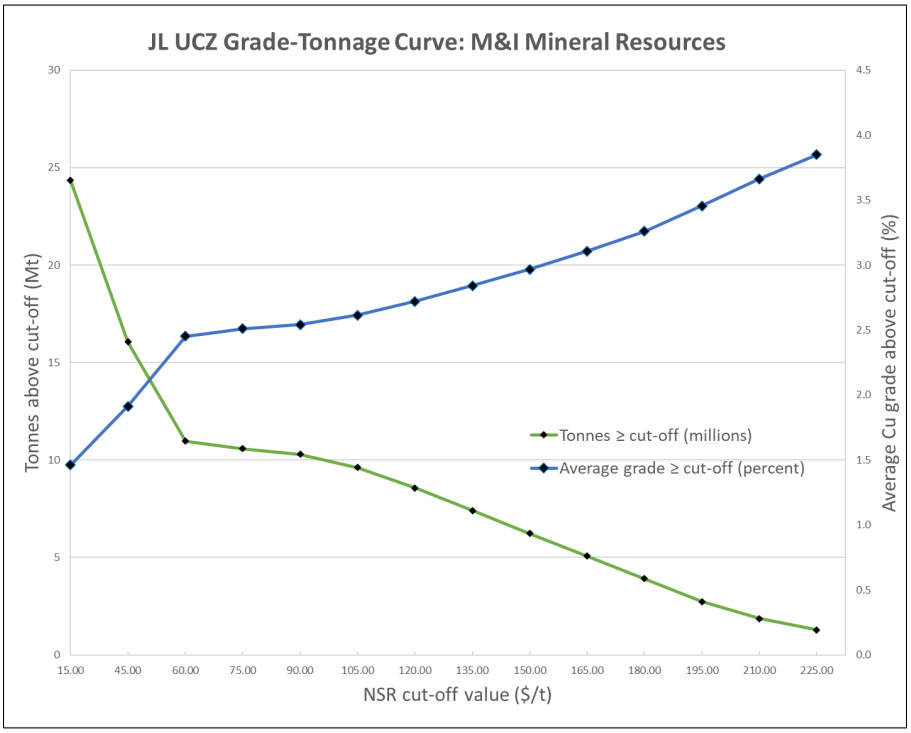


Figure 14.31 Grade-Tonnage Curve for the JL UCZ showing Measured and Indicated Mineral Resources

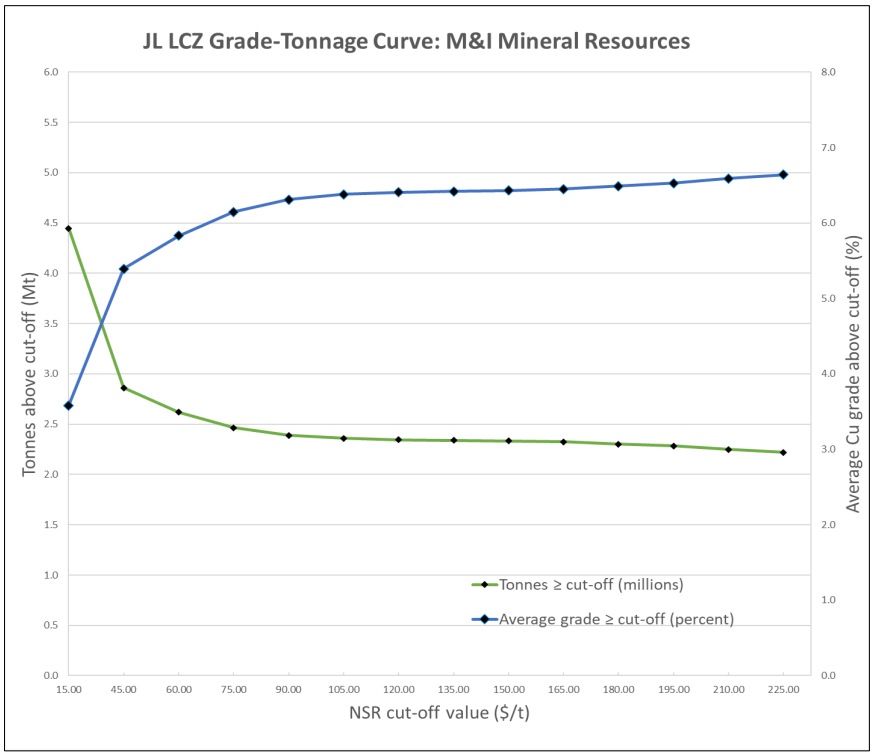


Figure 14.32 Grade-Tonnage Curve for the JL LCZ Main showing Measured and Indicated Mineral Resources

14.12.2 *Lowry deposit*

Grade-tonnage curves were calculated to assess the sensitivity of Mineral Resources to changes in CoV. Summary tabulated Mineral Resource sensitivity for Measured and Indicated classified material is presented in Table 14.24. Inferred material is not included in the graphs or table below. Figure 14.33 and Figure 14.34 show grade-tonnage curves for the Lowry LMCZ and LLCZ zones, respectively. Note that the Lowry LUCZ has only Inferred material and is not summarized below. The Lowry NSR cut-off value of US$110.00/t is highlighted in bold.

Lowry Middle Copper Zone		
NSR Cutoff ($/t)	Cu (%)	Tonnage (Mt)
15	1.2	15.6
45	1.5	11.1
60	1.8	8.5
75	2.1	6.6
90	2.3	5.6
105	2.4	5.0
110	**2.4**	**4.8**
120	2.5	4.4
135	2.7	3.7
150	2.8	3.2
165	3.0	2.7
180	3.2	2.2
195	3.4	1.7
210	3.5	1.5
225	3.7	1.2

Lowry Lower Copper Zone		
NSR Cutoff ($/t)	Cu (%)	Tonnage (Mt)
15	0.8	12.3
45	1.1	6.7
60	1.5	4.0
75	1.8	2.9
90	1.9	2.3
105	2.1	1.9
110	**2.1**	**1.8**
120	2.3	1.5
135	2.4	1.2
150	2.5	1.1
165	2.6	0.9
180	2.8	0.7
195	2.9	0.6
210	3.1	0.4
225	3.2	0.3

Table 14.24 Lowry Mineral Resource Sensitivity by Zone – Measured and Indicated

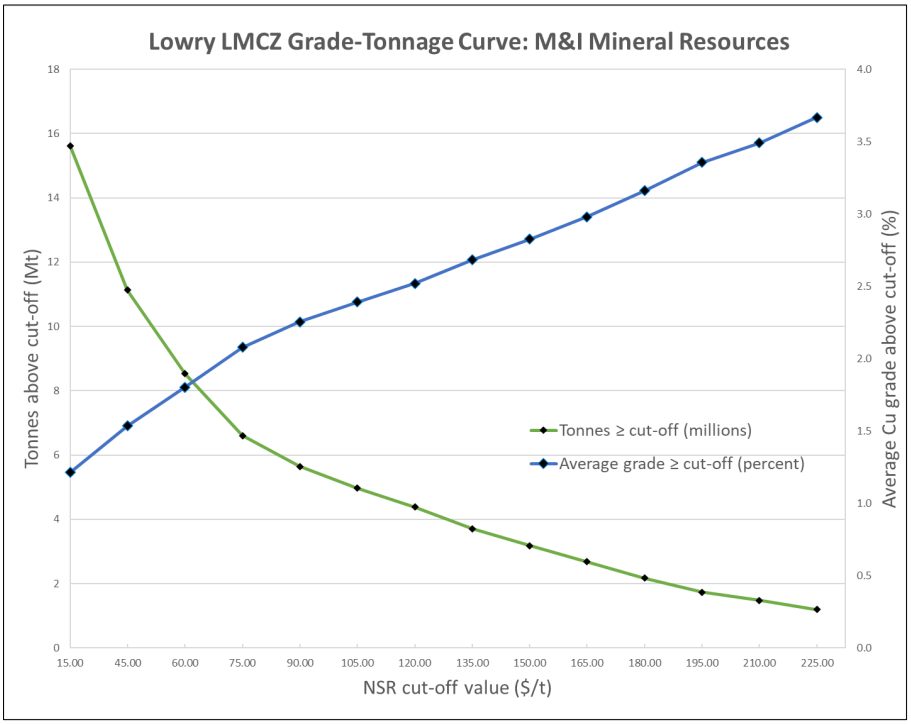


Figure 14.33 Grade-Tonnage Curve for the Lowry LMCZ Measured & Indicated Mineral Resources

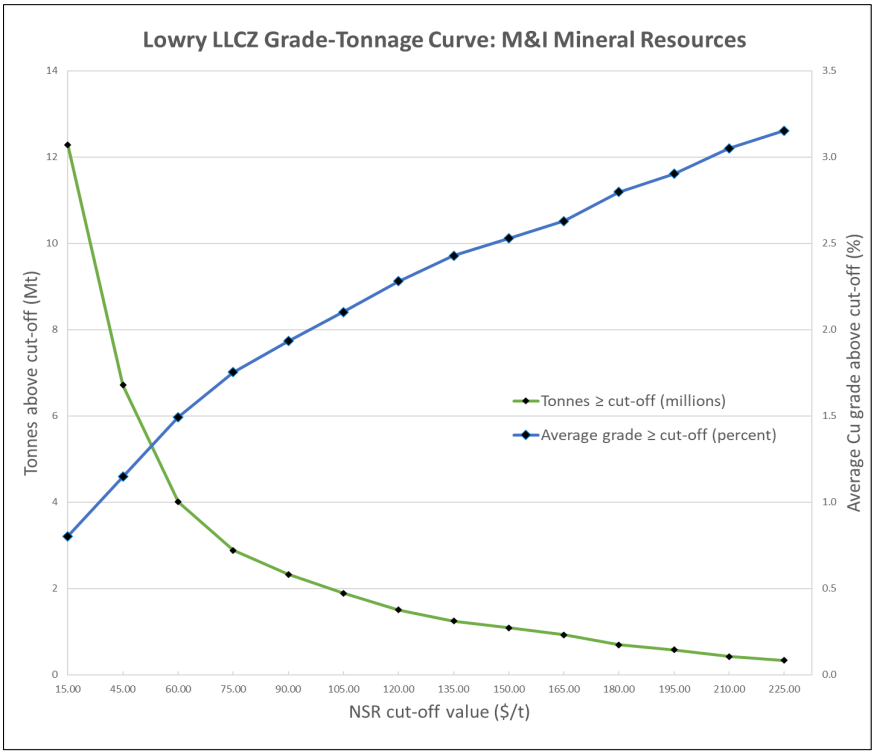


Figure 14.34 Grade-Tonnage Curve for the Lowry LLCZ Measured & Indicated Mineral Resources

14.13 Relevant Factors

For the Johnny Lee and Lowry deposits, 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. Previous legal challenges have been resolved by SRA through the Montana Supreme Court.

Limited mineralized material located beneath the McGuire Parcel has been excluded from the estimation of 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 CoV, may materially affect the Mineral Resources.

Future economic assessment could result in a change in the CoV, which would result in a change in the tonnage of available minable material. Mineralization represented by the resource block model was evaluated for RPEEE considering UG mining methods by applying SRA-provided input parameters. SRK did not independently audit recovery, processing costs, or other assumptions for deriving CoV, but does consider the inputs to be reasonable based on experience with other projects.

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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 a minimum PFS level of analysis.

This study includes adequate information on mining, processing, metallurgical, economic, and other relevant factors that demonstrate, at the time of reporting, that economic extraction of a Mineral Reserve mine plan 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 outlined 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

Two separate Mineral Resource block models were generated for the analysis, one each for the JL LCZ and JL UCZ areas. The models used are described in Section 14 (effective date 5th November 2025). The block model fields used in the Mineral Reserve estimation as summarised in Table 15.1.

Field	JL LCZ	JL UCZ
Grade - Cu	Cu_pct_LCZ	Cu_pct_UCZ
Grade - Metallurgically Recovered Cu	Cu_rec_LCZ	Cu_rec_UCZ
Grade - Ag	Ag_ppm_LCZ	Ag_ppm_UCZ
Grade - As	As_ppm_LCZ	As_ppm_UCZ
Grade - Bi	Bi_ppm_LCZ	Bi_ppm_UCZ
Grade - Co	Co_ppm_LCZ	Co_ppm_UCZ
Grade - Fe	Fe_pct_LCZ	Fe_pct_UCZ
Grade - Mo	Mo_ppm_LCZ	Mo_ppm_UCZ
Grade - Pb	Pb_ppm_LCZ	Pb_ppm_UCZ
Grade - S	S_pct_LCZ	S_pct_UCZ
Grade - Zn	Zn_ppm_LCZ	Zn_ppm_UCZ
Density	SG	SG_UCZ
Mineral Resource Classification	Class_LCZ	Class_UCZ

Table 15.1 Mineral Resource Block Model Fields Used in Mineral Reserve Estimation

The NSR for each block in the model was determined using the inputs in Table 15.2, with the metallurgically recovered Cu grades applied for the value calculation. For the Mineral Reserve estimate, the NSR value, Cu grade and metallurgically recovered Cu grades for blocks classified as Inferred were coded to zero. Penalties were excluded from the NSR calculation but considered in the financial modelling to test project economics.

Item	Units	Value
Cu Price	$/lb	4.70
Concentrate Grade	% Cu	22.5
Moisture Content	% Moisture	10.0
Cu Min Deduction	% Cu	1.0
Treatment Charge	$/dmt Concentrate	50.00
Refining Charge	$/lb Cu Payable	0.05
Freight	$/dmt Concentrate	174.27
Penalties	None applied in NSR calculation	
Royalties	% NSR	2.0

Table 15.2 NSR Inputs for Reserve Calculations

The NSR value $/t was calculated applying the following formula:

$$NSR = [\text{Recovered \& Payable Cu Metal}] * [\text{Cu Price}] - [\text{Treatment and Refining Cost}] - [\text{Concentrate Freight Cost}] - [\text{Royalty}]$$

15.3 Cut-off Value

Mine design shapes were generated based on a NSR Cut-Off Value (COV) greater than $110/t (operating cost) after dilution and mining recoveries are applied. The parameters used for the calculation of the COV are shown in Table 15.2. 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 mineral resources for conversion. For further details on the actual NSR calculation and the operating cost estimate see Sections 19, 21 and 22.

An NSR cut-off value of US$45/t (incremental operating cost) was used to determine whether material should be sent to waste or processed, representing the cost of processing ore including site general and administration costs. This lower grade ore is planned to be stockpiled and processed opportunistically when sufficient treatment plant capacity is available after treatment of the highest grade ore.

These initial parameters were used to define mineable shapes. Following this, all shapes were reviewed to determine if they were economic.

Operating Cost Area	Cost ($/t)
Mining	65
Processing/General and Administrative	45
Total	110

Note: *Assumptions in this table are for Mineral Reserve estimate only*

Table 15.3 Inputs for Cut-off Value Calculation

15.4 Mining Shapes

Preliminary mining shapes were created using Datamine Software's Mineable Shape Optimiser® (MSO) on the MRE block models with an NSR COV of $110/t and minimum dimensions of 5 mH x 5 mW (i.e. ore development dimensions). With these guides, development drives were designed in Deswik CAD software to maximise economic ore extraction with the proposed cut-and-fill and drift-and-fill mining methods.

A final check was undertaken on each area to ensure economic viability.

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.
- 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

An additional 5% dilution at waste grade was applied mathematically in the schedule to all unsupported ore shapes.

No other external dilution was applied to the remaining ore shapes (supported jumbo development).

15.5.2 Internal Dilution

Additional sources of dilution include sub-economic material that falls within the minimum dimension mineable shape. This also includes material classified as Inferred Resources for which a waste grade and NSR value was applied.

15.5.3 Mining Recovery

A 90% mining recovery was applied to all material being extracted from unsupported shapes.

In-situ barrier pillars were modelled in the JL UCZ area for global stability risk management, with ultimate recoveries of 50% (at end of mine life) or 0% (i.e. sterilised completely) based on geotechnical analysis.

All other ore material was development ore with an assumed mining recovery of 100%.

15.6 Risk to Reserve

Key identified risks to extraction of the estimated Mineral Reserve are:

- A water-bearing geological unit is present north of the Buttress Fault which is the boundary to the JL LCZ mineral resource. The interpretation of the structural framework of the JL LCZ area is early stage, therefore location and hydrogeological characteristics of structure has not been determined. Further work is required to decrease JL LCZ structural and hydrogeological uncertainty.
- Achievement of lateral development rates will dictate ore production schedules. Development rates can be negatively affected by latent conditions such unexpectedly poor ground conditions, or higher groundwater inflows than anticipated. Further analysis is recommended in future detailed studies to reduce these risks.

- The extraction ratio assumptions of the flat-dipping and relatively shallow JL UCZ area will require further refinement in future studies, and disciplined management during operations, to mitigate potential instability towards the end of the mine life.
- The project plans to use experienced expatriate crews and managers for the first 36 months to meet development targets and train local staff. If work permits or visas cannot be obtained, it could delay the project and negatively affect its financial performance.
- Costs have been estimated excluding tariffs. There is a risk of material cost inflation due to addition of US government imposts to specialised underground mining equipment, parts and consumables which must be sourced from overseas where no adequate local option exists.
- Permits for the Black Butte Copper Project are substantially in place having received the Final Mine Operating Permit from the MDEQ. The PFS supporting this Mineral Reserve aligns with the Mine Operating Permit and only mines the areas that are currently permitted (the Johnny Lee deposit).
- The largest risk to the JL UCZ Mineral Reserve is the metallurgical recovery which is 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 15.4.

Deposit	Class	Tonnes (Mt)	Cu Grade (%)	Cu Metal (kt)
Johnny Lee Deposit	Proven	-	-	-
	Probable	9.5	2.9	270
	Total	**9.5**	**2.9**	**270**

Notes:

1. *The Qualified Person for the Mineral Reserve estimate is Shane McLeay FAusIMM.*
2. *Effective date: November 14, 2025. 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 $4.70 /lb Cu price and NSR cut-off values of $110/t (operating cost) and $45/t (incremental operating cost).*
4. *Tonnages are rounded to the nearest 100 kt, metal grades are rounded to one decimal place, and metal tonnes are rounded the nearest 10kt. All units are metric.*
5. *Rounding as required by reporting guidelines may result in summation differences.*
6. *Average LOM Metallurgical Recovery is 95% for JL LCZ, 78% for JL UCZ, and 86% combined.*
7. *Mineral Reserves are based on the Mineral Resource estimate effective November 5, 2025.*
8. *Mineral Reserves are a subset of Mineral Resources.*

Table 15.4 Mineral Reserve Estimate Johnny Lee deposit

The Mineral Reserves identified in Table 15.4 comply with CIM definitions and standards for a NI 43-101 Technical Report. The Mineral Reserve Estimate is based on the Measured and Indicated categories of the Mineral Resource contained within the mine design. The Inferred Mineral Resources do not contribute to the financial performance of the project and are treated in the same way as waste. The Mineral Reserve Estimate has considered all modifying factors appropriate to the mine. The reference point at which the Mineral Reserves are defined is where the ore is delivered to the process plant primary crusher.

Although there is Measured material included in the Mineral Resource used as the basis of the design, no Proven Mineral Reserves are estimated as the QP considers the level of analysis to date does not yet provide sufficient confidence in the modifying factors to justify this classification. All Mineral Reserves have been classified as Probable in this estimate.

Table 15.5 below shows the Mineral Reserves broken out by zone.

Zone	Class	Diluted Tonnes (Mt)	Cu Mined Grade (%)	Mined Cu Metal (kt)
JL UCZ Total	Proven	-	-	-
	Probable	6.7	2.2	140
	Total	6.7	2.2	140
JL LCZ Total	Proven	-	-	-
	Probable	2.8	4.6	130
	Total	2.8	4.6	130
Project Total	***Proven***	-	-	-
	Probable	***9.5***	***2.9***	***270***
	Total	***9.5***	***2.9***	***270***

Notes:

1. *Tonnages are rounded to the nearest 100 kt, metal grades are rounded to one decimal place, and metal tonnes are rounded the nearest 10kt. All units are metric.*
2. *Rounding as required by reporting guidelines may result in summation differences.*
3. *Mineral Reserves were estimated using a $4.70 /lb Cu price and NSR cut-off values of $110/t (operating cost) and $45/t (incremental operating cost).*

Table 15.5 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 (JL UCZ) and the Johnny Lee Lower Copper Zone (JL LCZ) are scheduled to be mined as part of this Study. Both zones are reasonably flat-lying 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 Data Sufficiency and Validation

A review of the data sufficiency bench-marked against the level of study and confidence requirements was undertaken on all data used for this study.

16.2.1.1 Data Sufficiency

A total of 4,783 m of drill core was used in the Black Butte underground mining assessment, with 893 m logged for the upper zone, 2,330 m logged for the lower zone, 151 m for the initial decline and 1,662 m for the main decline. 254 m from hole SC24-311 was utilised in the analysis for both the lower zone, as well as the main decline analysis. The data used by Entech in this study was a combination of geotechnical logging completed by others, as well as a large portion of photo logging undertaken by Entech (re-logging holes in greater detail). A total of 1607 structural measurements of high confidence were used in the analysis. Validation was undertaken by Entech during a site visit on Monday 18 August 2025.

Comments on the coverage of the drilling is provided:

- The extent of drill hole coverage with domain geotechnical logging is satisfactory (commensurate with resource confidence) for Black Butte Copper Project.
- As additional drilling is undertaken, adequate geotechnical logging ensuring appropriate logging intervals should be completed to ensure sufficient data is available in these areas.

A suite of rock properties testing has been undertaken for the Black Butte Copper Project. Comments on the coverage of the drilling and rock properties testing is provided:

- Coverage and types of tests will require improvement in coverage across the deposit at various depths and across all rock types.
- Additional geotechnical samples should be collected and tested in the future, ideally before mining commences with a focus on increasing uniaxial compressive strength (UCS) data and a corresponding uniaxial tensile strength (UTS), elastic constants (Young's Modulus and Poisson's Ratio) and a suite of direct shear tests (DST) test work should be undertaken.

16.2.2 Geology

The Black Butte Copper 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 that was deposited along the fault controlled, northern margin of the Helena Embayment.

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

A geological map of the area is shown below in Figure 16.1 with the detailed stratigraphy displayed in Figure 16.2.

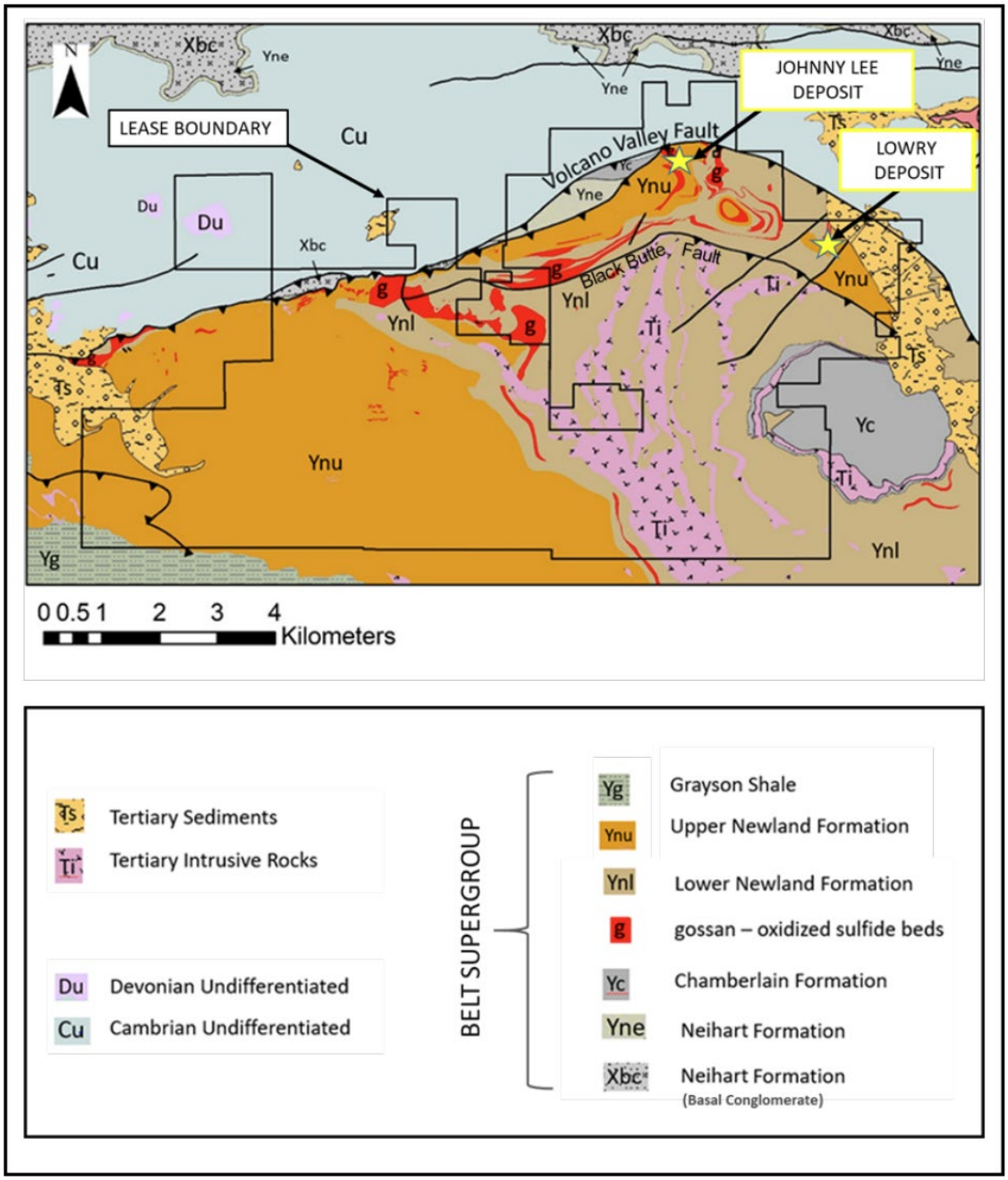


Figure 16.1 Regional Geological Map of the Black Butte Copper Project Site (SRA, 2026)

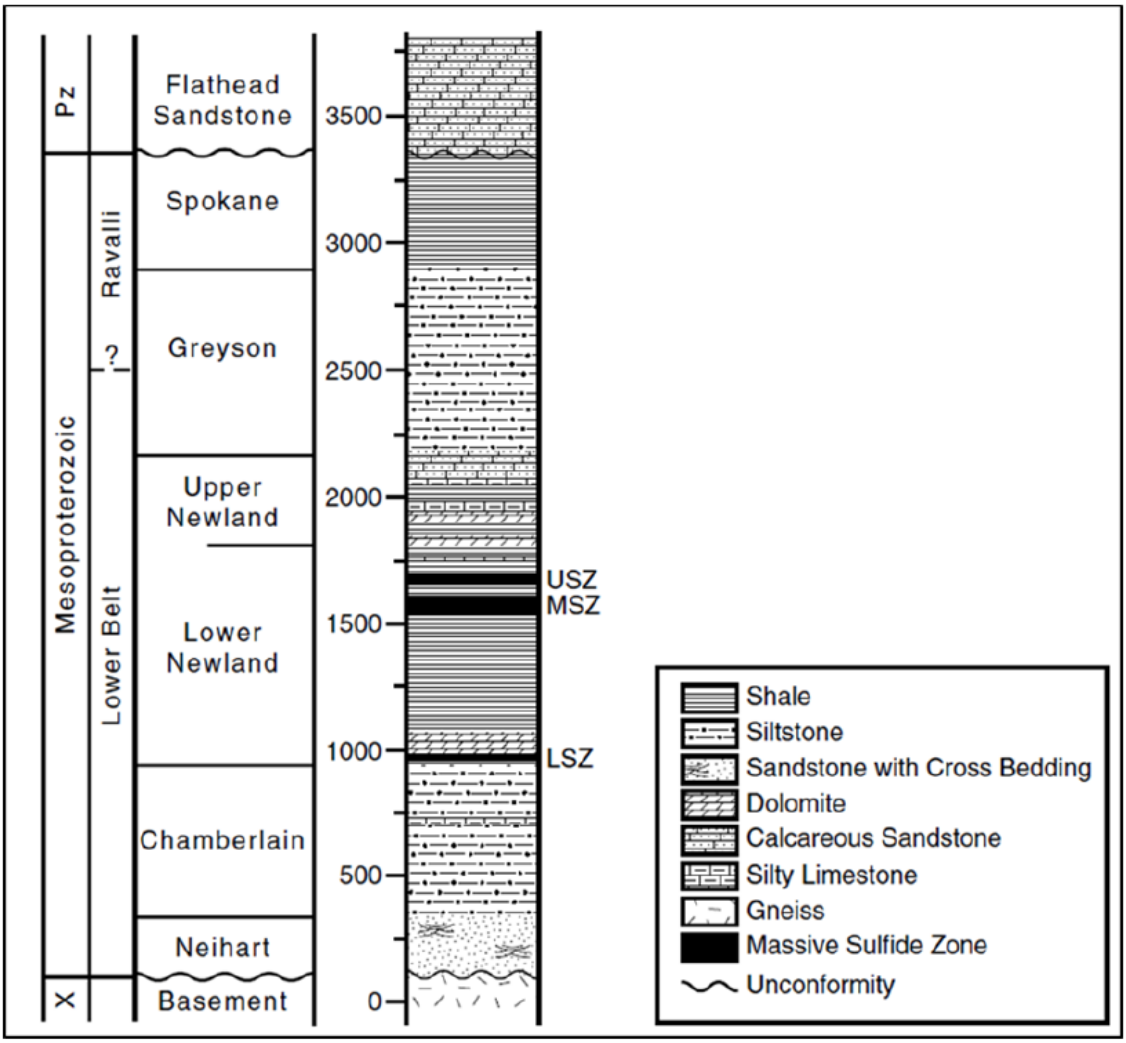


Figure 16.2 Project area stratigraphy with thickness scale (SRA, 2022)

16.2.2.1 Major Structures

Two major structural features have been identified in the vicinity of the Johnny Lee ore zones – the Volcano Valley Fault (VVF) and the Buttress Fault, which are described in turn below. Figure 16.3 provides a cross section of how the two major faults interact with the JL UCZ and JL LCZ.

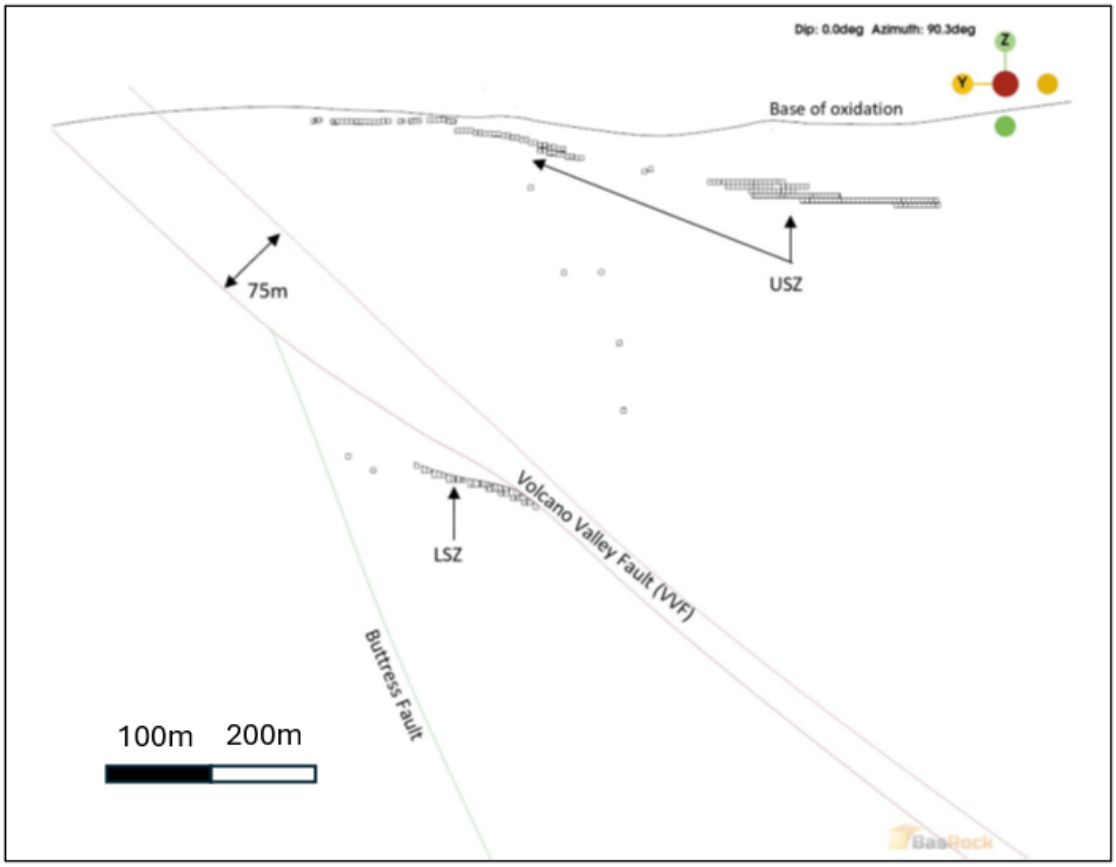


Figure 16.3 Cross-section view looking east, displaying the major structures (Entech, 2025)

16.2.2.1.1 Volcano Valley Fault

The VVF is the most prominent known structure within the Black Butte Copper Project area and is located in the hanging wall of the JL LCZ and at the northern end of the JL UCZ. This fault offsets the upper and lower orebodies. The VVF consists of a very wide zone, often up to 75 m of highly brecciated and graphitic mass, with graphite and clay mineral coatings and zones of clayey gouge.

16.2.2.1.2 Buttress Fault

The Buttress Fault is a normal fault located in the footwall of the JL LCZ. The Chamberlain Shale occurs in the hanging wall of the fault, whilst the Neihart Quartzite occurs along the footwall of the fault. The fault has a limited thickness of ~10 cm, with narrow gouge zones.

16.2.3 Geotechnical Data

Diamond drilling has been undertaken over a number of years throughout the deposits at the Black Butte Copper Project. Collection of specific geotechnical parameters by Sandfire, has occurred on a large portion of holes at appropriate logging intervals. However, more recent data has been collected over larger interval lengths. To improve data confidence in the upper sulphide zone, several holes relevant to the deposit and

previously logged by Sandfire were selected for photo validation logging. To improve the data confidence in the lower sulphide zone, several holes were selected for geotechnical photo logging. The data provided by Sandfire and the additional photo logging undertaken by Entech allowed for investigation of the ground conditions specific to the Black Butte underground project.

The geotechnical data collection program consisted of geotechnical photo logging undertaken by Entech, with onsite geotechnical logging, sampling and material properties testing undertaken by Sandfire/Tintina, or contractors engaged by Tintina. The extent of drill hole and material properties testing coverage is considered sufficient to undertake the geotechnical assessment for the phase of study.

Additional material properties testing would reduce the variability in the understanding of material properties and their behaviour with a focus on increasing uniaxial compressive strength (UCS) data and a corresponding uniaxial tensile strength (UTS), elastic constants (Young's Modulus and Poisson's Ratio) and commence collection of direct shear tests (DST) test work.

16.2.3.1 Diamond Drilling

A total of 56 diamond drill holes, located in the vicinity of the proposed Black Butte Copper Project and totalling 4,783 m with 893 m logged for the upper zone, 2,330 m logged for the lower zone, 151 m for the initial decline and 1662 m for the main decline, were used for the collection of detailed geotechnical data, including rock mass and structure characterisation, and oriented structure data. 254 m from hole SC24-311 was utilised in the analysis for both the lower zone, as well as the main decline analysis. Locations of drill holes used in the analysis are shown in Figure 16.4 and Figure 16.5.

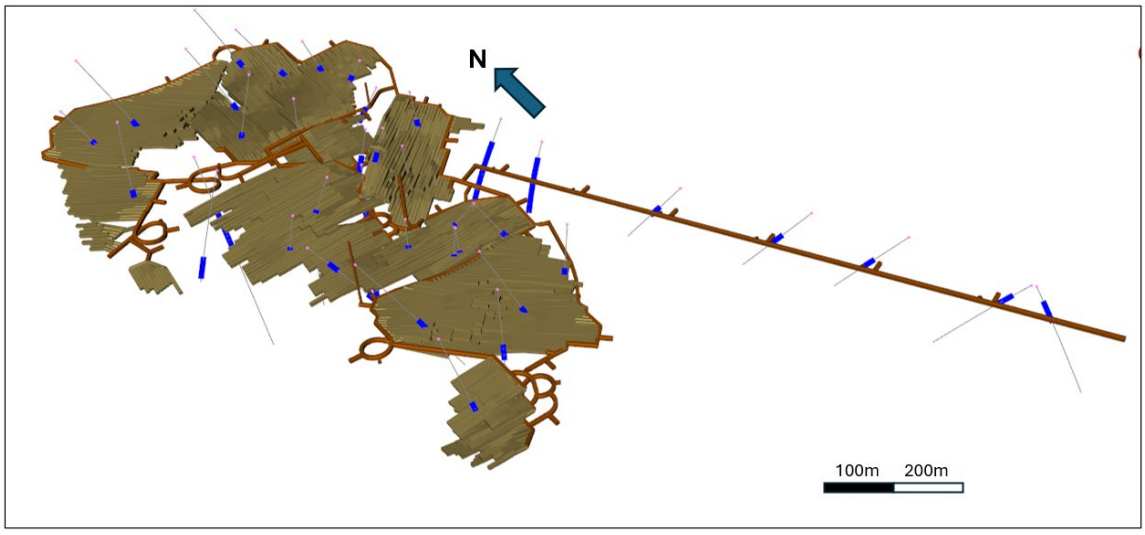


Figure 16.4 Oblique plan view of the proposed JL UCZ and initial decline development showing the distribution of geotechnical drilling data collected (Entech 2025)

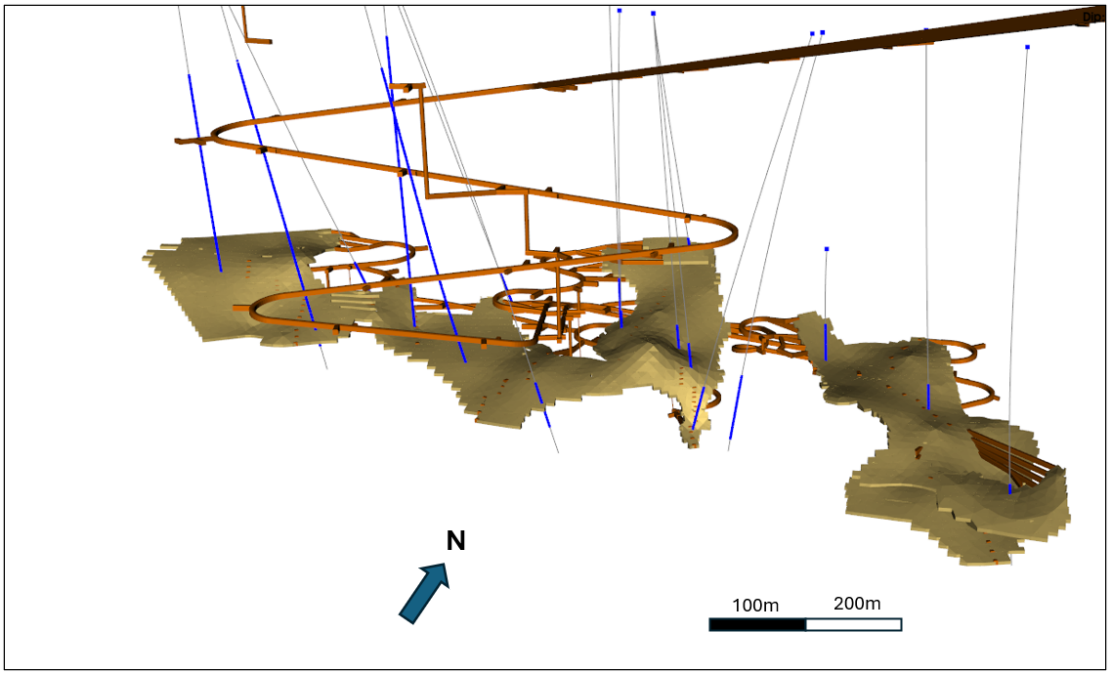


Figure 16.5 View looking north of the proposed JL LCZ showing the distribution of geotechnical drilling data collected (Entech 2025)

16.2.3.2 Laboratory Test Work

Several laboratory material properties test work programs were undertaken between 2015 and 2021 on drill core samples selected from diamond drill holes, with locations shown in Figure 16.6. The consolidated dataset from all laboratory testing programs comprises the following:

- 38 x Uniaxial Compressive Strength (UCS).
- 16 x Triaxial Compressive Strength (TCS).
- 36 x Young's Modulus and Poisson's Ratio.
- 36 x Uniaxial Tensile Strength (UTS).
- 33 x Slake Durability, and
- 29 x CERCHAR Abrasivity Tests.
- 1 x S & P Wave.

Test results with a failure mode along a structure i.e. vein, joint or bedding, were excluded from the test result summaries as they are not representative of intact rock properties.

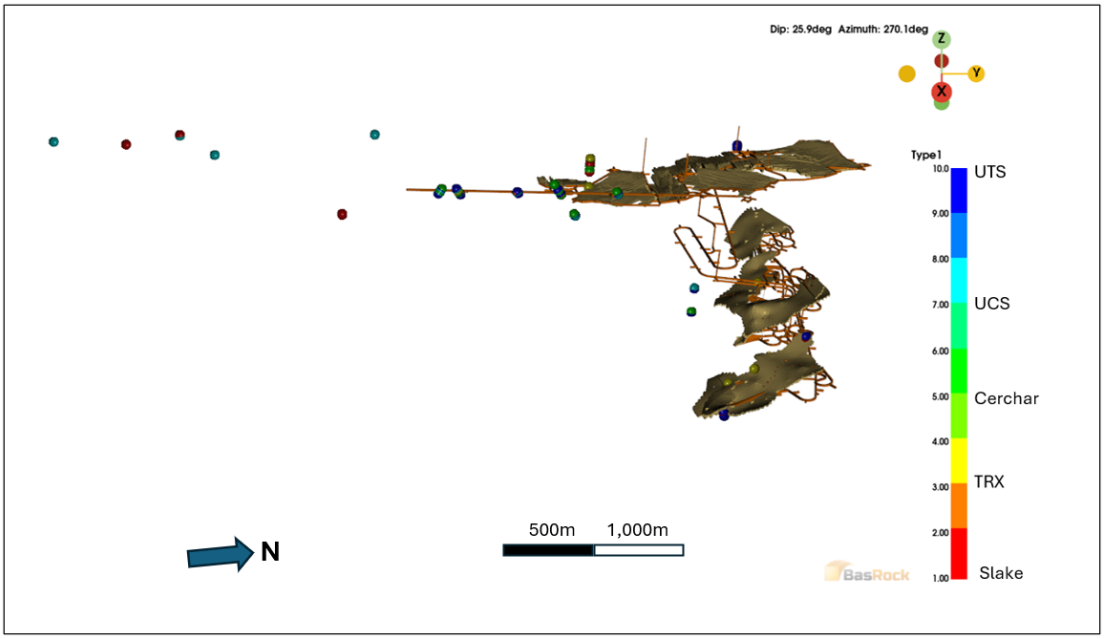


Figure 16.6 View looking west showing the sample locations (Entech, 2025)

16.2.3.3 Structural Data

The structural model contains structure observed at the drill core scale. Structure data was measured from the defects within the orientated diamond drill core from the selected diamond drill holes. A total of 1607 structural measurements of high confidence were used in the analysis.

16.2.3.3.1 Structure

Rocscience Dips® 8.029 software was used to plot and analyze the structure data and group it into structure sets. The analysis was carried out combining all data together, as well as for the separate lithology units. Stereonet analysis is presented below for JL UCZ and JL LCZ. A stereographic projection of global defect measurements from Black Butte is shown in Figure 16.7.

This data was used in the ground support analysis for the Main Decline, with the data for the JL UCZ and JL LCZ used within the respective mining areas.

The global structure sets defined at Black Butte are summarised as follows:

1. 09° / 221° - flat south-west dipping joint set.
2. 35° / 010° - moderately north dipping joint set.
3. 55° / 276° - moderately west dipping joint set.
4. 58° / 103°- moderately east dipping joint set.
5. 85° / 208° - steeply south west dipping joint set.

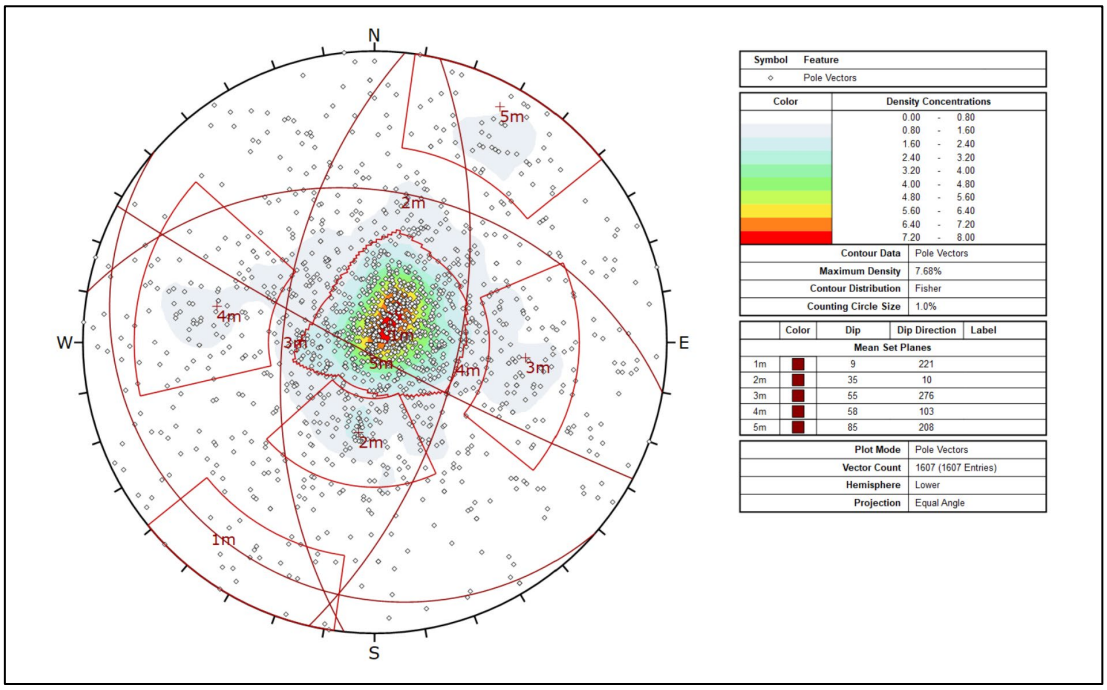


Figure 16.7 Stereonet plot displaying orientated structure data for Black Butte underground

16.2.3.3.2 JL UCZ

The stereonet projection of the defect measurements for the JL UCZ is shown in Figure 16.8. This data was used in the ground support analysis for all drive profiles expected to be present in the JL UCZ, as well as inputting into the stable span assessment in production areas.

The structure sets defined for the JL UCZ are summarised as follows:

1. 12° / 183° - flat south dipping joint set.
2. 53° / 279° - moderately west dipping joint set.
3. 77° / 100° - steeply east dipping joint set.
4. 78° / 156°- steeply south-east dipping joint set.
5. 63° / 024° - moderate to steeply north dipping joint set.

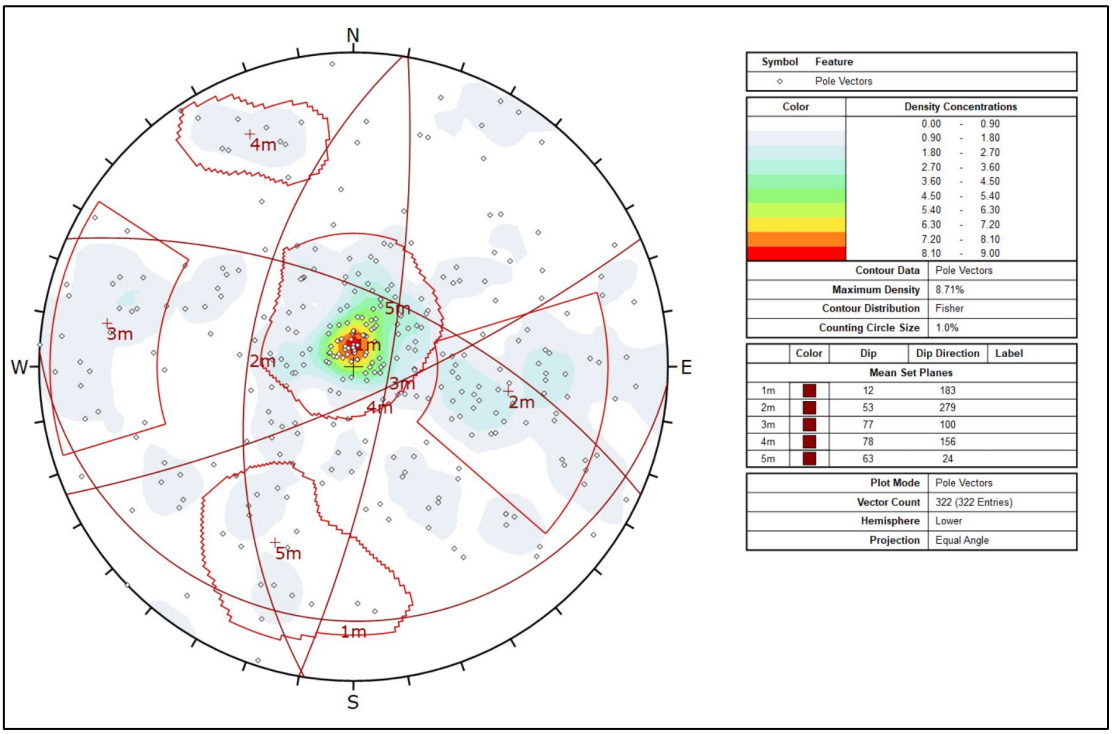


Figure 16.8 Stereonet plot displaying orientated structure for the JL UCZ

16.2.3.3.3 JL LCZ

The stereonet projection of the defect measurements for the JL LCZ is shown in Figure 16.9.

This data was used in the ground support analysis for all drive profiles expected to be present in the JL LCZ, as well as inputting into the stable span assessment in production areas.

The structure sets defined for the JL LCZ are summarised as follows:

1. 24° / 109° - flat south-east dipping joint set.
2. 30° / 215° - flat to moderately south-west dipping joint set.
3. 24° / 336° - flat north-west dipping joint set.
4. 52° / 028°- moderately north dipping joint set.
5. 61° / 172° - moderate to steeply south dipping joint set.
6. 80° / 255° - steeply west dipping joint set.
7. 81° / 336° - steeply north dipping joint set.

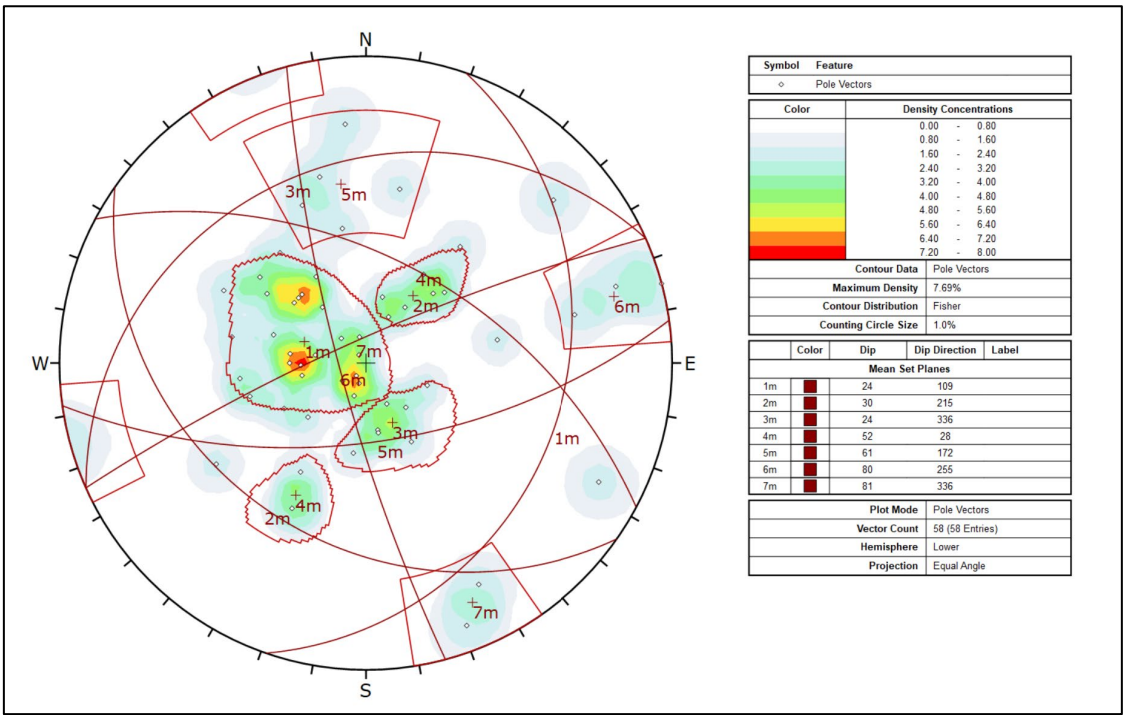


Figure 16.9 Stereonet plot displaying orientated structure for the JL LCZ

16.2.3.4 Rock Mass Characterisation

Barton et al (1974) developed a rock mass classification system (Q-system) based on many case histories of underground excavations. The Q-system classifies a rock mass based on several factors including Rock Quality Designator (RQD), joint properties including its roughness and alteration, the presence of water, and the Stress Reduction Factor (SRF). Q is represented in the following equation:

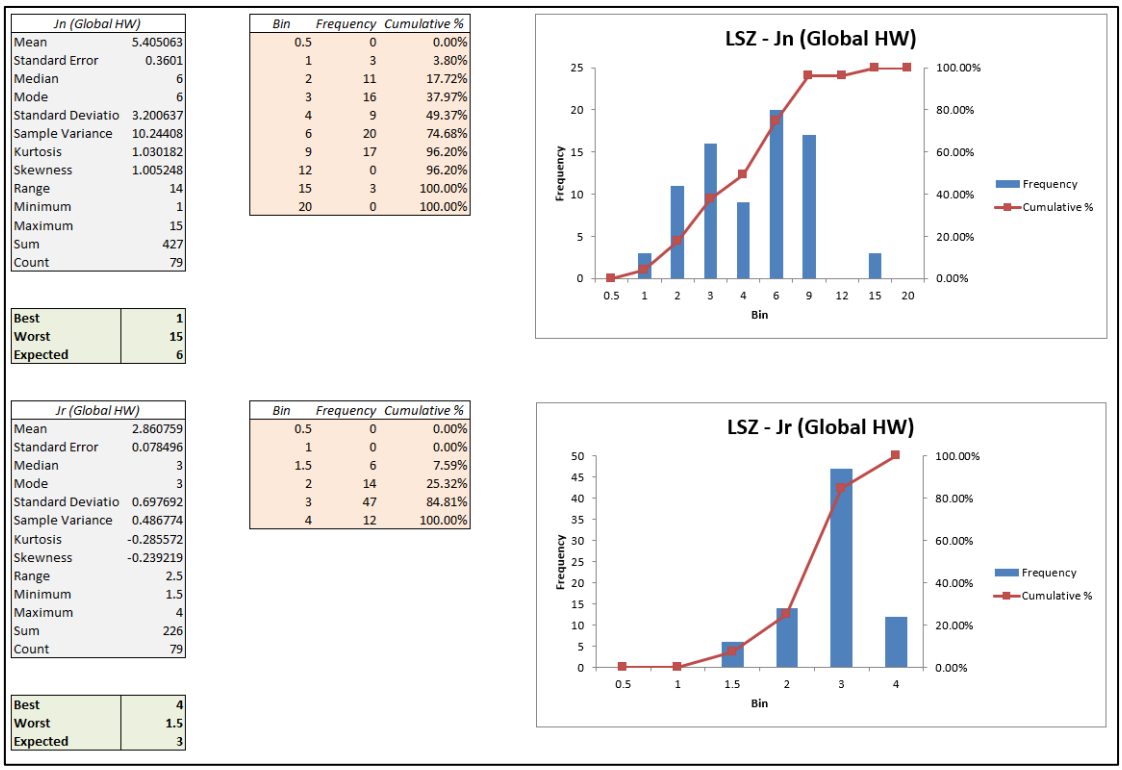
$$Q = RQD/Jn \times Jr/Ja \times Jw/SRF$$

Where:

- RQD Rock Quality Designator, the cumulative sum of core pieces with length >10 cm within a core run, represented as a cumulative percentage.
- Jn Joint set number, which takes into consideration the number of discontinuity sets present within the rock mass.
- Jr Joint roughness number, which takes into consideration small and large-scale asperities and undulations on the joint surface.
- Ja Joint alteration number, which takes into consideration the chemical alteration of a joint surface affecting its frictional and strength properties.

- Jw — Joint water properties, which takes into account the presence of water and the effect that water pressure has on the discontinuity.
- SRF — Stress Reduction Factor, which takes into account the effect of overstressing causing damage to a rock mass, or the presence of large structures such as faults and shears which will reduce the overall rock mass strength.

When determining worst, most likely and best-case rock mass conditions, each of the individual input parameters (RQD, Jn, Jr & Ja) have been assessed by rock type and by domain for each of the JL UCZ and JL LCZ. In general, *expected* has been determined as the mean RQD, and in the case of Jn, Jr and Ja as the most frequently occurring value/number (Figure 16.10). Worst and best-case values represent the upper and lower values encountered from the logging.



Jn (Global HW)	
Mean	5.405063
Standard Error	0.3601
Median	6
Mode	6
Standard Deviatio	3.200637
Sample Variance	10.24408
Kurtosis	1.030182
Skewness	1.005248
Range	14
Minimum	1
Maximum	15
Sum	427
Count	79

Bin	Frequency	Cumulative %
0.5	0	0.00%
1	3	3.80%
2	11	17.72%
3	16	37.97%
4	9	49.37%
6	20	74.68%
9	17	96.20%
12	0	96.20%
15	3	100.00%
20	0	100.00%

Best	**1**
Worst	**15**
Expected	**6**

Jr (Global HW)	
Mean	2.860759
Standard Error	0.078496
Median	3
Mode	3
Standard Deviatio	0.697692
Sample Variance	0.486774
Kurtosis	-0.285572
Skewness	-0.239219
Range	2.5
Minimum	1.5
Maximum	4
Sum	226
Count	79

Bin	Frequency	Cumulative %
0.5	0	0.00%
1	0	0.00%
1.5	6	7.59%
2	14	25.32%
3	47	84.81%
4	12	100.00%

Best	**4**
Worst	**1.5**
Expected	**3**

Figure 16.10 Example of Jn and Jr descriptive statistics for Black Butte rock mass characterisation

When querying the drill hole data by geotechnical domain, a composite of the drill hole database in 1 m intervals was first created. The composited drill hole intervals were then extracted by domain (i.e. Footwall (FW), Ore (ORE), Hangingwall (HW) and Development. The *immediate* FW and *immediate* HW zones are defined as a 7 m wide zone of rock either side of the current mineralised wireframe. A summary of the rock mass conditions for the geotechnical domains (HW, ORE, and FW) for the different lithologies expected to be encountered at Black Butte underground mine, for both the JL UCZ and JL LCZ, are summarised in Table 16.1 to Table 16.4.

JL UCZ - HW Global									
	RQD	Jn	Jr	Ja	Jw	SRF	Q'	Q	Rock Mass Class
Best	100	1	4	2	1	1	200.00	200.00	Very Good
Worst	10	20	1	15	1	5	0.03	0.01	Exceptionally Poor
Expected	55	3	1	3	1	1	6.11	6.11	Fair
JL UCZ - Ore Global									
	RQD	Jn	Jr	Ja	Jw	SRF	Q'	Q	Rock Mass Class
Best	100	1	4	2	1	1	200.00	200,00	Very Good
Worst	10	20	0.5	12	1	7.5	0.02	0.00	Exceptionally Poor
Expected	50	6	1.5	3	1	1	5.00	5.00	Fair
JL UCZ - FW Global									
	RQD	Jn	Jr	Ja	Jw	SRF	Q'	Q	Rock Mass Class
Best	94	1	4	2	1	1	188.00	188.00	Very Good
Worst	10	20	1	15	1	7.5	0.033	0.004	Exceptionally Poor
Expected	51	6	1.5	3	1	1	4.50	4.50	Fair

Table 16.1 Summary of rock mass conditions for the JL UCZ

JL LCZ - HW Global									
	RQD	Jn	Jr	Ja	Jw	SRF	Q'	Q	Rock Mass Class
Best	100	1	4	1	1	1	400.00	400.00	Exceptionally Good
Worst	52	15	1.5	12	1	5	0.43	0.09	Extremely Poor
Expected	92	6	3	2	1	1	23.0	23.0	Good
JL LCZ - Ore Global									
	RQD	Jn	Jr	Ja	Jw	SRF	Q'	Q	Rock Mass Class
Best	100	1	4	1	1	1	400.00	400.00	Exceptionally Good
Worst	33	15	1.5	8	1	1	0.41	0.08	Very Poor
Expected	95	6	3	2	1	1	23.75	23.75	Good
JL LCZ - FW Global									
	RQD	Jn	Jr	Ja	Jw	SRF	Q'	Q	Rock Mass Class
Best	100	1	4	1	1	1	400.00	400.00	Exceptionally Good
Worst	39	15	1.5	12	1	5	0.325	0.065	Extremely Poor
Expected	94	3	3	1	1	1	94.00	94.00	Very Good

Table 16.2 Summary of rock mass conditions for the JL LCZ

HW VVF									
	RQD	Jn	Jr	Ja	Jw	SRF	Q'	Q	Rock Mass Class
Best	100	4	3	2	1	1	37.5	37.5	Good
Worst	77	15	1.5	12	1	5	0.64	0.13	Very Poor
Expected	91	15	3	8	1	5	2.28	0.46	Very Poor

Table 16.3 Summary of rock mass conditions for the VVF

Initial Decline									
	RQD	Jn	Jr	Ja	Jw	SRF	Q'	Q	Rock Mass Class
Best	100	3	4	2	1	1	66.67	66.67	Very Good
Worst	10	20	1	7	1	5	0.07	0.01	Extremely Poor
Expected	42	6	1	3	1	1	2.33	2.33	Poor
Main Decline: Upper 1700 m – 1500 m									
	RQD	Jn	Jr	Ja	Jw	SRF	Q'	Q	Rock Mass Class
Best	100	1	4	1	1	1	400.00	400.00	Exceptionally Good
Worst	10	15	0.5	12	1	7.5	0.03	0.004	Exceptionally Poor
Expected	77	4	3	2	1	1	28.88	28.88	Good
Main Decline: FVV									
	RQD	Jn	Jr	Ja	Jw	SRF	Q'	Q	Rock Mass Class
Best	100	2	4	1	1	1	200.00	200.00	Exceptionally Good
Worst	10	15	0.5	12	1	7.5	0.028	0.004	Exceptionally Poor
Expected	68	4	2	4	1	1	8.50	8.50	Fair
Main Decline: Lower 1500 m – 1300 m									
	RQD	Jn	Jr	Ja	Jw	SRF	Q'	Q	Rock Mass Class
Best	100	1	4	1	1	1	400.00	400.00	Exceptionally Good
Worst	10	15	0.5	12	1	7.5	0.03	0.004	Exceptionally Poor
Expected	83	4	3	2	1	1	31.13	31.13	Good

Table 16.4 Summary of rock mass conditions for Decline development

16.2.4 Mine Design Criteria

The mining method outlined in Section 16.3 aligns with and takes into consideration all geotechnical requirements and recommendations.

16.2.5 Stable Unsupported Span Assessment

Mathews (1980) proposed a method for determining stable stope spans based on the modified Barton's Tunnelling Quality Index Q'. This was further refined by Potvin (1988) to include a further 175 case histories and has since undergone further modifications. The Modified Stability Method (MSG) is a widely used method of stope span determination.

The method is principally based on the use of Q', and three other factors to account for stress (Factor A), structural orientation (Factor B) and gravity effects (Factor C), giving the modified stability number (N').

$$N' = Q' \times A \times B \times C$$

Where:

Q' = RQD/Jn x Jr/Ja

- Factor A is a ratio of intact rock strength to induced mining stresses, degrading the overall N' value.
- Factor B measures effect of least favourably oriented joint set affecting stope surface.
- Factor C measures effect of gravity on potential failure modes (slabbing, sliding failures).

The stability number is used in conjunction with the hydraulic radius (a factor relating shape and dimensions) to assess stability on an empirical chart.

The approach of using the *expected* values was chosen to examine the stope spans for the expected stability/instability of principally hanging wall spans for a given hydraulic radius. *Expected* values have been used for span determination for maximum unsupported stable span according to the Mathews Potvin Stability Graph (Figure 16.11). *Expected* values are generally close in value to the arithmetic *Average.* Entech uses the line defining stable, unsupported and transitional stopes when assessing stope spans.

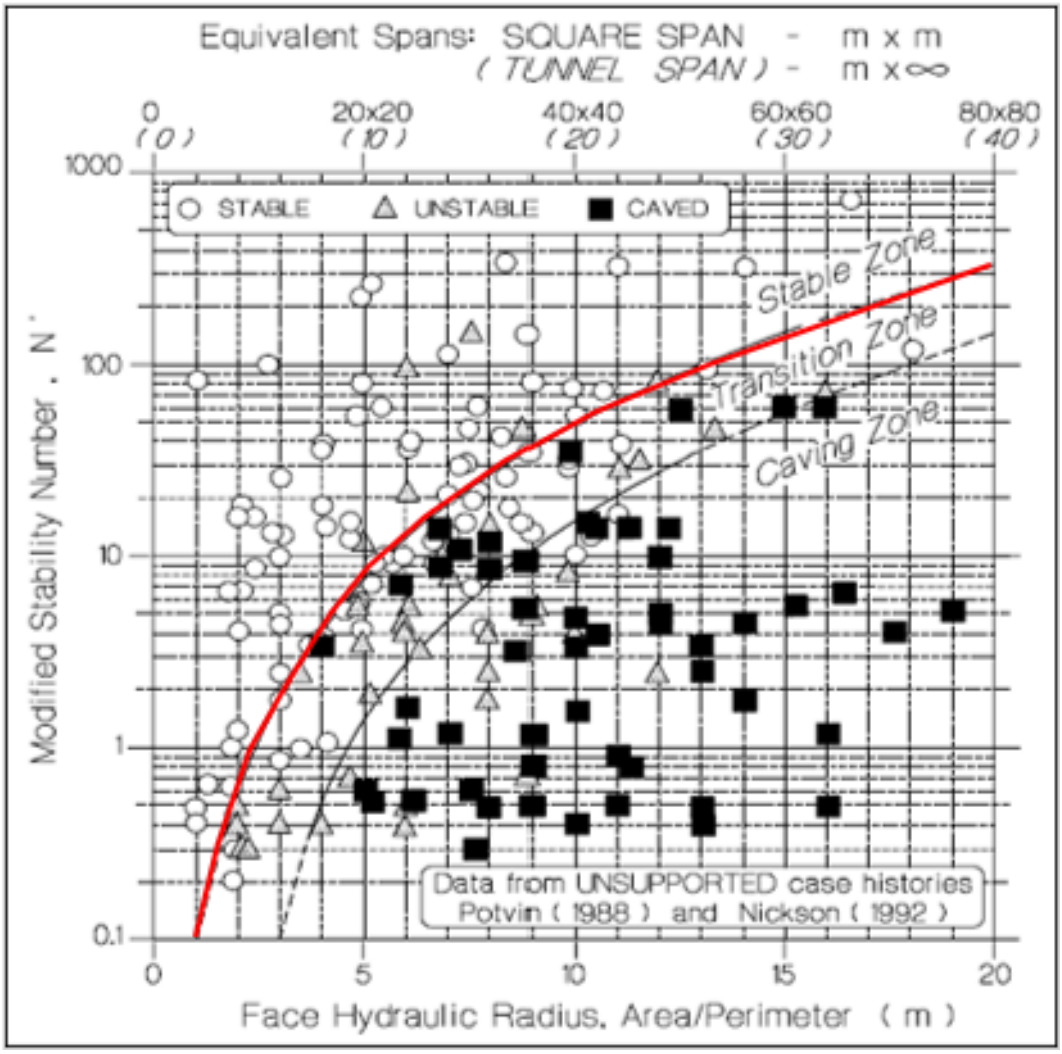


Figure 16.11 Potvin Stability Graph for Stable Unsupported Stope Spans. The red line indicates the Stable Unsupported Limit for which stope spans have been designed upon

The Hydraulic Radius was then determined for each mining area. Re-arranging the Hydraulic Radius (HR) equation allows for the calculation of Maximum Allowable Spans (i.e. we know the HR value and one of the span dimensions, therefore we can calculate the maximum allowable unsupported strike span).

The spans detailed in Table 16.5 are only indicative at this stage and a best estimate based on the data collected to date. They are considered appropriate for this study and initial mine planning and design activities but should be reviewed and updated once more detail is gained on the rock mass conditions during development mining and from the first areas to be stripped.

Zone	Parameters	Hanging Wall	Ore	Footwall
JL UCZ	Q' / N'	6.1 / 2.6	5.0 / 2.1	4.5 / 1.9
	HR	3.4	3.1	2.7
	Known Dimension (m)	5 m	5 m	5 m
	Allowable Strike Length	**30 m**	**30 m**	**30 m**
	Dilution Estimates	**0 – 0.5 m**	**0 – 0.5 m**	**0 – 0.5 m**
JL LCZ	Q' / N'	23.0 / 10.1	23.75 / 10.5	94.0 / 41.4
	HR	5.5	5.5	9.0
	Known Dimension (m)	5 m	5 m	5 m
	Allowable Strike Length	**30 m**	**30 m**	**30 m**
	Dilution Estimates	**0 – 0.5 m**	**0 – 0.5 m**	**0 – 0.5 m**
VVF	Q' / N'	2.3 / 1.0	-	-
	HR	2.5	-	-
	Known Dimension (m)	5 m	-	-
	Allowable Strike Length	**15 m**	-	-
	Dilution Estimates	**0.5 – 1.0 m**	-	-

Table 16.5 Summary of calculated maximum allowable spans

16.2.6 Pillars

Typically, in laterally extensive and flat dipping orebodies, the use of barrier pillars is used to compartmentalize the orebody into discrete mining areas in the event of a 'pillar-run'. In the JL UCZ, the use of barrier pillars is recommended for several reasons due to the spatial extent of the orebody, proximity to ground surface, and protection of main accessways through the orebody to maintain access long-term. Barrier pillars have been placed strategically across the JL UCZ to limit the distance between pillars to ~ 200 m. Barrier pillars are generally designed to be indestructible and have therefore been designed at ~30 m plan width from edge to edge. The width of the barrier pillars is a function of orebody thickness, depth of cover, and strength of ore/hangingwall/footwall rock units, amongst other factors.

Figure 16.12 displays the JL UCZ and barrier pillars that have been factored into the mine design for the purpose of this study. Final barrier pillar dimensions may require refinement once mining commences and greater orebody knowledge is gained.

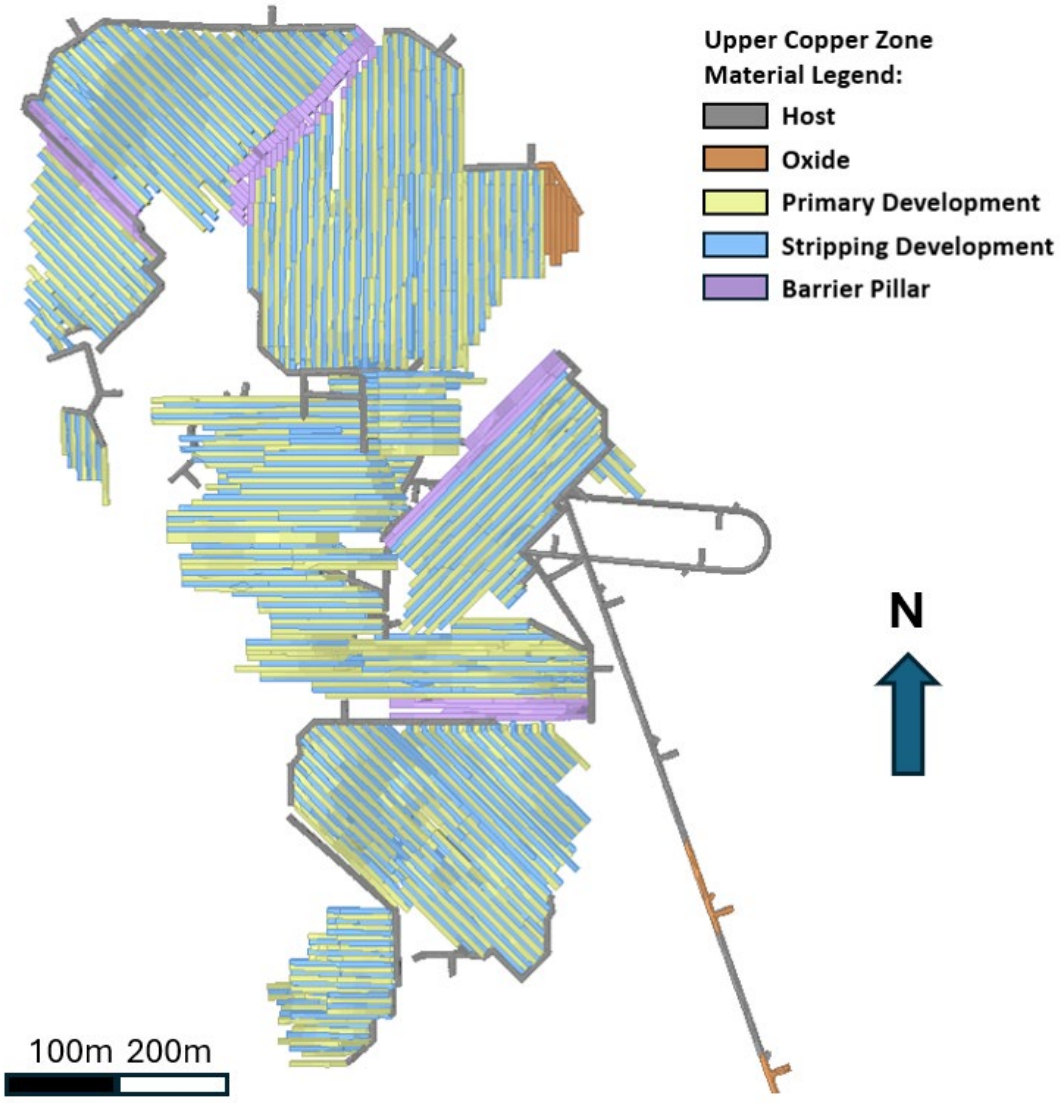


Figure 16.12 Plan view of JL UCZ showing placement of Barrier Pillars within the mine design (Entech, 2025)

16.2.7 Numerical Modelling

Numerical modelling of the Black Butte Underground Mine was undertaken using Rocscience RS3. Two linear, elastic models were built separating the JL UCZ and JL LCZ using planned development shapes and lithological and structural wireframes. The material properties were derived from the laboratory tests of samples while in-situ stress conditions were derived from Mine Design Engineering Report (Kalenchuk *et al*, 2015) in lieu of site-specific data.

The alternating primary – secondary sequencing with immediate paste fill is shown to be effective in restoring stresses to equilibrium which maintains confinement and reduces the risk of ground failure. The interaction

between the footwall and hanging wall lode in the JL UCZ are stable, provided that the footwall lode will be extracted first and tight paste filling of drives is undertaken.

In the JL LCZ, moderate stress distribution is observed around decline and level accesses. In addition, the influence of the VVF may impact the stability of capital infrastructure due to the reported low strength factors in these areas along decline and vertical development. When it comes to ore drive development, the weak material associated with VVF is mitigated by the application of immediate paste fill after the excavation of each ore drive.

Overall, the linear elastic model confirms the viability of the proposed mining sequence but also highlights the need for updated material strengths, in-situ stress data and detailed weathering profiles to improve confidence in the modelling results.

16.2.8 Ground Support

There are two main approaches to the selection and design of appropriate support patterns for an underground excavation, being *Empirical* and *Deterministic*. In the case of Black Butte mine, both techniques have been employed to determine the ground support and reinforcement requirements, particularly for ensuring adequate bolt density and surface coverage is taken into consideration for poorer rock mass conditions.

The Empirical approach requires that data points (visual observations or actual measurements) are collected and compared against a chart or equation which has been calibrated from many industry observations (such as 'support worked' or 'support failed') to establish repeatable trends.

In the case of deterministic support design, the orientation, length, spacing and nature of the discontinuities within the rock mass are directly sampled and recorded. These discontinuities are then used to estimate the size and geometry of potentially unstable wedges. A support pattern is then designed which is capable of restraining these unstable blocks. Typically, the support is applied in such a way that it has capacity which is in excess of its requirements, providing a measurable factor of safety against block failure.

For the kinematic assessment of wedges, the major structural sets identified were used to analyse the stability of the back (roof) and walls of typical excavations using the software package UnWedge®. UnWedge analyses combinations of 3 major planes at any one time and calculates all kinematically feasible wedges that can form within the excavation under consideration. For Black Butte, a number of structural sets were identified, with UnWedge selecting the 3 wedge forming joints in a particular orientation. To confirm all sets, mapping of development drives needs to be undertaken once underground mining commences in each lithology to confirm joint sets present. Mapping should be undertaken in varying orientations to ensure all potential sets are adequately sampled.

From the empirical and deterministic analysis, ground support standards were developed. For Black Butte, the ground support regime includes MDX/resin bolts as the primary support element for capital development until the JL UCZ turn out, then friction bolts will be the primary support element for the remainder of capital development. For all operating development, friction bolts will be the primary support element. Mesh is recommended as the primary surface support in fair ground or better, whilst the addition of fibrecrete will be required in poor ground and worse.

Figure 16.13 to Figure 16.16 show examples of the ground support standards developed for Black Butte. Table 16.6 provides details of the ground support elements for a range of rock mass conditions for each of the proposed development profiles.

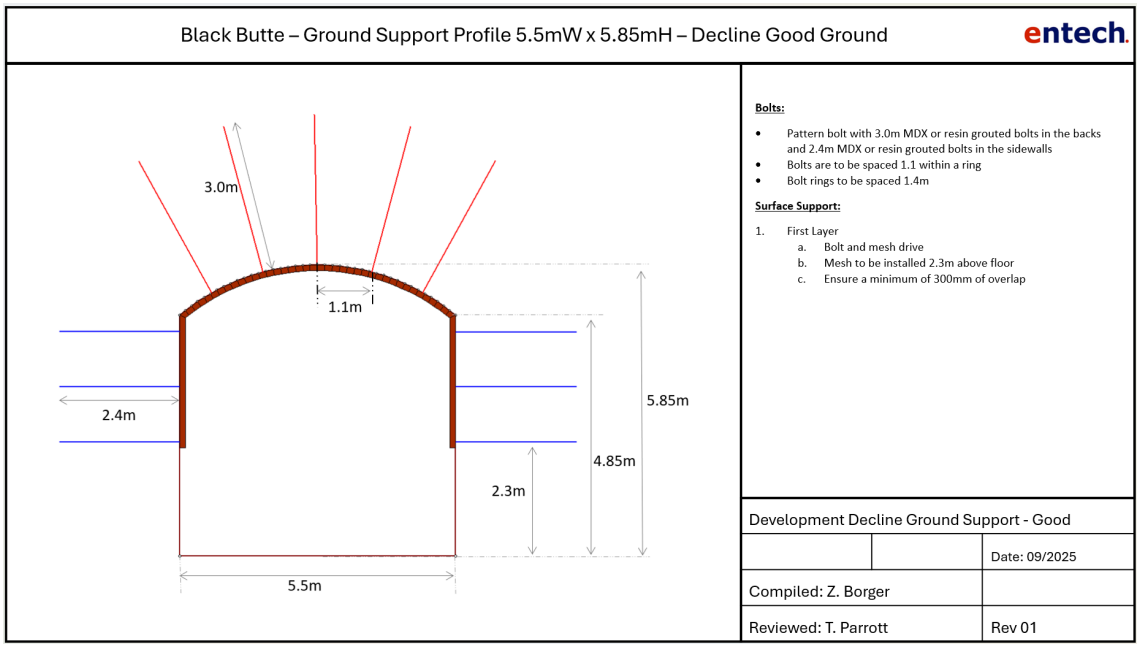


Figure 16.13 Ground support layout for Decline development - Good Ground

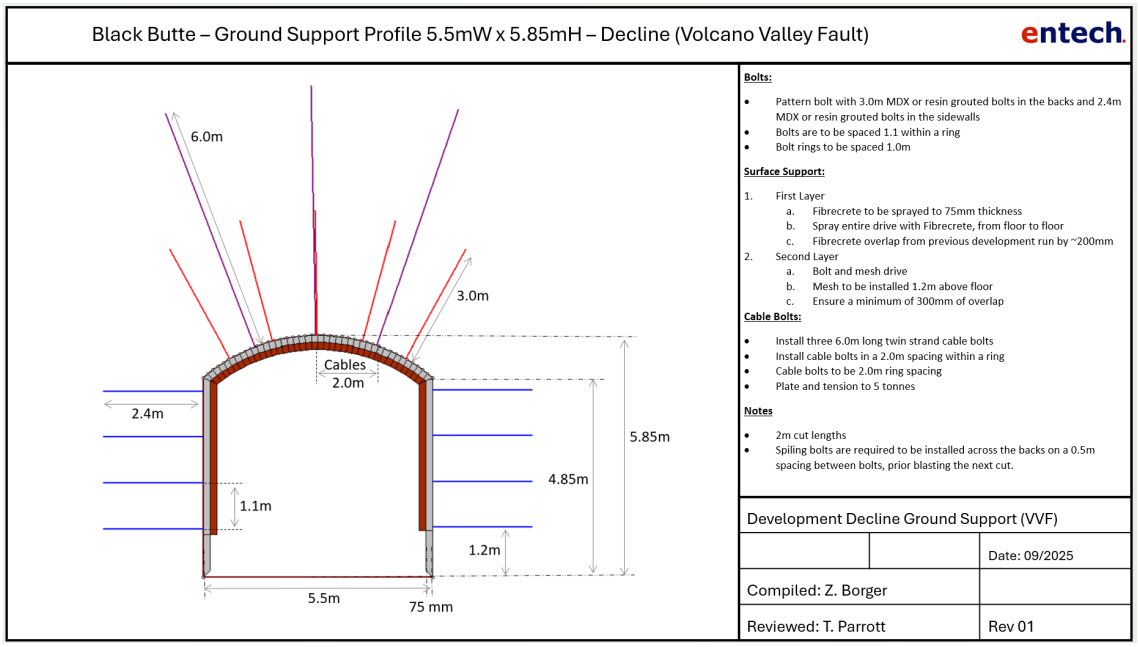


Figure 16.14 Ground support layout for Decline development - VVF

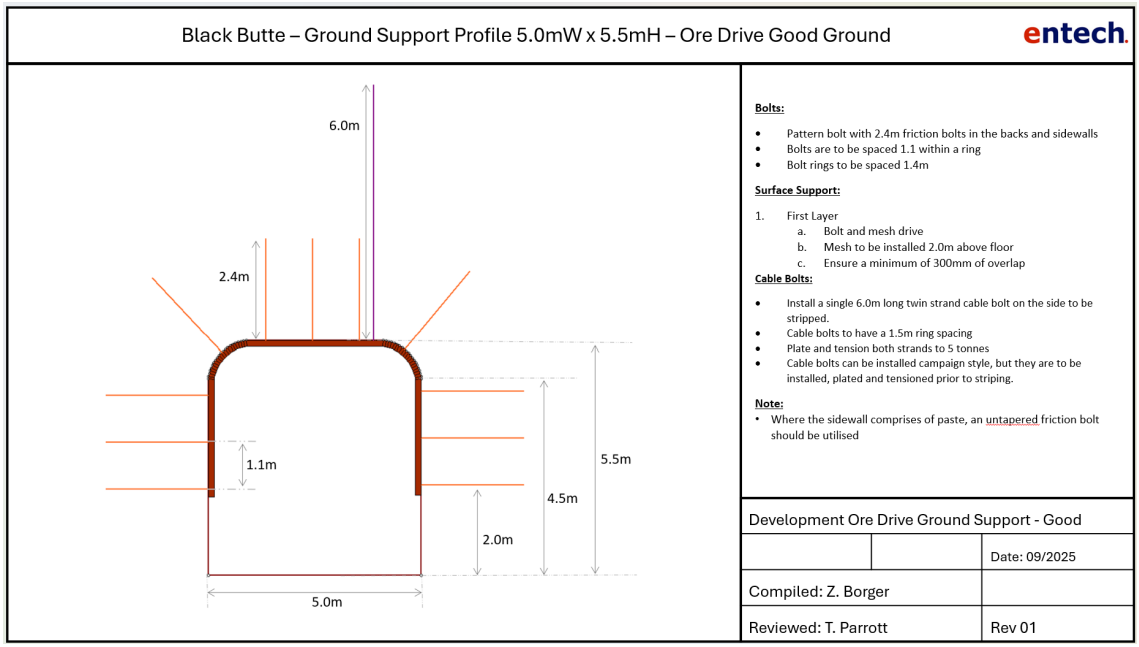


Figure 16.15 Ground support layout for ore drive development - good ground

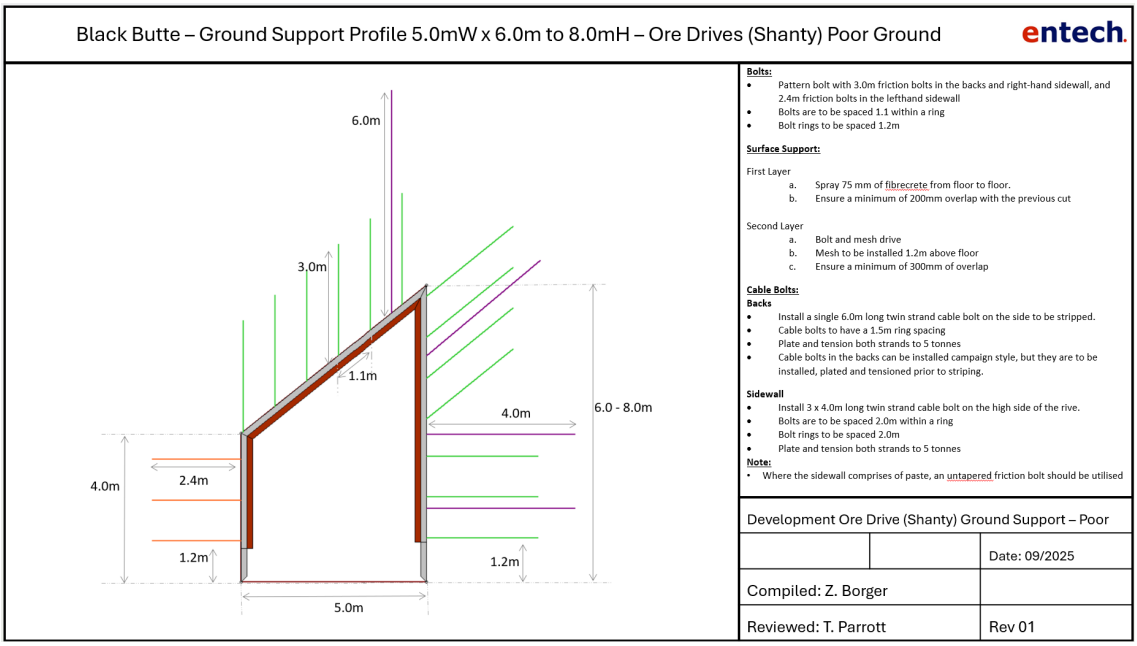


Figure 16.16 Ground support layout for ore drive development in JL LCZ - poor ground

Development	Ground Conditions	Reinforcement Elements	Bolt spacing	Surface support
5.5mW x 5.85mH (down to JL UCZ turn-off)	Good	3.0 m MDX/resin grouted bolt (backs) 2.4 m MDX/resin grouted bolt (sidewalls)	1.1 m ring spacing 1.4 m between rings	Mesh to 2.3 m from the floor
	Fair	3.0 m MDX/resin grouted bolt (backs) 2.4 m MDX/resin grouted bolt (sidewalls)	1.1 m ring spacing 1.2 m between rings	Mesh to 1.2 m from the floor
	Poor	3.0 m MDX/resin grouted bolt (backs) 2.4 m MDX/resin grouted bolt (sidewalls)	1.1 m ring spacing 1.2 m between rings	75 mm floor to floor Mesh to 1.2 m from the floor
5.5mW x 5.85mH (past JL UCZ turn-off to bottom of mine)	Good	3.0 m friction bolt (backs) 2.4 m friction bolt (sidewalls)	1.1 m ring spacing 1.4 m between rings	Mesh to 2.3 m from the floor
	Fair	3.0 m friction bolt (backs) 2.4 m friction bolt (sidewalls)	1.1 m ring spacing 1.2 m between rings	Mesh to 1.2 m from the floor
	Poor	3.0 m friction bolt (backs) 2.4 m friction bolt (sidewalls)	1.1 m ring spacing 1.2 m between rings	75 mm floor to floor Mesh to 1.2 m from the floor

Development	Ground Conditions	Reinforcement Elements	Bolt spacing	Surface support
	Volcano Valley Fault	3.0 m MDX/resin grouted bolt (backs) 2.4 m MDX/resin grouted bolt (sidewalls) 6.0 m twin strand cable bolts (backs)	Bolts 1.1 m ring spacing 1.0 m between rings Cable Bolts 2.0 m ring spacing 2.0 m between rings	75 mm floor to floor Mesh to 1.2 m from the floor
5.5mW x 5.5mH Capital Development (down to JL UCZ turn off and including capital development within JL UCZ)	Good	3.0 m MDX/resin grouted bolt (backs) 2.4 m MDX/resin grouted bolt (sidewalls)	1.1 m ring spacing 1.4 m between rings	Mesh to 2.4 m from the floor
	Fair	3.0 m MDX/resin grouted bolt (backs) 2.4 m MDX/resin grouted bolt (sidewalls)	1.1 m ring spacing 1.2 m between rings	Mesh to 1.3 m from the floor
	Poor	3.0 m MDX/resin grouted bolt (backs) 2.4 m MDX/resin grouted bolt (sidewalls)	1.1 m ring spacing 1.2 m between rings	75 mm floor to floor Mesh to 1.3 m from the floor
5.5mW x 5.5mH Capital Development (past JL UCZ turn-off to bottom of mine)	Good	3.0 m friction bolt (backs) 2.4 m friction bolt (sidewalls)	1.1 m ring spacing 1.4 m between rings	Mesh to 2.4 m from the floor
	Fair	3.0 m friction bolt (backs) 2.4 m friction bolt (sidewalls)	1.1 m ring spacing 1.2 m between rings	Mesh to 1.3 m from the floor
	Poor	3.0 m friction bolt (backs) 2.4 m friction bolt (sidewalls)	1.1 m ring spacing 1.2 m between rings	75 mm floor to floor Mesh to 1.3 m from the floor
5.0mW x 5.5mH Ore Drive development	Good	2.4 m friction bolt 1 x 6.0 m twin strand cable bolt to be installed in the side to be stripped	1.1 m ring spacing 1.4 m between rings	Mesh to 2.0 m from the floor
	Poor	2.4 m friction bolt 1 x 6.0 m twin strand cable bolt to be installed in the side to be stripped	1.1 m ring spacing 1.2 m between rings	75 mm floor to floor Mesh to 2.0 m from the floor
5.0mW x 5.5mH Ore Drive development	Good	2.4 m friction bolt 1 x 6.0 m twin strand cable bolt to be installed in the side to be stripped	1.1 m ring spacing 1.4 m between rings	Mesh to 1.5 m from the floor
	Poor	2.4 m friction bolt 1 x 6.0 m twin strand cable bolt to be installed in the side to be stripped	1.1 m ring spacing 1.2 m between rings	75 mm floor to floor Mesh to 1.2 m from the floor

Development	Ground Conditions	Reinforcement Elements	Bolt spacing	Surface support
5.0mW x 5.0mH Capital Access	Good	2.4 m MDX/resin grouted bolt	1.1 m ring spacing 1.4 m between rings	Mesh to 2.7 m from the floor
	Fair	2.4 m MDX/resin grouted bolt	1.1 m ring spacing 1.2 m between rings	Mesh to 1.5 m from the floor
	Poor	2.4 m MDX/resin grouted bolt	1.1 m ring spacing 1.2 m between rings	75 mm floor to floor Mesh to 1.2 m from the floor
5.0mW x 5.0mH Operating Access	Good	2.4 m friction bolt	1.1 m ring spacing 1.4 m between rings	Mesh to 2.7 m from the floor
	Fair	2.4 m friction bolt	1.1 m ring spacing 1.2 m between rings	Mesh to 1.5 m from the floor
	Poor	2.4 m friction bolt	1.1 m ring spacing 1.2 m between rings	75 mm floor to floor Mesh to 1.5 m from the floor
5.0mW x below 6.0mH Ore Drive (Shanty)	Good	3.0 m friction bolt (backs and right-hand wall) 2.4 m friction bolt (left hand wall) 1 x 6.0 m twin strand cable bolt to be installed in the side to be stripped	1.1 m ring spacing 1.4 m between rings	Mesh to 1.2 m from the floor
	Poor	3.0 m friction bolt (backs and right-hand wall) 2.4 m friction bolt (left hand wall) 1 x 6.0 m twin strand cable bolt to be installed in the side to be stripped	1.1 m ring spacing 1.2 m between rings	75 mm floor to floor Mesh to 1.2 m from the floor
5.0mW x 6.0mH to 8.0mH Ore Drive (Shanty)	Good	3.0 m friction bolt (backs and right-hand wall) 2.4 m friction bolt (left hand wall) 3 x 4.0 m twin strand cable bolts on the right hand wall 1 x 6.0 m twin strand cable bolt to be installed in the side to be stripped	1.1 m ring spacing 1.4 m between rings	Mesh to 1.2 m from the floor

Development	Ground Conditions	Reinforcement Elements	Bolt spacing	Surface support
	Poor	3.0 m friction bolt (backs and right-hand wall) 2.4 m friction bolt (left hand wall) 3 x 4.0 m twin strand cable bolts on the right hand wall 1 x 6.0 m twin strand cable bolt to be installed in the side to be stripped	1.1 m ring spacing 1.2 m between rings	75 mm floor to floor Mesh to 1.2 m from the floor
4.5mW x 4.5mH Sump	All	2.4 m friction bolt	1.1 m ring spacing 1.4 m between rings	Mesh to 2.1 m from the floor

Table 16.6 Summary of ground support standards for Black Butte underground mine

16.2.9 Geotechnical Recommendations

The following recommendations for future works are provided and should be carried out when appropriate:

- Further geotechnical data collection involving geotechnical logging of ongoing resource definition holes should be undertaken to provide a more comprehensive dataset in advance of mining. Where drilling is sparse, separate geotechnical holes may be required to allow rock mass characterisation of any key infrastructure areas.
- Ongoing rock properties testing should be conducted across all lithological units and mining domains to help verify the current rock properties assessed in this study and build an adequate site database. A focus should be made on collecting samples for UCS/UCSE and corresponding UTS samples and commence collection of samples for DST.
- Initial stripping should be rigorously reviewed, and adjustments made to design, maximum allowable spans and extraction sequencing as required. Learnings from production performance should be fed back into the design process.
- Mapping of occurrences of any water ingress and testing of water for UG corrosion potential.
- Detailed geotechnical mapping of ore drives is undertaken for input into stripping dimensions and design, and determining a more accurate characterisation of the rock mass with potential of drill hole orientation bias therefore removed.
- Continually updating the numerical modelling with the extraction and sequence, and as mining progresses.
- As mining progresses, a review of the ground support systems recommended here should be undertaken to incorporate updated geotechnical data to improve and justify the standards recommended here. This review needs to be undertaken immediately if issues with ground support/stability or QAQC results are encountered during mining and/or following any fall of ground events.

16.3 Selected Mining Methods

The mining methods selected are a mix of Drift and Fill (DF) and Cut and Fill (CF) mechanised mining methods with cemented pastefill used for post-mining void support.

Lateral development drilling will be conducted using electric-hydraulic jumbo drill rigs. Blasting will be carried out using emulsion explosives. Diesel-powered trackless load-haul-dump (LHD) units will load blasted material onto diesel trucks for haulage to surface. Longhole drill rigs will be used for DF production stripping and service holes.

An underground mining contractor is planned to be engaged to execute the initial 36 months of the mining schedule thereafter replaced by an owners mining team.

16.3.1 Upper Copper Zone Mining Methods

The JL UCZ is a flat dipping deposit with multiple stacked parallel lenses. It will be extracted using the DF mining method.

The DF panels will initially be developed from the haulage decline to the east, via access drifts developed around the perimeter for egress, ventilation and mining practicality. 5.0 mW x 5.0 mH primary ore drifts spaced 15 m apart will be developed from one access drift to the end of the panel. An unsupported 5.0 mW x 5.0 mH stripping drift adjacent to the primary drift will then be mined on retreat using longhole blasting and remote loading, to a maximum length of ~ 30 m along the drive based on geotechnical analysis. Following this, a paste barricade will be installed and the void paste filled, either from the wall, perimeter drive or adjacent primary drive. This cycle continues back to the barrier pillars at the access.

Once the primary ore drives are mined, stripped and filled, secondary ore drives at similar dimensions will be developed in the 10 m gaps between the filled primary voids. The same process will be conducted for these as that carried out for the primary drives.

Where lodes are stacked, the lower panel drives are mined and filled prior to commencement of the upper panel (overhand sequence).

The typical mining production sequence is shown in Figure 16.17.

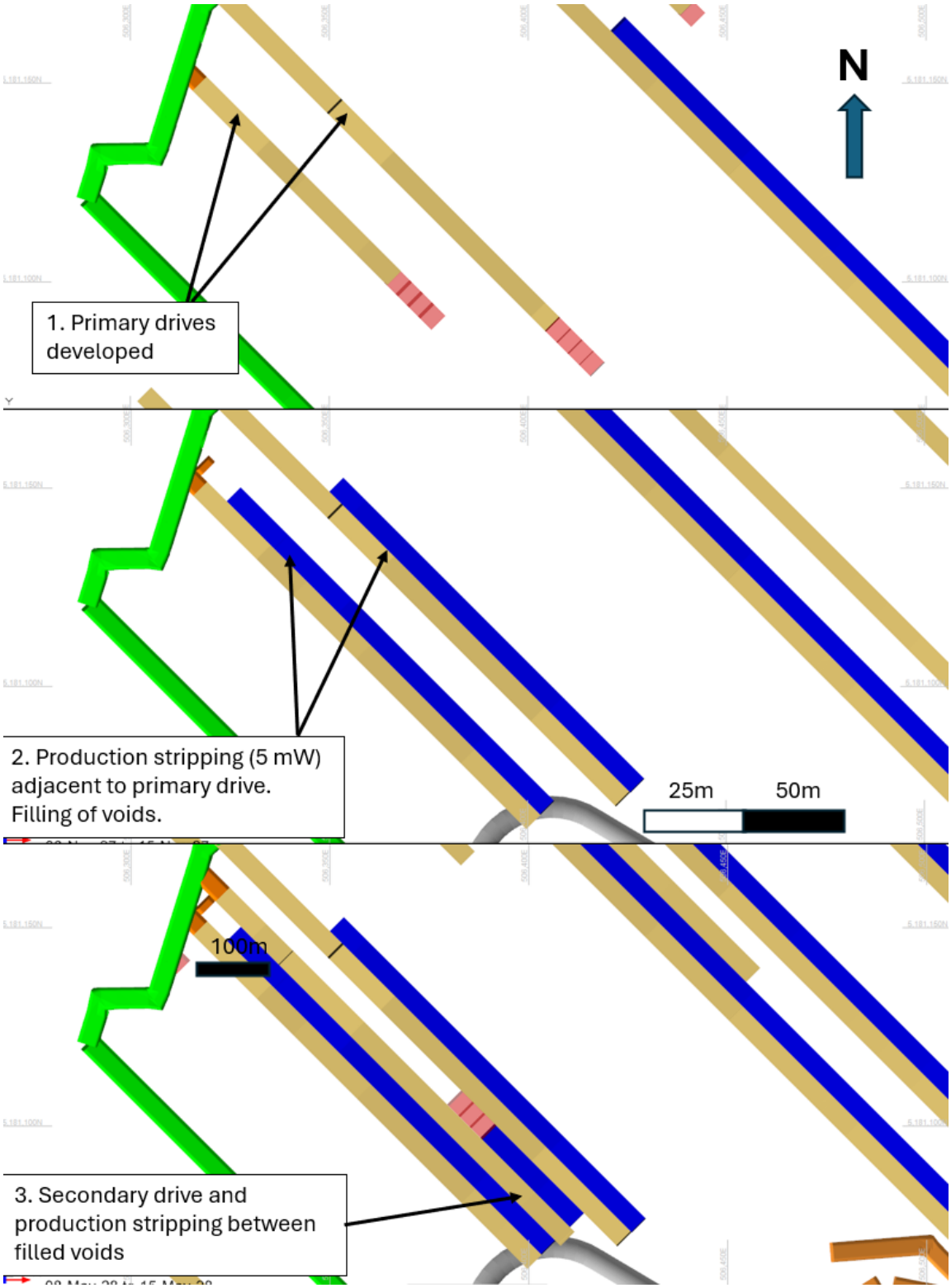


Figure 16.17 JL UCZ Production Mining Sequence (Plan View) (Entech, 2025)

16.3.2 *Lower Copper Zone Mining Method*

Where geotechnical conditions allow, the DF method applied in the JL UCZ is also applied to the JL LCZ. However, the majority of the JL LCZ mining area is planned to be mined using CF. Due to the orebody geometry dipping slightly steeper than the JL UCZ and no parallel lenses, multiple lifts will be taken from a single access point. The access will be driven from the JL LCZ ramp located in the orebody FW. In this method, extraction will commence from the lowest lift and progress up vertically. In each lift, 5.0 mW x 5.0 mH 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 JL LCZ. Figure 16.18 and Figure 16.19 below show a typical plan and section view schematic respectively of the JL LCZ mining.

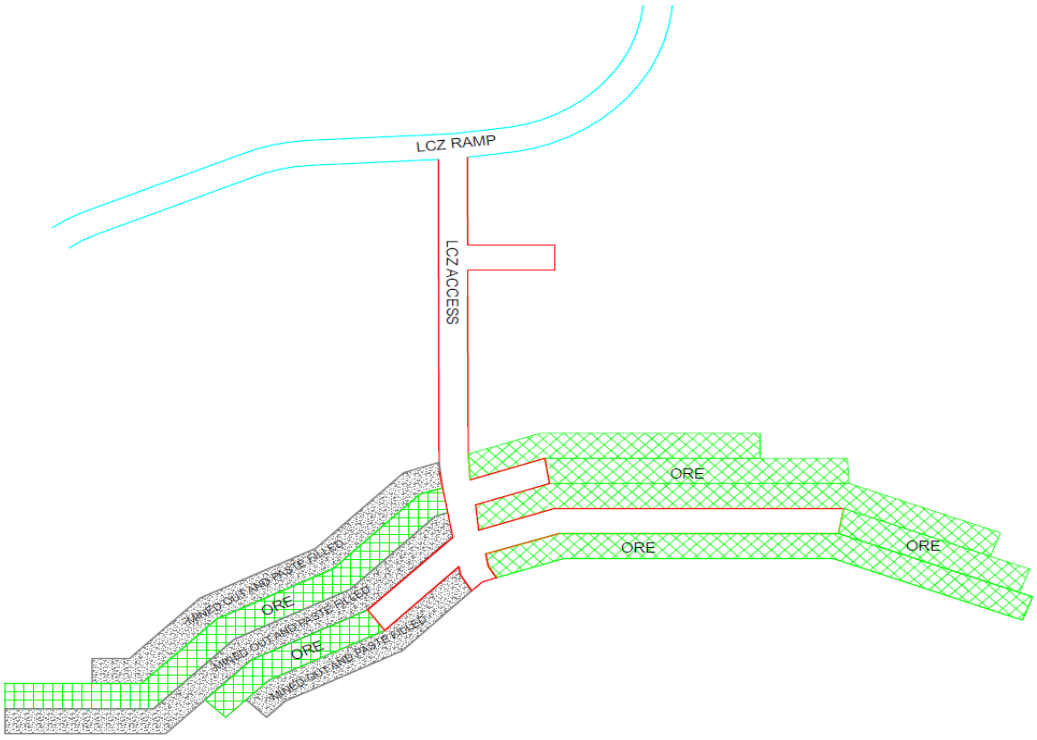


Figure 16.18 Plan View Schematic JL LCZ Ore Extraction (SRA 2022)

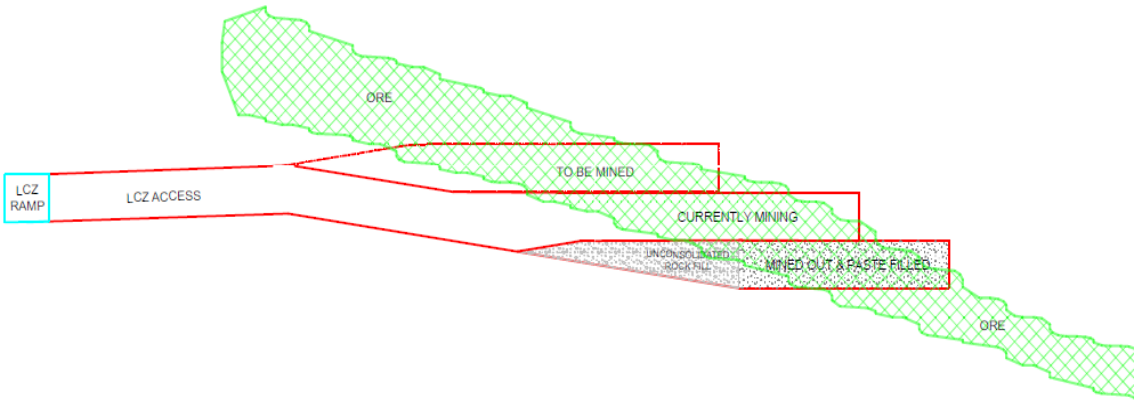


Figure 16.19 Section View Schematic JL LCZ Mining Method (SRA, 2022)

16.3.3 Operational Strategy

The operational strategy applied for this study assumes engagement of an experienced underground mining contractor for the initial 36 months of the mine plan. This will allow leverage of contractor experience in high-speed development and early site setup, with a switch to an owner-operator approach in the production phase to optimise ore quality control and mining costs.

Contractor input was provided in a request-for-quotation (RFQ) process involving multiple experienced and capable international mining contractors. These inputs were applied in the study for scheduling, fleet and personnel estimates, and costing.

16.4 Mine Design

The Johnny Lee deposit will be accessed by a single 5.5 mW x 5.8 mH Main Decline driven from an already established portal access point at surface. Stockpiles and sumps will be developed at 150 m intervals to assist in decline development rates.

The decline is designed at a maximum gradient of -14.3% (1-in-7 down) from the surface and will pass to the east of the JL UCZ. A single access drift will be driven from the decline into the JL UCZ. From this access drift an initial vertical ventilation connection to surface will be established via blind sinking methods, to allow for the shaft to be mined by the time development reaches this location for rapid establishment of a primary ventilation circuit. This access drift will subsequently serve as a take-off point for access perimeter drifts for mining the JL UCZ panels.

The decline will continue to spiral below the JL UCZ connecting to the return air and fresh air/ 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 fresh air intake and the second acting as the return airway and secondary egress, will be driven through the VVFZ to access the JL LCZ. Diamond drilling platforms have been planned along this decline for grade control drilling of the JL LCZ orebody.

The JL LCZ is made up of three zones, each of which will be accessed from a decline driven in the FW of the zones. Access drifts will be driven off the decline at regular intervals to access the ore. Each access will service three to four 5.0 mH lifts from a single decline turnout and stockpile. Lift access drives were designed at a maximum gradient as per the decline to allow for efficient use of the proposed fleet. Ore drifts were designed at 5.0 mW x 5.0 mH to allow the use of large-profile equipment, with shanty profiles against the HW contact to minimise waste dilution where appropriate.

All large diameter raisebore and surface shafts will be excavated by specialist contractors. All surface shafts were assumed to be blind sunk for scheduling and geotechnical stability purposes.

Figure 16.20 is an isometric view of the UG design looking from the SE.

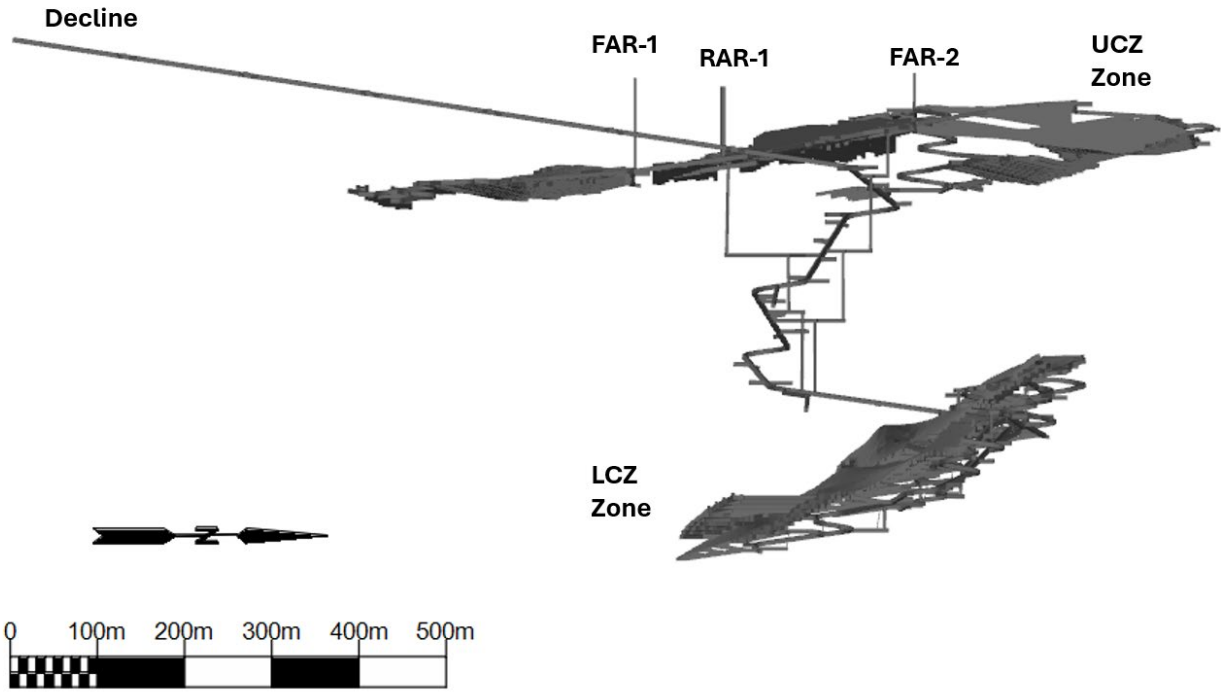


Figure 16.20 Isometric View UG Design from the SE (Entech, 2025)

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 backs and allow for maximum extraction of the ore.

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 paste 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 JL UCZ access. From there a borehole will distribute the paste to the JL LCZ FW decline where it will be piped to the final distribution. A second line will be installed into the JL 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/Bunds

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 16.21 below shows a similar style paste wall under construction prior to the application of shotcrete.



Figure 16.21 Paste Wall prior to Shotcrete Application (Minefill, 2022)

In cases where ore drives have a notable negative gradient from the fill point i.e. parts of the JL UCZ, paste wall barricades will be replaced with waste bunds.

16.6 Materials Handling

All material will be mucked using 21-tonne LHD's into stockpiles for loading and haulage to surface using 60 tonne trucks. Waste will be stockpiled on the waste pad until it can be deposited in the CTF as all waste has been designated as PAF. Ore will be stockpiled either on the surface ROM pad or LG/waste stockpile and fed to the surface crusher using a Front-End Loader (FEL)as required.

16.7 Ventilation

16.7.1 Primary Ventilation

The primary ventilation system at Black Butte mine will be structured to provide independent airflow to each of the JL UCZ and JL LCZ mining zones (Figure 16.22).

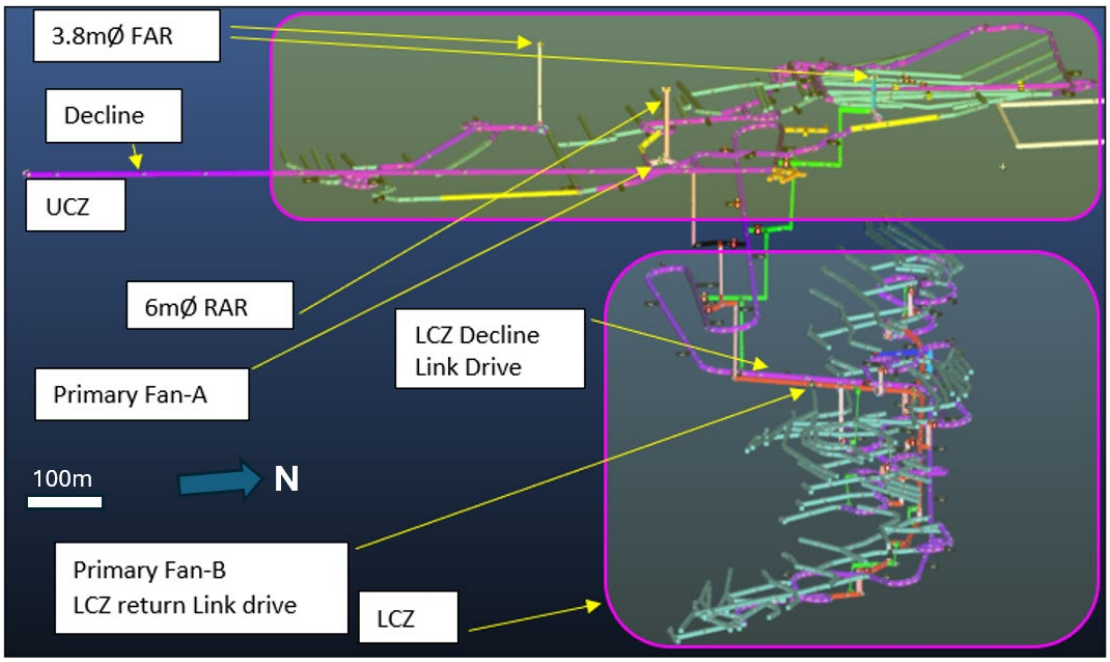


Figure 16.22 Primary Ventilation Plan Schematic (Entech 2025)

Each section is planned to be independently connected to a single ~30 m long, 6 mØ exhaust rise to surface (RAR-1). This will deliver 300 m³/s of airflow to the respective districts for a combined total of 600 m³/s. The primary fans are to be located underground to avoid surface noise restrictions. The JL UCZ fans will be located at the base of RAR-1, while the JL LCZ fans are to be located in the return link drive parallel to the decline.

It is proposed that a bank of 3-4 x 1.6 m Ø fans be installed at each station to maximise flexibility and redundancy whilst reducing costs by using direct drive 'off-the-shelf' fans (as opposed to custom made large diameter fans). This system will be capable of manipulating the airflow balance up to 100 m³/s between the JL UCZ and JL LCZ to provide operational flexibility. The nominal performance of each station is as follows:

- Station A (JL UCZ) - 3 x 300 HP (220 kW) 1.8 m diameter axial flow fans installed in a barricade.
  - Fan duty 300 m³/s @ 650 Pa, (~350 kW maximum power draw).
- Station B (JL LCZ) - 4 x 400 HP (300 kW) 1.8 m diameter axial flow fans installed in a barricade.
  - Fan duty 300 m³/s @ 2,600 Pa, (~1,100 kW maximum power draw).

Fresh airflow into the mine will be down the access decline as well as via two 3.8 m Ø surface fresh air rises (Figure 16.22). Fresh air for the JL LCZ will flow down the decline and a raise system parallel with the decline to the point where development crosses the VVFZ. At this point, all fresh air will be channelled through the access decline heading for ~160 m from the base of the fresh air raise system (Figure 16.23), through the VVFZ, before it splits to service the three declines proposed for JL LCZ.

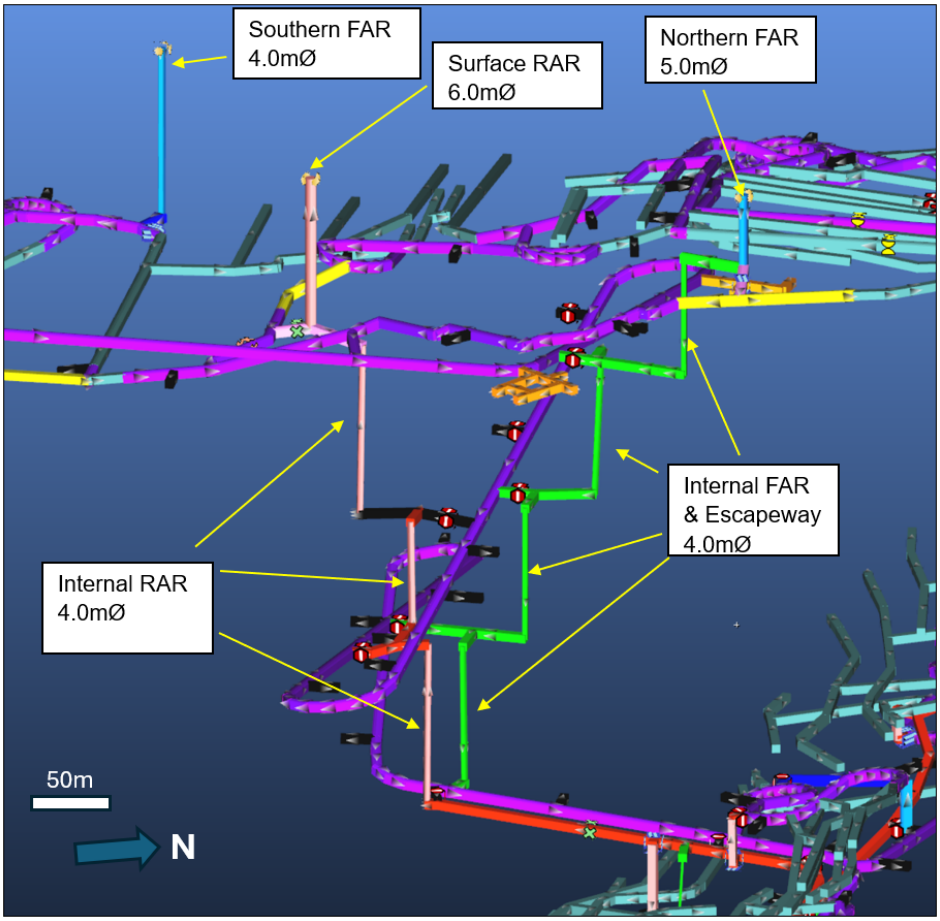


Figure 16.23 JL LCZ Fresh and Return Air Route (Entech 2025)

Airspeeds in the link-drive are expected to approach 9 m/s when there is a maximum of 300 m³/s flow to the JL LCZ. This will require appropriate dust management strategies in this portion of the access decline, or a secondary intake to be provided based on trade-off studies.

The proposed JL UCZ primary ventilation system has been designed to work on a flow-through ring-drive system due to the flat, tabular nature of the orebody. There are three primary ring loops (North/Central/South) each with a 100 m³/s airflow allocation. This system works to minimize re-entry delays by dividing up the JL UCZ into discrete ventilation districts, allowing airflow in any one district to be altered to meet production demands.

The JL LCZ was also divided into three ventilation districts based on the declines (West/Central/East). Each district is planned to be ventilated in series, with return air access off the decline as orebody topology makes on-level returns impractical. Each district was nominally allocated 100 m³/s, which may be altered depending on production demands.

At mine commencement, single heading ventilation prior to breakthrough of the RAR-1 will be via a single twin-stage 150HP (300 HP, or 220 kW total) 1400 mm Ø fan. At 500 m chainage, a second twin-stage 150 HP fan (with air duct) will then be installed in parallel. This system will suffice until installation of the Station A primary fans.

Based on the ventilation analysis, mine heating is forecast to be required in the colder months (November-February) to ensure safe working conditions are maintained underground. A diesel heating system was assumed for study costing.

16.7.2 Secondary Ventilation

A mix of twin stage 150 HP and 75 HP fans are anticipated to be used to ventilate up to 20 independent activities. 75 HP fans may have up to two open duct ends, while the 150 HP fans may have up to three to ensure sufficient airflow to operate a LH621 LHD. In working areas where LHD's are not in operation, a minimum of 7.5 m^3/s will be required to ensure sufficient air velocity in a standard 5 m x 5 m development drive.

16.8 Sequence and Schedule

The main decline and majority of capital UG infrastructure will be developed by a contractor during the initial 36-month period. All excavations after this time are planned to be developed by the owner.

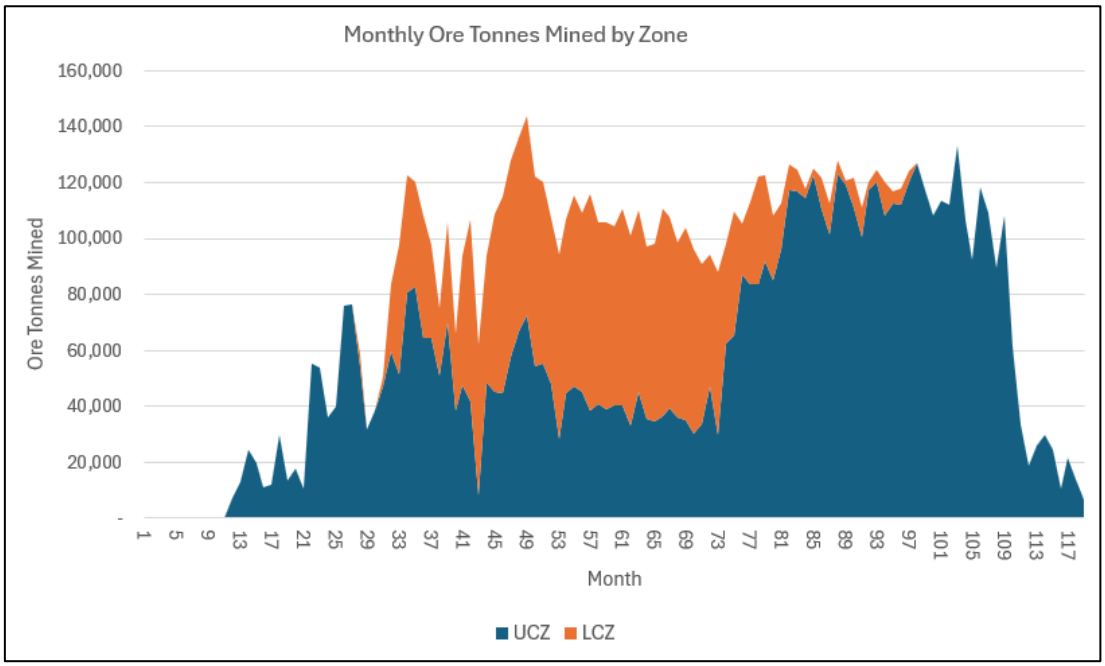
At commencement, the mining contractor will excavate and support the portal and commence development of the Main Decline. Simultaneously, sinking of RAR-1 will commence.

When the Main Decline reaches the JL UCZ take-off, a second heading will be developed to break into the RAR-1 which will be complete to design depth. This will establish a primary ventilation circuit in Month 9 and allow expansion of fleet operations.

The Main Decline will remain the focus of JL LCZ lateral development efforts for the initial 36-month period, to drive down to the higher-grade JL LCZ orebody as quickly as possible. Lateral development will also be mined in the JL UCZ, to establish FAR-1 and FAR-2 as well as to begin stockpiling ore for the mill.

Once the JL LCZ has been accessed the owner will shift a portion of the fleet to the JL LCZ and commence production in that area. Production will continue in both areas until they are depleted. The overall mine life from commencing development of the decline is just under 10 years. The production period is seven years.

The production mix is initially solely JL UCZ ore while the decline is being developed and the JL LCZ is being established. From approximately Month 37, the JL LCZ area is prioritised, and the JL UCZ area is mined at a rate to fill the remaining milling capacity. As the JL LCZ area is exhausted, mining focus shifts to the JL UCZ for the remainder of the mine plan, as shown in Figure 16.24.



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 16.7.

Parameter	Total	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11
Lateral Development (km)	**83.4**		1.5	8.8	12.1	13.8	11.6	9.9	8.7	7.9	7.6	1.6
Vertical Development (m)	**1,293**		-	643	386	265	-	-	-	-	-	-
Waste Tonnes (kt)	**1,799**		124	561	507	348	136	42	30	23	24	4
Paste Backfill ('000 m^3)	**2,331**		-	-	197	196	334	332	383	393	367	129
Ore Tonnes (Mt)	**9.5**		0.0	0.3	0.9	1.2	1.4	1.2	1.3	1.4	1.4	0.4
Ore Head Grade (% Cu)	**2.9**		2.0	1.7	2.8	3.1	3.5	3.7	3.0	2.4	2.2	2.1
Ore Metal (kt Cu)	**270**		0	5	25	37	47	45	40	34	30	7

Table 16.7 Mine Physicals Summary

The nominal annual production rate is 1.2-1.4 Mtpa (~110-120 kt/month) as shown in Figure 16.24.

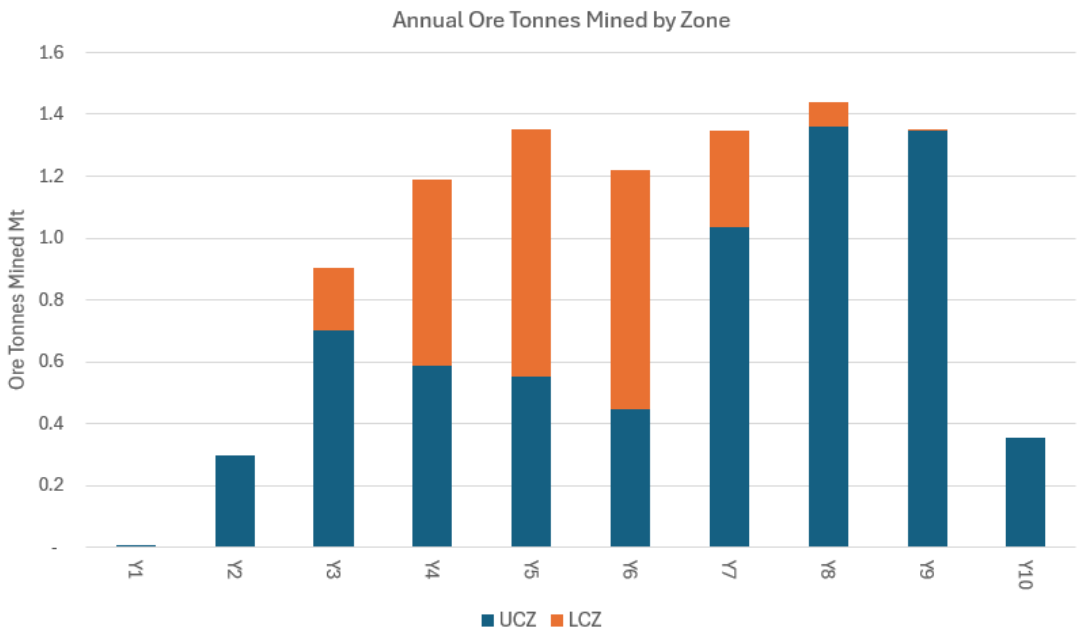


Figure 16.24 Ore Tonnes Mined Split by Zone

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 total owner and contractor key equipment fleet estimate by year over the life of the Project is illustrated in Table 16.8 (owner fleet from Y4). Contractor fleet estimates were sourced from the RFQ process, and the owner fleet determined from first principles with benchmarked productivity assumptions.

Equipment	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11
Jumbo		1	4	4	6	6	5	4	4	3	3
Loader		1	4	5	5	5	4	4	4	4	4
Haul Truck		1	3	5	6	6	5	4	4	4	4
Production Drill		-	1	1	1	1	1	1	1	1	1
Shotcrete Sprayer		1	2	2	2	1	1	1	1	1	1
Shotcrete Truck		1	2	2	2	2	2	1	1	1	1
Charge Rig		1	3	3	3	2	2	2	2	2	2
Integrated Tool Carrier/Multilift		2	4	5	5	4	3	3	3	3	3
Water Truck		1	1	1	1	1	1	1	1	1	1
Grader		1	1	1	1	1	1	1	1	1	1
Light Vehicles		13	21	22	22	19	18	17	18	17	17

Table 16.8 Forecast Underground Mine Equipment Fleet

16.11 Organization and Structure

Based on contractor RFQ submissions the proposed structure of the contractor will most likely be a hybrid arrangement with both fly-in/fly-out international expatriates and locally sourced personnel on-site during the initial 36-month development phase.

Owner personnel are planned to be sourced in-country.

All shifts were assumed to be 12 hours in duration, with the mine running seven days/week, 365 days/year. Personnel will be employed on a residential, drive-in/drive-out (DIDO) or fly-in/fly-out (FIFO) basis with accommodation provided at an SRA-supplied near-site accommodation village.

The proposed mine organization and structure is summarized in Table 16.9 below.

Position	Rotation (days on-days off)	Pre-Production Average (First 36 Months)	Production Average	Peak
Contractor Personnel - Expat				
Mining Contractor Staff	Expat 35-28 Local 8-6	28	0	32
Mining Contractor Operators	Expat 42-42 Local 7-7	107	0	174
Mining Contractor Maintenance	Expat 42-21 Local 7-7	37	0	56
Technical Services and Management - Owner				
Geology Manager	8-6	1	1	1
Senior Mine Geologist	8-6	1	1	1
Mine Geologist	8-6	1	1	1
Production Geologist	8-6	1	2	2
Core Tech	8-6	3	0	4
Senior Geotechnical Engineer	8-6	1	1	1
Geotechnical Engineer	8-6	1	1	1
Mining Manager (Registered UGM)	8-6	1	1	1
Senior Mining Engineer/Alt UG Manager	8-6	1	1	1
Technical Superintendent	8-6	1	1	1
Production Engineer	8-6	1	2	2
Planning Engineer	8-6	2	2	2

Position	Rotation (days on-days off)	Pre-Production Average (First 36 Months)	Production Average	Peak
Ventilation Engineer/Officer	8-6	1	2	2
Mine Technician	8-6	2	2	2
Senior Surveyor	8-6	1	1	1
Surveyor	8-6	2	2	2
Mine Operations - Owner				
Mine Foreman	8-6	0	1	1
Mining Supervisors	7-7	0	5	8
Site Administration	8-6	0	1	1
OH&S Supervisor/Coordinator	8-6	0	1	1
OHS/Trainers	8-6	0	2	2
Maintenance Superintendent	5-2	0	1	1
Maintenance Foreman	8-6	0	1	1
Underground Electrical Superintendent	7-7	0	1	1
Maintenance Administrator	8-6	0	1	1
Purchasing Officer	8-6	0	2	2
Maintenance Planner	8-6	0	1	1
Owner UG Mining Operators	7-7	0	115	170
Owner UG Mining Maintenance	7-7	0	32	48

Table 16.9 Organization and Structure Summary

16.12 Underground Infrastructure and Services

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 travel ways to provide enhanced communications and control.

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 one of the fresh air raises.

Each feeder will be connected to substations that will step down the voltage to 1,100 V for finalized distribution and usage.

Forecast mine power usage is shown in Table 16.10.

	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11
Power Consumption (MWh)		6,432	25,466	36,021	48,412	43,312	41,375	37,979	36,730	24,130	19,408
Average Power Draw (MW)		0.7	2.9	4.1	5.5	4.9	4.7	4.3	4.2	2.8	2.2

Table 16.10 Forecast Mine Electrical Power Consumption

16.12.3 Compressed Air

The contractor rates include provision of a central air compressor and compressed air reticulation through the workings during the initial 36 months of operation.

A central mine air compressor is planned to be retained for the owner-operator production period, using the installed reticulation system.

16.12.4 Potable Water

UG 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 an adequate buffer capacity. Peak water inflow is estimated to be 3,000 m^3 per day (550 gal/min or 35 L/s) when all capital development has been completed and full production is achieved.

The main dewatering station consisting of 2 x 90 kW centrifugal pumps will be located off the decline below the JL UCZ. All water in the mine will report to the main sump where it will be decanted and filtered prior to being pumped to the water treatment plant on surface through a steel rising main routed through FAR-1. A portion of the water will be stored at an UG water recycle dam and used as service water. The designed capacity of this primary pumping system is 40 L/s.

A 'travelling' 55 kW positive displacement pump will be advanced with the Main Decline development to pump to the main sump from the face. When decline distance exceeds the effective pumping distance of this pump, a permanent primary pump station consisting of 2 x 55 kW positive displacement pumps will be installed to stage the water flow up to the main dewatering station. Once the declines are complete there will be five pump stations in this arrangement, dropping back to two as the JL LCZ mining area is completed.

Water at development faces and in the production areas will drain to temporary sumps for collection and pumping by electric centrifugal submersible pumps to the nearest primary pump station.

There are geologic domains permissive of carrying water within the vicinity of the workings, particularly the Neihart Quartizite, located north of the Buttress Fault, which is currently interpreted to be located within 50m of the LCZ capital development. Hydrological/hydrogeological water inflow forecasts have been determined based on analysis commensurate with the level of study. However these do not assume that major water bearing structures are intersected. If intersected, there is a risk that excessive water inflow may impact on development rates and increase pumping costs. Further analysis is recommended and planned in forward works to increase confidence in the location and potential impact of water bearing structures.

16.12.6 Service Water

Service water will be supplied from two main water tanks located at underground access portal. This will be sourced from site water bores prior to treated water from underground dewatering being reused as required. Service water will be distributed through the main access declines HDPE pipe. Pressure management throughout is facilitated by installation of pressure reduction valves.

16.12.7 Refuelling and Service Bay

No underground workshops are planned due to the relatively shallow nature of the works. Minor maintenance such as daily servicing will be conducted at the work area or surface workshop. Major servicing and repairs will take place at the surface workshop.

Fleet units taken to surface regularly will be refuelled at the surface diesel dispensing point. Relocatable refuelling pods will be installed in suitable areas connected to the return air system as the mine progresses for other fleet equipment.

16.12.8 Explosives Magazine and Use

A surface magazine is planned which will be located in the southwest corner of the mining permit area in an area where topsoil and subsoil stockpiles are planned. Stockpiles will be reconfigured to act as bunds to the magazine.

The main UG storage facility is planned to be located off the Main Decline in a location protected from vehicle traffic. This facility will be equipped with a five-tonne jib crane that will facilitate the delivery of explosive products from the surface storage magazine. Emulsion is planned to be used as the bulk explosive.

16.13 Grade Control

16.13.1 Introduction

Grade control (GC) drillhole spacing requirements have been determined to support the conversion of areas of the Johnny Lee deposit currently classified as Indicated Resources to the Measured Resource category. For the JL UCZ, GC drilling is planned be spaced at approximately 15 m East–West and 20 m North–South. In the JL LCZ, GC holes are planned to be spaced at 20 m between fans (East–West) and approximately 15 m between holes within each fan (North–South). These spacings meet the criteria adopted for Measured Resource confidence and have been utilised to develop the grade control drilling program for the deposit.

16.13.2 Upper Copper Zone

Given the shallow depth of the deposit and lack of suitable UG platforms for drilling, it is assumed that all drillholes will be completed from surface. Drillholes are planned to be collared using an RC tri-cone bit to advance through overburden and weathered material and then diamond drilled once competent rock is encountered.

Suitable surface collar positions were identified using existing or previously permitted drill pads. To minimize environmental impact, drill planning was undertaken to allow as many drillholes as possible to be completed from each surface pad. It is estimated that approximately 15 to 25 pads will be required to complete the JL UCZ GC drilling program.

The planned JL UCZ GC drill program comprises a total of 110,000 m of drilling, to be completed over three campaigns of 44,000 m, 33,000 m and 33,000 m, respectively. The initial and largest campaign will focus on the JL UCZ capital development ring drives and initial ore production areas.

16.13.3 Lower Copper Zone

Drilling for the JL LCZ will be undertaken from various development drives accessed from the Main Decline, totalling approximately 700 m. A total of 48,400 m of grade control drilling is planned for the JL LCZ. Of this, 42,000 m will be drilled from the hangingwall to ensure timely delivery of GC drilling ahead of production. The remaining 6,400 m will be drilled from the footwall using shorter holes from existing capital development.

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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 t/h will provide for the annualized 1.2 Mt/a treatment of underground (UG) sulphide mineralization, producing 116 kt/a 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 copper rougher and scavenger flotation circuit. Circuit modifications from the previous study have moved the Jameson cell from the pre-cleaner duty to a flash cleaner duty prior to the regrind stage to float any liberated copper minerals.

Concentrate from copper rougher and scavenger flotation stage will be pumped to a Jameson cell acting as a pre-cleaner (flash) cell prior to regrind. Concentrate recovered by the pre-cleaner cell will report directly to the final concentrate sump. Jameson cell tailings will then report to the regrind circuit. Product from the regrind circuit will then be cleaned by three stages of cleaning in conventional flotation cells. Copper scavenger tailings, together with the first copper 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 copper cleaner combined with the pre-cleaner (flash) concentrate, will form the final copper concentrate at an average grade of 22.5% Cu. The copper 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.

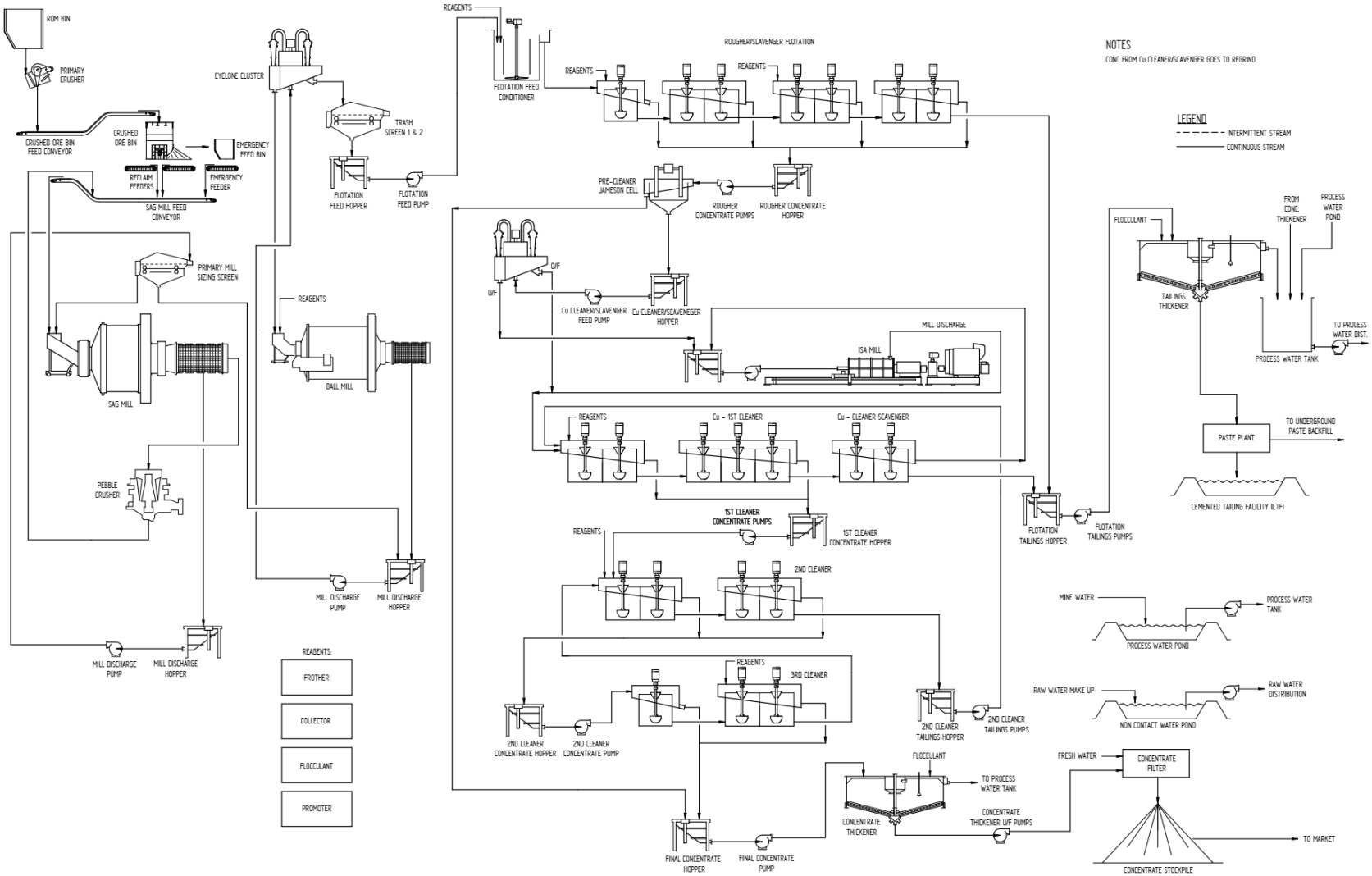


Figure 17.1 Plant Flowsheet

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	t/a	1,204,500
Feed Grades: ▪ Cu (LOM).	 %	 2.9
Copper Concentrate: ▪ Cu Grade. ▪ Cu Recovery (LOM). ▪ Concentrate produced (LOM ave).	 % % t/a	 22.5 85.8 129,000
Operating Hours: ▪ Crushing Circuit – UG Ore. ▪ Milling Circuit – UG Ore.	 hours hours	 5,694 8,059
Treatment Rate: ▪ Crushing-UG Ore. ▪ Milling Circuit – UG Ore.	 t/h t/h	 270 150
Primary Grind Size (P_{80}). Concentrate Regrind Grind Size (P_{80}).	µm µm	38 10-15

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. The copper plant tails will be converted to paste for use as underground (UG) backfill or stored in a new surface cemented tailings facility (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 separate stockpile fingers according to ore type and grade to facilitate blending. Stockpiled material will be reclaimed by a Front-End Loader (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

Method	Units	Value (80th Percentile)	Design Value
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 Copper Flotation Circuit

The copper minerals, chalcopyrite and tennantite, will be recovered from the pyrite and non-sulphide gangue throughout the copper 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 copper 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 copper 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 the Jameson re-cleaner (flash) cell being a E2532/6 rectangular J-cell. The circuit has been modified to produce a final grade concentrate of 26.2% Cu and improve the overall recovery by effectively recovering 53% of the copper prior to the regrind stage. J-cell tails will be directed to the regrind circuit, 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 IsaMill™. The regrind feed cyclone overflow will be combined with the regrind mill discharge and pumped to the 10 m³ pre-cleaner conditioning tank.

Copper cleaning will be conducted in three stages and consist of:

- An open circuit first cleaner/cleaner scavenger, and
- Closed circuit second and third cleaners.

The first cleaner flotation stage will consist of five conventional 16 m³ cleaner cells followed by two 16 m³ cleaner scavenger cells. The concentrate from the first copper cleaner cells will be pumped to the second copper cleaner cells for further upgrading, while the concentrate from the cleaner scavenger cells will be recirculated to the rougher/scavenger regrind circuit after the cyclones, to avoid misreporting around the regrind mill. 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 four 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 copper cleaning stage (3rd cleaners) will be combined with the Jameson cell concentrate to produce the final concentrate of 22.5% Cu with a combined recovery of 85.8% Cu (LOM average). 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 flash cell concentrate.

17.3.6 Concentrate Production

17.3.6.1 Concentrate Thickening

Final copper concentrate slurry, filtrate from the Cu filter and spillage from the Cu concentrate area, will be pumped to the feedwell of the copper concentrate thickener. A 12 m diameter high-rate copper 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 copper 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 copper concentrate thickener overflow will be pumped to the tailings thickener overflow tank.

17.3.6.2 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 Lorax type PF plate and frame type pressure filter which has been selected due to the fine particle size distribution of the concentrate. A narrow chamber depth of 33 mm has been used for sizing based on the test work results to realize the target concentrate moisture content of 10%. A PF60-60 has been selected for the required duty. The filter will operate automatically via a dedicated vendor supplied programmable logic controller (PLC) and operator interface system linked to the plant control system. The filter filtrate will be pumped to concentrate thickener.

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 filter feed pumps will transfer the material at a feed density of 55% to 60% solids to two horizontal plate and frame filters (Ishigaki MCDGC 2100 x 100/40). 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.

17.4.1.1 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.

17.4.1.2 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.

17.4.1.3 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.

17.4.1.4 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.

17.4.1.5 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.

17.4.1.6 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.

17.4.1.7 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.

17.4.1.8 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.

17.4.1.9 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 (MCCs). 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 PLCs 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

17.5.2.1 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.

17.5.2.2 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.

17.5.2.3 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 100,000 m^3 and will be used to store process water as the main storage facility.

17.5.2.4 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.

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 and general administration is 114 employees. The processing structure provided is considered adequate to maintain and operate a plant of this size and complexity.

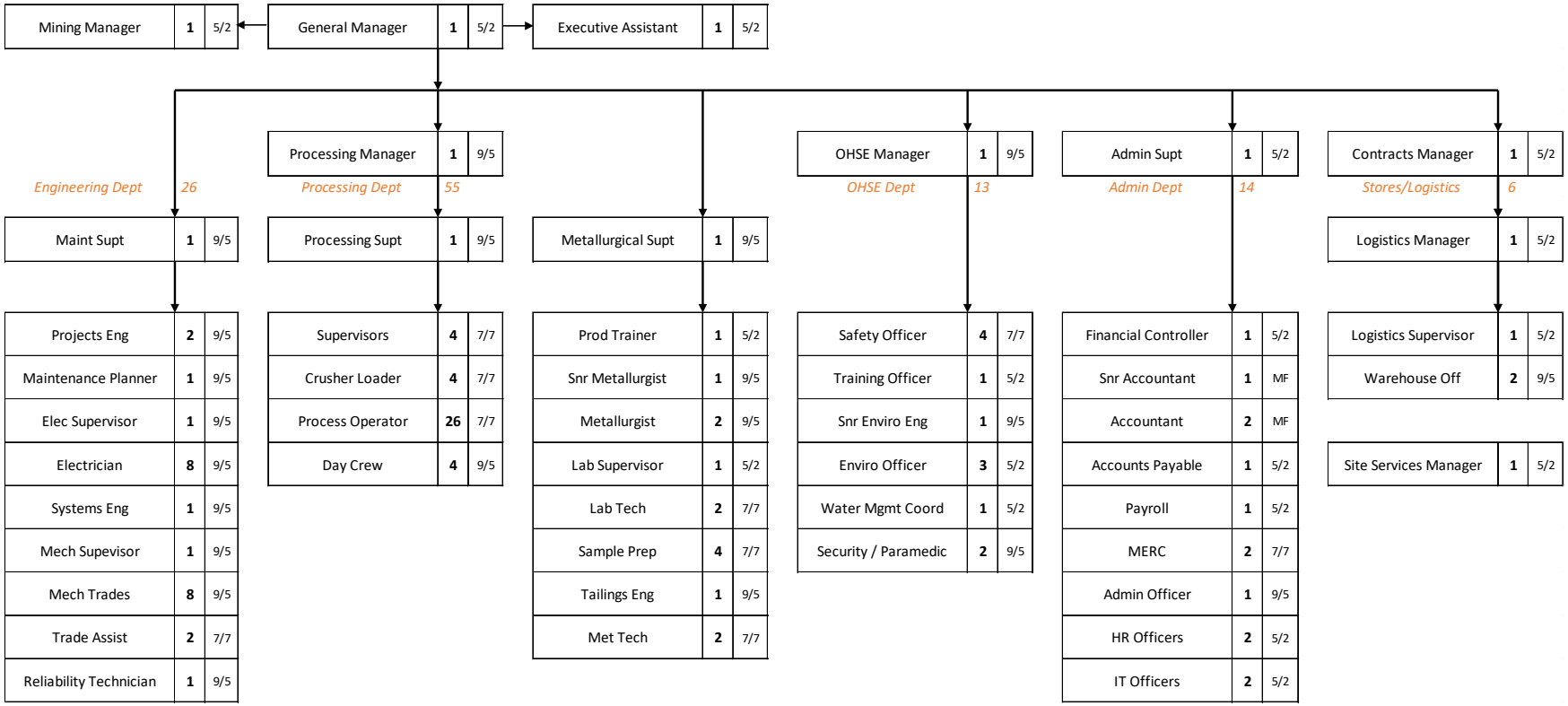


Figure 17.2 Processing and G&A Organizational Structure

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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:
  - Process 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).
  - Process Water Pond (PWP).

- Brine/Contact Water Pond (BCWP).
- Non-Contact Water Reservoir (NCWR).
- Cemented Tailings Facility (CTF), and
- Site supported equipment.

The locations of the main Project facilities are shown in Figure 18.1.

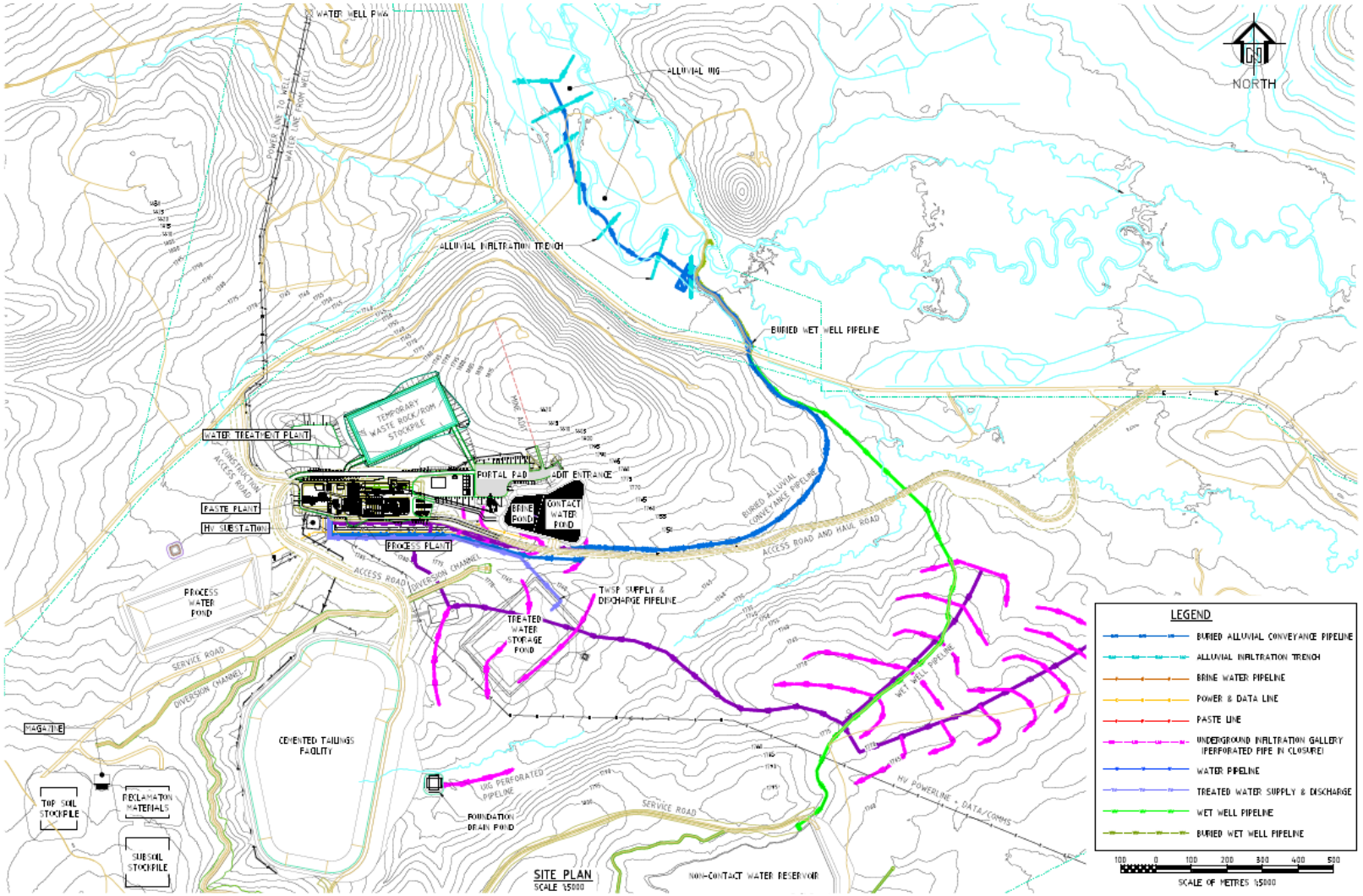


Figure 18.1 Overall Site Plan

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), FS design drawings (Feasibility Study (Johnny Lee Deposit) and Mineral Resource Estimate Update (Lowry Deposit) -Technical Report NI 43-101, 2020) and layout updates made for the PFS 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 grid 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 (2020b; 2020d, 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 17 August 2020 focused on the BCWP (Figure 18.2), portal pad, WRS pad, associated access roads, equipment laydown areas, and borrow pit located in the northern part of the CTF excavation footprint.

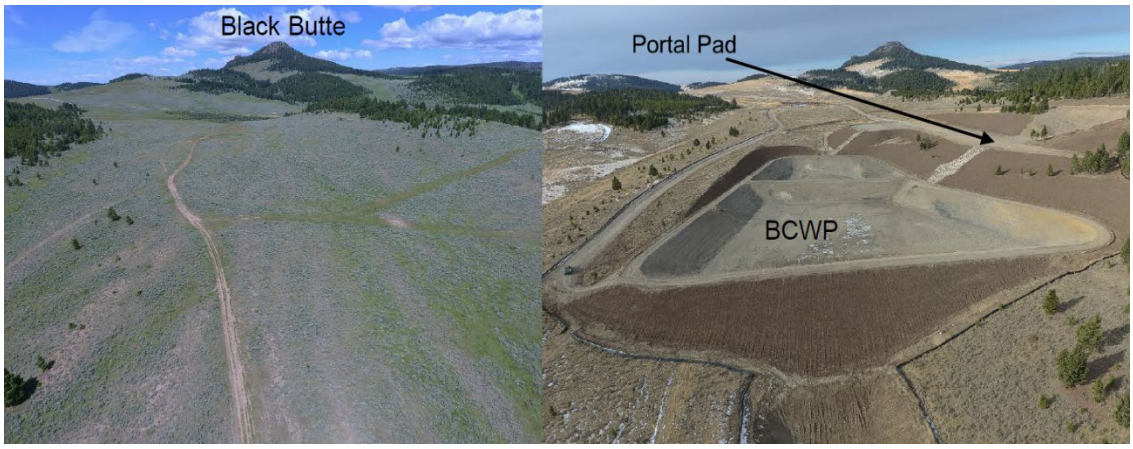


Figure 18.2 Photo looking west at the Brine-Contact Water Pond & Portal Pad

The 2020 Phase 1 as-built construction facilities were constructed from 17 August 2020 to 17 December 2020 and contained a total surface disturbance of 57.4 acres that included: portal pad and brine-contact water pond (BCWP) excavations (and installing the rock drains between them), establishing most of the material stockpiles, constructing the required temporary roads to these facilities (excluding the road to Sheep Creek County road), and aggregate production (crushing and screening of granodiorite) from a borrow pit.

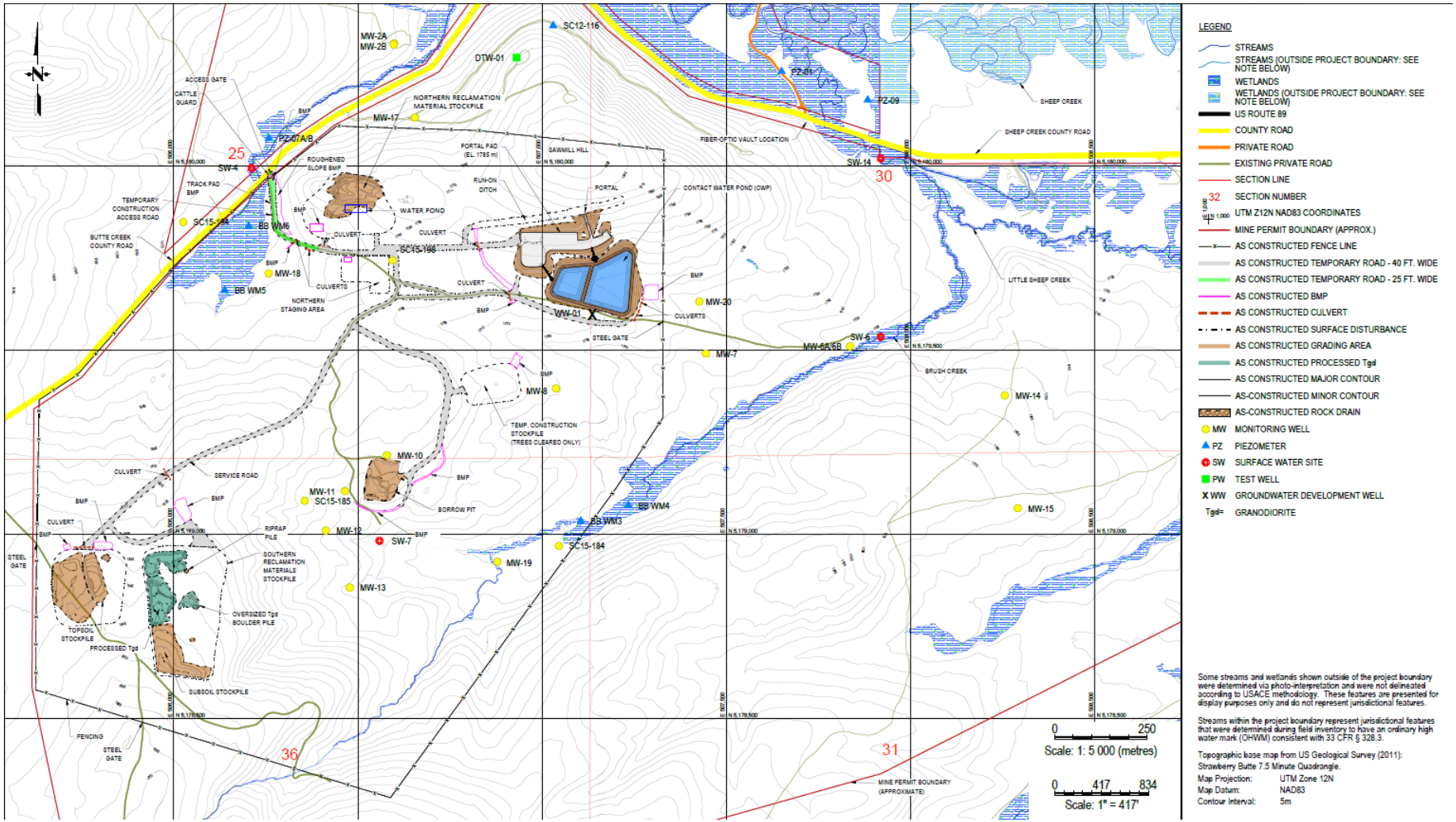


Figure 18.3 As-constructed 2020 Phase 1 Construction

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, 2017a) 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 proposed 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 proposed water management facilities include the CTF, BCWP, PWP, NCWR and the TWSP. The water management facilities (with the exception of those already constructed) will be reviewed during the next phase of the project to ensure operational stability. 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 18.4).

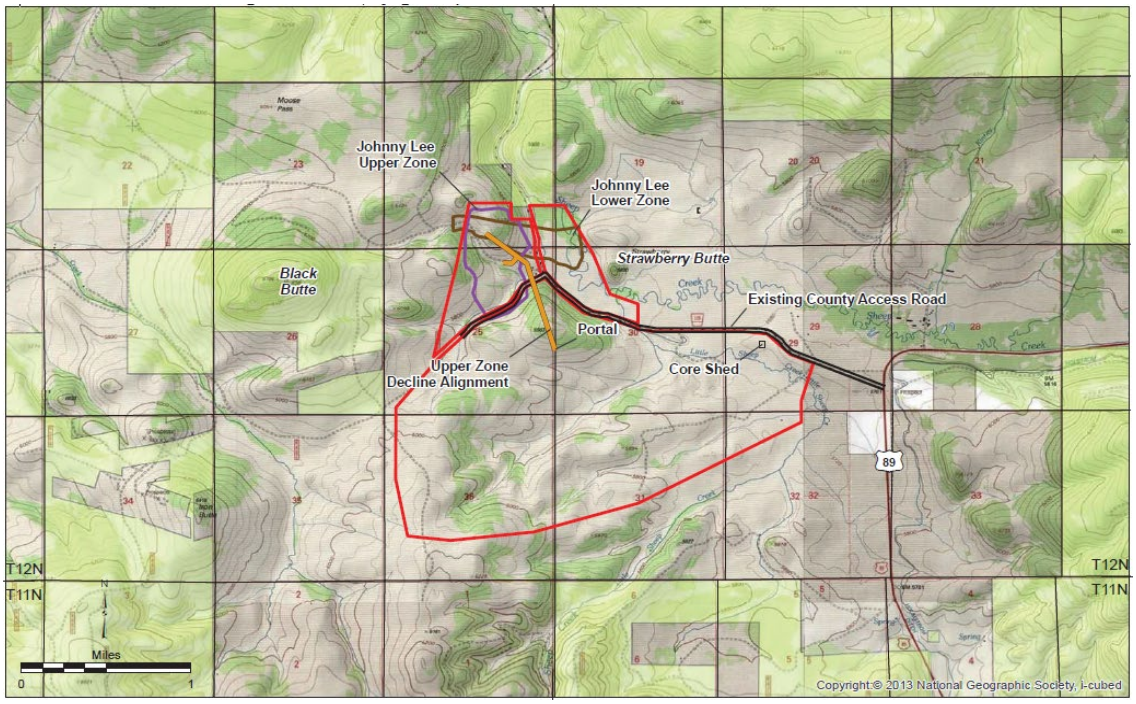


Figure 18.4 Site Vicinity Map with Mine Permit Boundary

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 km/h (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 km/h (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 km/h (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.

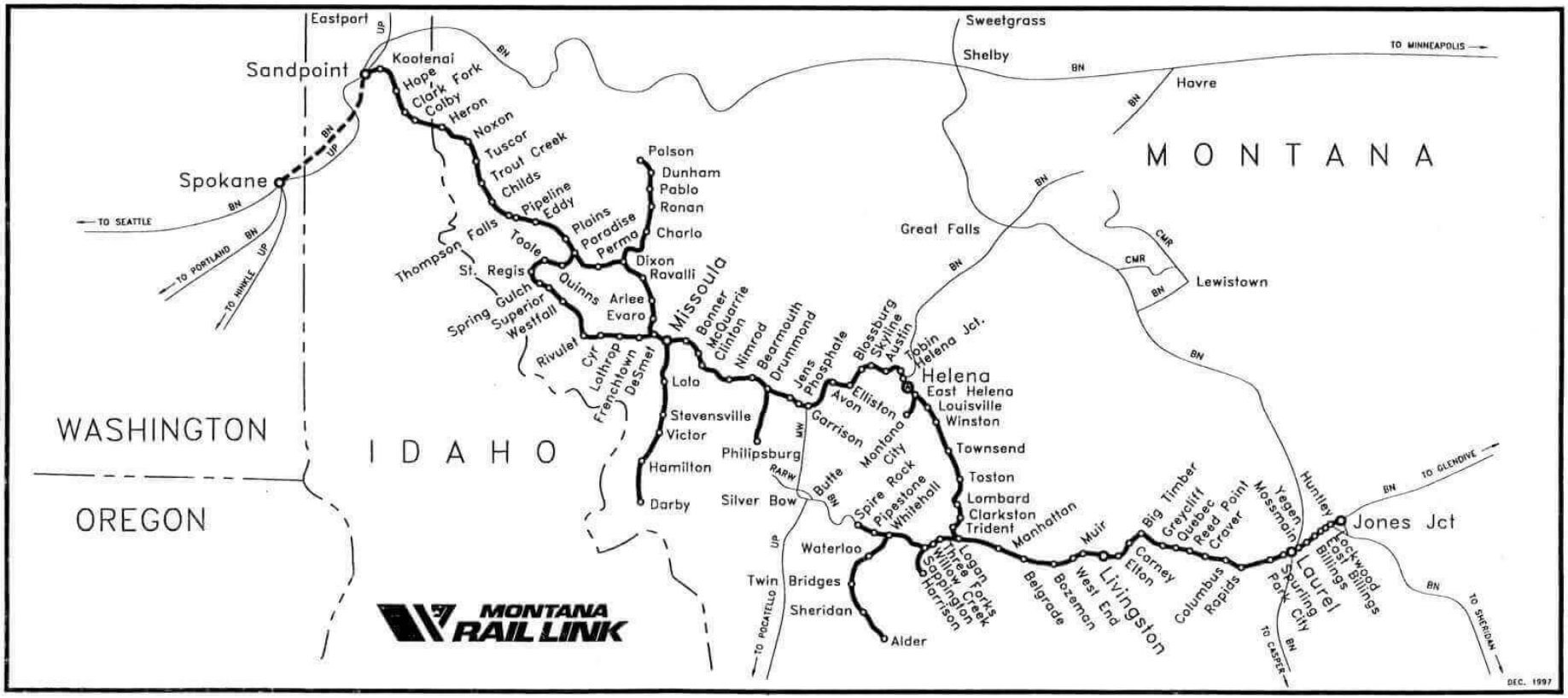


Figure 18.5 Montana Rail Link

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, 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 (L/d) or 3 million gallons per day (gpd) of process water. The site requires 32,595 L/d (8,575 gpd) or 23 Litres per minute (L/min) (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 L/min (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 L/s (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 L/s (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 L/s (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 13 March 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.

18.5.3.1 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.

18.5.3.2 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 and 2025) 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.

18.5.4.1 Construction Phase

Groundwater from mine dewatering and water collected in the CWP will require treatment at an estimated maximum flow rate of approximately 946 L/min (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 L/min (250 gpm). If this situation occurs, the flow in excess of the 946 L/min 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 L/min (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.

18.5.4.2 Operational Phase

The operational phase WTP will have an influent capacity of 2,226 L/min (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 L/min (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 L/min (498 gpm) will be treated by the RO system, producing 1,536 L/min (406 gpm) permeate water and 345 L/min (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.

18.5.4.3 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 L/min (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).

18.5.4.4 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.

18.5.4.5 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.

18.5.4.6 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.

18.5.4.7 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.

18.5.4.8 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 17 March 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 ft) 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

18.7.1.1 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.

18.7.1.2 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.

18.7.1.3 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.

18.7.1.4 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.

18.7.1.5 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.

18.7.1.6 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

18.7.2.1 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 (ASTs) 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 fueling 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-fueling and spillage. Manual fire suppressant equipment will be installed at the fueling station.

Additionally, a fuel and lubricant truck will be used to dispense fuel to mobile equipment and the smaller dedicated service diesel storage tanks.

18.7.2.2 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.

18.7.2.3 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 L/d (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	L/d		gpd	L/d
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 plate and frame filters, each with 100% capacity. 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.

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, 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 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

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 (1.5 mm) liner underlain by 15 oz/yd^2 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 can store up to a maximum of 700 kt of waste rock. A total of 1,799kt 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 yd^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 4.2 ha (10.4 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 1,114 kt 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 100,000 m^3 (130,800 yd^3).

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 yd^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 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 yd^3).

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	14

Equipment Description	Quantity
Site Services Equipment	
Hiab Truck – 5 T	1
ROM Loader (CAT980)	1
Skid Steer Loader	1
Concentrate Loader (CAT950)	1
Forklift – 3 T (Warehouse)	1
Mobile Crane Truck – 20 T	1
Misc. Small Equipment	
Mobile Air Compressor (5,200 L/min @ & Bar)	1
Diesel Welder – 7.2 kVa 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

Copper 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 (in this case the Port of Everett, Washington has been selected) and shipped to copper smelters for further refinement or to a blending facility.

19.2 Market Studies (Strategy)

No formal market studies or product valuations were completed as part of the 2025 PFS. Market price assumptions and payment terms were based on a review of public information, industry consensus, standard practices, and specific information from comparable global operations.

The potential buyers for the Black Butte Copper Project concentrates are expected to be domestic and global smelters, and trading companies. SRA is engaging with several potential buyers to build awareness of the project, projected volumes and expected quality of the cu concentrate. Formal offtake negotiations will be undertaken to develop firm commitments for product sales closer to the project construction and commissioning phase.

The initial copper concentrate product sales are expected to be contracted on a spot and short-term basis until production quality and quantity stabilises. Beyond this, it is expected that 50-70% of planned production will be placed on longer term contracts to multiple customers.

At this PFS stage there are no sales and marketing agency agreements in place.

19.3 Commodity Price Projections

Project economics were estimated based on the long-term metal price summarized in Table 19.1. This price was established on consensus data of the long-term real price forecasts from nine major financial institutions, in November 2025

Metal Price	Unit	Value
Copper	USD/lb	4.70

Table 19.1 Metal Price Assumptions

19.4 Refining Charges, Payment Terms & Penalties

The economic analysis applies treatment and refining charges and payment terms based on pricing used by Sandfire to calculate reserves. The copper concentrate treatment and refining terms used for the economic analysis are shown in Table 19.2.

Treatment and Refining Charges	Unit	Value
Treatment of Cu Concentrate	USD/dmt	50.00
Refining of Cu Concentrate	USD/payable lb	0.05

Table 19.2 Treatment and Refining Terms

Copper concentrate road and port costs are estimated to be USD 193.6/t of concentrate shipped. This cost considers truck transport from the site to a rail header located at Townsend, rail transport to port, and shipping to Europe/China. Ground transport, including container lease, railcar lease, and port charges are estimated to be USD 145.4/t of concentrate shipped, while sea freight is estimated at USD 48.2/t of concentrate.

Analyses of the copper concentrates from variability testing of the JL UCZ composites have reported potentially deleterious levels of arsenic. Various arsenic containing minerals have been identified through mineralogical analysis, with only some associated with copper reporting to concentrate. All samples tested from the JL LCZ showed low arsenic in copper concentrate. Arsenic is forecast to range between 1,900 to 7,520 ppm in copper concentrate over the life of the project (estimated in the mine plan based on plant feed blend).

An arsenic penalty of USD 2.75/dmt per 1,000 ppm arsenic in copper concentrate exceeding 3,500 ppm has been applied to all LOM production periods.

19.5 Contracts

No material contracts for property development, including mining, concentrate transportation, handling/storage, stevedoring, offtake sales, or hedging exist at this PFS stage of project development.

19.6 Comments on Market Studies and Contracts

The QP notes that the pricing and payment terms used in the cash flow analysis are reasonably aligned with consensus estimates and benchmarks from comparable operations.

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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 (Pressacco *et al*, 2019) 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, plans and costs.

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 2020a) and the Mine Operating Permit (MOP) (Tintina Montana, 2017), with additional information (Zieg, 2018; Hydrometrics, 2018a; Hydrometrics, 2025) developed to refine some of the details related to water management.

Additional evaluation was completed with preliminary data available for the Lowry Deposit, including a geochemical data review completed by Enviromin, Inc. (Enviromin, 2023), and a hydrologic/hydrogeologic, study, data review and preliminary analysis completed by Hydrometrics, Inc. (Hydrometrics, 2023).

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 Johnny Lee 2020 FS, the 2025 PFS update, and MOP, and their 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 1,706 m and 1,860 meters (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-Gieger 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.

20.2.1.1 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 (Figure 20.1). Sheep Creek, a fifth-order stream, flows out of the Little Belt Mountains approximately 27 km (17 miles) upstream from the Project. Sheep Creek drains an area of 503 km^2 (194 square miles) and runs approximately 55 river km (34 river miles) from its headwaters down to the Smith River, which in turn is a tributary to the Missouri 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 (Figure 20.1). The Holmstrom Ditch, located upstream of the Project was constructed in 1935 to divert water from Sheep Creek for irrigation and continues to operate seasonally (Hydrometrics 2017). Moose Creek flows into Sheep Creek downstream of the project site and is an important component of the water rights mitigation plan.

Baseline water resource monitoring for the Project began in 2011 and has expanded over time. Currently, a total of 14 surface water sites are monitored as part of the water resource monitoring plan (Figure 20.2). Surface water monitoring includes collection of flow measurements, field parameters at all sites and water quality samples at nine sites. The highest stream flows in Sheep Creek (Station 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 3.7 m^3/s (130 cfs) in 2021 to greater than 17.0 m^3/s (600 cfs) in 2014, 2018, and 2020. 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. Stream flows in 2021 and 2022 are among the lowest observed in recent years (Hydrometrics, 2022), and reflect the moderate to extreme drought conditions during those years (National Drought Mitigation Center, 2023).

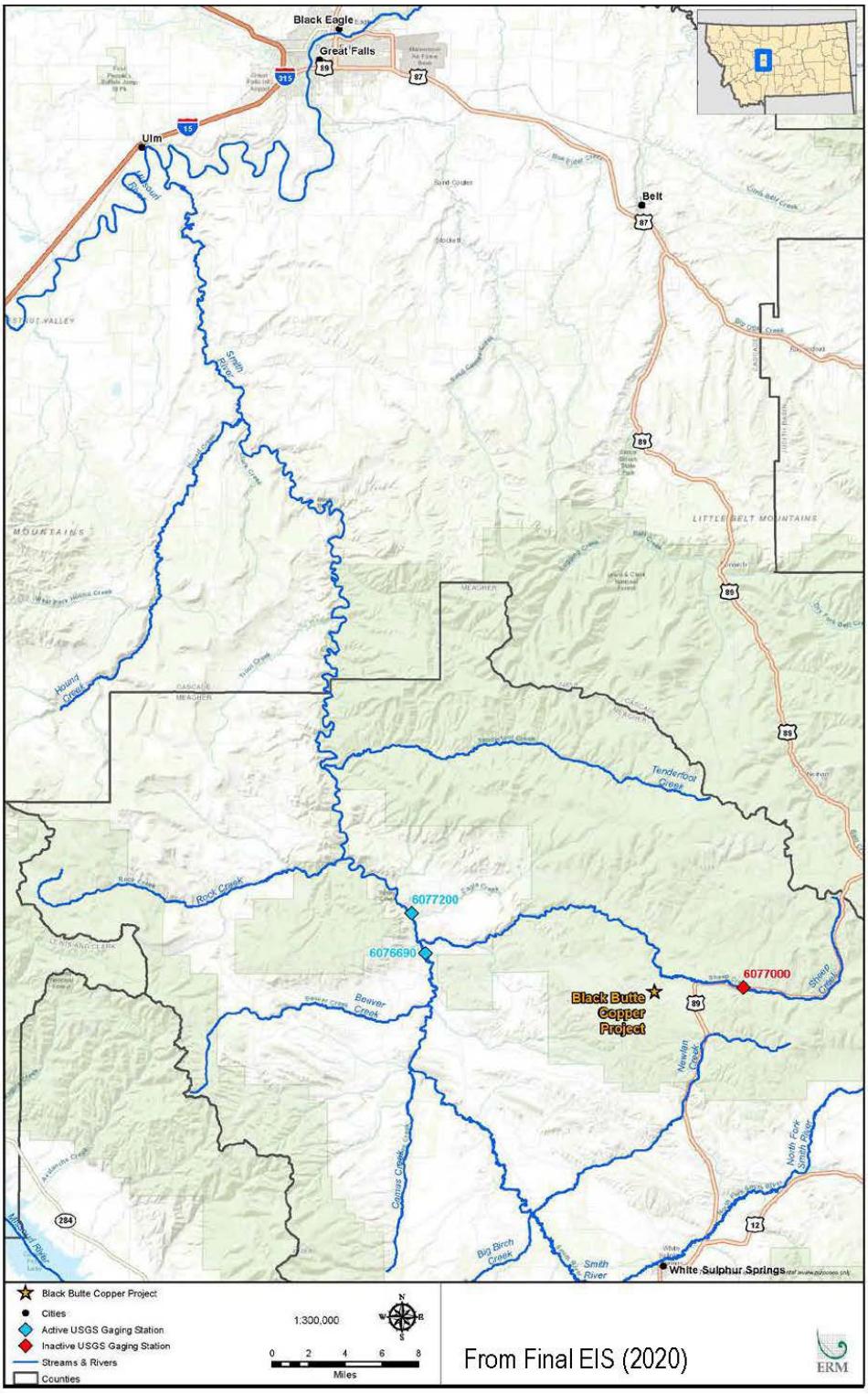


Figure 20.1 Plan Map Showing Rivers and Gaging Stations in the Project and Region

Surface water quality results show near neutral to slightly alkaline pH values (6.4 to 8.9 s.u.), and low to moderate specific conductance (57 to 595 micro mhos per centimeter (umhos/cm)). 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, as well as infrequent exceedances of total thallium, most commonly during winter and drought periods.

Sheep Creek is included in DEQ's 303(d) list of impaired streams for dissolved aluminum and Escherichia coli (E. coli), with sources listed as grazing in riparian zones, disturbances associated with human activities, and natural sources (MDEQ, 2020a).

20.2.1.1.1 Lowry

The current surface water monitoring program associated with the Johnny Lee deposit (Figure 20.2) provides the majority of the surface water monitoring needs for the Lowry deposit, except in the headwaters of Little Sheep Creek. A minimum of one additional surface water site should be established near the headwaters of Little Sheep Creek upstream of potential impacts from the Lowry deposit. As of this report, this additional surface water site has not been established yet.

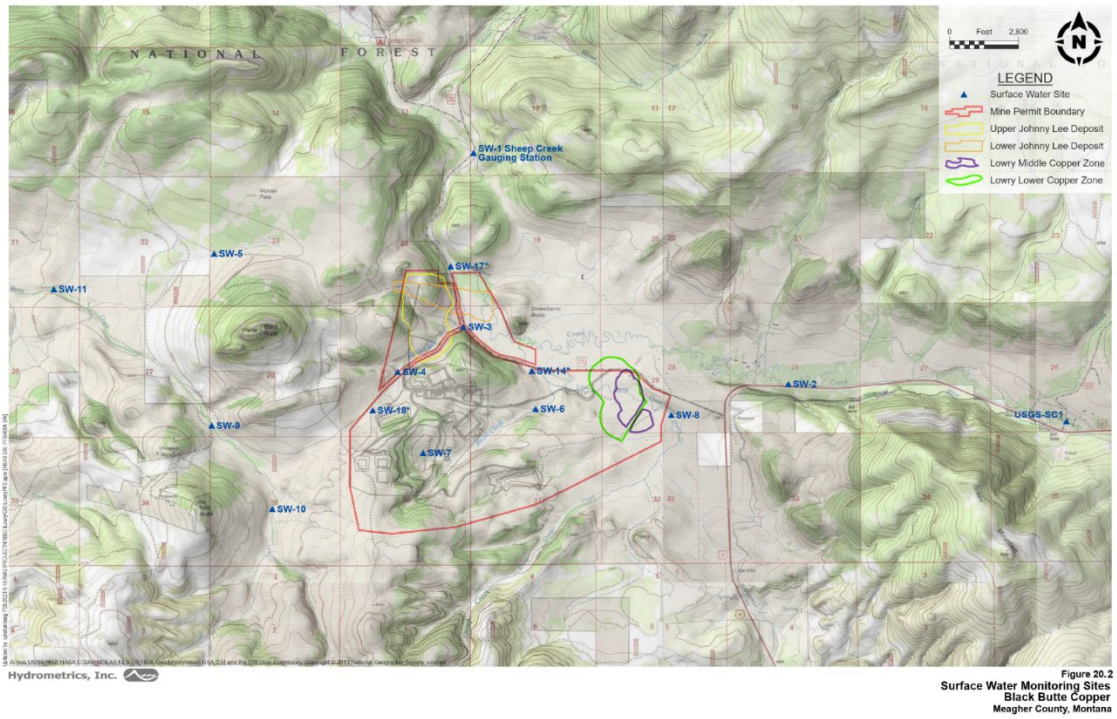


Figure 20.2 Surface Water Monitoring Locations

20.2.1.2 Springs and Seeps

The baseline spring and seep survey was conducted in 2011 and 2012 within a 2.4 km (1.5 mile) radius of the then-proposed exploration adit site, excluding privately-owned lands not leased to Tintina.

Current spring and seep monitoring includes 20 spring sites and 10 seep sites (see Figure 20.3). 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 generally re-infiltrate into the channels within a few hundred feet of the source. Of the identified springs, seven have been developed to supply livestock watering tanks.

Field parameters are collected at all spring and seep sites on a quarterly to annual basis. Flow measurements are collected quarterly to annually at 20 spring sites, and annual water quality samples are collected at 11 spring sites. Spring and seep water quality results show slightly acidic to slightly alkaline pH (5.3 s.u. to 8.9 s.u.), and low to moderate specific conductance (2.5 umhos/cm to 699 umhos/cm). Spring flow rates in the Project area ranged from no flow during certain dry or frozen periods in the year to greater than 545 m^3/day (100 gpm).

20.2.1.2.1 Lowry

The current spring and seep monitoring program for the Project provides an initial baseline for areas to the west of the Lowry deposit. However, an additional spring and seep survey will need to be conducted within a suggested 2.4 km (1.5 mile) radius of the boundary of the Lowry deposit, shown on Figure 20.4. The spring and seep survey should consist of a desktop survey using aerial photography, water right database, and other resources to identify potential sites and a field investigation to identify all springs and seeps within the area.

20.2.1.3 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. A preliminary evaluation of the groundwater inflow impacts of the updated 2025 mine model was also conducted by Hydrometrics (2025) to understand if there were significant changes. At this stage, no significant impacts are expected, but additional modelling is recommended.

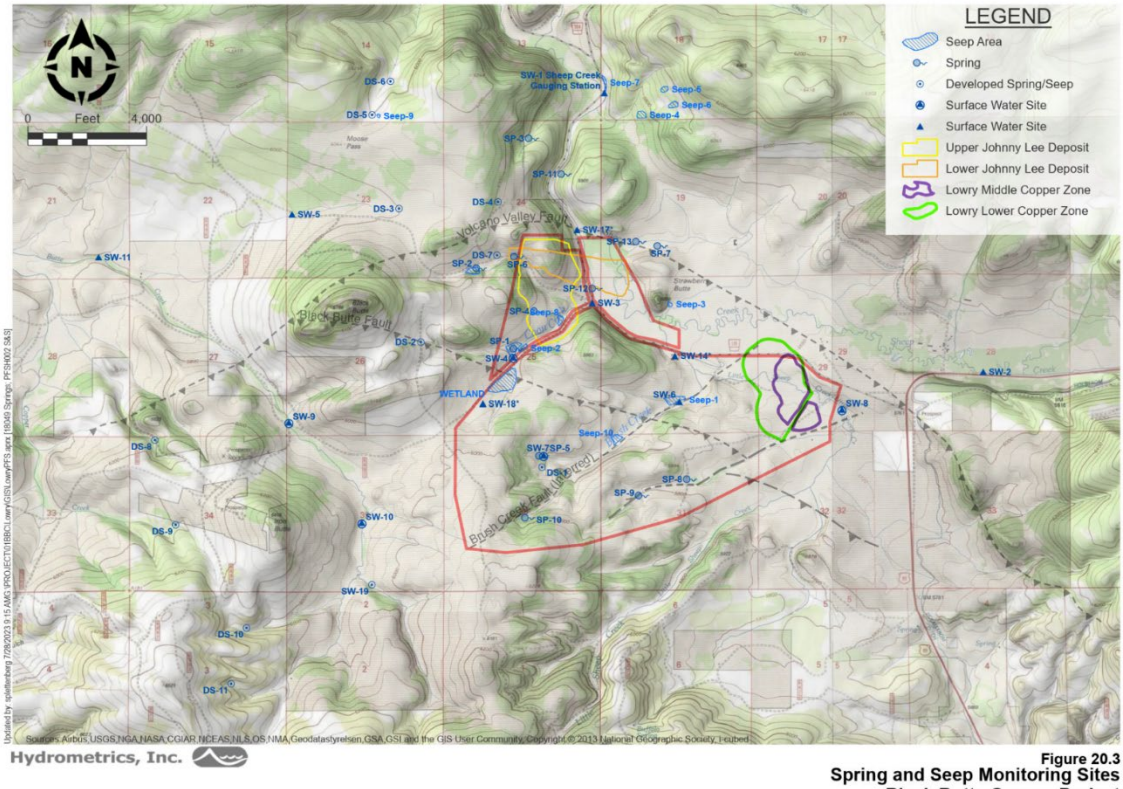


Figure 20.3
Spring and Seep Monitoring Sites
Black Butte Copper Project
Meagher County, Montana

Figure 20.3 Surface Springs and Seeps Map

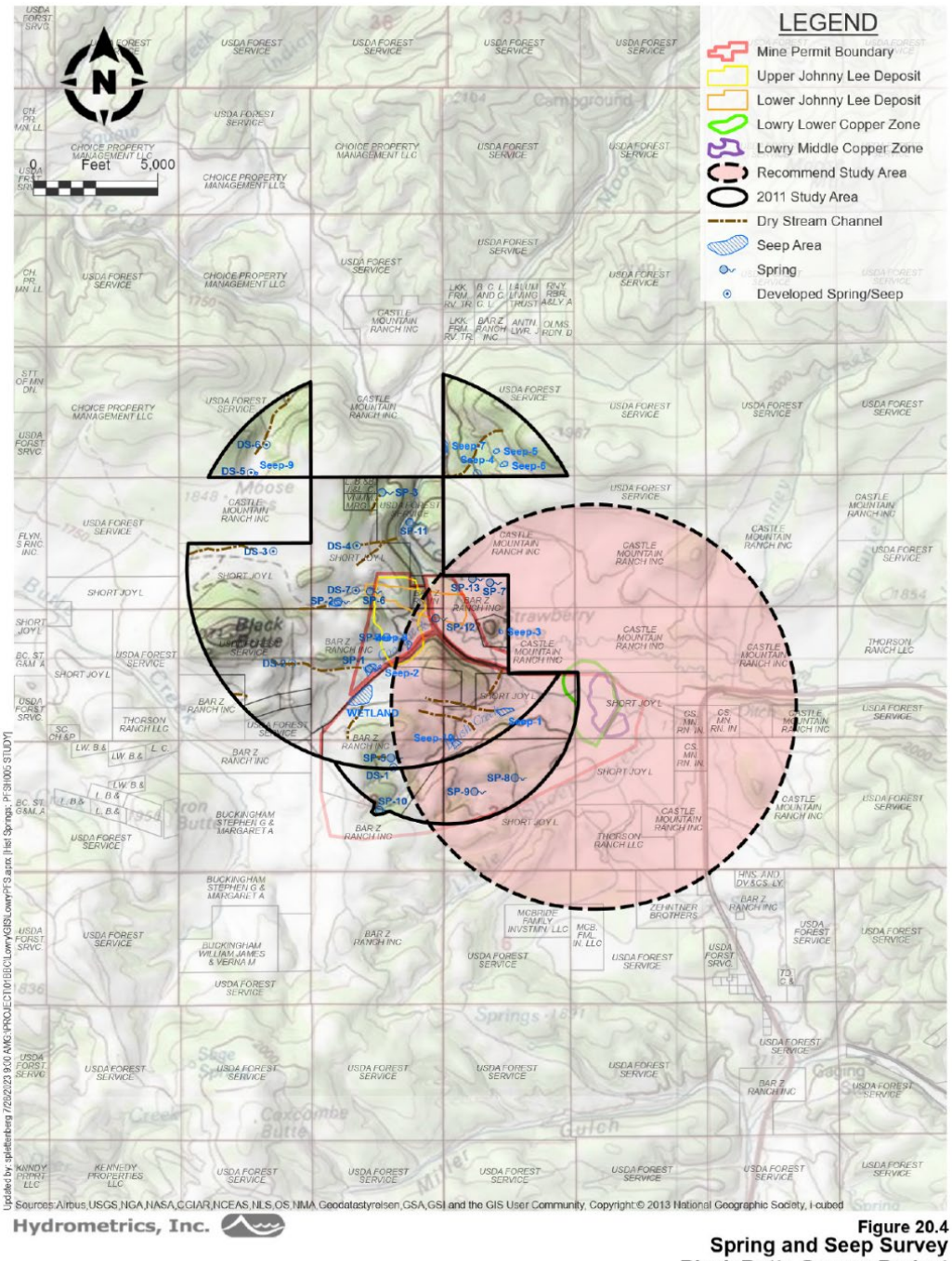


Figure 20.4
Spring and Seep Survey
Black Butte Copper Project
Meagher County, Montana

Figure 20.4 Surface Springs and Seep Survey Map

Unit	Description	Thickness (m, (ft.))	Hydraulic Conductivity* (cm/sec, (ft./day))	Storage Coefficient	Source of Hydraulic properties
Geologically-Based Hydrostratigraphic Units					
Sheep Creek Alluvium, Quaternary (QA)	coarse-grained sand and gravel alluvium	17	5 (17ft)	0.07 (200 ft/day)	slug test; literature
Flathead Sandstone *(Cf)*	sandstone bedrock	100	31 (100 ft)	10^{-9} to $5x10^{-4}$ (10^{-5} to 1.5 ft/day)	literature
Lower Newland Formation Shallow *(Ynl A)*	calcareous and bib-calcareous shale and siltstone bedrock	30-50	9-15 (30-50 ft)	$3.5x10^{-4}$ to $8.1x10^{-4}$ (1 to 2.3 ft/day) GM $5.3 x10^{-4}$ (1.5 ft/day)	pumping test
Upper Sulfide Zone	highly mineralized zone	30-150	9-46 (30-150)	$3.6 x10^{-6}$ to $2.5 x10^{-4}$ (0.01 to 0.7 ft/day) GM $2.8 x10^{-5}$ (0.08 ft/day)	pumping test
Upper Copper Zone	Cu mineralized zone				
Lower Copper Zone	Cu mineralized zone	30-50	9-15 (30-50 ft)	$6.7 x10^{-9}$ ($1.9 x 10^{-4}$ ft/day)	pumping test
Lower Newland Formation deep (*Ynl B*)	dolomitic and non-dolomitic shale and siltstone bedrock	150 in FW; to 2,000	46 in FS; to 610 (150-2000 ft)	$3.6 x10^{-7}$ to $2.5 x10^{-6}$ (0.001 to 0.007 ft/day)	pumping test
Chamberlain Formation Shale (*Yc*)	siliceous, locally arenaceous shale	500	152 (500 ft)	$3.6 x10^{-7}$ to $2.5 x10^{-6}$ (0.001 to 0.007 ft/day)	assumed
Neihart Formation Quartzite (*Yne*)	recrystallized sandstone	800	244 (800 ft)	Low; N.A.	assumed
Crystalline Bedrock (*Xbc*)	metamorphic crystalline rock	To depth	To depth	10^{-7} to 10^{-5} (10^{-3} to 10^{-1} ft/day)	literature
Structurally Defined Hydrostratigraphic Units					
Volcano Valley Fault	fault; clay gouge core; variable associated fracturing	46 (150 ft)	$2.5x10^{-7}$ to $5.3x10^{-9}$ ($7.1 x 10^{-4}$ to $1.5 x 10^{-5}$ ft/day) avg. 9.9x10-9 ($2.8 x 10^{-5}$ ft/day)	NA	permeameter testing
Black Butte Fault		3-4 (10–14 ft)			assumed
Buttress Fault		1.5 (5 ft)			
Brush Creek Fault		13.4 (44 ft)			

*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 Hydrometrics' Baseline Water Resources Report (2017).*

Table 20.1 Hydraulic Properties of Hydrostratigraphic Units in the Vicinity of the Johnny Lee deposit

In the vicinity of the Johnny Lee deposit, groundwater flow is controlled by the hydraulic characteristics of the geological units described in Part 7 and structural features (faults). Figure 16.1 provides a generalized cross-section through the Johnny Lee development, which is represented in the conceptual groundwater model as the Hydrostratigraphic Units (HSU) shown in Figure 20.5 (Hydrometrics, 2016b). Table 20.1 summarizes the physical and hydraulic characteristics of the principal HSUs that were evaluated during the Johnny Lee FS.

Hydrogeologic investigations and monitoring focused on the first five HSUs. The principal HSUs consist of the alluvium in the Sheep Creek drainage, and four HSUs within the Lower Newland Fm (Ynl A, JL USZ, Ynl B, and JL LCZ). The highest hydraulic conductivity of the principal HSUs is in the alluvium at (61 m/day (200 ft/day)). The HSUs within the Lower Newland Fm show a general decreasing trend in hydraulic conductivity with depth (0.3 to 0.7 m/day (1 to 2.3 ft/day) for Ynl A to 5.8 x 10^{-5} m/day (1.9 x 10^{-4} ft/day) for the JL LCZ). Faults running through the mine site in the vicinity of the Johnny Lee deposit were also evaluated as potential controls on groundwater flow, either as barriers to flow or preferential flow paths for groundwater flow (Table 20.1). The faults have much lower hydraulic conductivities than most of the HSUs, indicating they are most likely barriers to groundwater flow rather than preferential pathways.

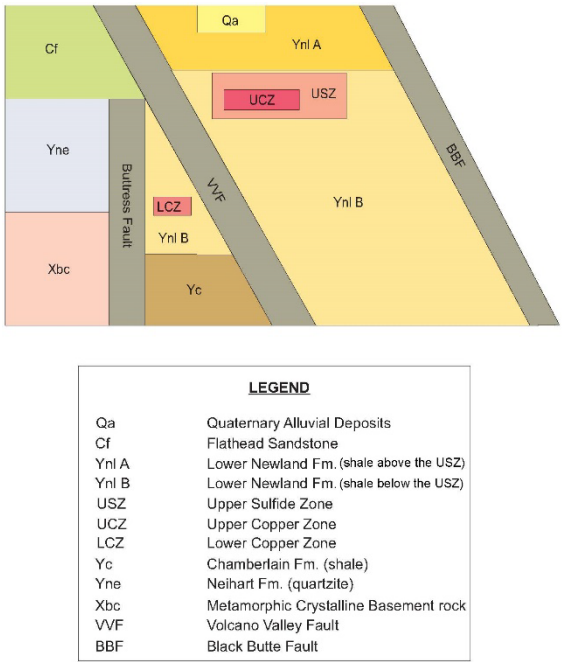


Figure 20.5 Hydrostratigraphic Units in the Vicinity of the Johnny Lee Deposit

Baseline groundwater monitoring started in 2011 and was expanded to include 31 wells and 15 piezometers through the first part of 2022. Seven wells were abandoned in 2022 as part of the Phase 1 construction. Currently, monitoring is conducted quarterly and includes water levels, field parameters, and water quality sampling at 25 monitoring wells. The locations of these wells and piezometers are shown on Figure 20.6.

Groundwater isoelevation maps were generated for both the regional and mine development areas. The regional map (Figure 20.6) 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 20.7, based on water level data collected from SRA's network of monitoring wells and piezometers.

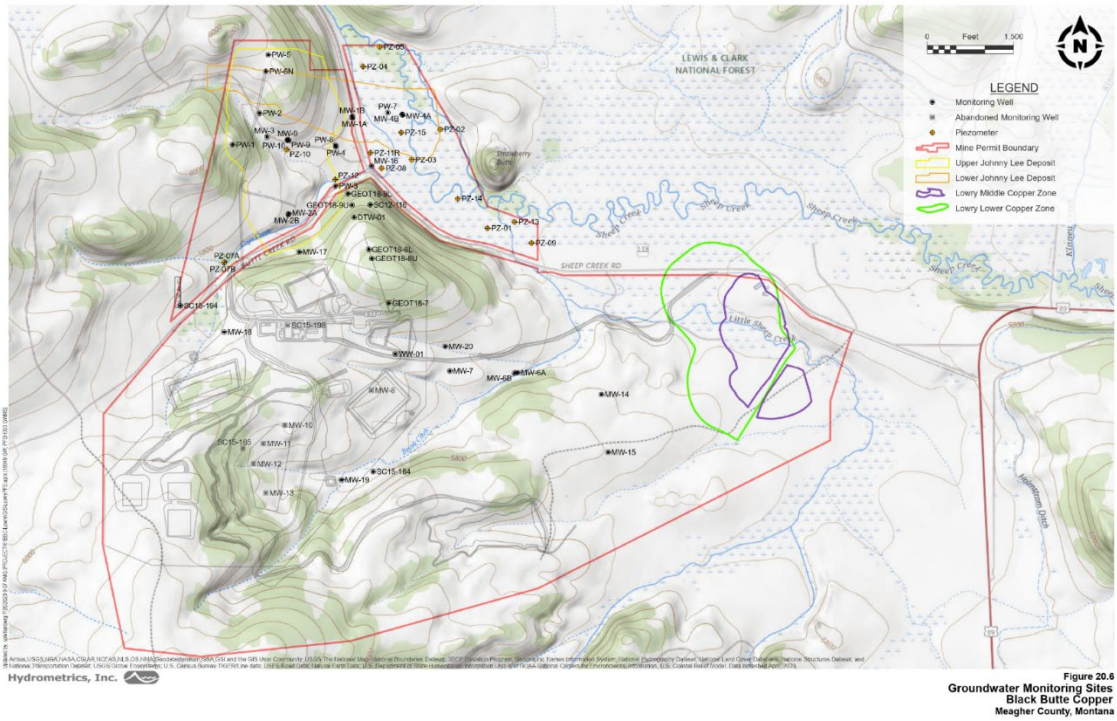


Figure 20.6 Groundwater Monitoring Sites

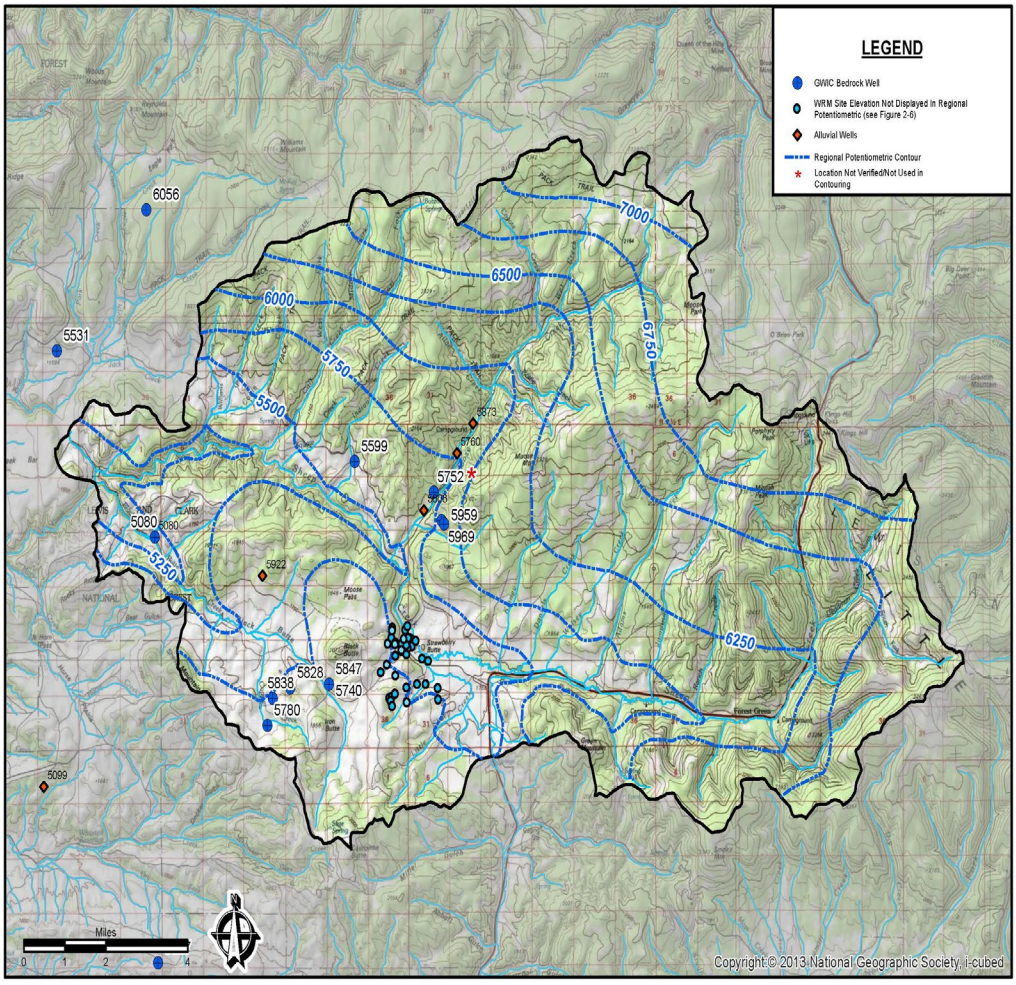


Figure 20.7 Regional Potentiometric Surface Map

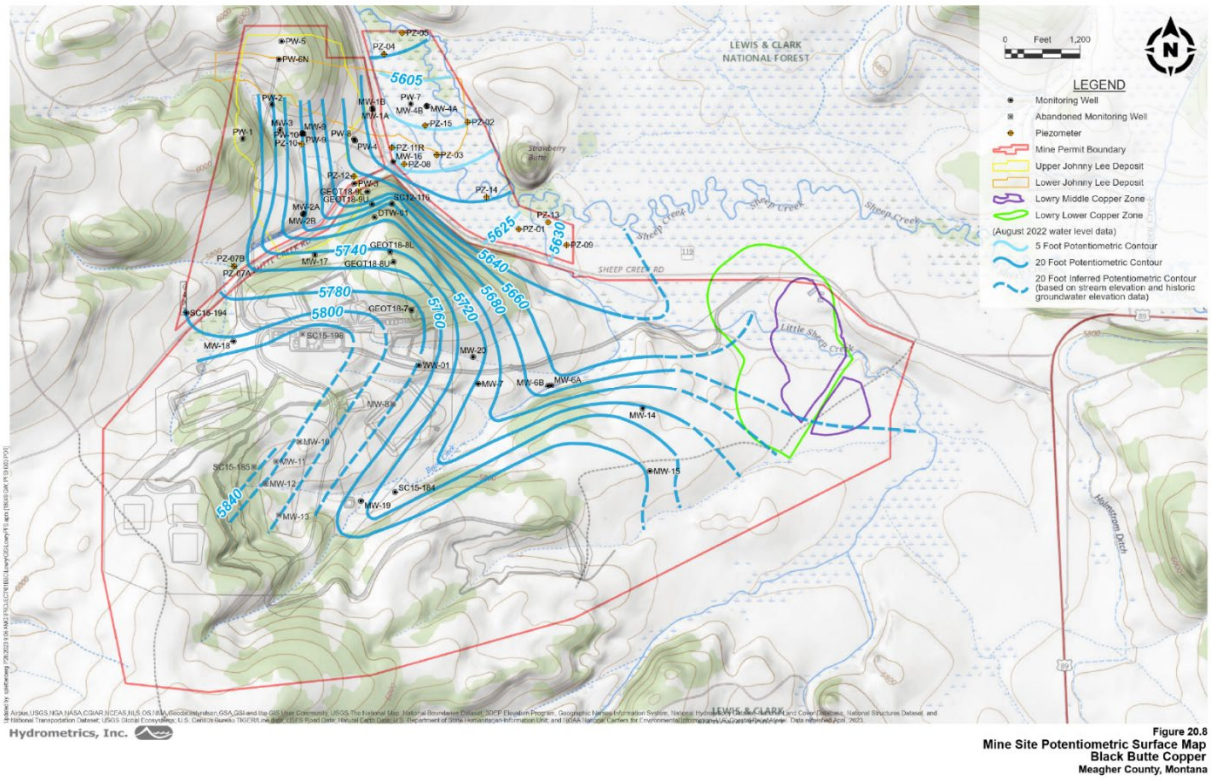


Figure 20.8 Mine Site Potentiometric Surface Map

Figure 20.8 depicts the bedrock potentiometric surface in the Lower Newland Formation near the proposed Johnny Lee Development and projected to the Lowry deposit, 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 near the Johnny Lee development and a north-eastward flow direction in the vicinity of the Lowry, 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 Johnny Lee 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, 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 for the Johnny Lee deposit varies depending on depth and proximity to the mineralized zone. Water quality of the principals HSUs is presented in Table 20.2. Wells completed in or near the USZ have the highest concentrations of dissolved solids and sulfate compared to the other hydrostratigraphic units, and consistently exceed the Human Health and secondary standards for arsenic, iron, manganese, strontium, and thallium.

Unit(s)	Geology	Wells	General Water Type	pH (s.u.)	Total Dissolved Solids (mg/L)	Sulfate (mg/L)	Exceedances[1]
Sheep Creek Alluvium (AL) and Overburden	coarse-grained sand and gravel alluvium	MW-1A, MW-4A, MW-6A	Calcium bicarbonate	6.24-8.57	176-328	1-21	Arsenic Iron Lead Manganese (MW-4A) Thallium
Flathead Sandstone *(Cf)*	sandstone bedrock	N/A					
Lower Newland Formation Shallow *(Ynl A)*, Lower Newland Formation deep (Ynl B)	shale and siltstone bedrock	MW-2A, MW-2B, MW-4B, MW-6B, MW-7, MW-8, MW-13, MW-14, MW-15, MW-16, MW-17, MW-18, MW-19, MW-20, SC15-194, SC15-198, PW-1, PW-3	Calcium-magnesium bicarbonate	6.26-8.74	151-684	1-77	Antimony Arsenic Iron (MW-16, PW-1[2], PW-3) Manganese (PW-1[2]) Thallium (MW-2B)
	granodiorite intrusives	MW-10, MW-11, MW-12, SC15-284, SC15-185	Calcium-magnesium bicarbonate	6.1-8.64	160-290	3-31	
	shale and siltstone bedrock, water quality similar to USZ	MW-1B, MW-9, PW-10	Calcium-magnesium sulfate	6.02-7.54	336-548	175-247	Arsenic (MW-1B, MW-9, PW-10) Iron (MW-1B) Manganese (PW-10) Strontium (PW-10) Thallium (MW-1B, MW-9)
Johnny Lee Upper Sulfide Zone [JL *USZ*], Johnny Lee Upper Copper Zone [JL UCZ]	highly mineralized zone	MW-3, PW-2, PW-4, PW-8, PW-9	Calcium-magnesium sulfate	6.11-7.88	284-607	50-280	Arsenic (MW-3, PW-4, PW-8, PW-9) Iron (MW-3, PW-2, PW-4, PW-9) Lead (PW-8[3]) Manganese (PW-4, PW-8) Strontium (MW-3, PW-4, PW-9) Thallium (PW-8)

Unit(s)	Geology	Wells	General Water Type	pH (s.u.)	Total Dissolved Solids (mg/L)	Sulfate (mg/L)	Exceedances[1]
Johnny Lee Lower Copper Zone [JL *LCZ*]	copper mineralized zone	PW-7	Sodium-bicarbonate	8.5-11.58	309-359	11-45	Antimony (infrequent) Iron (infrequent)
Chamberlain Formation Shale (*Yc*)	siliceous, locally arenaceous shale	N/A					
Neihart Formation Quartzite (*Yne*)	recrystallized sandstone	PW-6	Sodium-bicarbonate	9.85-9.96	160-178	5-11	N/A[4]
Crystalline Bedrock (*Xbc*)	metamorphic crystalline rock	N/A					

Notes:

1. *Parameters that have exceeded the Human Health Standard, chronic aquatic life standard for iron (1 mg/L) or health-based guidance for manganese (0.1 mg/L). Wells with consistent exceedances are noted in parentheses.*
2. *Based on one sample collected from PW-1.*
3. *Lead exceedances observed at PW-8 during the long-term pumping test. Results may be indicative of water quality from dewatered conditions.*
4. *Based on two samples collected from PW-6.*

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 HSUs. 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.

20.2.1.3.1 Lowry

The Lowry deposit is over 2,000 meters east of the Johnny Lee deposit. An initial hydrogeologic investigation was conducted during 2021 and 2022. The investigation included packer testing and VWP installation to provide preliminary hydraulic conductivity estimates and pressure heads within the mineralized zone for the Lowry (Hydrometrics, 2021). The results are summarized below in Table 20.3. The geometric-median hydraulic conductivity estimates in the vicinity of the Lowry deposit are similar to those measured in the Johnny Lee area.

Formation	Hydraulic Conductivity (cm/s)			Hydraulic Conductivity (ft/day)		
	Low	Geo-Mean	High	Low	Geo-Mean	High
MSZ	$2.5x10^{-7}$	$2.3x10^{-6}$	$4.9x10^{-5}$	$7.1x10^{-4}$	$6.5x10^{-3}$	0.14
LLSZ	$2.7x10^{-7}$	$8.8x10^{-7}$	$4.2x10^{-6}$	$7.6x10^{-4}$	$2.5x10^{-3}$	0.012
YNL	$5.3x10^{-7}$	$5.6x10^{-7}$	$6.7x10^{-7}$	$1.5x10^{-3}$	$1.6x10^{-3}$	$1.9x10^{-3}$

Table 20.3 Preliminary Hydraulic Conductivity Estimates for Lowry

Packer tests are short-term tests that assess aquifer characteristics at discrete test intervals and provide a good initial estimate of the aquifer properties; however, they do not evaluate the bulk characteristics of the system or its behaviour over time, which are key factors in determining long-term dewatering rates and potential impacts to water resources. Additional hydrogeologic investigation(s) are necessary to further characterize the groundwater system(s) in the vicinity of the Lowry. Further characterization of groundwater levels and aquifer characteristics will allow for a more accurate evaluation of groundwater elevations, direction of flow, and inflow rates to the Lowry expansion. Similar to what was conducted for the Johnny Lee, the investigation will consist of installing numerous wells and conducting both long-term (5+ days) and short-term (1-3 days) pumping tests, as well as monitoring of water levels and groundwater chemistry. The additional investigation(s) will provide key information to assess groundwater inflow to the mine and evaluate the interaction of groundwater and surface water and will be used in the permitting of the Lowry expansion. The proposed investigation will target the following features of the Lowry:

- MSZ: within the brecciated and competent massive sulfides of the Middle Sulfide Zone.
- Rose Fault: within fault in a location known to produce artesian flows.
- LLSZ: the fractures of the Lowry Lower Copper Zone Overlying units: including Tertiary sand and gravels (monitoring wells), shallow bedrock (test wells).
- Upper Sulfide Zone and other units that may influence mine dewatering.
- Potential access decline location to Lowry: in the major units that will be encountered in the mine access to the Lowry deposit.

The data collected from the planned Lowry aquifer characterization and expanded water resource monitoring programs will be integrated into the FS level characterization of the groundwater system in the vicinity of the Lowry deposit. This data will be used to support mine permitting and to develop a 3-dimensional numerical groundwater flow model to evaluate projected mine dewatering rates and their effects on water resources. Anticipated permitting requirements include a major amendment to the mine operating permit, likely modifications to the MPDES discharge permit, and potential updates to the 404/310 permitting and water rights.

20.2.1.4 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 permit 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 20.9. The predominant soil type is a loam (Cheadle channery loams), a soil with roughly equal proportions of sand, silt, and clay.

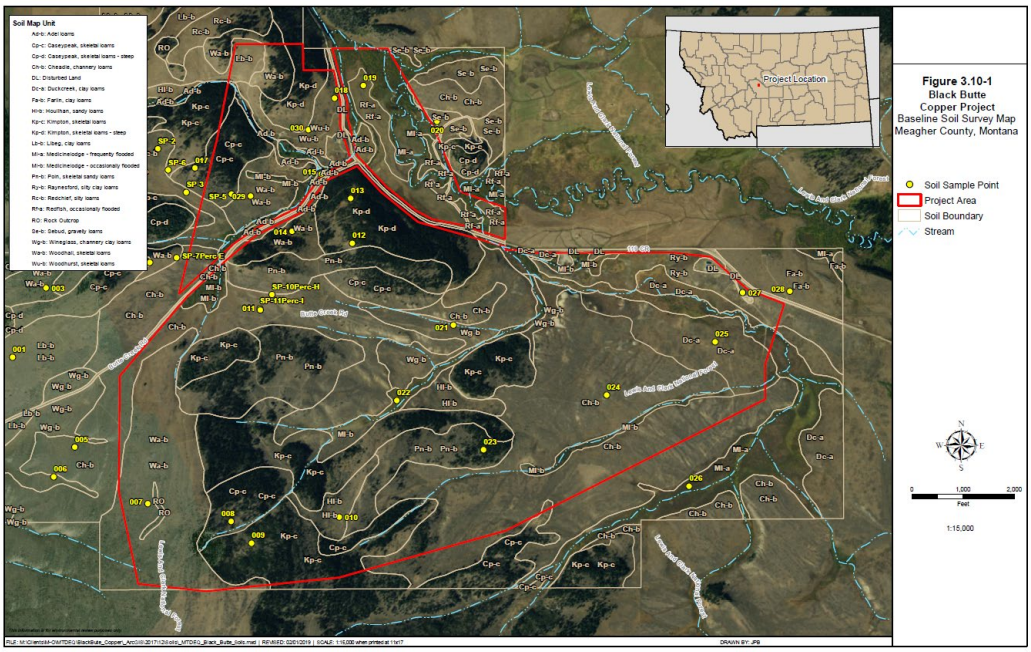


Figure 20.9 Baseline Soil Survey Map

20.2.1.5 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 km (46 miles) by air 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 mining activities will be completely underground, and the mine will be 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 underground 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 equipment (e.g. haul trucks), stationary equipment, portable equipment, and support vehicles. Fugitive emissions from crushing and conveying activities will be managed with best available technologies.

Quantitative modelling was conducted to evaluate the potential air quality impacts of the planned activities, 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 to demonstrate that no exceedances are expected based on models.

Air dispersion modelling will be modified to include possible sources from mining of the Lowry deposit. Due to minimal added surface activities, no significant air quality changes are expected.

20.2.1.6 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 and Lowry copper resources which SRA intends to mine. Land uses are predominantly agricultural, including

livestock grazing and hay production. In addition, recreational uses include fishing and big game hunters within the Sheep Creek drainage.

The Bar Z Ranch and Short Ranch own 100% of mineral rights of the lands containing the proposed mine and related facilities. Tintina Montana Inc. owns the surface rights of most of the area containing mine surface 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.

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, Newlan Reservoir, Lake Sutherlin, and Bair Reservoir. 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.

20.2.1.7 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.

Updated plan maps will be generated during the permitting process to include projections for the Lowry deposit area. Visual impacts expected for the Lowry are limited to small vent raise outlets and light vehicle access roadways.

20.2.1.8 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. An updated transportation study will be completed during the permitting process to consider possible changes in association with development of the Lowry resource.

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 Burlington Northern-Santa Fe (BNSF) line 64 km (40 miles) W/SW of White Sulphur Springs at Townsend, MT. In addition, another 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.

20.2.1.9 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, ATVs, and occasional air traffic. Dominant nighttime 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

20.2.2.1 Vegetation

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 three federally listed Threatened and Endangered (T&E) plant species in Montana, the Whitebark pine (*Pinus albicaulis),* Spalding's Catchfly (*Silene spadingii)*, and Ute ladies'-tresses (*Spiranthes diluvialis)*; each of which are listed as threatened. Spalding's Catchfly and Ute ladies'-tresses are not found within the Project region. Whitebark pine has been observed regionally near the Project area, but at much higher elevations. There is no suitable habitat for whitebark pine within or near the Project.

Montana also maintains 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 17 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*).

20.2.2.2 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 (Environmental Laboratory, 1987). The 2014 wetland and waterbody baseline survey identified 122 ha (328.8 acres) of wetlands within the wetland analysis area (Figure 20.10). 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 3 October 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, 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, do not require Section 404 permit authorization to impact these wetland features.

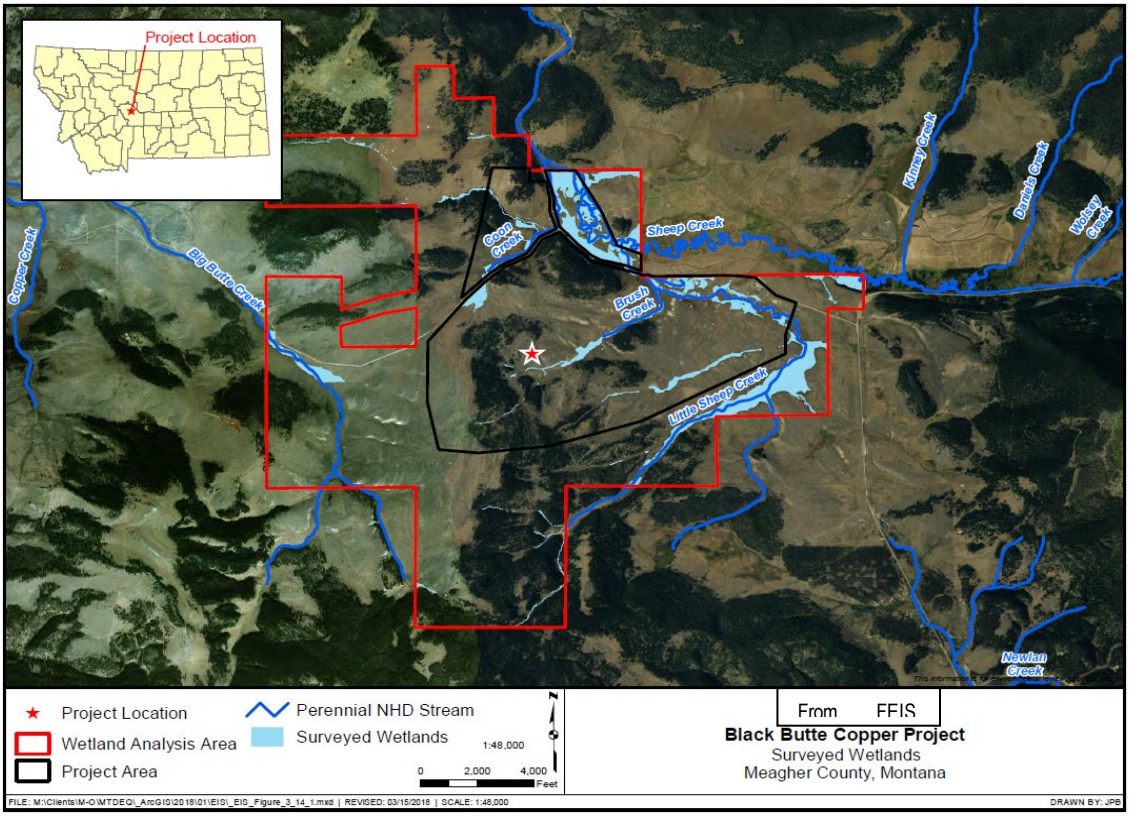


Figure 20.10 Surveyed Wetlands Map

20.2.2.3 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 through 2025. Baseline 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. In 2017, Moose Creek was added to the seasonal baseline survey area as an additional reference reach. Montana Biological Survey issues an updated report annually that includes seasonal survey results.

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 (78 mi^2).

Tenderfoot Creek, a 64 km (40 mile) long tributary to the Smith River has a total watershed area of 281 km^2 (108 mi^2) and was chosen as the off-site control reach. An estimated watershed area of 203 km^2 (78 mi^2) is drained above the reference reach. Montana Biological Survey (Stagliano 2018) 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.

20.2.2.4 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 (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, 2014b), 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. Future biodiversity studies may want to consider environmental DNA analysis. Nevertheless, the study area is considered to support good wildlife species richness. This evaluation recorded no amphibians. However, a Columbian spotted frog, a Western toad, 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. Eleven 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 grey 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.

The USFSW (2018) updated the list of listed or candidate species under the Endangered Species Act for Meagher County to include: Canada lynx (listed threatened), Grizzly bear (listed threatened), and Wolverine (proposed). Grizzly bears have been rarely observed in Meagher County and are not common in the Little Belt mountains.

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 (Archaeology) 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 minimal 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 Project will impact wetlands that are considered waters of the US and is therefore permitted by the US Army Corps of Engineers (USACE). The area under jurisdiction of the USACE (0.34 ha or 0.85 ac) was evaluated under Section 106 of the National Historic Preservation Act. 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 (Tetra Tech, 2015) and summarized in Appendix I of the MOP application. Additional surveys were conducted 2019 by Tetra Tech, and in 2020 by GANDA and Kleinfelder (Kleinfelder/Garcia and Associates, 2021). 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 artifacts 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 Northern Cheyenne), SRA voluntarily re-designed the access road and the nearby alluvial conveyance pipeline to avoid one of the lithic scatter sites that would have been impacted.

In the Final EIS, SRA voluntarily agreed that any cultural resources eligible for listing the NRHP will be either mitigated or avoided during construction of the mine facilities. Archaeological 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 archaeological 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 that will be generated by the Johnny Lee and Lowry mines are a major factor in the evaluation of the potential impacts of the mine on the environment during operation and closure. Waste rock and tailings geochemistry of the Johnny Lee deposit was characterized by Enviromin and summarized in Appendix D of the MOP application (Enviromin, 2017a). A geochemical characterization of waste rock from the Lowry deposit was performed by Enviromin in 2023 based on a comparison of total metal concentrations. In 2025, Enviromin, now RESPEC, provided a statement confirming that the 2025 PFS revisions to the mine plan are unlikely to require major new geochemical testing, though limited supplemental analysis may be needed if new geochemical variation is identified (RESPEC, 2025). Components of the geochemical characterization for these deposits are described below.

In general, geochemical characterization programs include a series of test methods developed to predict acid generation and metal leaching potential of mined materials, beginning with short-term static tests of a larger number of samples, followed by longer-term kinetic tests of fewer samples. The geochemical data can be incorporated into hydrogeochemical models to predict water quality for the future mining operation, during operations and closure.

The Johnny Lee deposit would mine approximately 8.8 million tonnes of copper-enriched rock with an average grade of 2.6 percent copper and 0.88 million tonnes of waste rock. Approximately 6.0 million tonnes of raw tailings would be produced. The tailings from the Johnny Lee deposit will be thickened and sent to a paste plant where mixture of cement and slag will be added to the tailings as a binder. The addition of binder provides structural strength to support underground workings and changes the physical properties of the material to a stable, non-flowable material with low hydraulic conductivities (approximately 10^{-9} meters per

second) in both surface and underground settings. The product, called cemented paste tailings, would be piped either to the underground mine to backfill workings or to a double-lined tailings basin, including the Cemented Tailings Facility (CTF) for the Johnny Lee deposit. Approximately half of the tailings will be returned underground as cemented tailings while the remainder would be stored in engineered structures on the surface. An extensive geochemical characterization program was conducted for the Johny Lee deposit to determine the acid generation and metal leaching potential of the waste rock, tailings and potentially reactive surfaces for all formations that would be intercepted by the mines, with a focus on the ore zones. This program also included a series of geochemical models to predict water quality during operation and post-closure (Enviromin, 2017a). Data from humidity cell tests (HCTs) were incorporated into predictive water quality models of mine operation and closure scenarios.

20.2.4.1 Johnny Lee Deposit Waste Rock

The acid generation and metal release potential of waste rock, construction rock, and tailings to be produced by mining of the Johnny Lee deposit were characterized using static tests (acid-base accounting [ABA], multi-element analysis [ICP], net acid generation [NAG], and static leach tests [SPLP]), and kinetic methods (HCTs). Mineralogical analyses of metal residence and asbestiform mineral analyses were also completed. Table 20.4 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 that was made to support the MOP application. These waste rock lithology percentages will be refined as the mine plan evolves.

A sufficient number of 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 percent) are unlikely to form acid, while the USZ and LZ FW waste rock have an uncertain acid generation potential or are likely to generate acid.

Metal mobility (SPLP) testing, used for Ynl A, Ynl B and the USZ (Table 20.4), can provide an indication of metals that have potential to be released from rock exposed to meteoric water. The degassing of carbon dioxide from these carbonate-rich rocks lead to supersaturated carbon dioxide concentrations that artificially elevated pH and likely influenced metal solubility. These results were used for metal mobility predictions for materials that could undergo sulfide oxidation and which, therefore, may produce low pH water. Metal mobility predictions were based instead on HCTs.

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 HCTs. The tests were run for 24 to 88 weeks, with the test length determined by the results. All HCTs 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 sulfide 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 HCTs was limited. Furthermore, metal release from waste rock HCTs 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 copper, lead, and nickel) exceeding groundwater standards.

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 sulfide 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 20.4 Geochemical Testing of Major Waste Rock and Near-surface Materials by Lithotype

20.2.4.2 *Johnny Lee Deposit Tailings*

Static test and kinetic tests were also performed on tailings produced from metallurgical pilot tests and paste tailings from pilot thickener tests (Table 20.5). 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 underground, and to change the physical properties of the material to a stable, non-flowable material with low hydraulic conductivities (approximately 10^{-9} meters per second) in both surface and underground 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 Materials	ABA	NAG	ICP	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] *Conventional unsaturated (Unsat) HCTs conducted on intact cement paste cylinders,*
[2] *An attempted test of 2% cemented paste tailings could not be completed.*

Table 20.5 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 20.6 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 HCTs to simulate subaerial weathering. They were also subjected to diffusion tests to simulate diffusion through cemented backfill in saturated underground 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	Modified ASTM method D5744 (intact cylinder in HCT)
	Cement paste in CTF, subaerial weathering, final closure lift	4% binder	Modified ASTM method D5744 (intact cylinder in 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)

Action Scenarios	Facility Represented	Tailings Characteristics	Test Method
Additional*	Cement paste in CTF, subaerial weathering	4% co-disposed with waste rock	Modified ASTM method D5744 (intact cylinder in 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.*

Table 20.6 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 HCTs indicate that all cemented paste tailings samples had potential to oxidize and to release sulfate, acidity, and metals if left exposed to air and water. Importantly, this was not observed immediately in all 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 sulfide oxidation.

20.2.4.3 Comparison of waste rock units in Johnny Lee and Lowry Deposits

The baseline geochemical characterization and modeling completed for the Johnny Lee deposit provides an initial understanding of the waste rock geochemistry and a foundation on which to base future work in support of the development of the Lowry deposit. Where possible, available geochemical testing results from the Johnny Lee deposit were used to enhance and support analyses of rock in and around the Lowry deposit. Initial results of statistical analyses using the available multi-element data representing the Lower Newland (Ynl) and Upper Sulfide Zone (USZ) waste rock units, suggest geochemical similarities, with potential variation across these deposits.

Statistical analysis including summary statistics, principal component analysis (PCA), and boxplots of multielement data from both deposits has been carried out to understand potential use of previous test data in the current study of the Lowry Deposit. PCA is used to assess the similarities and differences in the overall chemistry for each rock unit. PCA is a statistical tool that allows analysis of multiple parameters at the same time and is useful for comparing large datasets. PCA ranks each parameter's influence on the variation in a dataset and groups them as "principal components." The first component accounts for as much of the variance as possible, and each subsequent component demonstrates a stepwise reduction in variance until all the variance (100%) is described (Anderson, 2003).

Figure 20.11 a is a PCA comparison of Ynl A and B in the Johnny Lee deposit, The overlapping green clusters show that these units are more chemically similar than different from one another. A PCA comparing the Ynl from the Johnny Lee and Lowry is shown in Figure 20.11b. While these clusters are not perfectly

overlapping, there is significant similarity between these two populations. This suggests that the data from the Johnny Lee testing program can serve as a surrogate for the geochemistry of the Lowry deposit.

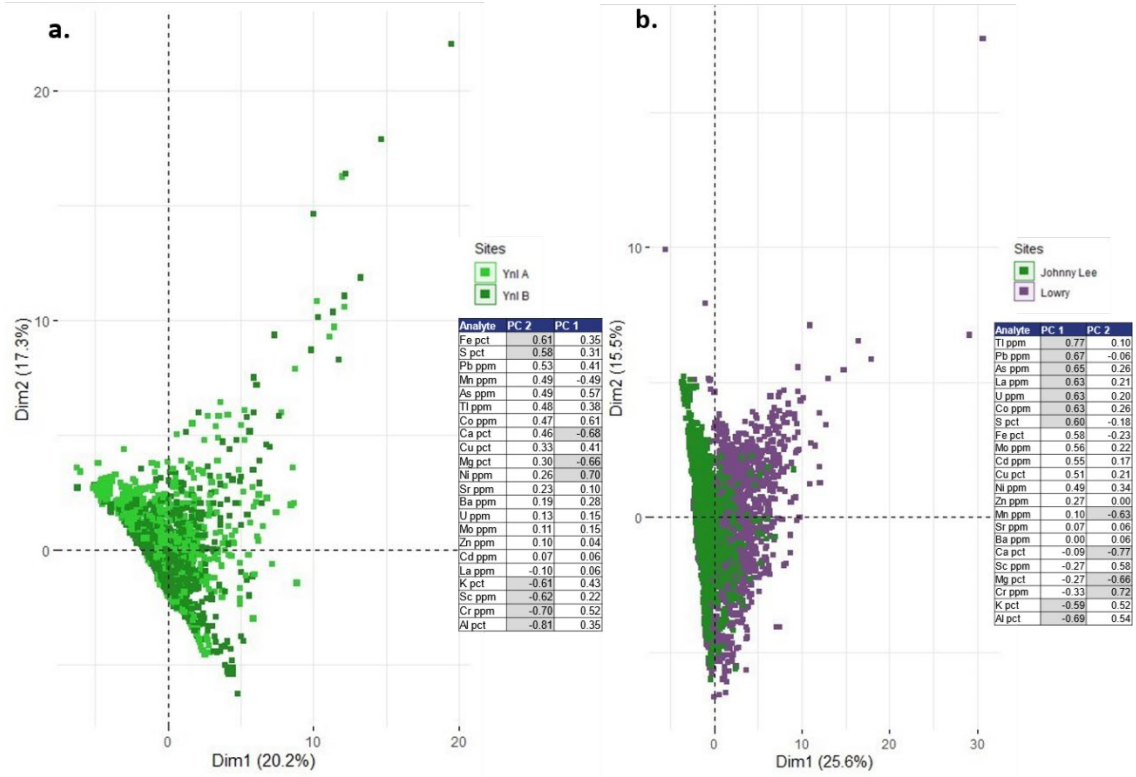


Figure 20.11 Principal Component Analysis of Ynl units in the Johnny Lee Deposit (a) and the Johnny Lee and Lowry Deposits (b)

Figure 20.12 compares boxplots of key analytes in the USZ from both the Johnny Lee and Lowry deposits. These boxplots demonstrate strong chemical similarities between these deposits, suggesting potential to use previous data in conjunction with supplemental testing of Lowry deposit rocks to represent the Upper Sulfide Zone characterization. Similarly, Figure 20.13 compares boxplots of key analytes in the Ynl from both deposits. These boxplots highlight chemical similarities, particularly in iron, sulfur, and magnesium, as well as some differences in the distribution of elements such as copper, thallium, and arsenic.

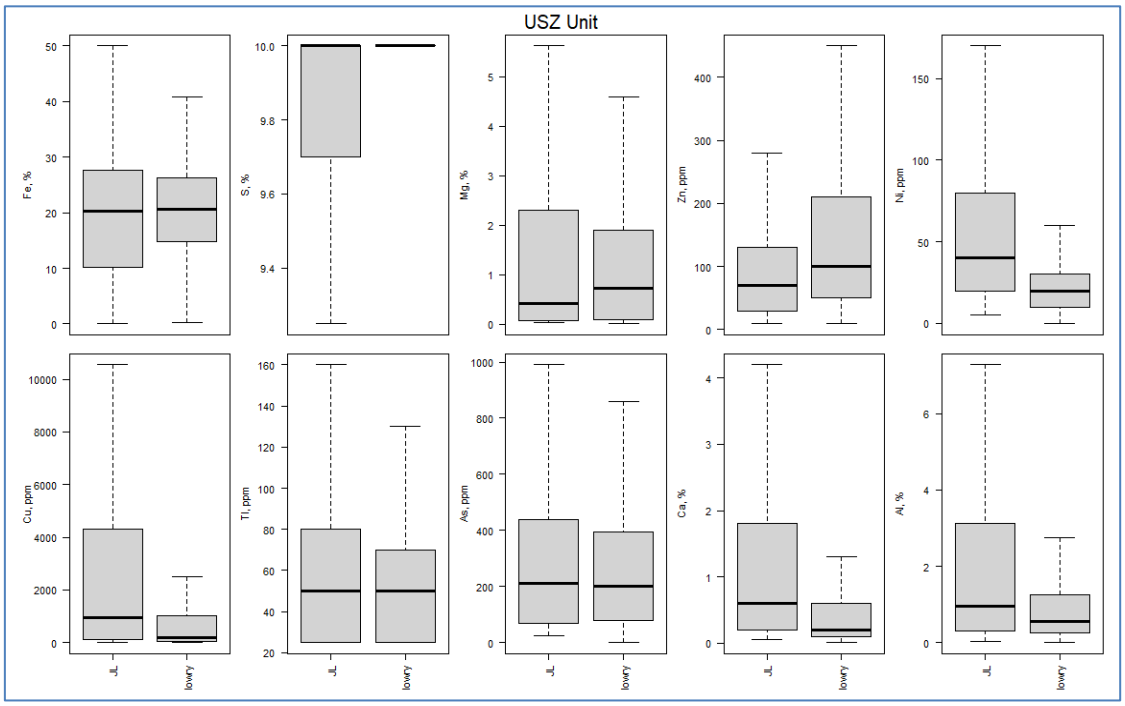


Figure 20.12 Boxplots of select parameters from multi-element testing of the USZ waste rock in the Johnny Lee (JL) and Lowry (L) deposits.

The PCA and boxplots of key elements in the USZ from the Johnny Lee and Lowry deposits demonstrate strong chemical similarities between these deposits, suggesting that previous data in conjunction with supplemental testing of Lowry deposit rocks is a valid geochemical evaluation of the Upper Sulfide Zone of the Lowry deposit. Additional comparisons of the Ynl unit, incorporating spatial layout, are recommended as new data become available, particularly in the area of the Lowry deposit access decline.

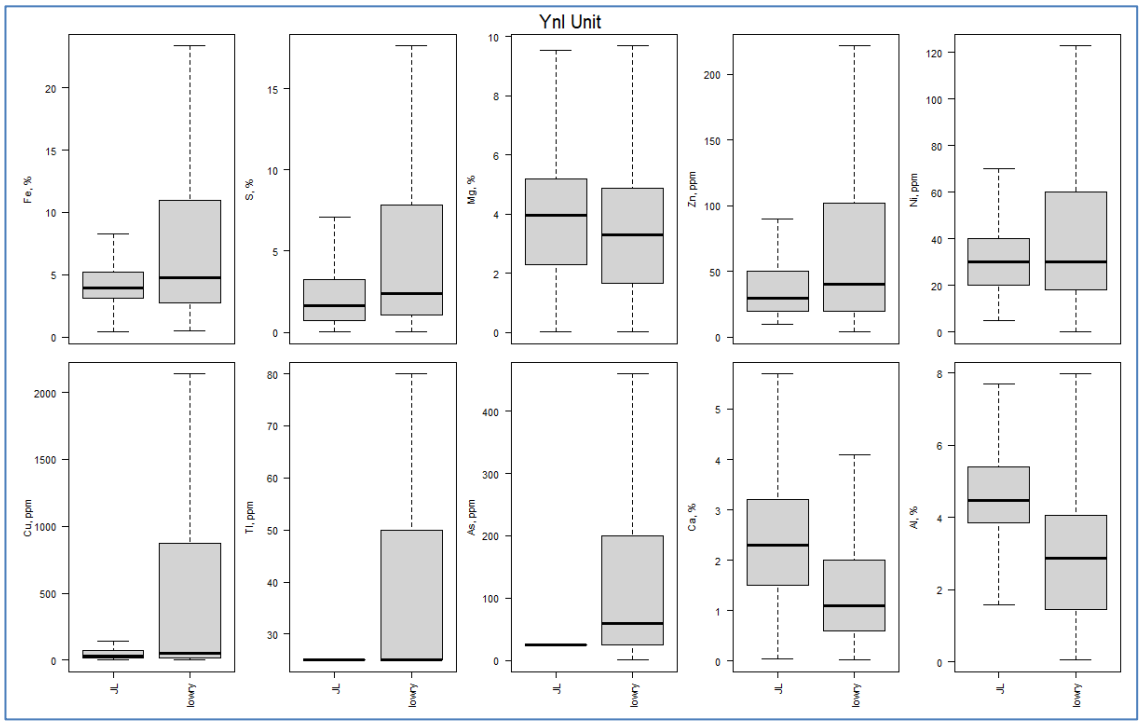


Figure 20.13 Boxplots of select parameters from multi-element testing of the Ynl waste rock in the Johnny Lee (JL) and Lowry (L) deposits.

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 6,195 km^2 (2,392 mi^2) and the population density is 0.8 people per mi^2, while the average for Montana in 2017 was 7.16 people per mi^2. Between 2010 and 2020 the population increased from 1,891 to 1,927.

Table 20.7 lists the population of White Sulphur Springs and other nearby towns and their distance from the Project site. Figure 1.1 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 (km)
White Sulphur Springs	955	0
Billings	117,116	240
Great Falls	60,442	158
Bozeman	53,293	127
Helena	32,091	123

Town or City	Population	Distance from White Sulphur Springs (km)
Livingston	8,040	115
Lewistown	5,952	184
Townsend	1,787	68
Three Forks	1,989	124
Harlowton	955	92
Belt	510	125
Clyde Park	332	82
Wilsall	197	69
Martinsdale	43	58
Ringling	45	35

Table 20.7 Population of Towns and Distance from Project Site

The demographics data for the county indicates that 32% of the population is over 65 and 18% is under the age of 18. Approximately 50% of the county's population is of employment age, compared to 60% for the state.

Meagher County is rural and the main industry is farming and ranching. Table 20.8 shows the industries in the county and trends in employment between 2011 and 2021. The total number of people employed in Meagher County in 2025 was 710.

Total Employment (Number of Jobs)	2011		2021		Number of Jobs - Change: 2011 to 2021
	Number of Jobs	Percent of Total	Number of Jobs	Percent of Total	
Full-time, year-round civilian employed population 16 years and over	491	--	670	--	179
Agriculture, forestry, fishing and hunting, and mining:	**139**	**28%**	**165**	**25%**	26
Agriculture, forestry, fishing and hunting	137	28%	157	23%	20
Mining, quarrying, and oil and gas extraction	2	0%	8	1%	6
Construction	57	12%	33	5%	-24
Manufacturing	3	1%	0	0%	-3
Wholesale trade	3	1%	0	0%	-3
Retail trade	27	5%	90	13%	63

Total Employment (Number of Jobs)	2011		2021		Number of Jobs - Change: 2011 to 2021
	Number of Jobs	Percent of Total	Number of Jobs	Percent of Total	
Transportation and warehousing, and utilities:	12	2%	53	8%	41
Transportation and warehousing	12	2%	28	4%	16
Utilities	0	0%	25	4%	25
Information	5	1%	0	0%	-5
Finance and insurance, and real estate and rental and leasing:	**21**	**4%**	**0**	**0%**	-21
Finance and insurance	21	4%	0	0%	-21
Real estate and rental and leasing	0	0%	0	0%	0
Professional, scientific, and management, and administrative and waste management services:	**10**	**2%**	**49**	**7%**	39
Professional, scientific, and technical services	5	1%	49	7%	44
Management of companies and enterprises	0	0%	0	0%	0
Administrative and support and waste management services	5	1%	0	0%	-5
Educational services, and health care and social assistance:	**81**	**16%**	**107**	**16%**	26
Educational services	26	5%	64	10%	38
Health care and social assistance	55	11%	43	6%	-12
Arts, entertainment, and recreation, and accommodation and food services:	**103**	**21%**	**16**	**2%**	-87
Arts, entertainment, and recreation	48	10%	0	0%	-48
Accommodation and food services	55	11%	16	2%	-39
Other services, except public administration	8	2%	19	3%	11
Public administration	22	4%	138	21%	116

Table 20.8 Meagher County Employment by Industry, 2011-2021

The unemployment rate is an indication of the potential available employees. Meagher County and Montana reported lower than average unemployment rates for June 2025 of 2.4% and 2.8%, respectively, compared to the US unemployment rate of 4.21%.

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 20.9 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
Per capita income in past 12 months	$34,766	$40,197	$43,300
Median household income	$56,215	$70,804	$83,730
Persons below poverty level, percent	10.7%	11.7%	10.6%
American Community Survey (ACS) 2023 5 Year Estimates			

Table 20.9 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 $442,147 and a surety bond of $607,419, totalling $1,049,566 in bonds, for release of the reclamation of earlier drill programs. These obligations will be released when the reclamation is completed by SRA and inspected and approved by the MDEQ.

Under Mine Operating Permit #00188, SRA currently holds a surety bond of $4,653,347, and a cash bond of $7,976, for reclamation of Phase I Construction activities, including ponds, borrow area and reclamation material stockpiles, roadways, and waste rock storage pad. In addition, there are approximately 53 ground water wells (37 monitoring wells/test wells, 1 water well, and 15 piezometers) currently in place that will ultimately need to be removed during closure and reclamation. All surety bonds are insured by Marsh McLennan Agency, formerly known as Payne-West Insurance. The MOP #00188 bond is undergoing a five-year review in 2025 and is expected to increase slightly.

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.

20.3.2.1 Air Quality

Quantitative modelling was conducted to evaluate the potential air quality impacts of the Project, 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 Project 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.

20.3.2.2 Groundwater, Seeps, and Springs

Operation of the mine dewatering system is expected to significantly lower groundwater levels around the mine, and to somewhat reduce base flow in nearby creeks based on water modelling work. Springs and seeps within the area are not expected to 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 20.4)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 HSUs. 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.

Potential impacts in the Lowry mine area will be fully analyzed and evaluated during future hydrologic characterization programs prior to mining.

20.3.2.3 Surface Water Quality and Flow

Potential impacts to surface water quality and flow are summarized in Table 20.10(MDEQ, 2020a). 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 and during high season flow (when flows in Sheep Creek exceed all beneficial uses).

20.3.2.3.1 Johnny Lee Deposit

Secondary impacts on base flow of Sheep Creek due to 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 is 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).

20.3.2.3.2 Lowry Deposit

Evaluation of impacts on base flow of Sheep Creek as a result of developing the Lowry deposit has not been conducted. Although surface water impacts have not been evaluated for the Lowry, it is assumed that any impacts will be offset by returning the water to the watershed or offsetting the impacts through currently held water rights or potential lease of other water rights in the area. A surface water impacts analysis and mitigation plan (if necessary) will be included in the Lowry Feasibility Study.

20.3.2.4 Water Supply

A critical issue in the ability of a mine to process ore is the availability of water. The mine will depend entirely on groundwater produced from the UG workings. The underground workings of the Johnny Lee deposit will produce an average of 2,725 m^3/day (500 gpm) (Hydrometrics, 2019a).

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. The use of water from mine dewatering is contingent on having sufficient water to mitigate adverse effects to other water rights. The mitigation plan includes use of both a high flow water right (when flows in Sheep Creek exceed downstream beneficial uses) and leased water rights to offset adverse effects from the use of water to process ore. The worst-case estimate assumes that the high flow water right will not be available, and mitigation will come from only the leased water rights. Below is a summary of the findings.

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 (reviewed and approved 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 (approved by 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 20.10 Potential Impacts Regarding Surface Water Quality and Flow – Johnny Lee Deposit

20.3.2.4.1 Lowry Deposit

The water produced from mining of the Lowry deposit will need to be defined based on a thorough hydrogeologic investigation. The total water produced from underground workings is dependent on the aquifer characteristics, development schedule, mining approach, and development sequencing.

20.3.2.5 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 for mining the Johnny Lee. Additional options for disposal of treated water were identified by Hydrometrics (2019a) and include deep well disposal, land application, and forced evaporation.

20.3.2.5.1 Lowry Deposit

Groundwater production from the Lowry deposit has the potential to exceed the UIG capacity. Initial additional options for disposal of treated water were identified by Hydrometrics and include deep well disposal, land application, year-round discharge to UIG, direct discharge to Sheep Creek and forced evaporation.

20.3.2.6 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.

20.3.2.7 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. An additional loss of use of a minimal area may occur for the Lowry deposit, for vent raises and light vehicle access roadways. For the Johnny Lee deposit, these direct impacts will last approximately 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.

20.3.2.8 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 will 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 will result in approximately 0.34 ha (0.85 acres) of permanently impacted wetlands from fill and dredging activities. Secondary impacts to wetlands from potential depletion of groundwater levels is possible. The most susceptible area is the wetlands near Coon Creek due to their proximity to the Johnny Lee development. The wetland monitoring program defines the baseline conditions of select wetlands in areas that have potential to have secondary impacts. This data will be compared to operational data and reference monitoring sites, located outside of areas that may have secondary impacts, and if necessary, wetland mitigation will include flooding the wetlands with water from the NCWR.

20.3.2.9 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 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.

20.3.2.10 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.

20.3.2.11 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.

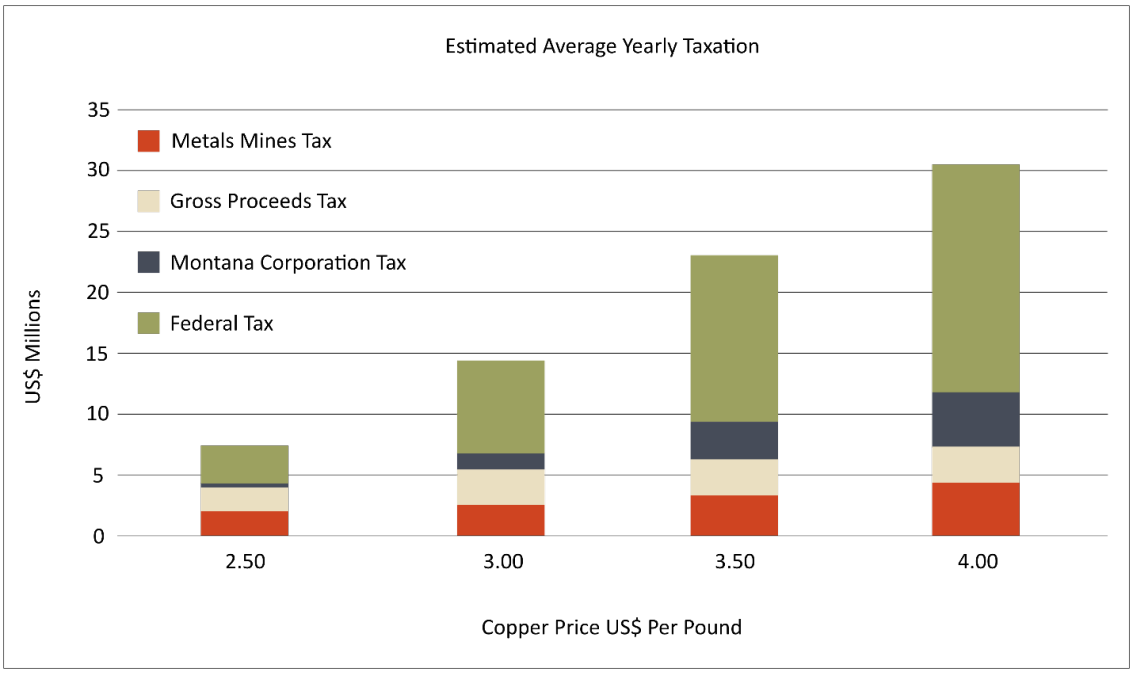


Figure 20.14 Estimated Average Yearly Taxation

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, Montana Pollutant Discharge Elimination System (MPDES) and construction and exploration storm water permit. A Draft EIS was published by the MDEQ on March 11, 2019, as required under Montana Environmental Policy Act (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.

In the future, an application for a major amendment to the Mine Operating Permit will need to be submitted to encompass the Lowry mining area. The amendment application will follow all requirements included within MMRA, MEPA, and other regulations as relevant.

A detailed list of all the Project permits and plans is provided in Table 20.11.

	Permit/Plan	Agency	Status
Mine Operating Permit	Exploration License	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 Construction Activity General Permit – Storm Water Discharges Associated with Exploration 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.	Approved
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	Completed
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	Noxious Weed Management Plan	Meagher County Noxious Weed Management	Approved
Cultural Resources	Historical Preservation Act	MT State Historical Preservation Office	Completed
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	Approved
	Explosives Permit	Bureau of Alcohol, Tobacco, Firearms and Explosives	Approved

Table 20.11 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 4.2). SRA owns portions of, and 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 mining claims contiguous with the fee simple (leased) lands. Figure 4.3 shows the proposed mine permit boundary and the location of the Johnny Lee and Lowry deposits 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 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.

SRA is currently employing Benchmark Gensuite software to track environmental performance and measure environmental compliance, in support of the EMS program. The company will continue to evaluate EMS systems and select the best fit for purpose solution in mine operations.

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

20.7.1 Tailings Properties

Acid Base Accounting (ABA) and Net Acid Generation (NAG) tests for the Johnny Lee deposit tailings and the geochemical similarities between the Lowry and Johnny Lee deposits, indicate that the tailings will have a strong potential to generate acid regardless of cement addition. The Humidity Cell Tests (HCT) indicate that all the Johnny Lee 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. 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 sulfide oxidation.

20.7.2 CTF Operation and Tailings Delivery

Waste rock generated over the life of mine will be stored on the lined WRS, to be used at the end of mine life to generate cemented paste tailings for underground closure. Remaining waste rock produced will be encapsulated in cemented paste tailings in a 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. Approximately 45% of the tailings will be mixed with 4% cement and binder and pumped back underground as backfill into mined out underground stopes and access headings during operations. The remaining 55% of tailings will be amended with between 0.5% - 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.

20.7.3 CTF Monitoring

Operation, monitoring, and record keeping of the CTF and other water and waste management facilities are described 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 has been included in the design and permitting phase. The CTF is not yet in operation and the TOMS Manual 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.

20.8.1.1.1 Johnny Lee Deposit

The permitted mine design for the Johnny Lee development includes an RO water treatment plant capable of treating up to 2725 m^3/day (500 gpm). The water treatment plant will be needed throughout mine construction, underground production mining, and during underground mine closure. Figure 20.15 shows the average annualized water balance schematic during mine operations (Zieg, 2018).

20.8.1.1.2 Lowry Deposit

Additional groundwater is anticipated to be produced from the Lowry deposit. A thorough hydrogeologic investigation must be conducted to characterize the hydrostratigraphic units in the vicinity of the Lowry deposit and to define the estimated mine inflow.

20.8.2 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 20.15. 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 5.2 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. Because the reservoir will contain natural surface water, it will not have the potential to chemically degrade groundwater.

20.8.2.1.1 Lowry

As the Lowry deposit continues to be evaluated, water supply and water rights will need to be reevaluated.

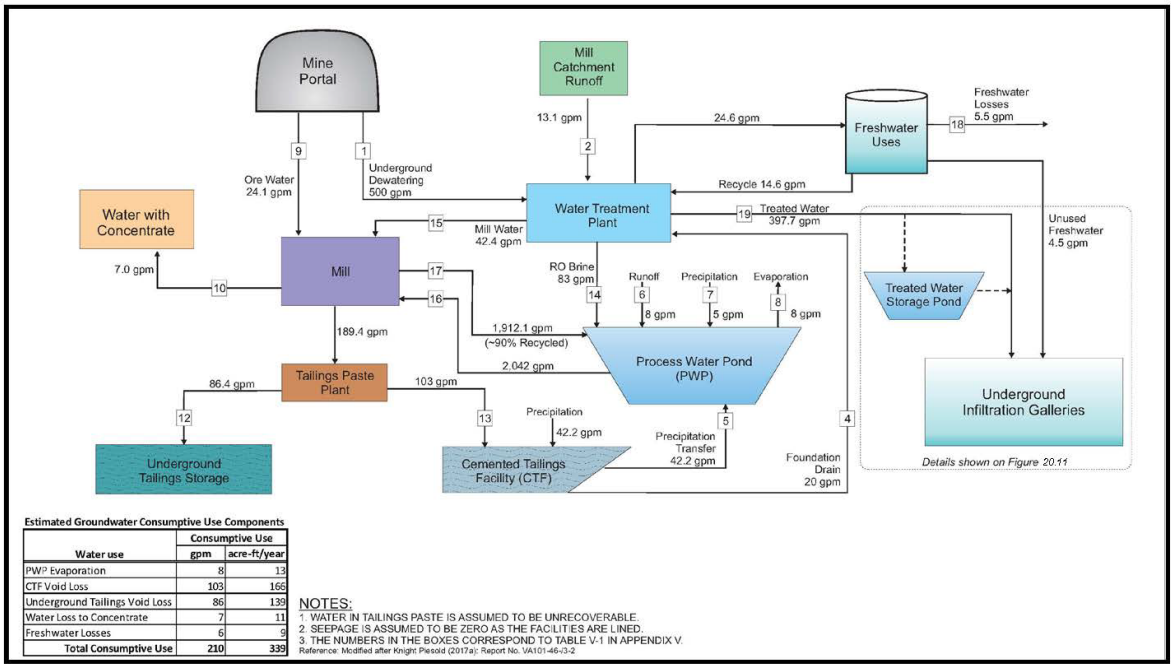


Figure 20.15 Average Annualized Water Balance During Mining Operations – Johnny Lee

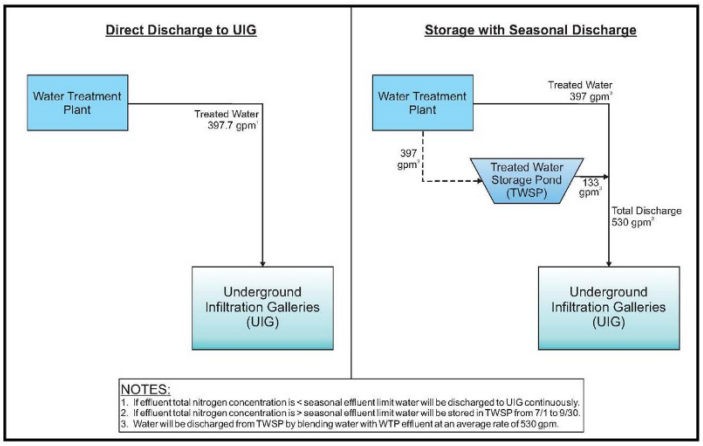


Figure 20.16 Average Annualized Water Balance During Mining Operations – Johnny Lee (part 2)

20.8.3 Water Balance

This section details the project water balance prepared for the Johnny Lee Feasibility Study and evaluates potential water balance limitations with respect to the Lowry deposit.

20.8.3.1 Water Balance of Permitted Development - Johnny Lee Deposit

As part of the MPDES Permit, a water balance was prepared based on average values at year 6 of mine operation. This water balance is based on development and mining of the Johnny Lee deposit and does not include any development or mining related to the Lowry deposit.

The water balance during mine operations is based on a calculated volume of groundwater from the mine dewatering system of the Johnny Lee deposit, average values for precipitation and temperature, and the capacity of principal facilities. The annualized water balance during operations is shown in Figure 20.15 and Figure 20.16. The water balance assumes an annual average mine inflow rate of 2,725 m^3/day (500 gpm). 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 limitations and assumptions are summarized below:

20.8.3.2 Process Water Pond (PWP)

- Minimum Operational Capacity: 80,000 m^3 (65 acre-ft) (Tintina Montana, 2017).
- Maximum Operational Capacity: 200,000 m^3 (162 acre-ft) (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.

20.8.3.3 Treated Water Storage Pond (TWSP)

- Sized to store average flow rate (2,207.6 m^3/day (400 gpm)) from WTP from July through September.
- Maximum Storage Capacity: (not including storm events): 204,000 m^3 (165 acre-ft).
- 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, 2018b).
- Water stored in TWSP from 1 July through 30 September.
- Average inflow rate: 2,167.8 m^3 day (398 gpm).
- Water discharged from TWSP from 1 October through 30 June.
- Average discharge rate: 725 m^3/day (133 gpm).
- A water storage year is considered as the period of July through June.

20.8.3.4 *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 20.8.4.

- 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 20.16). 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 2,169 m^3/day (398 gpm) to the UIG, and seasonal discharge of 2,889 m^3/day (530 gpm).

20.8.3.5 *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.

20.8.3.6 *Sensitivity Analysis – Johnny Lee*

Since the Johnny Lee 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, Figure 20.15 and Figure 20.16 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 of the Johnny Lee workings 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 of the Johnny Lee workings 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 (210 gpm). 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 issued MPDES permit. The annual average net precipitation to the CTF and PWP is about 257 m^3/day (47 gpm). This results in a dewatering rate of approximately 889 m^3/day (163 gpm) 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 (163 gpm).

Overall, the monthly water balance analysis prepared for the Johnny Lee deposit show shows 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 (525 gpm). 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 (163 gpm) (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.

20.8.3.7 Lowry Deposit

A detailed water balance and sensitivity analysis for the Lowry deposit has not been completed at this stage as there are numerous factors that still need to be determined. The two key factors that need to be determined prior to developing a detailed water balance for a potential Lowry expansion; 1) if the CTF will need to be expanded and what the final dimensions would be, and 2) if the Lowry will be developed concurrently or in sequence with the Johnny Lee development. Some key findings that can be taken from the Johnny Lee water balance and applied to the Lowry are summarized below:

- If the dewatering rates in the Lowry deposit are similar to flows in the Johnny Lee, then the flows would remain above the low dewatering rate analysed in the Johnny Lee sensitivity water balance.
- If the dewatering rates in Lowry are higher than in the Johnny Lee, it would potentially require increased capacity for the WTP, discharge capacity, and additional discharge options and/or increased storage of treated water, above the currently permitted facilities.

20.8.4 Water Treatment

A WTP will be used during construction, operations, and closure to treat mine process water and contact water. Figure 20.17 and Figure 20.18 show the permitted design for the Johnny Lee Deposit (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.

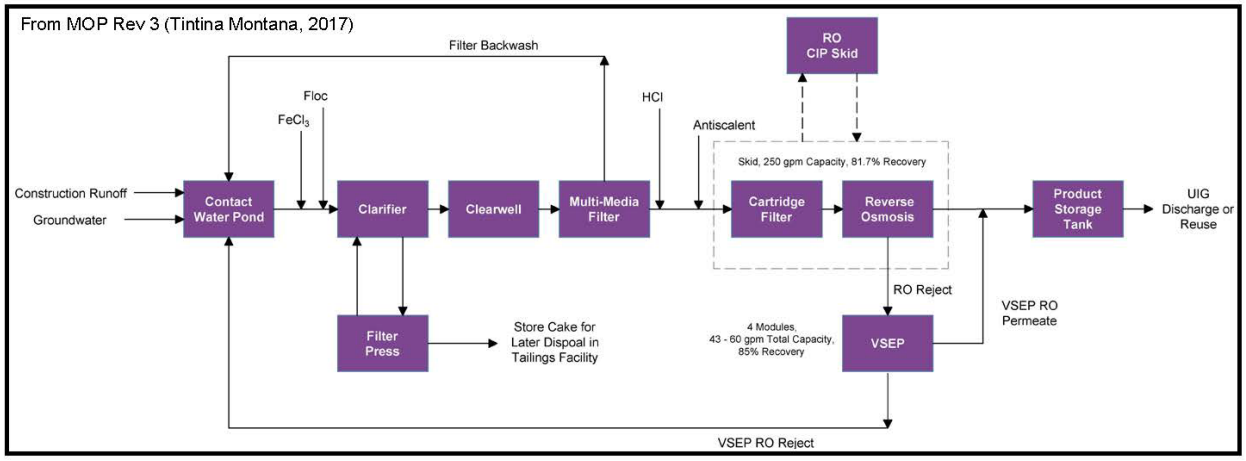


Figure 20.17 Water Treatment Construction Phase Process Flow Diagram

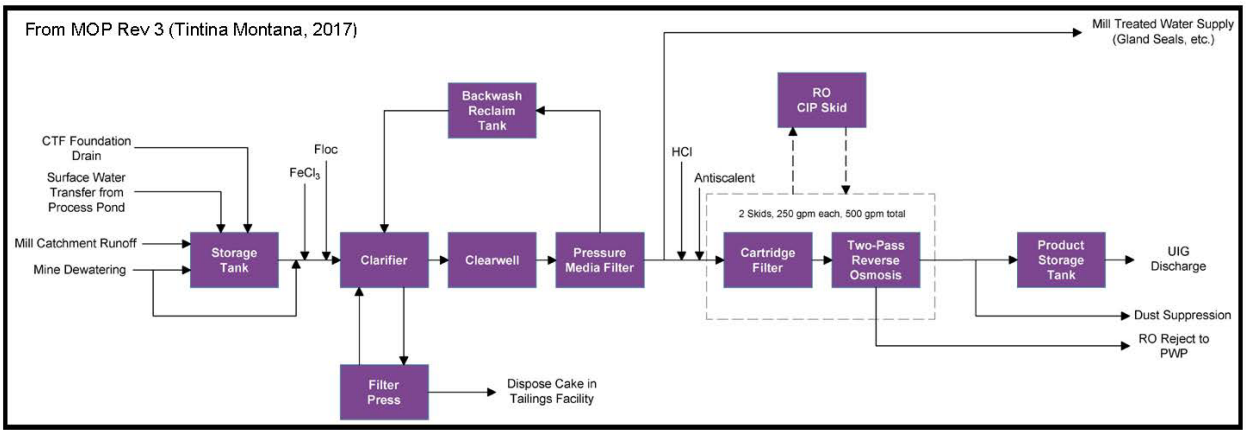


Figure 20.18 Water Treatment Operational Phase Process Flow Diagram

20.8.5 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 maximum total design discharge capacity of the UIG is 3,134 m^3/day (575 gpm).

Discharge of treated water will be contingent on the total nitrogen effluent limit as described in the Montana Pollutant Discharge Elimination System (MPDES). The MPDES 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. However, if the effluent nitrogen concentration exceeds the seasonal discharge limit, the water will be stored temporarily in the TWSP from 1 July to 30 September. Starting 1 October, the stored water will be routed back to the WTP and blended with the WTP effluent prior to discharge to the alluvial UIG. The planned location of the UIG is outside all designated wetland areas, and its length is oriented perpendicular to the groundwater flow direction.

20.8.5.1.1 Johnny Lee Deposit

The water balance analysis completed for the Johnny Lee deposit determined that year-round discharge to the UIG would average 2,169 m^3/day (398 gpm). However, if the effluent nitrogen concentration exceeds the seasonal discharge limits and water is temporarily stored from July through September, the discharge rate would average 2,889 m^3/day (530 gpm), Figure 20.15, which is below the maximum capacity of 3,134 m^3/day (575 gpm). The depth to the groundwater table in the UIG area once the mine has been developed would be approximately 2.4 to 4.0 m (8 to 13 feet).

20.8.6 Storm Water

Storm water management is comprised of 13 stormwater outfalls (002-014) included in the MPDES discharge permit. Compliance with the MPDES permit will be maintained using BMPs as outlined in the permit application. Numerous mine facilities identified in the site facilities map are covered by the storm water regulations including power plant (substation) areas and reclaimed areas 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.

20.8.7 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, 2018a): 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 (BMPs) will be implemented prior to and during construction at the Project. Erosion control BMPs reduce erosion by stabilizing exposed soil, or by reducing surface run-off flow velocities. There are generally two types of erosion control BMPs: source control BMPs for protection of exposed surfaces, and conveyance BMPs for control of run-off. Examples of BMPs that will be implemented are included in both the Integrated Permit application and are from "Water Quality BMPs for Montana Forests" (MSU Extension Service, 2015).

Development of the Lowry deposit will likely result in additional stormwater permitting for disturbances, such as around vent raises and expansion of surface facilities, if necessary. The additional stormwater management and permitting are not anticipated to be significant.

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. Each are 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, wildlife, vegetation, 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 (2020b) 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 monthly, quarterly and annual 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. Additional sites will be added in the vicinity of the Lowry deposit, as described in Section 20.9. 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 facility can be reduced or that the closure objectives have been met and monitoring can be eliminated.

20.9.2.1 Lowry

The list below, along with the proposed hydrologic test wells, outlines the requirements for a water monitoring program at the Lowry deposit:

- An additional spring and seep survey must be conducted within a 2.4 km (1.5 mile) radius of the Lowry deposit.
- At a minimum, one additional surface water monitoring site is necessary upstream of the Lowry deposit on Little Sheep Creek.
- Additional monitoring wells, approximately 3-5, are necessary to properly characterize critical hydrostratigraphic units in the vicinity of the Lowry deposit, including unconsolidated deposits near surface and units overlying the Lowry deposit. 1-2 of these wells may provide dual purpose for the hydrologic testing and baseline monitoring.

- Test wells installed in the Lowry deposit will be used to provide water quality within the mineralized zone.
- Groundwater monitoring near facilities is currently sufficient. Additional monitoring may be necessary depending on final design of expanded facilities, if required.

A minimum of 1 year of baseline monitoring is required and a minimum of 5 years is advised to have a data set that captures trends and natural fluctuations in water quality and quantity over time and with different climatic conditions. To permit the Lowry deposit, monitoring of these additional sites should commence at least 5 years prior to the target permit application date.

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 and approval 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 other 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 Johnny Lee 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 have been 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 BMPs, and revegetation measures to prevent air and water pollution.
- Surveying and monitoring 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 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 Closire Costs and Reclamation Bond Requirements

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 Johnny Lee Project as described in the MOP Application (Revision 3), subsequent MOP updates (Zieg, 2018) and the Final EIS (MDEQ, 2020). The full Johnny Lee Project bond cost estimate is for the LOM that includes the early phase activities, all

other surface mine facilities and the underground 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^3.
- No water or brine in the BCWP at the start.
- 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.

In 2020, SRCE (McIntosh, 2020) calculated amounts of $24.3 M for the full Johnny Lee Project Bond (LOM) mine closure activities that will be required to be paid by SRA prior to the start of the Phase 2 Development construction activities (see below discussion). The actual bond amount will be updated before the Phase 2 Construction process begins. Additional closure costs for the Lowry deposit may include:

- A larger CTF or second CTF.
- Rinsing of the potentially acid generating litho-units, water treatment and brine disposal.
- Hydraulic plugs and barriers (including reclamation material backfill) and monitoring wells.
- An expanded monitoring program.

The difference in closure costs for the surface facilities would be minimal as the same facilities would be used for both deposits. 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, which will be approximately 36% of the total estimated cost.

The ROD document (MDEQ, 2020b) separates the Mine Development Construction activities into two Phases with five stipulations. Phase 1 includes preliminary construction and earthworks activities, and Phase 2 includes all mining, processing, and operational activities representing the full Project bond amount. SRA requested the MDEQ to consider an incremental bonding payment approach into 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). 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 Johnny

Lee Prefeasibility 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, 2020a).

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 for closure of surface 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.

Bond amounts are also evaluated when material project changes are permitted, at the discretion of MDEQ. It is expected that a bond adjustment will be required in conjunction with permitting of the Lowry mine area.

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, pipeline 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 underground 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). These volumes would increase according to the need for additional storage volume of cemented tailings. 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.

Minimal soil disturbance, salvaging, or stockpiling is expected for development of the Lowry mine area. Small areas will be disturbed for establishment of vent raises and light vehicle access roads. Any topsoil and subsoil encountered will be salvaged and added to existing stockpiles and used for reclamation during the closure phase.

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 underground workings are two of the most important activities and are summarized below.

20.10.7.1 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.

20.10.7.2 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, the sulfide zones of the mine would be separated by hydraulic barriers (paste walls). Each sulfide zone would be filled with cemented paste tailings to limit the potential for groundwater interaction with potentially oxidized sulfide surfaces and to separate them from non-sulfidic zones that will be flooded. For the Johnny Lee deposit, an estimated 14 hydraulic barriers—both plugs and walls, which are masses of concrete—would be installed in the underground workings. The primary purpose of the hydraulic barriers is to segment the mine workings based upon sulfide content to facilitate selective backfilling 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 volumes of treatment water to be generated during underground mine closure are derived from assumptions in the MOP Application. 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. Off-site disposal of brine could begin anytime during the underground mine closure process and may need to be completed prior to starting to close and reclaim the BCWP. During the underground mine closure process, adequate storage capacity will be maintained in the BCWP and the PWP.

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 have been estimated to a PFS-level, with an accuracy of ±25%. The costs are presented in US dollars ($) and are based on prices obtained during the third quarter of 2025 (3Q25).

21.1.1 Underground Mine Operating Costs

Operating costs were estimated based on applying fixed and variable estimates to the mine plan schedule physicals. These costs were developed to a PFS-level of accuracy and were based on the following:

- A contractor request for quotation (RFQ) process for pre-production activities in the first 36 months of the mine plan, and
- Owner-operator cost estimates in the subsequent production phase, built up from first principle estimates and supplier quotations.

Operating costs include:

- Operating development costs related to the drilling, blasting, material movement, and ground support installation in ore drifts, including production accesses and associated drifts in waste.
- Ore stripping costs related to the drilling, blasting, remote mucking and haulage of all material from unsupported stripping activities.
- Paste backfill costs related to paste materials, reticulation system, construction of paste barricades and maintenance of the underground paste system.
- The operating cost allocation of mine services provision including contractor fixed costs, owner fleet equipment costs, owner operational and maintenance labour, logistics, power, grade control activities, ventilation and air heating, and diesel.
- The operating cost allocation of mine owner overhead costs including technical services, management and support, mining department general and administration costs, and site accommodation, and
- Grade control activities.

All operating costs incurred in the pre-production phase are classified as capital costs. The cost estimate does not account for tariffs or visas.

The operating cost estimate has been developed based on the Project schedule presented in Section 16.

The average UG mining operating cost estimate excluding capitalised pre-production costs is $79 million per annum (during the production phase), or $66/t ore mined, as summarised in Table 21.1.

Operating Cost Centre	Total Cost $ million	Annual Cost $ million	Unit Cost $/t ore
Development	249	31	26
Stoping	75	9	8
Pastefill	63	8	7
Mine Services	165	21	17
Mine Overheads	52	7	6
Grade Control	18	2	2
TOTAL	**622**	**79**	**66**

Table 21.1 UG Mine Operating Costs Summary

21.1.1.1 Major Operating Items

Major components to the operating cost estimates are summarised as follows.

21.1.1.1.1 Variable Costs

Costs per activity unit were determined based on quotes from suppliers and equipment manufacturers and include:

- Consumables (excluding diesel).
- Maintenance.
- Lubricants.
- Tires, and
- Ground engaging tools.

21.1.1.1.2 Equipment Fixed Costs

Owner costs including financing, insurance, and depreciation were calculated as a $/month for each item of equipment and plant. Contractor fixed costs were provided in the RFQ.

Owner fleet level requirements were estimated based on the mining schedule and reasonable productivity assumptions relevant to the jurisdiction. Contractor fleet estimates were based on information provided as part of the RFQ process.

21.1.1.1.3 Labor Fixed Costs

UG owner mining staffing levels were built up based on a first principles estimate of productivities and varying requirements 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 were included in the estimate.

Contractor staffing level estimates were based on information provided as part of the RFQ process.

Costs for Contractor and Owner personnel accommodation on-site are included in this area.

21.1.1.1.4 Fuel Costs

Fuel costs have been calculated based on equipment consumption, including heating systems and temporary gensets at mine start-up. The fuel price used was $0.74/L diesel.

21.1.1.1.5 Power Costs

UG mine power costs include allowances for fleet items, ventilation fans, infrastructure and dewatering. The power price used was $0.082/kWh.

21.1.1.1.6 Other Costs

This item includes mining department general and administration (G&A) costs such as office supplies, software maintenance, and technical services consumables and equipment.

Table 21.2 and Figure 21.1 illustrate the mining operating cost broken down by major component.

Cost Centre	Total Cost $ million	Annual Cost $ million	Unit Cost $/t
Variable Costs	262	33	28
Equipment Fixed Costs	59	7	6
Labor Fixed Costs	211	27	22
Fuel	24	3	3
Power	22	3	2
Other	45	6	5
TOTAL	**622**	**79**	**66**

Table 21.2 Mining Operating Cost by Major Component

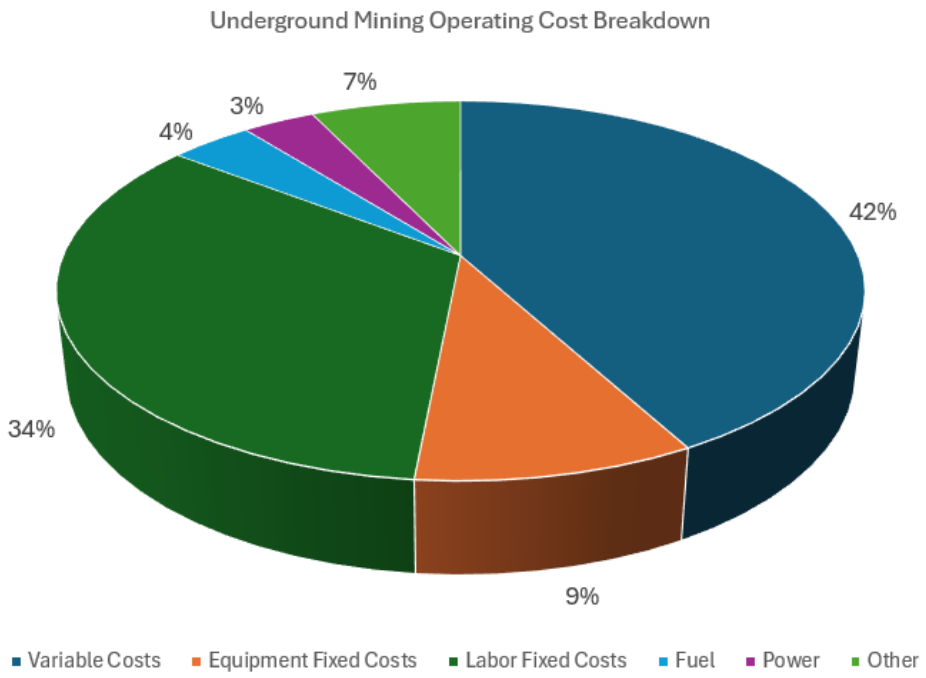


Figure 21.1 Mining Operating Cost Breakdown

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 a PFS-level, with an accuracy of ±25%.

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 1.2 Mt/a.

The costs have been compiled from a variety of sources including:

- First principle estimates.
- Suppliers' budget quotations.
- GR Engineering Services (GRES) data based on similar operations.
- Metallurgical test work results.

The operating cost estimate is $44.8 million per annum or $37.3/t milled. These costs exclude process plant sustaining capital which is presented in Section 21.3.2.

Cost Centre	Annual Cost $ million	Unit Cost $/t
Processing Costs – Operations		
Salaries/Labor	8.9	7.45
Power	6.2	5.21
Reagents	5.3	4.46
Grinding Media	4.8	3.99
Paste Consumables	2.1	1.78
Other Consumables	1.9	1.59
Maintenance	4.4	3.64
General	1.9	1.58
Laboratory	0.4	0.37
Processing Costs – Administration		
G&A	8.7	7.24
TOTAL	**44.8***	**37.31**

** May not align due to cumulative rounding*

Table 21.3 Annual Processing Costs – Concentrator

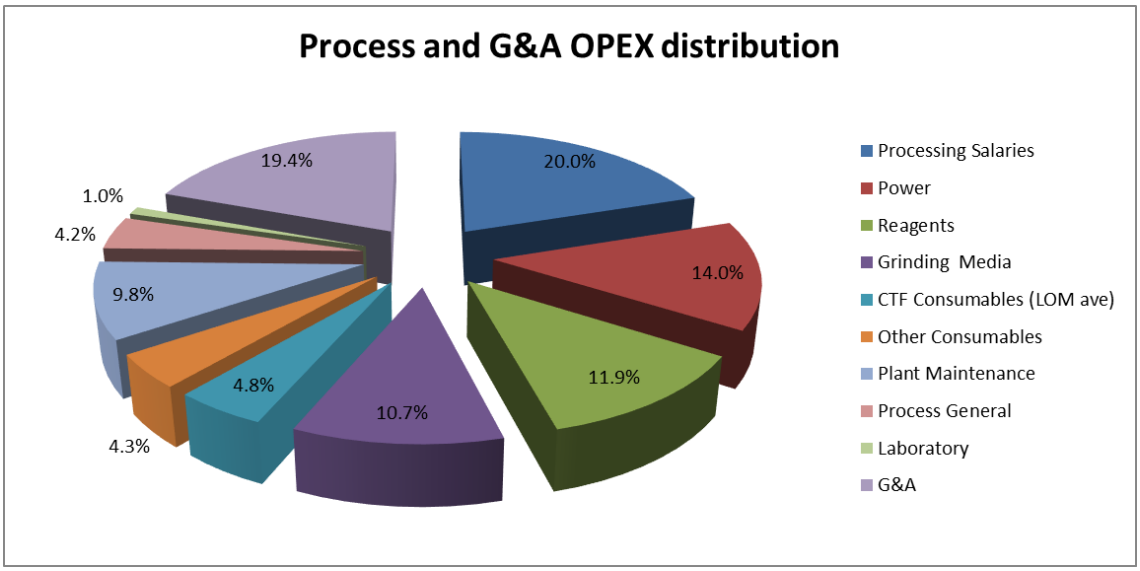


Figure 21.2 Annual Processing Costs - Concentrator

21.1.2.1 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 ±25%.

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.
- Escalation.
- Project financing costs.
- Interest charges.
- Political risk.
- Insurance.
- Land compensation/land Owner's costs.
- Subsidies to the local community.
- Rehabilitation costs.
- Amortisation and depreciation charges.

21.1.2.1.1 Concentrate

- Concentrate transport from site by truck in sea containers has been estimated separately from the Process and G&A operating costs.
- Product marketing costs have been excluded.
- Treatment and refining of the concentrate smelters have been excluded from concentrate transport costs and separately added to the financial model.

21.1.2.1.2 Tailings Storage

- Future wall lifts on the cemented tailings facility (CTF) have not been included within the operating cost and have been included as sustaining capital.

21.1.2.1.3 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.

21.1.2.1.4 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.
- Labor rates were obtained from a national survey of mining operations, with rates taken from the Rockey Mountain Region summary.
- Other allowances included in the annual salaries were:
  - Health insurance premiums (based on an estimated average family status made up of singles, couples with no children, single parents and families).
  - Annual leave (vacation leave) accrual based on service duration.
  - Payroll tax.
  - Medicaid, and
  - 401k allowances.
- Accommodation costs have been allowed for within the G&A costs for all workers requiring accommodation to be provided. An estimate 20–25% of the total workforce requirements were assumed to be locally available (and don't require accommodation to be provided).

21.1.2.1.5 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.

21.1.2.1.6 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 $0.082/kWh.

21.1.2.1.7 General and Administration

- General and administration costs have been estimated from other similar scale operations and include allowances covering:
  - Training and safety supplies.
  - IT software.
  - Consultants.
  - Local government fees.
  - Audit fees.
  - Insurance.

- Internet and other telecommunications costs.
- Cleaning and laundry allowances.
- Stationary and general freight allowances.

21.1.2.2 Major Operating Items

The major operating costs are detailed in Table 21.3 and consist of the following:

21.1.2.2.1 Labor

The process operational labor costs account for 20% of the overall cost to operation and maintain the processing plant. These costs are based on data sourced from government departments, National surveys and correspondence with SRA, for Montana and US based mining and processing operations.

Cost Category	Number of Employees No.	Base Salary/Wage $/a	On Cost $/a	Total Cost $/a
Processing				
Laboratory	6	375,000	167,916	542,916
Process Plant Management	12	1,506,200	454,707	1,960,907
Process Plant Operations	30	2,280,854	906,182	3,187,036
Plant Maintenance	23	2,458,100	792,974	3,251,074
Administration				
(Including Procurement, HSE)	32	3,109,000	1,049,367	4,158,367
TOTAL	**103**	**9,729,154**	**3,371,146**	**13,100,300**

Table 21.4 Processing Salaries and On-Costs

21.1.2.2.2 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.
- The paste plant uses 7.8 MWh/a of combined usage.

Area Description	Installed kW	Consumed kW	Annual Usage kWh
Primary Crushing	396	267	1,556,010
Ore Storage and Handling	159	91	461,862
Grinding and Classification	7625	5,734	44,913,342
Flotation Regrind	2,744	1,569	12,395,676
Concentrate Dewatering and Containerisation	291	182	1,154,024
Tailings Disposal	149	74	560,755
Reagents	65	42	176,229
Paste Plant	2,446	1,621	7,844,639
Air Services	720	295	2,121,532
Water Services	882	478	2,724,008
Admin, Lighting, Workshops, Misc. #	584	385	2,294,726
TOTAL	**16,060**	**10,736**	**76,202,803**
Total Cost $			**6,247,777**
Total Cost $/t Ore			**5.21**

Table 21.5 Power by Plant Area

21.1.2.2.3 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 (t)	Unit Cost	Annual Cost	
Reagents							
Hydrated Lime	t	g/t ore	1320	1,584	285	451,741	0.38
MSP	t	g/t ore	435	522	3,087	1,611,414	1.34
Dextrin	t	g/t ore	345	414	1,550	641,700	0.53
Areophine 3477	t	g/t ore	129	155	3,480	538,704	0.45
W31	t	g/t ore	102	122	4,340	531,216	0.44
MIBC (AeroFroth 70)	t	g/t ore	136	163	4,340	708,288	0.59
Flocculant Tails Thickener	t	g/t tailings	30	32	8,690	278,514	0.23
Flocculant Cu Cons Thickener	t	g/t conc	30	4	8,690	34,326	0.03
Flocculant Paste	t	g/t tailings	45	48	8,690	417,770	0.35
Anti-scalent	t	g/t ore	10	12	7,040	84,480	0.07
Paste Plant							
Cement (LOM ave)	t	kg/t tail	1.0	4,420	221	978,445	0.82
Binder (LOM ave)	t	kg/t tail	1.0	4,420	256	1,132,544	0.94

Table 21.6 Reagent Operating Costs (LOM ave)

21.1.2.2.4 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 supplied by Orway Mineral Consultants (OMC) as part of their comminution review. Hi-chrome (stainless steel) grinding media has been selected for the ball mill. 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 (mild steel)	t	kg/t ore	0.84	1,003	1,630	1,635,216	1.36
Ball mill balls (hi-chrome)	t	kg/t ore	1.36	1,634	1,925	3,146,220	2.62
Regrind mill media (ceramic)	t	kg/t conc	0.28	36.9	2,352	86,713	0.07

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 copper concentrate regrind application from database information.

21.1.2.2.5 Concentrate Freight

Concentrate freight is one of the largest costs to the operating cost and has been based on $193.6/wmt for ground transport and shipping costs:

- Ground freight (a combination of trucking and rail) including container rental, trucking from site to the rail terminal, rail freight, railcar lease and port charges was estimated at $145.4/wmt.
- Sea transport to smelters was estimated separately using current sea freight rates (with various shipment sizes) was estimated at $48.2/wmt.
- All transport costs have been based on 10% concentrate moisture.

Based on the life of mine (LOM) production rates, concentrate transport costs equal $221 million over the life of the project, or $23.37/t ore.

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.

21.1.4 *Operating Cost Overall*

The combined mining and processing costs for the Project, given in Table 21.8, are $123.4 million per annum and $103.1/t (mined ore to mill). This excludes concentrate transport costs.

Cost Centre	Annual Cost $ million	Unit Cost $/t
Processing Costs – Mining		
Variable Costs	33.1	27.65
Equipment Fixed Costs	7.4	6.22
Labor Fixed Costs	26.7	22.30
Fuel	3.1	2.59
Power	2.7	2.28
Other	5.6	4.71
Sub-Total	**78.6**	**65.76**
Processing Costs – Operations		
Salaries/Labor	8.9	7.45
Power	6.2	5.21
Reagents	5.3	4.46
Grinding Media	4.8	3.99
Paste Consumables	2.1	1.78
Other Consumables	1.9	1.59
Maintenance	4.4	3.64
General	1.9	1.58
Laboratory	0.4	0.37
General & Administration		
G&A	8.7	7.24
Sub-Total	**44.8**	**37.31**
TOTAL	**123.4**	**103.1**

Table 21.8 Combined Operating Costs

21.2 Capital Cost Estimate

21.2.1 *Introduction*

The Project capital cost estimate has been developed for the PFS based upon an EPCM approach for the construction and commissioning of the Project facilities. This includes 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, Owner's costs, 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, labour and mining costs was obtained during 3Q25. Any equipment quoted in foreign currencies has been converted using the foreign exchange rates and presented in US$. The capital cost estimates presented in this document are considered to have an overall accuracy of ±25% 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:

- GRES: process plant, plant infrastructure, non-process infrastructure and compilation of overall capital estimate.
- Entech: underground mine development and infrastructure, mine fleet and mine ancillary costs.
- Knight Piesold: Earthworks quantities used for the CTF estimate.
- Fergus Electric Co-op: High voltage overhead power transmission lines.
- 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
Australian dollars (AUD)	1.48
Euros (EUR)	0.86
Chinese yuan (CNY)	7.18

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 $ million
Mining	155
Process & Infrastructure	293
Owner's Costs	26
Total Project	**474**

Note: *Capital cost details for each Area in this table are included in Tables 21.11 and 21.12.*

Table 21.10 Total Project Capital Cost Summary

21.2.5 Capital Cost Estimate Summary – Mining

The total UG mining capital cost estimate is $224 million (excluding contingency) as summarised in Table 21.11.

Item	Pre-Production Cost $ million	Sustaining Cost $ million	Total Cost $ million
Lateral Development	42	38	80
Vertical Development	14	4	18
Mine Services	35	20	56
UG Infrastructure	10	3	13
Mine Overheads	6	3	9
Capitalised Pre-Production Operating Costs	48	0	48
TOTAL	**155**	**69**	**224**

Table 21.11 Total Mining Capital Cost Summary

This capital cost estimate includes the following:

- Capital lateral development, including permanent declines, accesses, and infrastructure development. These costs were estimated from information provided from the contractor RFQ process.
- Capital vertical development, including ventilation raises from surface, ventilation raises collared underground, and secondary egress ladderway raises. These costs were estimated from information provided from the contractor RFQ process.
- The capital cost allocation of mine services provision including contractor fixed costs, owner fleet equipment costs, owner operational and maintenance labour, logistics, power, grade control activities, ventilation and air heating, and diesel. These costs were estimated from information provided from the contractor RFQ process, estimated from first principles, or obtained from supplier quotes.
- Capital UG infrastructure including ventilation fans, dewatering systems, electrical distribution and reticulation, technical services equipment, communications, contractor mobilisation and demobilisation, and contractor site establishment. These costs were estimated from information provided from the contractor RFQ process, estimated from first principles, or obtained from supplier quotes.
- The capital cost allocation of mine owner overhead costs including technical services, management and support, general and administration costs, and site accommodation. These costs were estimated from information provided from the contractor RFQ process or obtained from supplier quotes.
- All operating costs (as detailed in Section 21.1.1 previously) incurred during the pre-production phase.

Of this $224 million estimate, $69 million (31%) is sustaining capital expensed after the pre-production phase.

21.2.6 *Capital Cost Estimate Summary – Process & Infrastructure*

Below details a summary by area of the capital cost estimate for the design, construction and commissioning of the Process plant and site infrastructure facilities.

Area	Capital Cost $ million
200 Earthworks	44
300 Process Plant	107
400 Services	76
500 EPCM	48

Area	Capital Cost $ million
600 Spares	6
800 Contractor Indirect Costs	13
TOTAL	**293**

Table 21.12 Site Process & Infrastructure Capital Cost Estimate Summary

21.2.6.1 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 was 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 was incorporated into the estimate. The installation rates include all charges necessary to deliver the requirements of the Project.

21.2.6.1.1 Earthworks

An overall bill of quantities was compiled for the various elements of the work, based on the previous FS.

Rates adopted for the estimate were based on budget pricing submissions from reputable contractors with experience in undertaking similar works. The rates are inclusive of equipment hire, operator and maintenance labour, consumables, materials, overhead and margin. The basis assumes a single contractor on multiple work fronts performing all the nominated earthworks for the Project (roads, plant, pipelines, CTF and water pond works).

The 2020 Phase 1 earthworks were constructed from 17 August 2020 to 17 December 2020 and contained a total surface disturbance of 57.4 acres (excluding 3.4 acres of SWPPP BMPs) that included: portal pad and brine-contact water pond (BCWP) excavations (and installing the rock drains between them), establishing most of the material stockpiles, constructing the required temporary roads to these facilities (excluding the road to Sheep Creek County road), and aggregate production (crushing and screening of granodiorite) from a borrow pit.

The Phase 1 earthworks completed post FS were accounted for in the quantity take-offs for the PFS.

21.2.6.1.2 Fencing

Fencing quantities were based upon preliminary engineering and quantity take-offs from the drawings. Rates adopted for the estimate were based on a budget pricing submission from a reputable contractor with experience in undertaking similar works.

21.2.6.1.3 Fuel Storage and Distribution

Allowances were made for diesel storage vessels and a containerized filtering and pumping facility.

The propane storage vessels were assumed to be supplied, owned and maintained by the propane gas supplier under a supply agreement.

21.2.6.1.4 Buildings

Costs for major process buildings were based on a budget pricing submission from a reputable contractor with experience in undertaking similar works. Costs from some minor process buildings were estimated based on recent pricing for similar items from the GRES database.

21.2.6.1.5 Emergency Power Generation

The estimate includes two Cummins DQKAN 2,500 kW diesel generators as backup emergency power generation, based on pricing from APASOF.

21.2.6.1.6 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. The costs for the high voltage switchyard were factored from the previous FS.

21.2.6.1.7 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.

21.2.6.1.8 Concrete

Concrete quantities were generally developed for each major area from similar designs for previous projects, where suited to the scope. Some concrete quantities were estimated.

Rates adopted for the estimate were based on a budget pricing submission from a reputable contractor with experience in undertaking similar works. The rates used were inclusive of site concrete supply and

installation were inclusive of equipment, labour, fuel, consumables, materials, subcontractor indirect costs and margin.

21.2.6.1.9 Structural Steelwork and Platework

Structural steel quantities were developed for each major area from similar designs for previous projects, where suited to the scope. Some steel quantities were estimated. Platework quantities were estimated from similar applications on previous projects. Tankage quantities were calculated.

Rates adopted for the estimate for structural steel were based on a fabricator submission for similar for three mass classifications of structural steelwork, conveyor trestle and truss steelwork, gridmesh, handrailing and stair treads. The rates are ex-works, inclusive of shop detailing, materials, fabrication labour, painting materials and labor, consumables, fabricator indirect costs and margin.

Rates adopted for the estimate for platework were based on a mix of a fabricator submission and database rates for similar works. The rates were ex-works, inclusive of materials, fabrication labour, painting materials and labor, consumables, fabricator indirect costs and margin.

21.2.6.1.10 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 the following major equipment in the plant:

- Primary crusher.
- Apron feeders.
- Pebble crusher.
- SAG mill.
- Ball mill.
- Mill liner handler.
- Vibrating screens.
- Regrind mill.
- Flotation cells.
- Flotation blowers.
- Hydrocyclones.
- Tailings and concentrate thickeners.
- Concentrate filter.
- Paste filters.
- Paste mixers.

- Sump, slurry and process pumps.
- Paste pumps.
- Water treatment plant.
- HVAC systems (radiant heaters and air handling units).
- Cranes and hoists, and
- Laboratory equipment.

Recent pricing for similar equipment from the GRES database was used to estimate costs for the following equipment items:

- Dust collectors.
- Conveyor components.
- Weightometers.
- Lime silo.
- Samplers and analysers.
- Cement and slag silos.
- Lime storage and mixing system.
- Flocculant plant.
- Compressed air equipment, and
- Weighbridge.

21.2.6.1.11 Piping

In-plant piping costs were factored based on the engineer's experience on similar-sized projects.

Overland piping quantities were calculated from process requirements and geographical locations of the water sources and the location of the CTF for the tailings. Material supply rates adopted for the estimate were based on supplier submissions for the various specifications of pipes, valves and fittings.

21.2.6.1.12 Electrical and Instrumentation

In-plant electrical distribution and instrumentation costs were factored based on the engineer's experience on similar-sized projects.

21.2.6.1.13 Transport and Freight

Sea freight has been included for imported equipment items, either based on the supplier's budget pricing (where received) or based on an allowance by GRES.

Costs to transport equipment and materials to site have generally been estimated based on the size or weight of each item, using current rates from a transport provider, or based on a factored allowance.

21.2.6.1.14 Installation Labor

Site installation labor hours were based on the estimated effort associated with the equipment, materials and fabricated items to be installed in each area of the plant.

Labor rates for each trade classification were based on a budget pricing submission from a reputable contractor with experience in undertaking similar works.

Crew rates for each trade were built-up to include an appropriate mix of supervision, skilled and unskilled personnel. The crew rates include raw labor cost, overtime, fringes and industry funds, payroll taxes and liability workman's compensation insurance and payroll costs but exclude paid holidays, sick leave, income protection insurance and any uplift for night shift (i.e. day works only).

An assessment was made of non-productive time, to account for pre-start meetings, job safety analysis, toolbox meetings, smoko, weather impacts and the proportion of leading hand non-productive time.

Allowances for personal protective equipment, clothing, consumables and accommodation were included in Site Indirects.

21.2.6.1.15 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 from 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.

21.2.7 Capital Cost Estimate Summary – Project Indirects

Indirect costs were estimated by GRES. The costs were based on budget pricing from suppliers and historical costs from previous projects.

21.2.7.1 Cranes and Equipment Costs

Costs developed for the required site cranes and construction equipment were based on estimates of the required durations to complete installation works in each area of the plant. Industry-standard charge-out rates for the various cranes and equipment types were then applied to the estimated hire durations. Rates were obtained from a number of local suppliers. The rates did not include fuel, which was estimated separately.

21.2.7.2 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.

21.2.7.3 Construction Fuel

Construction fuel has been allowed for the duration of construction at $0.96/L ($3.65 per gallon). Construction fuel is included in the earthworks and concrete all-in rates.

21.2.7.4 Mobilization and Demobilization

Estimates of the mobilization and demobilization costs for each of the construction contractors have been included in the capital cost estimate. These costs have been based on data sourced from recent projects and make allowance for materials, equipment and personnel.

21.2.7.5 R&R Flights, Meals, and Accommodation

A mining village will be constructed and provide some accommodation and messing facilities. It has been assumed that 20% of the construction workforce will come from the local workforce, for whom accommodation and messing is not required. Earthworks were assumed to be supplied by local contractors, who will be responsible for their own accommodation. An allowance of $70/day has been allowed for construction labour where accommodation is provided. The allowance includes the supply of meals.

Flights, meals and accommodation have been included for 80% of direct labor and 100% of indirect construction personnel for the duration of the construction period. Flights, meals and accommodation for the EPCM contractor's personnel have been allowed for in the estimate.

21.2.7.6 *Engineering, Procurement and Construction Management*

The estimate for engineering, procurement, construction management and commissioning services was based on the hours required for a multi-disciplined project delivery team, consisting of suitably qualified and experienced personnel, to successfully complete the project scope within the scheduled timeframe. The involvement of each design discipline was estimated based on the complexity of the individual tasks and benchmarked against EPCM costs incurred on recent similar projects

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 estimate.

21.2.7.7 *Initial Fills*

The first fill quantities were derived from the process design criteria and rates from the operating cost estimate. A provisional sum was made for first fill of oils and lubricants and for general supplies.

21.2.7.8 *Spares*

Commissioning spares were included in the estimate, based on supplier submissions, for the crushers, grinding mills, and regrind mill. An allowance of 0.5% of the mechanical and electrical supply cost was made for commissioning spares for the remainder of the process plant equipment.

Mechanical capital spares were included in the estimate, based on supplier submissions, for the crushers, screens, grinding mills, flotation cells, thickeners, hydrocyclones, air blowers and regrind mill. An allowance of 2% of the mechanical supply cost was made for insurance spares for the remainder of the process plant equipment.

Electrical capital spares are included in the estimate as 2% of the electrical supply costs.

21.2.7.9 *Owner's Costs*

The Owner's costs included scope items that fell outside of the consultants' scope of work and included a mixture of pre-production labor, Owner's construction supervision, project insurances, and various other miscellaneous costs.

An allowance for accommodation facilities was included in the Owner's costs and could be used for temporary or permanent accommodation as required.

A summary of the Owner's costs is presented in Table 21.13.

Item	Cost $ million
Technical Support	0.4
Construction Management	4.3
Construction General Expenses	2.6
Pre-operational Expenses (Process)	2.9
Pre-operational Expenses (G&A)	5.5
Village Accommodation	7.5
Contingency	2.3
TOTAL	**25.5**

Table 21.13 Owner's Capital Costs

21.2.8 Contingency and Escalation

Contingency costs were estimated by GRES for the Infrastructure and Process Plant. The summary contingency costs are shown in Table 21.14.

Item	Cost $ million
Mining Contingency	0
Process & Infrastructure Contingency	25
Owner's Costs Contingency	2
TOTAL	**27**

Table 21.14 Contingency and Escalation Cost Summary

21.2.8.1 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 7.5–10% 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.

21.2.8.2 Escalation

Escalation is excluded from the estimate.

21.2.9 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

A breakdown of the $69 million mining sustaining capital estimate over LOM is provided in Table 21.15.

Item	Sustaining Cost $ million
Lateral Development	38
Vertical Development	4

Item	Sustaining Cost $ million
Capitalised Mine Services	20
UG Infrastructure	3
Mine Overheads	3
TOTAL	**69**

Table 21.15 LOM Mining Sustaining Capital Estimate

21.3.2 Process Plant and Infrastructure Sustaining Capital

A provision for minor projects of 0.5% of the plant CAPEX was applied over the LOM, equating to $2.2 million. Two allowances for mill refurbishment or repairs were also listed as 10% of the mill capital costs, allowing for $0.6 million over LOM. Other sustaining CAPEX items within the LOM included the design and construction of the CTF phase 2 lift, an allowance to replace surface vehicles and replacement of IT equipment. A breakdown of the sustaining capital estimate over LOM is provided in Table 21.16.

Item	Sustaining Cost $ million
Design and construct CTF Phase 2	8.6
Minor projects provisions	2.2
Mill repairs	0.6
Vehicle replacement	0.5
IT equipment replacement	1.0
TOTAL	**12.8**

Table 21.16 LOM Process and G&A Sustaining Capital

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 $30 million.

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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 Entech 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 life of mine (LOM) economic outputs. The economic model taxation treatment was reviewed by Assure CPA, 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 "PFS Case".

For the PFS, SRA has undertaken exploration and drilling to develop an updated Mineral Resource estimate reported in this document. 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 PFS plant feed estimate for the UG has increased by approximately 0.7 Mt compared to the 2020 FS amount of 8.8 Mt.

The economic analysis for the PFS uses a copper price of $4.70/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 successfully converting Inferred Resources to Measured or Indicated Resources, the addition of the Lowry Deposit 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 on 5 November 2025 by SRK Consulting. Entech have used these Measured and Indicated Mineral Resources to estimate a Mineral Reserves from the JL UCZ and the JL LCZ totalling 9.5 Mt averaging 2.9% Cu as determined on 14 November 2025. The Mineral Reserves are scheduled to be produced over a LOM of approximately eight years through the 1.2 Mt/a 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 PFS has been based on average monthly plant feed copper grades (based on the developed mine schedule) and recovery values developed from metallurgical testwork conducted on the JL UCZ and JL LCZ, and additional blending testwork between the two zones. Blending testwork conducted on a range of JL UCZ and JL LCZ ore showed the blended recovery arithmetically matched the forecast recovery from individual tests. Based on the blending results, the JL UCZ and JL LCZ recovery models were calculated independent of each other, with a combined recovery developed within the mining schedule.

The JL UCZ recovery model developed during the previously reported PFS was again used. As has been previously reported, the recovery model combined LCT results and cleaner variability tests to cover a greater range of mineralised ore.

JL UCZ 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 JL LCZ metallurgical recovery was derived from variability testing conducted during the PFS discussed in Section 13. The JL LCZ has been previously reported as having higher copper recovery (93.3% to 96.6%), improved copper concentrate grades and lower deleterious elements compared to the JL UCZ. Blending testwork confirmed combining the JL UCZ and JL LCZ ore didn't "poison" the JL LCZ ore in anyway. The PFS variability testwork showed a range of copper recoveries of 94.4% to 97.4% with an average recovery of 96.2%. To account for the ideal conditions under laboratory testing, a 1% copper recovery deduction was made, with an average copper recovery of 95.2% applied to the JL LCZ.

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 86.1%. Monthly based variability ranged from 77.4% to 92.4% 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.

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.

The economic model uses a treatment charge of $50/t and a refining charge of 5.0 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. 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*

Project economics were estimated based on the long-term metal price summarized in Table 22.1. This price was based on consensus data (top nine financial institutions) for the long-term real price forecasts from November 2025.

Metal Price	Unit	Value
Copper	USD/lb	4.70

Table 22.1 Metal price Assumptions

22.2.7 *Operating Costs*

For the full production years, the combined mining, processing and G&A costs for the Project average $123.4 million per annum or $103.1/t (mined ore to mill). 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 $474 million (including contingency).

Area	Capital Cost $M
Mining	155
Process & Infrastructure	293
Owners	26
Total Project	**474**

Table 22.2 Summary of Project Capital Cost

22.2.9 Royalties and Taxes

The economic model assumes that $10 million 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 $3.9 million prepaid royalty balance prior to production.

Project tax liability is based on Federal Income Tax (21.0%), Montana State Corporate Tax (6.75%), Montana Metal Mines License Tax (1.81%), Montana Metal Mines Gross Proceeds Tax (3%), Montana Class 4 (1.4%) and Class 8 property taxes (1.5% first $6.0 million thereafter 3.0%) and a local mill levy of 4%.

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. Project economics are calculated on a 100% equity basis.

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 therefore the economic outcomes are shown in real terms.

22.3 Economic Evaluation Results

The main inputs and outputs of the economic model for the PFS Case are based on the PFS base price deck described in Section 22.2.6. Table 22.3 shows an annual summary of key economic evaluation inputs and outcomes.

The Project has demonstrated positive economics based on the PFS Case and the assumptions used. The NPV together with a positive IRR supports the Project's viability and provides positive support to undertake the next phase of study. The recommendations and proposed forward program are discussed further in Section 26.

Description	Units	Total	Yr1	Yr2	Yr3	Yr4	Yr5	Yr6	Yr7	Yr8	Yr9	Yr10	Yr11	Yr12
Physicals														
Mined ore tonnes	Mt	9.5	-	0.0	0.3	0.9	1.2	1.4	1.2	1.3	1.4	1.4	0.4	
Processed tonnes	Mt	9.5				1.1	1.2	1.2	1.2	1.2	1.2	1.2	1.2	
Cu recovery	%	86.1	-	-	-	83.7	89.2	91.4	91.6	85.6	80.7	78.3	79.7	-
Cu - recovered	kt	234	-	-	-	24	33	41	42	32	25	22	15	
Cu - payable	kt	226	-	-	-	23	32	40	40	31	24	21	14	
Revenue														
Gross revenue	USDm	2,338	-	-	-	241	334	410	416	321	247	219	150	
Net revenue	USDm	2,035	-	-	-	210	291	357	363	280	214	190	130	
Operating Costs														
Mining	USDm	624	-	-	-	93	100	88	84	79	75	71	34	-
Processing	USDm	275	-	-	-	33	35	35	35	35	35	35	34	-
G&A	USDm	73	-	-	-	9	9	9	9	9	9	9	9	-
Royalty	USDm	51	-	-	-	14	6	7	7	6	4	4	3	-
Total operating costs	USDm	1,023	-	-	-	149	150	139	135	128	123	119	80	-
Operating cashflow	USDm	1,011	-	-	-	61	141	218	228	151	91	71	50	-
Capital Costs														
Development capital	USDm	474	21	233	219	1								
Sustaining capital	USDm	82	-	-	-	48	25	6	1	1	1	0	0	-
Closure	USDm	30	-	-	-	-	-	-	-	-	-	-	-	30
Total capital costs	USDm	585	21	233	219	49	25	6	1	1	1	0	0	30

Description	Units	Total	Yr1	Yr2	Yr3	Yr4	Yr5	Yr6	Yr7	Yr8	Yr9	Yr10	Yr11	Yr12
Economics														
Pre-tax FCF	USDm	426	-	-	-	11	70	136	153	69	2	-12.4	28	-30.0
Tax	USDm	67	-	-	-	6	4	18	27	10	0	0	1	-
Post-tax FCF	USDm	359	-	-	-	5	66	117	126	58	2	-12.8	27	-30.0
NPV8	USDm	98.8												
IRR	%	11.3												

Table 22.3 Black Butte Copper Project PFS Case Economic Evaluation Summary

22.3.1 NPV

The Project has a pre-tax NPV of $142.9 million and a post-tax NPV of $98.8 million using an 8% discount rate.

22.3.2 IRR

The Project has a pre-tax IRR of 13.6% and a post-tax IRR of 11.3%.

22.3.3 Payback

The projected cash flow will allow for the Project construction capital to be paid back in approximately four years from process plant commissioning.

22.4 Sensitivities

The Project NPV is most sensitive to the copper price. Other significant factors influencing the Project returns include the capital and mine operating costs.

22.4.1 Sensitivity Analysis

Table 22.4 and Table 22.5 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 @ 8%
	Worst	Best	Worst	Best	Point
Cu Selling Price	-10%	10%	($2.4)	$288	$143
Operating Costs - Mining	10%	-10%	$103	$182	$143
Operating Costs - Process	10%	-10%	$126	$160	$143
Operating Costs – G&A	10%	-10%	$138	$147	$143
Capital Costs – Total	10%	-10%	$92.5	$193	$143
Discount Rate	10%	-10%	$123	$164	$143

Table 22.4 Black Butte Copper Project PFS Case Pre-Tax NPV Sensitivity Impacts

Sensitivity Variables	Confidence Ranges		$ Millions	Post-Tax	NPV @ 8%
	Worst	Best	Worst	Best	Point
Cu Selling Price	-10%	10%	($27)	$217	$99
Operating Costs - Mining	10%	-10%	$65	$131	$99
Operating Costs - Process	10%	-10%	$84	$113	$99
Operating Costs – G&A	10%	-10%	$95	$103	$99
Capital Costs - Total	10%	-10%	$52	$143	$99
Discount Rate	10%	-10%	$81	$118	$99

Table 22.5 Black Butte Copper Project PFS Case After-Tax NPV Sensitivity Impacts

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23. ADJACENT PROPERTIES

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 gossan from oxidization of Newland Formation sulfide beds 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.

Additionally, there are multiple identified exploration prospects in the vicinity of the Johnny Lee and Lowry deposits that exist within the SRA-controlled land package. SRA provided details of these exploration prospects in Section 9.

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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.
- Design and construction of the accommodation village and any additional temporary construction accommodation requirements.
- 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 PFS, 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 PFS determined that most 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.

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 and a newly constructed accommodation village.

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 PFS Report.

SANDFIRE RESOURCES AMERICA INC. | BLACK BUTTE COPPER PROJECT | PRE-FEASIBILITY STUDY | PROJECT IMPLEMENTATION SCHEDULE

ID	Task Name	Duration	Duration (weeks)
0	**BLACK BUTTE COPPER PROJECT**	**1038 dys**	**148**
1	Project start	0 wks	0
2	Mining mobilisation	0 wks	0
3	Mining commences	0 wks	0
4	Process plant - award EPCM	0 wks	0
5	Process plant - construction complete	0 wks	0
6	Process plant - practical completion	0 wks	0
7	**MINE DEVELOPMENT**	**1038 dys**	**148**
27	**PROCESS PLANT DESIGN AND CONSTRUCTION**	**703 dys**	**100**
28	Award of Plant Contact	0 wks	0
29	**Client Deliverables**	**584 dys**	**83**
38	**Project Milestones**	**689 dys**	**98**
50	**Off-site EPCM Works**	**567 dys**	**81**
612	**Site Construction Works**	**567 dys**	**81**
613	**Preliminary Activities**	**7 dys**	**1**
616	**Plant Bulk Earthworks**	**112 dys**	**16**
622	**Infrastructure Earthworks**	**336 dys**	**48**
633	**Concrete Works**	**161 dys**	**23**
643	**Steel Tank Site Works**	**91 dys**	**13**
646	**Structural Steel Erection**	**238 dys**	**34**
657	**Platework Installations**	**210 dys**	**30**
665	**Mechanical Installations**	**294 dys**	**42**
677	**Piping Installations**	**273 dys**	**39**
686	**Building Installations**	**168 dys**	**24**
689	**Electrical installations**	**392 dys**	**56**
727	**Commissioning**	**283 dys**	**40**
746	**ORE COMMISSIONING**	**14 dys**	**2**
748	Training and handover	2 wks	2
749	Demobilisation	0 wks	0

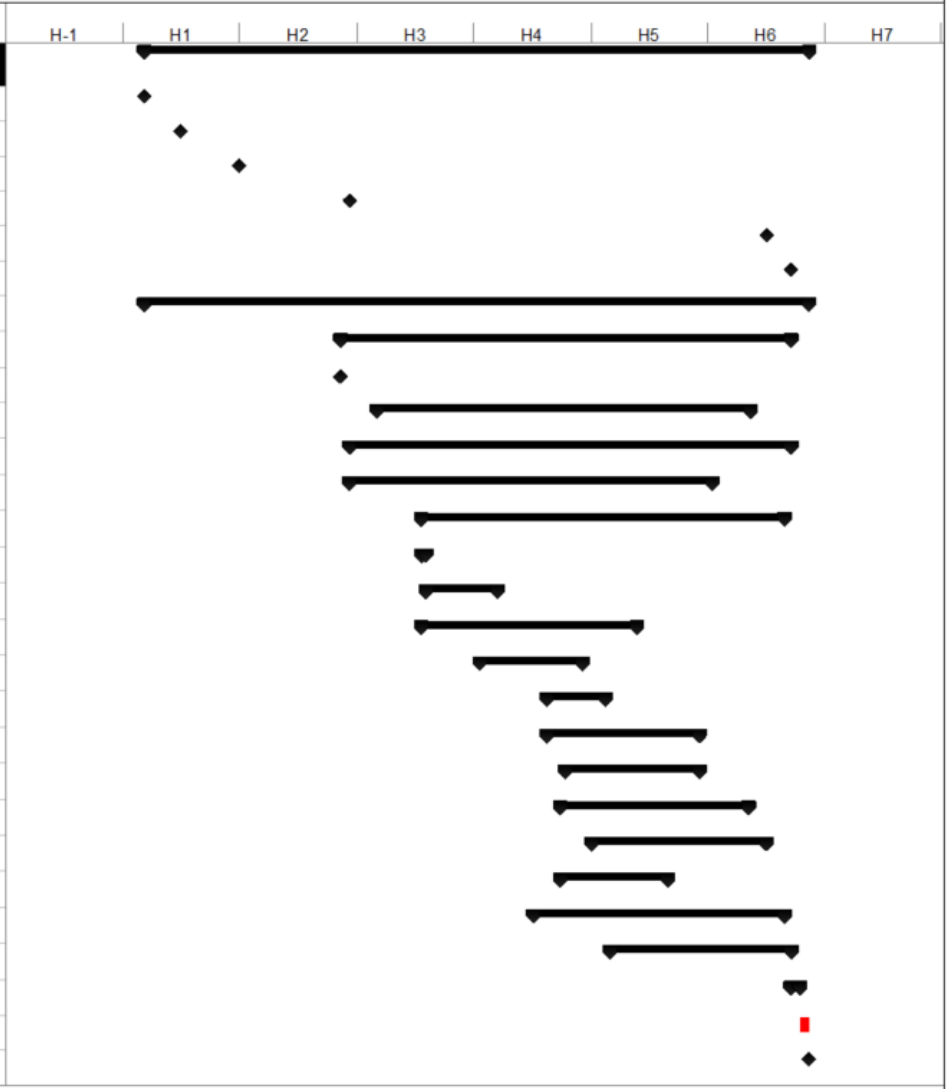


Figure 24.1 Project Implementation Schedule

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 PFS.

Item	Potential Manufacture Location as tendered	Delivery Period (Weeks)	Logistics duration (Weeks)
Paste Filters	Japan	61	8
SAG and Ball Mill	Various global locations	52	8
Concentrate Filters	Various global locations	52	8
Flotation Cells	Finland / China	48	8
Paste Pumps	Germany	48	8
Water Treatment Plant	USA	48	2
Regrind Mill	Germany / India / Brazil	44	8
Tailings and Concentrate Thickeners	China	36	8
Jameson Cell	SE Asia / Australia	35	8
Apron Feeders	Türkiye / USA	30	8
Cyclone Clusters	USA	30	2
Vibrating Screens	South Africa	26	8
Slurry Pumps	USA	24	2
Emergency Genset	USA	24	2
Pebble Crusher	Brazil	18	8
Jaw Crusher	Brazil	16	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 29 months after contract award, with ramp-up to 90% of full production over a period of approximately three months, 100% over six 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 paste filter. The order for commencement of supply is scheduled to be placed within

the first twelve weeks of the Project after award. The duration of this critical path, from contract award to completion of the installation, is approximately 91 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 29 months (contract award to demobilization).

24.10.3.1 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

24.10.4.1 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.

24.10.4.2 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 180 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.

25.2 Geology and Mineral Resource

It is the opinion of the QP for Geology and Mineral Resources, Berkley Tracy, P.Geo (PGO# 3024) that geological interpretations, drilling, sampling, analytical, modelling, and estimation procedures and methods represent good industry practice and align with CIM guidelines (CIM, 2014) and NI 43-101 disclosure standards. The QA/QC procedures and results have been reviewed and considered adequate. The resulting validated assay data were deemed appropriate for use in determining Mineral Resources. Block model estimation and validation is reasonable and appropriate given the style of mineralization and nature of the Johnny Lee and Lowry deposits.

Mineral Resources with an effective date of 5 November 2025 at the Johnny Lee deposit consist of two zones: the JL UCZ and the JL LCZ. Combined Measured and Indicated Mineral Resources at the JL UCZ and JL LCZ total: 18.9 Mt averaging 2.4% Cu totalling 459.8 kt of total Cu metal using an NSR cut-off value of US$45/t. There is an additional 3.4 Mt averaging 1.9% Cu totalling 64.5 kt of Cu metal of combined Inferred Mineral Resources at Johnny Lee at the same NSR cut-off value.

It is recommended that additional drilling and analytical testing at closer spacing within the Johnny Lee UCZ and JL LCZ will provide improved confidence in grade, recovery, and geological continuity toward increasing the confidence of Mineral Resources. To date, the 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 JL UCZ and JL LCZ areas are not recommended beyond infill drilling to convert resources to higher-confidence classification categories. The QP for Mineral Resources suggests testing the Johnny Lee deposit area for potential deeper mineralization, as a similar geological environment exists at depth as what currently hosts economic copper mineralization.

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 at the Lowry deposit, with an effective date of 5 November 2025, consist of three zones: LUCZ, LMCZ, and LLCZ. Combined Measured and Indicated Mineral Resources at the three Lowry zones total: 6.6 Mt averaging 2.4% Cu totalling 154 kt of total Cu metal using an NSR cut-off value of US$110/t. Additional Inferred Resources total: 2.8 Mt averaging 2.1% Cu totalling 58 kt of total Cu metal at the same NSR cut-off value.

The following are recommendations for additional work at the Lowry deposit:

- Additional infill and step-out drilling in both the LMCZ and LLCZ to further define mineralization.
- Hydrogeological studies including pump testing.
- Detailed lithostratigraphic and structural modelling in 3D, and
- Additional recovery and metallurgical studies to continue refining the recovery assumptions.

25.3 Mining and Mineral Reserve

25.3.1 Johnny Lee Deposit

It is the opinion of the QP responsible for the mine design, mine planning, and reserve calculations, Shane McLeay, that the level of information and work is appropriate for a PFS and represents good industry practice that aligns with NI 43-101 reporting requirements. 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 mechanized Cut and Fill, Drift and Fill and longhole stripping. A NSR value modelling approach, based on Cu grade, metallurgical recovery, As content and payabilities, 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 PFS.

All Mineral Reserves have been classified as Probable due to the level of uncertainty in the applied modifying factors. The total Mineral Reserve is estimated to be 9.5 Mt @ 2.9% Cu for 270,000 t of contained Cu. Mining production occurs over a nine year period with a peak production rate of 1.4 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, various water management facilities, a CTF, waste rock storage, paste plant, 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 PFS which identified risks in the various disciplines (i.e. geology, mining, metallurgy, environmental, etc.) that could have a negative impact on the project economics.

High and moderate risks identified 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.

ID	Risk	Rank
RI-007	Risk that resource material does not convert to mining tonnes	Moderate
RI-012	Falling copper price, leading to revenue decline	High
RI-031	Inability to consistently tight-fill with paste fill in flat-lying ore bodies	Moderate
RI-035	Shortage of skilled / experienced mining personnel and adequate housing requirements	Moderate
RI-037	Inability to achieve scheduled development rates for decline development rate	Moderate
RI-053	Unable to gain approval for permanent accommodation required for the project.	Moderate
RI-077	High / uncertain tariffs imposed - impact on equipment costs and OPEX	Moderate
RI-079	Unable to export of copper concentrate	Moderate
RI-081	Signification variation to capital expenditure	Moderate
RI-091	Global events (e.g., pandemic, geopolitical conflict)	Moderate
RI-097	Inadequate local services and peripheral infrastructure requirements	Moderate
RI-099	If no local workforce available, will need to hire from outside community	Moderate

Table 25.1 Moderate and High-Risks identified during the PFS

The only high-risk item identified in the study was a failing copper price, leading to reduced revenue. Moderate risks identified included the ability to consistently tight-fill paste, shortage of local skilled mining personnel, local housing availability and uncertainty around tariffs and the impact on capital cost of equipment.

Limited mineralized material located beneath the McGuire Parcel has been excluded from the estimation of Mineral Resources, as this land is not currently under control of SRA. The QP for Mineral Resources recognizes that unforeseen factors resulting in changes to the current assumptions on price, costs or other factors effecting the cut-off value (CoV), may materially affect the Mineral Resources. Future economic assessment could result in a change in the CoV, which would result in a change in the tonnage of available minable material. Mineralization represented by the resource block model was evaluated for RPEEE considering UG mining methods by applying SRA-provided input parameters that are considered to be reasonable based on experience with other projects.

The QP for Mineral Reserves acknowledges that sufficiently adverse movements in inputs, particularly realised commodity revenues, will negatively impact the Mineral Reserves estimate and may render the Project sub-economic. The QP considers that commodity pricing is the critical risk to the Project.

25.7 Opportunities

While preparing the Prefeasibility 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 exploration prospects exists in the district proximal to the planned Black Butte Copper Project mine facilities.
- Additional geo-metallurgical test work targeting JL UCZ to better understand copper and arsenic recovery drivers.
- Simplification of the logistic cycle for concentrate delivery to port can reduce operating cost.

25.8 Conclusions

The PFS 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 should proceed to Feasibility Study (FS) using the considerations and recommendations as outlined in the PFS.

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26. RECOMMENDATIONS

26.1 Geology and Mineral Resources

26.1.1 Johnny Lee Deposit

26.1.1.1 Geology and Mineralization

The QP for Mineral Resources recommends additional drilling and analytical testing at closer spacing within both the JL UCZ and JL 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, SRK would recommend additional drilling in the Sawmill Hill exploration 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.

26.1.1.2 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

26.1.2.1 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 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.

26.1.2.2 Drilling

The QP of Mineral Resources recommends additional drilling in the Lowry deposit to improve the spatial distribution of data by domain with emphasis on the LLCZ. 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.

Future drilling campaigns should collect additional metallurgical samples to refine the existing assumptions on Cu recovery in the Lowry deposit.

26.1.2.3 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 Additional District-wide Exploration

Several of the identified prospects within the surrounding district are controlled by SRA and 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

26.2.1.1 Johnny Lee Deposit

Entech recommends that further geotechnical data collection involving geotechnical logging of ongoing resource definition holes should be undertaken to provide a more comprehensive dataset in advance of mining. Ongoing rock properties testing should be conducted across all lithological units and mining domains to help verify the current rock properties assessed in this study and build an adequate site database. Further investigation is warranted on the major structures present around the workings to better quantify the potential risks to operations, including additional groundwater ingress. Appropriate geotechnical investigation for the proposed surface shaft locations is also recommended.

26.2.2 Mining Methods

26.2.2.1 Johnny Lee Deposit

The mining method, mine design and filling strategy should be further optimised and planned at higher granularity in forward works to identify and manage operational risks. Mine ventilation, costing and scheduling should also be conducted to a greater degree of accuracy, commensurate with the level of detail of forward works study progression.

26.3 Metallurgy and Mineral Process Design

26.3.1 Continuous Improvement

26.3.1.1 Johnny Lee Deposit

Additional testwork for the JL UCZ should be considered to better understand key drivers impacting both copper recovery and arsenic recovery to concentrate. Analysis of the drilling database assays and previous metallurgical testwork indicates variability of arsenic mineralisation in the JL UCZ impacts the arsenic in concentrate. Developing an algorithm to link grade control assays and arsenic mineralisation will enable better identification of problem areas that require blending to reduce penalties.

26.4 Project Infrastructure

Subject to ongoing Mine Operating Permit activities 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:

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.

26.4.1 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.

26.4.2 Power Supply

As the Project advances through the feasibility study, initiate studies with Fergus Electric and Northwestern Energy on transmission line upgrades required.

26.4.3 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

It is also recommended for SRA to continue its strong community engagement and information transparency as mine construction and permitting activities continue.

26.5.1 Lowry Deposit

As mentioned in Section 20, additional monitoring and test wells should be installed to refine the information from the Johnny Lee deposit studies on water levels, direction of groundwater flow and aquifer characteristics. These data, in conjunction with a mine plan, will inform water balances and mine water management plan.

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.

26.7 Recommended Work Program Costs

The total costs for the recommended work program to progress toward an FS are estimated to be approximately US$10.6 million (excluding owners costs), as summarized in Table 26.1. SRA anticipates budgeting this estimated spend over a two-year period, pending availability of financing and board approval.

Discipline	Program Description	Cost (US$ Million)	No Further Work is Recommended Reason
Property Description and Ownership	-	-	Acceptable understanding
Geology and Mineralization	Resource	0.5	
Exploration, Development and Operations	Infill Drilling	4.7	
Mineral Processing and Metallurgical	Testwork	0.5	
Technical Studies	Update Disclosure	2.7	
Mineral Reserve Estimate	Mine Engineering	1.0	
Mining Methods	Geotechnical	0.2	

Discipline	Program Description	Cost (US$ Million)	No Further Work is Recommended Reason
Recovery Methods	Testwork	0.3	
Project Infrastructure	Baseline studies	0.4	
Environmental Studies and Permitting	Baseline studies	0.3	
Total		**10.6**	

Source: SRA, 2026

Table 26.1 Summary of Costs for Recommended Work

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28. LIST OF ABBREVIATIONS

28.1 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

Abbreviation	Unit or Term
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
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
kg/m^3	Kilograms per cubic metre

Abbreviation	Unit or Term
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 Sulphur Zone
LMCZ	Lowry Middle Copper Zone
LMSZ	Lowry Middle Sulphur 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
m/sec	metres per second
Ma	Mega annum

Abbreviation	Unit or Term
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
NCWR	Non-Contact Water Reservoir
NERC	North American Electrical Reliability Corporation
NGI	Norwegian Geotechnical Institute

Abbreviation	Unit or Term
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
QP	Qualified Person
RAR	Return Air Recirculation
RBF	Radial Base Function
RC	Rotary Circulation drilling

Abbreviation	Unit or Term
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
Tgd	Tertiary granodiorite
t/h or tph	tonnes per hour
t/m^2h	tonnes per square metres hour
Tn	tennantite
TOMS	Tailings Operations, Monitoring and Surveillance

Abbreviation	Unit or Term
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

28.2 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.

Term	Definition
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.
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.

Term	Definition
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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APPENDIX A PROPERTY DESCRIPTION

APPENDIX A PROPERTY DESCRIPTION

Property Description

Serial Number	Lead Serial Number	Claim Name	County	Disposition	Case Type	Last Assessment Year	Location Date	Meridian Township Range Section	
MMC220173	MMC220161	BLK 13	MEAGHER	ACTIVE	LODE	2019	03/25/2008	20 0120N 0060E 027	NE
MMC220174	MMC220161	BLK 14	MEAGHER	ACTIVE	LODE	2019	03/25/2008	20 0120N 0060E 027	NE
MMC220175	MMC220161	BLK 15	MEAGHER	ACTIVE	LODE	2019	03/25/2008	20 0120N 0060E 027	NE
MMC220176	MMC220161	BLK 16	MEAGHER	ACTIVE	LODE	2019	03/25/2008	20 0120N 0060E 027	NE
MMC220177	MMC220161	BLK 17	MEAGHER	ACTIVE	LODE	2019	03/25/2008	20 0120N 0060E 027	NE
MMC220178	MMC220161	BLK 18	MEAGHER	ACTIVE	LODE	2019	03/25/2008	20 0120N 0060E 027	NE
MMC220179	MMC220161	BLK 19	MEAGHER	ACTIVE	LODE	2019	03/25/2008	20 0120N 0060E 027	NE
MMC220180	MMC220161	BLK 20	MEAGHER	ACTIVE	LODE	2019	03/25/2008	20 0120N 0060E 027	NE
MMC220181	MMC220161	BLK 21	MEAGHER	ACTIVE	LODE	2019	03/25/2008	20 0120N 0060E 026	NW
								20 0120N 0060E 027	NE
MMC220182	MMC220161	BLK 22	MEAGHER	ACTIVE	LODE	2019	03/25/2008	20 0120N 0060E 026	NW
								20 0120N 0060E 027	NE
MMC220183	MMC220161	BLK 23	MEAGHER	ACTIVE	LODE	2019	03/25/2008	20 0120N 0060E 026	NW
MMC220184	MMC220161	BLK 24	MEAGHER	ACTIVE	LODE	2019	03/25/2008	20 0120N 0060E 026	NW
MMC220185	MMC220161	BLK 25	MEAGHER	ACTIVE	LODE	2019	03/25/2008	20 0120N 0060E 026	NW
MMC220186	MMC220161	BLK 26	MEAGHER	ACTIVE	LODE	2019	03/25/2008	20 0120N 0060E 026	NW
MMC220187	MMC220161	BLK 27	MEAGHER	ACTIVE	LODE	2019	03/25/2008	20 0120N 0060E 026	NW
MMC220188	MMC220161	BLK 28	MEAGHER	ACTIVE	LODE	2019	03/25/2008	20 0120N 0060E 026	NW
MMC220189	MMC220161	BLK 29	MEAGHER	ACTIVE	LODE	2019	03/25/2008	20 0120N 0060E 026	NE,NW
MMC220190	MMC220161	BLK 30	MEAGHER	ACTIVE	LODE	2019	03/25/2008	20 0120N 0060E 026	NE,NW
MMC220191	MMC220161	BLK 31	MEAGHER	ACTIVE	LODE	2019	03/25/2008	20 0120N 0060E 026	NE
MMC220192	MMC220161	BLK 32	MEAGHER	ACTIVE	LODE	2019	03/25/2008	20 0120N 0060E 026	NE
MMC220193	MMC220161	BLK 33	MEAGHER	ACTIVE	LODE	2019	03/25/2008	20 0120N 0060E 026	NE
MMC220194	MMC220161	BLK 34	MEAGHER	ACTIVE	LODE	2019	03/25/2008	20 0120N 0060E 026	NE
MMC220195	MMC220161	BLK 35	MEAGHER	ACTIVE	LODE	2019	03/25/2008	20 0120N 0060E 026	NE
MMC220196	MMC220161	BLK 36	MEAGHER	ACTIVE	LODE	2019	03/25/2008	20 0120N 0060E 026	NE
MMC220197	MMC220161	BLK 37	MEAGHER	ACTIVE	LODE	2019	03/25/2008	20 0120N 0060E 026	NE
MMC220198	MMC220161	BLK 38	MEAGHER	ACTIVE	LODE	2019	03/25/2008	20 0120N 0060E 026	NE
MMC220199	MMC220161	BLK 39	MEAGHER	ACTIVE	LODE	2019	03/25/2008	20 0120N 0060E 027	SE
MMC220200	MMC220161	BLK 40	MEAGHER	ACTIVE	LODE	2019	03/25/2008	20 0120N 0060E 027	SE
MMC220201	MMC220161	BLK 41	MEAGHER	ACTIVE	LODE	2019	03/25/2008	20 0120N 0060E 027	SE
MMC220202	MMC220161	BLK 42	MEAGHER	ACTIVE	LODE	2019	03/25/2008	20 0120N 0060E 027	SE
MMC220203	MMC220161	BLK 43	MEAGHER	ACTIVE	LODE	2019	03/25/2008	20 0120N 0060E 026	SE
MMC220204	MMC220161	BLK 44	MEAGHER	ACTIVE	LODE	2019	03/25/2008	20 0120N 0060E 026	SE
MMC220205	MMC220161	BLK 45	MEAGHER	ACTIVE	LODE	2019	03/25/2008	20 0120N 0060E 026	SE
MMC220206	MMC220161	BLK 46	MEAGHER	ACTIVE	LODE	2019	03/25/2008	20 0120N 0060E 026	SE
MMC220207	MMC220161	BLK 47	MEAGHER	ACTIVE	LODE	2019	03/25/2008	20 0120N 0060E 026	SE
MMC220208	MMC220161	BLK 48	MEAGHER	ACTIVE	LODE	2019	03/25/2008	20 0120N 0060E 026	SE

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MMC220209	MMC220161	BLK 49	MEAGHER	ACTIVE	LODE	2019	03/25/2008	20 0120N 0060E 026	SE
MMC220210	MMC220161	BLK 50	MEAGHER	ACTIVE	LODE	2019	03/25/2008	20 0120N 0060E 026	SE
MMC220211	MMC220161	BLK 51	MEAGHER	ACTIVE	LODE	2019	03/25/2008	20 0120N 0060E 026	SE
MMC220212	MMC220161	BLK 52	MEAGHER	ACTIVE	LODE	2019	03/25/2008	20 0120N 0060E 026	SE
MMC223234	MMC223234	SB 1	MEAGHER	ACTIVE	LODE	2019	11/05/2010	20 0120N 0060E 032	NW
MMC223235	MMC223234	SB 2	MEAGHER	ACTIVE	LODE	2019	11/05/2010	20 0120N 0060E 032	NW
MMC223236	MMC223234	SB 3	MEAGHER	ACTIVE	LODE	2019	11/05/2010	20 0120N 0060E 032	NW
MMC223237	MMC223234	SB 4	MEAGHER	ACTIVE	LODE	2019	11/05/2010	20 0120N 0060E 032	NW
MMC223238	MMC223234	SB 5	MEAGHER	ACTIVE	LODE	2019	11/05/2010	20 0120N 0060E 032	NE
MMC223239	MMC223234	SB 6	MEAGHER	ACTIVE	LODE	2019	11/05/2010	20 0120N 0060E 032	NE
MMC223240	MMC223234	SB 7	MEAGHER	ACTIVE	LODE	2019	11/05/2010	20 0120N 0060E 032	NE
MMC223241	MMC223234	SB 8	MEAGHER	ACTIVE	LODE	2019	11/05/2010	20 0120N 0060E 032	NE
MMC223242	MMC223234	SB 9	MEAGHER	ACTIVE	LODE	2019	11/05/2010	20 0120N 0060E 032	NW
MMC223243	MMC223234	SB 10	MEAGHER	ACTIVE	LODE	2019	11/05/2010	20 0120N 0060E 032	NW
MMC223244	MMC223234	SB 11	MEAGHER	ACTIVE	LODE	2019	11/05/2010	20 0120N 0060E 032	NW
MMC223245	MMC223234	SB 12	MEAGHER	ACTIVE	LODE	2019	11/05/2010	20 0120N 0060E 032	NW
MMC223246	MMC223234	SB 13	MEAGHER	ACTIVE	LODE	2019	11/05/2010	20 0120N 0060E 032	NE
MMC223247	MMC223234	SB 14	MEAGHER	ACTIVE	LODE	2019	11/05/2010	20 0120N 0060E 032	NE
MMC223248	MMC223234	SB 15	MEAGHER	ACTIVE	LODE	2019	11/05/2010	20 0120N 0060E 032	NE
MMC223249	MMC223234	SB 16	MEAGHER	ACTIVE	LODE	2019	11/05/2010	20 0120N 0060E 032	NE
MMC223250	MMC223234	SB 17	MEAGHER	ACTIVE	LODE	2019	11/05/2010	20 0120N 0060E 032	NW,SW
MMC223251	MMC223234	SB 18	MEAGHER	ACTIVE	LODE	2019	11/05/2010	20 0120N 0060E 032	SW
MMC223252	MMC223234	SB 19	MEAGHER	ACTIVE	LODE	2019	11/05/2010	20 0120N 0060E 032	NW,SW
MMC223253	MMC223234	SB 20	MEAGHER	ACTIVE	LODE	2019	11/05/2010	20 0120N 0060E 032	SW
MMC223254	MMC223234	SB 21	MEAGHER	ACTIVE	LODE	2019	11/05/2010	20 0120N 0060E 032	NE,SE
MMC223255	MMC223234	SB 22	MEAGHER	ACTIVE	LODE	2019	11/05/2010	20 0120N 0060E 032	SE
MMC223256	MMC223234	SB 23	MEAGHER	ACTIVE	LODE	2019	11/05/2010	20 0120N 0060E 032	NE,SE
MMC223257	MMC223234	SB 24	MEAGHER	ACTIVE	LODE	2019	11/05/2010	20 0120N 0060E 032	SE
MMC223258	MMC223234	SB 25	MEAGHER	ACTIVE	LODE	2019	11/05/2010	20 0120N 0060E 032	SW
MMC223259	MMC223234	SB 26	MEAGHER	ACTIVE	LODE	2019	11/05/2010	20 0120N 0060E 032	SW
MMC223260	MMC223234	SB 27	MEAGHER	ACTIVE	LODE	2019	11/05/2010	20 0120N 0060E 032	SW
MMC223261	MMC223234	SB 28	MEAGHER	ACTIVE	LODE	2019	11/05/2010	20 0120N 0060E 032	SW
MMC223262	MMC223234	SB 29	MEAGHER	ACTIVE	LODE	2019	11/05/2010	20 0120N 0060E 032	SE
MMC223263	MMC223234	SB 30	MEAGHER	ACTIVE	LODE	2019	11/05/2010	20 0120N 0060E 032	SE
MMC223264	MMC223234	SB 31	MEAGHER	ACTIVE	LODE	2019	11/05/2010	20 0120N 0060E 032	SE
MMC223265	MMC223234	SB 32	MEAGHER	ACTIVE	LODE	2019	11/05/2010	20 0120N 0060E 032	SE
MMC223266	MMC223234	SB 33	MEAGHER	ACTIVE	LODE	2019	11/05/2010	20 0110N 0060E 005	NW
								20 0120N 0060E 032	SW

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MMC223267	MMC223234	SB 34	MEAGHER	ACTIVE	LODE	2019	11/05/2010	20 0110N 0060E 005	NW
								20 0120N 0060E 032	SW
MMC223268	MMC223234	SB 35	MEAGHER	ACTIVE	LODE	2019	11/05/2010	20 0110N 0060E 005	NE
								20 0120N 0060E 032	SE
MMC223269	MMC223234	SB 36	MEAGHER	ACTIVE	LODE	2019	11/05/2010	20 0110N 0060E 005	NE
								20 0120N 0060E 032	SE
MMC223270	MMC223234	SB 37	MEAGHER	ACTIVE	LODE	2019	11/04/2010	20 0120N 0060E 028	SE
MMC223271	MMC223234	SB 38	MEAGHER	ACTIVE	LODE	2019	11/04/2010	20 0120N 0060E 028	SE
MMC223272	MMC223234	SB 39	MEAGHER	ACTIVE	LODE	2019	11/04/2010	20 0120N 0060E 028	SE
MMC223273	MMC223234	SB 40	MEAGHER	ACTIVE	LODE	2019	11/04/2010	20 0120N 0060E 028	SE
MMC223274	MMC223234	SB 41	MEAGHER	ACTIVE	LODE	2019	11/04/2010	20 0120N 0060E 028	SE
MMC223275	MMC223234	SB 42	MEAGHER	ACTIVE	LODE	2019	11/04/2010	20 0120N 0060E 028	SE
MMC223276	MMC223234	SB 43	MEAGHER	ACTIVE	LODE	2019	11/04/2010	20 0120N 0060E 034	NW
MMC223277	MMC223234	SB 44	MEAGHER	ACTIVE	LODE	2019	11/04/2010	20 0120N 0060E 034	NW
MMC223278	MMC223234	SB 45	MEAGHER	ACTIVE	LODE	2019	11/04/2010	20 0120N 0060E 034	SW
MMC223279	MMC223234	SB 46	MEAGHER	ACTIVE	LODE	2019	11/04/2010	20 0120N 0060E 034	SW
MMC223280	MMC223234	SB 47	MEAGHER	ACTIVE	LODE	2019	11/04/2010	20 0120N 0060E 034	SW
MMC223281	MMC223234	SB 48	MEAGHER	ACTIVE	LODE	2019	11/04/2010	20 0120N 0060E 034	SW
MMC223282	MMC223234	SB 49	MEAGHER	ACTIVE	LODE	2019	11/04/2010	20 0120N 0060E 034	NW
MMC223283	MMC223234	SB 50	MEAGHER	ACTIVE	LODE	2019	11/04/2010	20 0120N 0060E 034	SW,SE
MMC223284	MMC223234	SB 51	MEAGHER	ACTIVE	LODE	2019	11/05/2010	20 0110N 0060E 003	NE
								20 0120N 0060E 034	SE
MMC223285	MMC223234	SB 52	MEAGHER	ACTIVE	LODE	2019	11/05/2010	20 0110N 0060E 003	NE
								20 0120N 0060E 034	SE
MMC223286	MMC223234	SB 53	MEAGHER	ACTIVE	LODE	2019	11/05/2010	20 0110N 0060E 003	NE
								20 0120N 0060E 034	SE
MMC223287	MMC223234	SB 54	MEAGHER	ACTIVE	LODE	2019	11/05/2010	20 0110N 0060E 003	NE
								20 0120N 0060E 034	SE
MMC223288	MMC223234	SB 55	MEAGHER	ACTIVE	LODE	2019	11/07/2010	20 0120N 0060E 034	NE,NW
MMC223289	MMC223234	SB 56	MEAGHER	ACTIVE	LODE	2019	11/07/2010	20 0120N 0060E 028	SE
MMC223580	MMC223580	BSP 1	MEAGHER	ACTIVE	LODE	2019	03/13/2011	20 0110N 0060E 004	NW
								20 0110N 0060E 005	NE
								20 0120N 0060E 032	SE
								20 0120N 0060E 033	SW
MMC223581	MMC223580	BSP 2	MEAGHER	ACTIVE	LODE	2019	03/13/2011	20 0110N 0060E 004	NW,SW
								20 0110N 0060E 005	NE,SE
MMC223582	MMC223580	BSP 3	MEAGHER	ACTIVE	LODE	2019	03/13/2011	20 0110N 0060E 004	NW
								20 0120N 0060E 033	SW

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MMC223583	MMC223580	BSP 4	MEAGHER	ACTIVE	LODE	2019	03/13/2011	20 0110N 0060E 004	NW,SW
MMC223584	MMC223580	BSP 5	MEAGHER	ACTIVE	LODE	2019	03/13/2011	20 0110N 0060E 004	NW
								20 0120N 0060E 033	SW
MMC223585	MMC223580	BSP 6	MEAGHER	ACTIVE	LODE	2019	03/13/2011	20 0110N 0060E 004	NW,SW
MMC223586	MMC223580	BSP 7	MEAGHER	ACTIVE	LODE	2019	03/13/2011	20 0110N 0060E 004	NW
MMC223587	MMC223580	BSP 8	MEAGHER	ACTIVE	LODE	2019	03/13/2011	20 0110N 0060E 004	NW,SW
MMC223588	MMC223580	BSP 9	MEAGHER	ACTIVE	LODE	2019	03/13/2011	20 0110N 0060E 004	NE,NW
MMC223589	MMC223580	BSP 10	MEAGHER	ACTIVE	LODE	2019	03/13/2011	20 0110N 0060E 004	NE,NW,SW,SE
MMC223590	MMC223580	BSP 11	MEAGHER	ACTIVE	LODE	2019	03/13/2011	20 0110N 0060E 004	NE
MMC223591	MMC223580	BSP 12	MEAGHER	ACTIVE	LODE	2019	03/13/2011	20 0110N 0060E 004	NE,SE
MMC223592	MMC223580	BSP 13	MEAGHER	ACTIVE	LODE	2019	03/13/2011	20 0110N 0060E 004	NE
MMC223593	MMC223580	BSP 14	MEAGHER	ACTIVE	LODE	2019	03/13/2011	20 0110N 0060E 004	NE,SE
MMC223594	MMC223580	BSP 15	MEAGHER	ACTIVE	LODE	2019	03/13/2011	20 0110N 0060E 004	NE
MMC223595	MMC223580	BSP 16	MEAGHER	ACTIVE	LODE	2019	03/13/2011	20 0110N 0060E 004	NE,SE
MMC223596	MMC223580	BSP 17	MEAGHER	ACTIVE	LODE	2019	03/13/2011	20 0110N 0060E 004	NE
MMC223597	MMC223580	BSP 18	MEAGHER	ACTIVE	LODE	2019	03/13/2011	20 0110N 0060E 004	NE,SE
MMC223598	MMC223580	BSP 19	MEAGHER	ACTIVE	LODE	2019	03/13/2011	20 0110N 0060E 003	NW
MMC223599	MMC223580	BSP 20	MEAGHER	ACTIVE	LODE	2019	03/13/2011	20 0110N 0060E 003	NW,SW
MMC223600	MMC223580	BSP 21	MEAGHER	ACTIVE	LODE	2019	03/14/2011	20 0110N 0060E 003	NW
MMC223601	MMC223580	BSP 22	MEAGHER	ACTIVE	LODE	2019	03/14/2011	20 0110N 0060E 003	NW,SW
MMC223602	MMC223580	BSP 23	MEAGHER	ACTIVE	LODE	2019	03/14/2011	20 0110N 0060E 003	NW
MMC223603	MMC223580	BSP 24	MEAGHER	ACTIVE	LODE	2019	03/14/2011	20 0110N 0060E 003	NW,SW
MMC223604	MMC223580	BSP 25	MEAGHER	ACTIVE	LODE	2019	03/14/2011	20 0110N 0060E 003	NW
MMC223605	MMC223580	BSP 26	MEAGHER	ACTIVE	LODE	2019	03/14/2011	20 0110N 0060E 003	NW,SW
MMC223606	MMC223580	BSP 27	MEAGHER	ACTIVE	LODE	2019	03/14/2011	20 0110N 0060E 003	NE,NW
MMC223607	MMC223580	BSP 28	MEAGHER	ACTIVE	LODE	2019	03/14/2011	20 0110N 0060E 003	NE,NW,SW,SE
MMC223608	MMC223580	BSP 29	MEAGHER	ACTIVE	LODE	2019	03/14/2011	20 0110N 0060E 003	NE
MMC223609	MMC223580	BSP 30	MEAGHER	ACTIVE	LODE	2019	03/14/2011	20 0110N 0060E 003	NE,SE
MMC223610	MMC223580	BSP 31	MEAGHER	ACTIVE	LODE	2019	03/14/2011	20 0110N 0060E 003	NE
MMC223611	MMC223580	BSP 32	MEAGHER	ACTIVE	LODE	2019	03/14/2011	20 0110N 0060E 003	NE,SE
MMC223612	MMC223580	BSP 33	MEAGHER	ACTIVE	LODE	2019	03/14/2011	20 0110N 0060E 003	NE
MMC223613	MMC223580	BSP 34	MEAGHER	ACTIVE	LODE	2019	03/14/2011	20 0110N 0060E 003	NE,SE
MMC223614	MMC223580	BSP 35	MEAGHER	ACTIVE	LODE	2019	03/14/2011	20 0110N 0060E 003	NE
MMC223615	MMC223580	BSP 36	MEAGHER	ACTIVE	LODE	2019	03/14/2011	20 0110N 0060E 003	NE,SE
MMC223616	MMC223580	BSP 37	MEAGHER	ACTIVE	LODE	2019	03/14/2011	20 0110N 0060E 003	SW
MMC223617	MMC223580	BSP 38	MEAGHER	ACTIVE	LODE	2019	03/14/2011	20 0110N 0060E 003	SW
								20 0110N 0060E 009	NE
								20 0110N 0060E 010	NW

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MMC223618	MMC223580	BSP 39	MEAGHER	ACTIVE	LODE	2019	03/14/2011	20 0110N 0060E 003	SW
MMC223619	MMC223580	BSP 40	MEAGHER	ACTIVE	LODE	2019	03/14/2011	20 0110N 0060E 003	SW
								20 0110N 0060E 010	NW
MMC223620	MMC223580	BSP 41	MEAGHER	ACTIVE	LODE	2019	03/14/2011	20 0110N 0060E 003	SW
MMC223621	MMC223580	BSP 42	MEAGHER	ACTIVE	LODE	2019	03/14/2011	20 0110N 0060E 003	SW
								20 0110N 0060E 010	NW
MMC223622	MMC223580	BSP 43	MEAGHER	ACTIVE	LODE	2019	03/14/2011	20 0110N 0060E 003	SW
MMC223623	MMC223580	BSP 44	MEAGHER	ACTIVE	LODE	2019	03/14/2011	20 0110N 0060E 003	SW
								20 0110N 0060E 010	NW
MMC223624	MMC223580	BSP 45	MEAGHER	ACTIVE	LODE	2019	03/14/2011	20 0110N 0060E 003	SW,SE
MMC223625	MMC223580	BSP 46	MEAGHER	ACTIVE	LODE	2019	03/14/2011	20 0110N 0060E 003	SW,SE
								20 0110N 0060E 010	NE,NW
MMC223626	MMC223580	BSP 47	MEAGHER	ACTIVE	LODE	2019	03/14/2011	20 0110N 0060E 003	SE
MMC223627	MMC223580	BSP 48	MEAGHER	ACTIVE	LODE	2019	03/14/2011	20 0110N 0060E 003	SE
								20 0110N 0060E 010	NE
MMC223628	MMC223580	BSP 49	MEAGHER	ACTIVE	LODE	2019	03/14/2011	20 0110N 0060E 003	SE
MMC223629	MMC223580	BSP 50	MEAGHER	ACTIVE	LODE	2019	03/14/2011	20 0110N 0060E 003	SE
								20 0110N 0060E 010	NE
MMC223630	MMC223580	BSP 51	MEAGHER	ACTIVE	LODE	2019	03/14/2011	20 0110N 0060E 003	SE
MMC223631	MMC223580	BSP 52	MEAGHER	ACTIVE	LODE	2019	03/14/2011	20 0110N 0060E 003	SE
								20 0110N 0060E 010	NE
MMC223632	MMC223580	BSP 53	MEAGHER	ACTIVE	LODE	2019	03/14/2011	20 0110N 0060E 003	SE
MMC223633	MMC223580	BSP 54	MEAGHER	ACTIVE	LODE	2019	03/14/2011	20 0110N 0060E 003	SE
								20 0110N 0060E 010	NE
MMC223634	MMC223580	BSP 55	MEAGHER	ACTIVE	LODE	2019	03/13/2011	20 0110N 0060E 002	SW
								20 0110N 0060E 003	SE
MMC223635	MMC223580	BSP 56	MEAGHER	ACTIVE	LODE	2019	03/13/2011	20 0110N 0060E 002	SW
MMC223636	MMC223580	BSP 57	MEAGHER	ACTIVE	LODE	2019	03/13/2011	20 0110N 0060E 002	SW
MMC223637	MMC223580	BSP 58	MEAGHER	ACTIVE	LODE	2019	03/13/2011	20 0110N 0060E 002	SW
MMC223638	MMC223580	BSP 59	MEAGHER	ACTIVE	LODE	2019	03/13/2011	20 0110N 0060E 002	SW,SE
MMC223639	MMC223580	BSP 60	MEAGHER	ACTIVE	LODE	2019	03/13/2011	20 0110N 0060E 002	SE
MMC223640	MMC223580	BSP 61	MEAGHER	ACTIVE	LODE	2019	03/13/2011	20 0110N 0060E 002	SE
MMC223641	MMC223580	BSP 62	MEAGHER	ACTIVE	LODE	2019	03/13/2011	20 0110N 0060E 002	SE
MMC223642	MMC223580	BSP 63	MEAGHER	ACTIVE	LODE	2019	03/13/2011	20 0110N 0060E 002	SE
MMC223643	MMC223580	BSP 64	MEAGHER	ACTIVE	LODE	2019	03/11/2011	20 0110N 0060E 010	NE,SE
								20 0110N 0060E 011	NW,SW
MMC223644	MMC223580	BSP 65	MEAGHER	ACTIVE	LODE	2019	03/11/2011	20 0110N 0060E 002	SW
								20 0110N 0060E 003	SE

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								20 0110N 0060E 010	NE
								20 0110N 0060E 011	NW
MMC223645	MMC223580	BSP 66	MEAGHER	ACTIVE	LODE	2019	03/11/2011	20 0110N 0060E 011	NW,SW
MMC223646	MMC223580	BSP 67	MEAGHER	ACTIVE	LODE	2019	03/11/2011	20 0110N 0060E 002	SW
								20 0110N 0060E 011	NW
MMC223647	MMC223580	BSP 68	MEAGHER	ACTIVE	LODE	2019	03/11/2011	20 0110N 0060E 011	NW,SW
MMC223648	MMC223580	BSP 69	MEAGHER	ACTIVE	LODE	2019	03/11/2011	20 0110N 0060E 002	SW
								20 0110N 0060E 011	NW
MMC223649	MMC223580	BSP 70	MEAGHER	ACTIVE	LODE	2019	03/11/2011	20 0110N 0060E 011	NW,SW
MMC223650	MMC223580	BSP 71	MEAGHER	ACTIVE	LODE	2019	03/11/2011	20 0110N 0060E 002	SW
								20 0110N 0060E 011	NW
MMC223651	MMC223580	BSP 72	MEAGHER	ACTIVE	LODE	2019	03/11/2011	20 0110N 0060E 011	NE,NW,SW,SE
MMC223652	MMC223580	BSP 73	MEAGHER	ACTIVE	LODE	2019	03/11/2011	20 0110N 0060E 002	SW,SE
								20 0110N 0060E 011	NE,NW
MMC223653	MMC223580	BSP 74	MEAGHER	ACTIVE	LODE	2019	03/11/2011	20 0110N 0060E 011	NE,SE
MMC223654	MMC223580	BSP 75	MEAGHER	ACTIVE	LODE	2019	03/11/2011	20 0110N 0060E 002	SE
								20 0110N 0060E 011	NE
MMC223655	MMC223580	BSP 76	MEAGHER	ACTIVE	LODE	2019	03/11/2011	20 0110N 0060E 011	NE,SE
MMC223656	MMC223580	BSP 77	MEAGHER	ACTIVE	LODE	2019	03/11/2011	20 0110N 0060E 002	SE
								20 0110N 0060E 011	NE
MMC223657	MMC223580	BSP 78	MEAGHER	ACTIVE	LODE	2019	03/11/2011	20 0110N 0060E 011	NE,SE
MMC223658	MMC223580	BSP 79	MEAGHER	ACTIVE	LODE	2019	03/11/2011	20 0110N 0060E 002	SE
								20 0110N 0060E 011	NE
MMC223659	MMC223580	BSP 80	MEAGHER	ACTIVE	LODE	2019	03/11/2011	20 0110N 0060E 011	NE,SE
MMC223660	MMC223580	BSP 81	MEAGHER	ACTIVE	LODE	2019	03/11/2011	20 0110N 0060E 002	SE
								20 0110N 0060E 011	NE
MMC223661	MMC223580	BSP 82	MEAGHER	ACTIVE	LODE	2019	03/11/2011	20 0110N 0060E 011	NE,SE
								20 0110N 0060E 012	NW,SW
MMC223662	MMC223580	BSP 83	MEAGHER	ACTIVE	LODE	2019	03/11/2011	20 0110N 0060E 001	SW
								20 0110N 0060E 002	SE
								20 0110N 0060E 011	NE
								20 0110N 0060E 012	NW
MMC223663	MMC223580	BSP 84	MEAGHER	ACTIVE	LODE	2019	03/11/2011	20 0110N 0060E 012	NW,SW
MMC223664	MMC223580	BSP 85	MEAGHER	ACTIVE	LODE	2019	03/11/2011	20 0110N 0060E 001	SW
								20 0110N 0060E 012	NW
MMC223665	MMC223580	BSP 86	MEAGHER	ACTIVE	LODE	2019	03/11/2011	20 0110N 0060E 012	NW,SW
MMC223666	MMC223580	BSP 87	MEAGHER	ACTIVE	LODE	2019	03/11/2011	20 0110N 0060E 001	SW
								20 0110N 0060E 012	NW

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MMC223667	MMC223580	BSP 88	MEAGHER	ACTIVE	LODE	2019	03/11/2011	20 0110N 0060E 012	NW,SW
MMC223668	MMC223580	BSP 89	MEAGHER	ACTIVE	LODE	2019	03/11/2011	20 0110N 0060E 001	SW
								20 0110N 0060E 012	NW
MMC223669	MMC223580	BSP 90	MEAGHER	ACTIVE	LODE	2019	03/11/2011	20 0110N 0060E 012	NE,NW,SW,SE
MMC223670	MMC223580	BSP 91	MEAGHER	ACTIVE	LODE	2019	03/11/2011	20 0110N 0060E 001	SW,SE
								20 0110N 0060E 012	NE,NW
MMC223671	MMC223580	BSP 92	MEAGHER	ACTIVE	LODE	2019	03/11/2011	20 0110N 0060E 012	NE,SE
MMC223672	MMC223580	BSP 93	MEAGHER	ACTIVE	LODE	2019	03/11/2011	20 0110N 0060E 001	SE
								20 0110N 0060E 012	NE
MMC223673	MMC223580	BSP 94	MEAGHER	ACTIVE	LODE	2019	03/11/2011	20 0110N 0060E 012	NE,SE
MMC223674	MMC223580	BSP 95	MEAGHER	ACTIVE	LODE	2019	03/11/2011	20 0110N 0060E 001	SE
								20 0110N 0060E 012	NE
MMC223675	MMC223580	BSP 96	MEAGHER	ACTIVE	LODE	2019	03/11/2011	20 0110N 0060E 012	NE,SE
MMC223676	MMC223580	BSP 97	MEAGHER	ACTIVE	LODE	2019	03/11/2011	20 0110N 0060E 001	SE
								20 0110N 0060E 012	NE
MMC223677	MMC223580	BSP 98	MEAGHER	ACTIVE	LODE	2019	03/11/2011	20 0110N 0060E 012	NE,SE
MMC223678	MMC223580	BSP 99	MEAGHER	ACTIVE	LODE	2019	03/11/2011	20 0110N 0060E 001	SE
								20 0110N 0060E 012	NE
MMC223679	MMC223580	BSP 100	MEAGHER	ACTIVE	LODE	2019	03/11/2011	20 0110N 0070E 007	NW,SW
MMC223680	MMC223580	BSP 101	MEAGHER	ACTIVE	LODE	2019	03/11/2011	20 0110N 0070E 006	SW
								20 0110N 0070E 007	NW
MMC223681	MMC223580	BSP 102	MEAGHER	ACTIVE	LODE	2019	03/11/2011	20 0110N 0070E 007	NW,SW
MMC223682	MMC223580	BSP 103	MEAGHER	ACTIVE	LODE	2019	03/11/2011	20 0110N 0070E 006	SW
								20 0110N 0070E 007	NW
MMC223683	MMC223580	BSP 104	MEAGHER	ACTIVE	LODE	2019	03/11/2011	20 0110N 0070E 007	NW,SW
MMC223684	MMC223580	BSP 105	MEAGHER	ACTIVE	LODE	2019	03/11/2011	20 0110N 0070E 006	SW
								20 0110N 0070E 007	NW
MMC223685	MMC223580	BSP 106	MEAGHER	ACTIVE	LODE	2019	03/11/2011	20 0110N 0070E 007	NW,SW
MMC223686	MMC223580	BSP 107	MEAGHER	ACTIVE	LODE	2019	03/11/2011	20 0110N 0070E 006	SW
								20 0110N 0070E 007	NW
MMC223687	MMC223580	BSP 108	MEAGHER	ACTIVE	LODE	2019	03/11/2011	20 0110N 0070E 007	NE,NW,SW,SE
MMC223688	MMC223580	BSP 109	MEAGHER	ACTIVE	LODE	2019	03/11/2011	20 0110N 0070E 006	SW,SE
								20 0110N 0070E 007	NE,NW
MMC223689	MMC223580	BSP 110	MEAGHER	ACTIVE	LODE	2019	03/11/2011	20 0110N 0070E 007	NE,SE
MMC223690	MMC223580	BSP 111	MEAGHER	ACTIVE	LODE	2019	03/11/2011	20 0110N 0070E 006	SE
								20 0110N 0070E 007	NE
MMC223691	MMC223580	BSP 112	MEAGHER	ACTIVE	LODE	2019	03/11/2011	20 0110N 0070E 007	NE,SE
MMC223692	MMC223580	BSP 113	MEAGHER	ACTIVE	LODE	2019	03/11/2011	20 0110N 0070E 006	SE

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								20 0110N 0070E 007	NE
MMC223693	MMC223580	BSP 114	MEAGHER	ACTIVE	LODE	2019	03/11/2011	20 0110N 0070E 007	NE,SE
MMC223694	MMC223580	BSP 115	MEAGHER	ACTIVE	LODE	2019	03/11/2011	20 0110N 0070E 006	SE
								20 0110N 0070E 007	NE
MMC223695	MMC223580	BSP 116	MEAGHER	ACTIVE	LODE	2019	03/11/2011	20 0110N 0070E 007	NE,SE
								20 0110N 0070E 008	NW,SW
MMC223696	MMC223580	BSP 117	MEAGHER	ACTIVE	LODE	2019	03/11/2011	20 0110N 0070E 005	SW
								20 0110N 0070E 006	SE
								20 0110N 0070E 007	NE
								20 0110N 0070E 008	NW
MMC223697	MMC223580	BSP 118	MEAGHER	ACTIVE	LODE	2019	03/10/2011	20 0110N 0060E 011	SE
								20 0110N 0060E 012	SW
								20 0110N 0060E 013	NW
								20 0110N 0060E 014	NE
MMC223698	MMC223580	BSP 119	MEAGHER	ACTIVE	LODE	2019	03/10/2011	20 0110N 0060E 011	SE
								20 0110N 0060E 012	SW
MMC223699	MMC223580	BSP 120	MEAGHER	ACTIVE	LODE	2019	03/10/2011	20 0110N 0060E 012	SW
								20 0110N 0060E 013	NW
MMC223700	MMC223580	BSP 121	MEAGHER	ACTIVE	LODE	2019	03/10/2011	20 0110N 0060E 012	SW
MMC223701	MMC223580	BSP 122	MEAGHER	ACTIVE	LODE	2019	03/10/2011	20 0110N 0060E 012	SW
								20 0110N 0060E 013	NW
MMC223702	MMC223580	BSP 123	MEAGHER	ACTIVE	LODE	2019	03/10/2011	20 0110N 0060E 012	SW
MMC223703	MMC223580	BSP 124	MEAGHER	ACTIVE	LODE	2019	03/10/2011	20 0110N 0060E 012	SW
								20 0110N 0060E 013	NW
MMC223704	MMC223580	BSP 125	MEAGHER	ACTIVE	LODE	2019	03/10/2011	20 0110N 0060E 012	SW
MMC223705	MMC223580	BSP 126	MEAGHER	ACTIVE	LODE	2019	03/10/2011	20 0110N 0060E 012	SW,SE
								20 0110N 0060E 013	NE,NW
MMC223706	MMC223580	BSP 127	MEAGHER	ACTIVE	LODE	2019	03/10/2011	20 0110N 0060E 012	SW,SE
MMC223707	MMC223580	BSP 128	MEAGHER	ACTIVE	LODE	2019	03/10/2011	20 0110N 0060E 012	SE
								20 0110N 0060E 013	NE
MMC223708	MMC223580	BSP 129	MEAGHER	ACTIVE	LODE	2019	03/10/2011	20 0110N 0060E 012	SE
MMC223709	MMC223580	BSP 130	MEAGHER	ACTIVE	LODE	2019	03/10/2011	20 0110N 0060E 012	SE
								20 0110N 0060E 013	NE
MMC223710	MMC223580	BSP 131	MEAGHER	ACTIVE	LODE	2019	03/10/2011	20 0110N 0060E 012	SE
MMC223711	MMC223580	BSP 132	MEAGHER	ACTIVE	LODE	2019	03/10/2011	20 0110N 0060E 012	SE
								20 0110N 0060E 013	NE
MMC223712	MMC223580	BSP 133	MEAGHER	ACTIVE	LODE	2019	03/10/2011	20 0110N 0060E 012	SE
MMC223713	MMC223580	BSP 134	MEAGHER	ACTIVE	LODE	2019	03/10/2011	20 0110N 0060E 012	SE

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								20 0110N 0060E 013	NE
MMC223714	MMC223580	BSP 135	MEAGHER	ACTIVE	LODE	2019	03/10/2011	20 0110N 0060E 012	SE
MMC223715	MMC223580	BSP 136	MEAGHER	ACTIVE	LODE	2019	03/10/2011	20 0110N 0070E 007	SW
								20 0110N 0070E 018	NW
MMC223716	MMC223580	BSP 137	MEAGHER	ACTIVE	LODE	2019	03/10/2011	20 0110N 0070E 007	SW
MMC223717	MMC223580	BSP 138	MEAGHER	ACTIVE	LODE	2019	03/10/2011	20 0110N 0070E 007	SW
								20 0110N 0070E 018	NW
MMC223718	MMC223580	BSP 139	MEAGHER	ACTIVE	LODE	2019	03/10/2011	20 0110N 0070E 007	SW
MMC223719	MMC223580	BSP 140	MEAGHER	ACTIVE	LODE	2019	03/10/2011	20 0110N 0070E 007	SW
								20 0110N 0070E 018	NW
MMC223720	MMC223580	BSP 141	MEAGHER	ACTIVE	LODE	2019	03/10/2011	20 0110N 0070E 007	SW
MMC223721	MMC223580	BSP 142	MEAGHER	ACTIVE	LODE	2019	03/10/2011	20 0110N 0070E 007	SW
								20 0110N 0070E 018	NW
MMC223722	MMC223580	BSP 143	MEAGHER	ACTIVE	LODE	2019	03/10/2011	20 0110N 0070E 007	SW
MMC223723	MMC223580	BSP 144	MEAGHER	ACTIVE	LODE	2019	03/10/2011	20 0110N 0070E 007	SW,SE
								20 0110N 0070E 018	NE,NW
MMC223724	MMC223580	BSP 145	MEAGHER	ACTIVE	LODE	2019	03/10/2011	20 0110N 0070E 007	SW,SE
MMC223725	MMC223580	BSP 146	MEAGHER	ACTIVE	LODE	2019	03/10/2011	20 0110N 0070E 007	SE
								20 0110N 0070E 018	NE
MMC223726	MMC223580	BSP 147	MEAGHER	ACTIVE	LODE	2019	03/10/2011	20 0110N 0070E 007	SE
MMC223727	MMC223580	BSP 148	MEAGHER	ACTIVE	LODE	2019	03/09/2011	20 0110N 0070E 007	SE
								20 0110N 0070E 018	NE
MMC223728	MMC223580	BSP 149	MEAGHER	ACTIVE	LODE	2019	03/09/2011	20 0110N 0070E 007	SE
MMC223729	MMC223580	BSP 150	MEAGHER	ACTIVE	LODE	2019	03/09/2011	20 0110N 0070E 007	SE
								20 0110N 0070E 018	NE
MMC223730	MMC223580	BSP 151	MEAGHER	ACTIVE	LODE	2019	03/09/2011	20 0110N 0070E 007	SE
MMC223731	MMC223580	BSP 152	MEAGHER	ACTIVE	LODE	2019	03/09/2011	20 0110N 0070E 007	SE
								20 0110N 0070E 008	SW
								20 0110N 0070E 017	NW
								20 0110N 0070E 018	NE
MMC223732	MMC223580	BSP 153	MEAGHER	ACTIVE	LODE	2019	03/09/2011	20 0110N 0070E 007	SE
								20 0110N 0070E 008	SW
MMC223733	MMC223580	BSP 154	MEAGHER	ACTIVE	LODE	2019	03/12/2011	20 0110N 0070E 006	SW
MMC223734	MMC223580	BSP 155	MEAGHER	ACTIVE	LODE	2019	03/12/2011	20 0110N 0070E 006	SW
MMC223735	MMC223580	BSP 156	MEAGHER	ACTIVE	LODE	2019	03/12/2011	20 0110N 0070E 006	SW
MMC223736	MMC223580	BSP 157	MEAGHER	ACTIVE	LODE	2019	03/12/2011	20 0110N 0070E 006	SW
MMC223737	MMC223580	BSP 158	MEAGHER	ACTIVE	LODE	2019	03/12/2011	20 0110N 0070E 006	SW
MMC223738	MMC223580	BSP 159	MEAGHER	ACTIVE	LODE	2019	03/12/2011	20 0110N 0070E 006	SW

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MMC223739	MMC223580	BSP 160	MEAGHER	ACTIVE	LODE	2019	03/12/2011	20 0110N 0070E 006	SW
MMC223740	MMC223580	BSP 161	MEAGHER	ACTIVE	LODE	2019	03/12/2011	20 0110N 0070E 006	SW
MMC223741	MMC223580	BSP 162	MEAGHER	ACTIVE	LODE	2019	03/12/2011	20 0110N 0070E 006	SW,SE
MMC223742	MMC223580	BSP 163	MEAGHER	ACTIVE	LODE	2019	03/12/2011	20 0110N 0070E 006	SW,SE
MMC223743	MMC223580	BSP 164	MEAGHER	ACTIVE	LODE	2019	03/12/2011	20 0110N 0070E 006	SE
MMC223744	MMC223580	BSP 165	MEAGHER	ACTIVE	LODE	2019	03/12/2011	20 0110N 0070E 006	SE
MMC223745	MMC223580	BSP 166	MEAGHER	ACTIVE	LODE	2019	03/12/2011	20 0110N 0070E 006	SE
MMC223746	MMC223580	BSP 167	MEAGHER	ACTIVE	LODE	2019	03/12/2011	20 0110N 0070E 006	SE
MMC223747	MMC223580	BSP 168	MEAGHER	ACTIVE	LODE	2019	03/12/2011	20 0110N 0070E 006	SE
MMC223748	MMC223580	BSP 169	MEAGHER	ACTIVE	LODE	2019	03/12/2011	20 0110N 0070E 006	SE
MMC223749	MMC223580	BSP 170	MEAGHER	ACTIVE	LODE	2019	03/12/2011	20 0110N 0070E 005	SW
								20 0110N 0070E 006	SE
MMC223750	MMC223580	BSP 171	MEAGHER	ACTIVE	LODE	2019	03/12/2011	20 0110N 0070E 006	SE
MMC223751	MMC223580	BSP 172	MEAGHER	ACTIVE	LODE	2019	03/12/2011	20 0110N 0070E 001	NE
								20 0110N 0070E 006	NW
MMC223752	MMC223580	BSP 173	MEAGHER	ACTIVE	LODE	2019	03/12/2011	20 0110N 0070E 001	NE
								20 0110N 0070E 006	NW
MMC223753	MMC223580	BSP 174	MEAGHER	ACTIVE	LODE	2019	03/12/2011	20 0110N 0070E 006	NW
MMC223754	MMC223580	BSP 175	MEAGHER	ACTIVE	LODE	2019	03/12/2011	20 0110N 0070E 006	NW
MMC223755	MMC223580	BSP 176	MEAGHER	ACTIVE	LODE	2019	03/12/2011	20 0110N 0070E 006	NW
MMC223756	MMC223580	BSP 177	MEAGHER	ACTIVE	LODE	2019	03/12/2011	20 0110N 0070E 006	NW
MMC223757	MMC223580	BSP 178	MEAGHER	ACTIVE	LODE	2019	03/12/2011	20 0110N 0070E 006	NW
MMC223758	MMC223580	BSP 179	MEAGHER	ACTIVE	LODE	2019	03/12/2011	20 0110N 0070E 006	NW
MMC223759	MMC223580	BSP 180	MEAGHER	ACTIVE	LODE	2019	03/12/2011	20 0110N 0070E 006	NE,NW
MMC223760	MMC223580	BSP 181	MEAGHER	ACTIVE	LODE	2019	03/12/2011	20 0110N 0070E 006	NE
MMC223761	MMC223580	BSP 182	MEAGHER	ACTIVE	LODE	2019	03/12/2011	20 0110N 0070E 006	NE
MMC223762	MMC223580	BSP 183	MEAGHER	ACTIVE	LODE	2019	03/12/2011	20 0110N 0070E 006	NE
MMC230573	MMC230573	TR 1	MEAGHER	ACTIVE	LODE	2019	03/16/2014	20 0120N 0060E 024	NE
MMC230574	MMC230573	TR 2	MEAGHER	ACTIVE	LODE	2019	03/16/2014	20 0120N 0060E 024	NE
MMC230575	MMC230573	TR 3	MEAGHER	ACTIVE	LODE	2019	03/16/2014	20 0120N 0060E 024	NE
MMC230576	MMC230573	TR 4	MEAGHER	ACTIVE	LODE	2019	03/16/2014	20 0120N 0060E 024	NE
MMC230577	MMC230573	TR 5	MEAGHER	ACTIVE	LODE	2019	03/16/2014	20 0120N 0060E 024	NE
MMC230578	MMC230573	TR 6	MEAGHER	ACTIVE	LODE	2019	03/16/2014	20 0120N 0060E 024	NE
MMC230579	MMC230573	TR 7	MEAGHER	ACTIVE	LODE	2019	03/15/2014	20 0120N 0060E 024	NE
MMC230580	MMC230573	TR 8	MEAGHER	ACTIVE	LODE	2019	03/15/2014	20 0120N 0060E 024	NE
MMC230581	MMC230573	TR 9	MEAGHER	ACTIVE	LODE	2019	03/15/2014	20 0120N 0060E 024	NE
MMC230582	MMC230573	TR 10	MEAGHER	ACTIVE	LODE	2019	03/15/2014	20 0120N 0060E 024	NE,SE
MMC230583	MMC230573	TR 11	MEAGHER	ACTIVE	LODE	2019	03/15/2014	20 0120N 0060E 024	SE

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MMC230584	MMC230573	TR 12	MEAGHER	ACTIVE	LODE	2019	03/15/2014	20 0120N 0060E 024	SE
MMC230946	MMC230946	AKZ 1	MEAGHER	ACTIVE	LODE	2019	10/28/2014	20 0110N 0060E 006	NE
MMC230947	MMC230946	AKZ 2	MEAGHER	ACTIVE	LODE	2019	10/28/2014	20 0110N 0060E 006	NE,SE
MMC230948	MMC230946	AKZ 3	MEAGHER	ACTIVE	LODE	2019	10/28/2014	20 0110N 0060E 006	NE
MMC230949	MMC230946	AKZ 4	MEAGHER	ACTIVE	LODE	2019	10/28/2014	20 0110N 0060E 006	NE,SE
MMC230950	MMC230946	AKZ 5	MEAGHER	ACTIVE	LODE	2019	10/28/2014	20 0110N 0060E 006	NE
MMC230951	MMC230946	AKZ 6	MEAGHER	ACTIVE	LODE	2019	10/28/2014	20 0110N 0060E 006	NE,SE
MMC230952	MMC230946	AKZ 7	MEAGHER	ACTIVE	LODE	2019	10/28/2014	20 0110N 0060E 005	NW
								20 0110N 0060E 006	NE
MMC230953	MMC230946	AKZ 8	MEAGHER	ACTIVE	LODE	2019	10/28/2014	20 0110N 0060E 005	NW
								20 0110N 0060E 006	NE,SE
MMC230954	MMC230946	AKZ 9	MEAGHER	ACTIVE	LODE	2019	10/28/2014	20 0110N 0060E 005	NW
								20 0120N 0060E 032	SW
MMC230955	MMC230946	AKZ 10	MEAGHER	ACTIVE	LODE	2019	10/28/2014	20 0110N 0060E 005	NW
								20 0110N 0060E 006	NE,SE
MMC230956	MMC230946	AKZ 11	MEAGHER	ACTIVE	LODE	2019	10/28/2014	20 0110N 0060E 005	NW
								20 0120N 0060E 032	SW
MMC230957	MMC230946	AKZ 12	MEAGHER	ACTIVE	LODE	2019	10/28/2014	20 0110N 0060E 005	NW,SW
MMC230958	MMC230946	AKZ 13	MEAGHER	ACTIVE	LODE	2019	10/28/2014	20 0110N 0060E 005	NW
								20 0120N 0060E 032	SW
MMC230959	MMC230946	AKZ 14	MEAGHER	ACTIVE	LODE	2019	10/28/2014	20 0110N 0060E 005	NW,SW
MMC230960	MMC230946	AKZ 15	MEAGHER	ACTIVE	LODE	2019	10/28/2014	20 0110N 0060E 005	NW
								20 0120N 0060E 032	SW
MMC230961	MMC230946	AKZ 16	MEAGHER	ACTIVE	LODE	2019	10/28/2014	20 0110N 0060E 005	NW,SW
MMC230962	MMC230946	AKZ 17	MEAGHER	ACTIVE	LODE	2019	10/28/2014	20 0110N 0060E 005	NE,NW
								20 0120N 0060E 032	SW,SE
MMC230963	MMC230946	AKZ 18	MEAGHER	ACTIVE	LODE	2019	10/28/2014	20 0110N 0060E 005	NE,NW,SW,SE
MMC230964	MMC230946	AKZ 19	MEAGHER	ACTIVE	LODE	2019	10/28/2014	20 0110N 0060E 005	NE
								20 0120N 0060E 032	SE
MMC230965	MMC230946	AKZ 20	MEAGHER	ACTIVE	LODE	2019	10/28/2014	20 0110N 0060E 005	NE,SE
MMC230966	MMC230946	AKZ 21	MEAGHER	ACTIVE	LODE	2019	10/28/2014	20 0110N 0060E 005	NE
								20 0120N 0060E 032	SE
MMC230967	MMC230946	AKZ 22	MEAGHER	ACTIVE	LODE	2019	10/28/2014	20 0110N 0060E 005	NE,SE
MMC230968	MMC230946	AKZ 23	MEAGHER	ACTIVE	LODE	2019	10/28/2014	20 0110N 0060E 005	NE
								20 0120N 0060E 032	SE
MMC230969	MMC230946	AKZ 24	MEAGHER	ACTIVE	LODE	2019	10/28/2014	20 0110N 0060E 005	NE,SE
MMC230970	MMC230946	AKZ 25	MEAGHER	ACTIVE	LODE	2019	10/28/2014	20 0110N 0060E 005	NE
								20 0120N 0060E 032	SE

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MMC230971	MMC230946	AKZ 26	MEAGHER	ACTIVE	LODE	2019	10/28/2014	20 0110N 0060E 005	NE,SE
MMC230972	MMC230946	AKZ 27	MEAGHER	ACTIVE	LODE	2019	10/30/2014	20 0110N 0060E 006	SE
MMC230973	MMC230946	AKZ 28	MEAGHER	ACTIVE	LODE	2019	10/30/2014	20 0110N 0060E 006	SE
								20 0110N 0060E 007	NE
MMC230974	MMC230946	AKZ 29	MEAGHER	ACTIVE	LODE	2019	10/30/2014	20 0110N 0060E 006	SE
MMC230975	MMC230946	AKZ 30	MEAGHER	ACTIVE	LODE	2019	10/30/2014	20 0110N 0060E 006	SE
								20 0110N 0060E 007	NE
MMC230976	MMC230946	AKZ 31	MEAGHER	ACTIVE	LODE	2019	10/30/2014	20 0110N 0060E 006	SE
MMC230977	MMC230946	AKZ 32	MEAGHER	ACTIVE	LODE	2019	10/30/2014	20 0110N 0060E 006	SE
								20 0110N 0060E 007	NE
MMC230978	MMC230946	AKZ 33	MEAGHER	ACTIVE	LODE	2019	10/30/2014	20 0110N 0060E 006	SE
MMC230979	MMC230946	AKZ 34	MEAGHER	ACTIVE	LODE	2019	10/30/2014	20 0110N 0060E 006	SE
								20 0110N 0060E 007	NE
MMC230980	MMC230946	AKZ 35	MEAGHER	ACTIVE	LODE	2019	10/30/2014	20 0110N 0060E 005	SW
								20 0110N 0060E 006	SE
MMC230981	MMC230946	AKZ 36	MEAGHER	ACTIVE	LODE	2019	10/30/2014	20 0110N 0060E 005	SW
								20 0110N 0060E 006	SE
								20 0110N 0060E 007	NE
								20 0110N 0060E 008	NW
MMC230982	MMC230946	AKZ 37	MEAGHER	ACTIVE	LODE	2019	10/25/2014	20 0110N 0060E 005	SW
MMC230983	MMC230946	AKZ 38	MEAGHER	ACTIVE	LODE	2019	10/25/2014	20 0110N 0060E 005	SW
								20 0110N 0060E 008	NW
MMC230984	MMC230946	AKZ 39	MEAGHER	ACTIVE	LODE	2019	10/25/2014	20 0110N 0060E 005	SW
MMC230985	MMC230946	AKZ 40	MEAGHER	ACTIVE	LODE	2019	10/25/2014	20 0110N 0060E 005	SW
								20 0110N 0060E 008	NW
MMC230986	MMC230946	AKZ 41	MEAGHER	ACTIVE	LODE	2019	10/25/2014	20 0110N 0060E 005	SW
MMC230987	MMC230946	AKZ 42	MEAGHER	ACTIVE	LODE	2019	10/25/2014	20 0110N 0060E 005	SW
								20 0110N 0060E 008	NW
MMC230988	MMC230946	AKZ 43	MEAGHER	ACTIVE	LODE	2019	10/25/2014	20 0110N 0060E 005	SW,SE
MMC230989	MMC230946	AKZ 44	MEAGHER	ACTIVE	LODE	2019	10/25/2014	20 0110N 0060E 005	SW,SE
								20 0110N 0060E 008	NE,NW
MMC230990	MMC230946	AKZ 45	MEAGHER	ACTIVE	LODE	2019	10/25/2014	20 0110N 0060E 005	SE
MMC230991	MMC230946	AKZ 46	MEAGHER	ACTIVE	LODE	2019	10/25/2014	20 0110N 0060E 005	SE
								20 0110N 0060E 008	NE
MMC230992	MMC230946	AKZ 47	MEAGHER	ACTIVE	LODE	2019	10/25/2014	20 0110N 0060E 005	SE
MMC230993	MMC230946	AKZ 48	MEAGHER	ACTIVE	LODE	2019	10/25/2014	20 0110N 0060E 005	SE
								20 0110N 0060E 008	NE
MMC230994	MMC230946	AKZ 49	MEAGHER	ACTIVE	LODE	2019	10/25/2014	20 0110N 0060E 005	SE

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MMC230995	MMC230946	AKZ 50	MEAGHER	ACTIVE	LODE	2019	10/25/2014	20 0110N 0060E 005	SE
								20 0110N 0060E 008	NE
MMC230996	MMC230946	AKZ 51	MEAGHER	ACTIVE	LODE	2019	10/25/2014	20 0110N 0060E 005	SE
MMC230997	MMC230946	AKZ 52	MEAGHER	ACTIVE	LODE	2019	10/25/2014	20 0110N 0060E 005	SE
								20 0110N 0060E 008	NE
MMC230998	MMC230946	AKZ 53	MEAGHER	ACTIVE	LODE	2019	10/25/2014	20 0110N 0060E 004	SW
								20 0110N 0060E 005	SE
MMC230999	MMC230946	AKZ 54	MEAGHER	ACTIVE	LODE	2019	10/25/2014	20 0110N 0060E 004	SW
								20 0110N 0060E 005	SE
MMC231000	MMC230946	AKZ 55	MEAGHER	ACTIVE	LODE	2019	10/25/2014	20 0110N 0060E 004	SW
MMC231001	MMC230946	AKZ 56	MEAGHER	ACTIVE	LODE	2019	10/25/2014	20 0110N 0060E 004	SW
								20 0110N 0060E 009	NW
MMC231002	MMC230946	AKZ 57	MEAGHER	ACTIVE	LODE	2019	10/25/2014	20 0110N 0060E 004	SW
MMC231003	MMC230946	AKZ 58	MEAGHER	ACTIVE	LODE	2019	10/25/2014	20 0110N 0060E 004	SW
								20 0110N 0060E 009	NW
MMC231004	MMC230946	AKZ 59	MEAGHER	ACTIVE	LODE	2019	10/25/2014	20 0110N 0060E 004	SW
MMC231005	MMC230946	AKZ 60	MEAGHER	ACTIVE	LODE	2019	10/25/2014	20 0110N 0060E 004	SW
								20 0110N 0060E 009	NW
MMC231006	MMC230946	AKZ 61	MEAGHER	ACTIVE	LODE	2019	10/25/2014	20 0110N 0060E 004	SW,SE
MMC231007	MMC230946	AKZ 62	MEAGHER	ACTIVE	LODE	2019	10/25/2014	20 0110N 0060E 004	SW,SE
								20 0110N 0060E 009	NE,NW
MMC231008	MMC230946	AKZ 63	MEAGHER	ACTIVE	LODE	2019	10/25/2014	20 0110N 0060E 004	SE
MMC231009	MMC230946	AKZ 64	MEAGHER	ACTIVE	LODE	2019	10/25/2014	20 0110N 0060E 004	SE
								20 0110N 0060E 009	NE
MMC231010	MMC230946	AKZ 65	MEAGHER	ACTIVE	LODE	2019	10/25/2014	20 0110N 0060E 004	SE
MMC231011	MMC230946	AKZ 66	MEAGHER	ACTIVE	LODE	2019	10/25/2014	20 0110N 0060E 004	SE
								20 0110N 0060E 009	NE
MMC231012	MMC230946	AKZ 67	MEAGHER	ACTIVE	LODE	2019	10/25/2014	20 0110N 0060E 004	SE
MMC231013	MMC230946	AKZ 68	MEAGHER	ACTIVE	LODE	2019	10/25/2014	20 0110N 0060E 004	SE
								20 0110N 0060E 009	NE
MMC231014	MMC230946	AKZ 69	MEAGHER	ACTIVE	LODE	2019	10/25/2014	20 0110N 0060E 003	SW
								20 0110N 0060E 004	SE
MMC231015	MMC230946	AKZ 70	MEAGHER	ACTIVE	LODE	2019	10/25/2014	20 0110N 0060E 003	SW
								20 0110N 0060E 004	SE
								20 0110N 0060E 009	NE
MMC231016	MMC230946	AKZ 71	MEAGHER	ACTIVE	LODE	2019	10/26/2014	20 0110N 0060E 007	NE
MMC231017	MMC230946	AKZ 72	MEAGHER	ACTIVE	LODE	2019	10/26/2014	20 0110N 0060E 007	NE,SE
MMC231018	MMC230946	AKZ 73	MEAGHER	ACTIVE	LODE	2019	10/26/2014	20 0110N 0060E 007	NE

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MMC231019	MMC230946	AKZ 74	MEAGHER	ACTIVE	LODE	2019	10/26/2014	20 0110N 0060E 007	NE,SE
MMC231020	MMC230946	AKZ 75	MEAGHER	ACTIVE	LODE	2019	10/26/2014	20 0110N 0060E 007	NE
MMC231021	MMC230946	AKZ 76	MEAGHER	ACTIVE	LODE	2019	10/26/2014	20 0110N 0060E 007	NE,SE
MMC231022	MMC230946	AKZ 77	MEAGHER	ACTIVE	LODE	2019	10/23/2014	20 0110N 0060E 007	NE
MMC231023	MMC230946	AKZ 78	MEAGHER	ACTIVE	LODE	2019	10/23/2014	20 0110N 0060E 007	NE,SE
MMC231024	MMC230946	AKZ 79	MEAGHER	ACTIVE	LODE	2019	10/23/2014	20 0110N 0060E 007	NE
								20 0110N 0060E 008	NW
MMC231025	MMC230946	AKZ 80	MEAGHER	ACTIVE	LODE	2019	10/23/2014	20 0110N 0060E 007	NE,SE
								20 0110N 0060E 008	NW,SW
MMC231026	MMC230946	AKZ 81	MEAGHER	ACTIVE	LODE	2019	10/23/2014	20 0110N 0060E 008	NW
MMC231027	MMC230946	AKZ 82	MEAGHER	ACTIVE	LODE	2019	10/23/2014	20 0110N 0060E 008	NW,SW
MMC231028	MMC230946	AKZ 83	MEAGHER	ACTIVE	LODE	2019	10/23/2014	20 0110N 0060E 008	NW
MMC231029	MMC230946	AKZ 84	MEAGHER	ACTIVE	LODE	2019	10/23/2014	20 0110N 0060E 008	NW,SW
MMC231030	MMC230946	AKZ 85	MEAGHER	ACTIVE	LODE	2019	10/23/2014	20 0110N 0060E 008	NW
MMC231031	MMC230946	AKZ 86	MEAGHER	ACTIVE	LODE	2019	10/23/2014	20 0110N 0060E 008	NW,SW
MMC231032	MMC230946	AKZ 87	MEAGHER	ACTIVE	LODE	2019	10/23/2014	20 0110N 0060E 008	NW
MMC231033	MMC230946	AKZ 88	MEAGHER	ACTIVE	LODE	2019	10/23/2014	20 0110N 0060E 008	NW,SW
MMC231034	MMC230946	AKZ 89	MEAGHER	ACTIVE	LODE	2019	10/23/2014	20 0110N 0060E 008	NE
MMC231035	MMC230946	AKZ 90	MEAGHER	ACTIVE	LODE	2019	10/23/2014	20 0110N 0060E 008	NE,SE
MMC231036	MMC230946	AKZ 91	MEAGHER	ACTIVE	LODE	2019	10/23/2014	20 0110N 0060E 008	NE
MMC231037	MMC230946	AKZ 92	MEAGHER	ACTIVE	LODE	2019	10/23/2014	20 0110N 0060E 008	NE,SE
MMC231038	MMC230946	AKZ 93	MEAGHER	ACTIVE	LODE	2019	10/23/2014	20 0110N 0060E 008	NE
MMC231039	MMC230946	AKZ 94	MEAGHER	ACTIVE	LODE	2019	10/23/2014	20 0110N 0060E 008	NE,SE
MMC231040	MMC230946	AKZ 95	MEAGHER	ACTIVE	LODE	2019	10/23/2014	20 0110N 0060E 008	NE
MMC231041	MMC230946	AKZ 96	MEAGHER	ACTIVE	LODE	2019	10/23/2014	20 0110N 0060E 008	NE,SE
MMC231042	MMC230946	AKZ 97	MEAGHER	ACTIVE	LODE	2019	10/23/2014	20 0110N 0060E 008	NE
								20 0110N 0060E 009	NW
MMC231043	MMC230946	AKZ 98	MEAGHER	ACTIVE	LODE	2019	10/23/2014	20 0110N 0060E 008	NE,SE
								20 0110N 0060E 009	NW,SW
MMC231044	MMC230946	AKZ 99	MEAGHER	ACTIVE	LODE	2019	10/29/2014	20 0110N 0060E 009	NW
MMC231045	MMC230946	AKZ 100	MEAGHER	ACTIVE	LODE	2019	10/29/2014	20 0110N 0060E 009	NW,SW
MMC231046	MMC230946	AKZ 101	MEAGHER	ACTIVE	LODE	2019	10/29/2014	20 0110N 0060E 009	NW
MMC231047	MMC230946	AKZ 102	MEAGHER	ACTIVE	LODE	2019	10/29/2014	20 0110N 0060E 009	NW,SW
MMC231048	MMC230946	AKZ 103	MEAGHER	ACTIVE	LODE	2019	10/29/2014	20 0110N 0060E 009	NW
MMC231049	MMC230946	AKZ 104	MEAGHER	ACTIVE	LODE	2019	10/29/2014	20 0110N 0060E 009	NW,SW
MMC231050	MMC230946	AKZ 105	MEAGHER	ACTIVE	LODE	2019	10/29/2014	20 0110N 0060E 009	NE,NW
MMC231051	MMC230946	AKZ 106	MEAGHER	ACTIVE	LODE	2019	10/29/2014	20 0110N 0060E 009	NE,NW,SW,SE
MMC231052	MMC230946	AKZ 107	MEAGHER	ACTIVE	LODE	2019	10/29/2014	20 0110N 0060E 009	NE

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MMC231053	MMC230946	AKZ 108	MEAGHER	ACTIVE	LODE	2019	10/29/2014	20 0110N 0060E 009	NE,SE
MMC231054	MMC230946	AKZ 109	MEAGHER	ACTIVE	LODE	2019	10/29/2014	20 0110N 0060E 009	NE
MMC231055	MMC230946	AKZ 110	MEAGHER	ACTIVE	LODE	2019	10/29/2014	20 0110N 0060E 009	NE,SE
MMC231056	MMC230946	AKZ 111	MEAGHER	ACTIVE	LODE	2019	10/24/2014	20 0110N 0060E 009	NE
MMC231057	MMC230946	AKZ 112	MEAGHER	ACTIVE	LODE	2019	10/24/2014	20 0110N 0060E 009	NE,SE
MMC231058	MMC230946	AKZ 113	MEAGHER	ACTIVE	LODE	2019	10/24/2014	20 0110N 0060E 009	NE
MMC231059	MMC230946	AKZ 114	MEAGHER	ACTIVE	LODE	2019	10/24/2014	20 0110N 0060E 009	NE,SE
MMC231060	MMC230946	AKZ 115	MEAGHER	ACTIVE	LODE	2019	10/24/2014	20 0110N 0060E 009	NE
								20 0110N 0060E 010	NW
MMC231061	MMC230946	AKZ 116	MEAGHER	ACTIVE	LODE	2019	10/24/2014	20 0110N 0060E 009	NE,SE
								20 0110N 0060E 010	NW,SW
MMC231062	MMC230946	AKZ 117	MEAGHER	ACTIVE	LODE	2019	10/24/2014	20 0110N 0060E 010	NW
MMC231063	MMC230946	AKZ 118	MEAGHER	ACTIVE	LODE	2019	10/24/2014	20 0110N 0060E 010	NW,SW
MMC231064	MMC230946	AKZ 119	MEAGHER	ACTIVE	LODE	2019	10/24/2014	20 0110N 0060E 010	NW
MMC231065	MMC230946	AKZ 120	MEAGHER	ACTIVE	LODE	2019	10/24/2014	20 0110N 0060E 010	NW,SW
MMC231066	MMC230946	AKZ 121	MEAGHER	ACTIVE	LODE	2019	10/24/2014	20 0110N 0060E 010	NW
MMC231067	MMC230946	AKZ 122	MEAGHER	ACTIVE	LODE	2019	10/24/2014	20 0110N 0060E 010	NW,SW
MMC231068	MMC230946	AKZ 123	MEAGHER	ACTIVE	LODE	2019	10/24/2014	20 0110N 0060E 010	NE,NW
MMC231069	MMC230946	AKZ 124	MEAGHER	ACTIVE	LODE	2019	10/24/2014	20 0110N 0060E 010	NE,NW,SW,SE
MMC231070	MMC230946	AKZ 125	MEAGHER	ACTIVE	LODE	2019	10/24/2014	20 0110N 0060E 010	NE
MMC231071	MMC230946	AKZ 126	MEAGHER	ACTIVE	LODE	2019	10/24/2014	20 0110N 0060E 010	NE,SE
MMC231072	MMC230946	AKZ 127	MEAGHER	ACTIVE	LODE	2019	10/24/2014	20 0110N 0060E 010	NE
MMC231073	MMC230946	AKZ 128	MEAGHER	ACTIVE	LODE	2019	10/24/2014	20 0110N 0060E 010	NE,SE
MMC231074	MMC230946	AKZ 129	MEAGHER	ACTIVE	LODE	2019	10/24/2014	20 0110N 0060E 010	NE
MMC231075	MMC230946	AKZ 130	MEAGHER	ACTIVE	LODE	2019	10/24/2014	20 0110N 0060E 010	NE,SE
MMC231076	MMC230946	AKZ 131	MEAGHER	ACTIVE	LODE	2019	10/24/2014	20 0110N 0060E 010	NE
MMC231077	MMC230946	AKZ 132	MEAGHER	ACTIVE	LODE	2019	10/24/2014	20 0110N 0060E 010	NE,SE
								20 0110N 0060E 011	NW,SW
MMC231078	MMC230946	AKZ 133	MEAGHER	ACTIVE	LODE	2019	10/26/2014	20 0110N 0060E 007	SE
MMC231079	MMC230946	AKZ 134	MEAGHER	ACTIVE	LODE	2019	10/26/2014	20 0110N 0060E 007	SE
MMC231080	MMC230946	AKZ 135	MEAGHER	ACTIVE	LODE	2019	10/26/2014	20 0110N 0060E 007	SE
MMC231081	MMC230946	AKZ 136	MEAGHER	ACTIVE	LODE	2019	10/26/2014	20 0110N 0060E 007	SE
MMC231082	MMC230946	AKZ 137	MEAGHER	ACTIVE	LODE	2019	10/26/2014	20 0110N 0060E 007	SE
								20 0110N 0060E 008	SW
MMC231083	MMC230946	AKZ 138	MEAGHER	ACTIVE	LODE	2019	10/26/2014	20 0110N 0060E 008	SW
MMC231084	MMC230946	AKZ 139	MEAGHER	ACTIVE	LODE	2019	10/26/2014	20 0110N 0060E 008	SW
MMC231085	MMC230946	AKZ 140	MEAGHER	ACTIVE	LODE	2019	10/26/2014	20 0110N 0060E 008	SW
MMC231086	MMC230946	AKZ 141	MEAGHER	ACTIVE	LODE	2019	10/26/2014	20 0110N 0060E 008	SW,SE

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MMC231087	MMC230946	AKZ 142	MEAGHER	ACTIVE	LODE	2019	10/26/2014	20 0110N 0060E 008	SE
MMC231088	MMC230946	AKZ 143	MEAGHER	ACTIVE	LODE	2019	10/23/2014	20 0110N 0060E 008	SE
MMC231089	MMC230946	AKZ 144	MEAGHER	ACTIVE	LODE	2019	10/23/2014	20 0110N 0060E 008	SE
MMC231090	MMC230946	AKZ 145	MEAGHER	ACTIVE	LODE	2019	10/23/2014	20 0110N 0060E 008	SE
MMC231091	MMC230946	AKZ 146	MEAGHER	ACTIVE	LODE	2019	10/23/2014	20 0110N 0060E 008	SE
								20 0110N 0060E 009	SW
MMC231092	MMC230946	AKZ 147	MEAGHER	ACTIVE	LODE	2019	10/23/2014	20 0110N 0060E 009	SW
MMC231093	MMC230946	AKZ 148	MEAGHER	ACTIVE	LODE	2019	10/23/2014	20 0110N 0060E 009	SW
MMC231094	MMC230946	AKZ 149	MEAGHER	ACTIVE	LODE	2019	10/23/2014	20 0110N 0060E 009	SW
MMC231095	MMC230946	AKZ 150	MEAGHER	ACTIVE	LODE	2019	10/23/2014	20 0110N 0060E 009	SW,SE
MMC231096	MMC230946	AKZ 151	MEAGHER	ACTIVE	LODE	2019	10/23/2014	20 0110N 0060E 009	SE
MMC231097	MMC230946	AKZ 152	MEAGHER	ACTIVE	LODE	2019	10/23/2014	20 0110N 0060E 009	SE
MMC231098	MMC230946	AKZ 153	MEAGHER	ACTIVE	LODE	2019	10/23/2014	20 0110N 0060E 009	SE
MMC231099	MMC230946	AKZ 154	MEAGHER	ACTIVE	LODE	2019	10/23/2014	20 0110N 0060E 009	SE
MMC231100	MMC230946	AKZ 155	MEAGHER	ACTIVE	LODE	2019	10/24/2014	20 0110N 0060E 009	SE
								20 0110N 0060E 010	SW
MMC231101	MMC230946	AKZ 156	MEAGHER	ACTIVE	LODE	2019	10/24/2014	20 0110N 0060E 010	SW
MMC231102	MMC230946	AKZ 157	MEAGHER	ACTIVE	LODE	2019	10/24/2014	20 0110N 0060E 010	SW
MMC231103	MMC230946	AKZ 158	MEAGHER	ACTIVE	LODE	2019	10/24/2014	20 0110N 0060E 010	SW
MMC231104	MMC230946	AKZ 159	MEAGHER	ACTIVE	LODE	2019	10/24/2014	20 0110N 0060E 010	SW,SE
MMC231105	MMC230946	AKZ 160	MEAGHER	ACTIVE	LODE	2019	10/24/2014	20 0110N 0060E 010	SE
MMC231106	MMC230946	AKZ 161	MEAGHER	ACTIVE	LODE	2019	10/24/2014	20 0110N 0060E 010	SE
MMC231107	MMC230946	AKZ 162	MEAGHER	ACTIVE	LODE	2019	10/24/2014	20 0110N 0060E 010	SE
MMC231108	MMC230946	AKZ 163	MEAGHER	ACTIVE	LODE	2019	10/24/2014	20 0110N 0060E 010	SE
MMC231109	MMC230946	AKZ 164	MEAGHER	ACTIVE	LODE	2019	10/27/2014	20 0110N 0060E 011	SW
								20 0110N 0060E 014	NW
MMC231110	MMC230946	AKZ 165	MEAGHER	ACTIVE	LODE	2019	10/27/2014	20 0110N 0060E 011	SW
MMC231111	MMC230946	AKZ 166	MEAGHER	ACTIVE	LODE	2019	10/27/2014	20 0110N 0060E 011	SW
								20 0110N 0060E 014	NW
MMC231112	MMC230946	AKZ 167	MEAGHER	ACTIVE	LODE	2019	10/27/2014	20 0110N 0060E 011	SW
MMC231113	MMC230946	AKZ 168	MEAGHER	ACTIVE	LODE	2019	10/27/2014	20 0110N 0060E 011	SW
								20 0110N 0060E 014	NW
MMC231114	MMC230946	AKZ 169	MEAGHER	ACTIVE	LODE	2019	10/27/2014	20 0110N 0060E 011	SW
MMC231115	MMC230946	AKZ 170	MEAGHER	ACTIVE	LODE	2019	10/27/2014	20 0110N 0060E 011	SW
								20 0110N 0060E 014	NW
MMC231116	MMC230946	AKZ 171	MEAGHER	ACTIVE	LODE	2019	10/27/2014	20 0110N 0060E 011	SW
MMC231117	MMC230946	AKZ 172	MEAGHER	ACTIVE	LODE	2019	10/27/2014	20 0110N 0060E 011	SW,SE
								20 0110N 0060E 014	NE,NW

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MMC231118	MMC230946	AKZ 173	MEAGHER	ACTIVE	LODE	2019	10/27/2014	20 0110N 0060E 011	SW,SE
MMC231119	MMC230946	AKZ 174	MEAGHER	ACTIVE	LODE	2019	10/27/2014	20 0110N 0060E 011	SE
								20 0110N 0060E 014	NE
MMC231120	MMC230946	AKZ 175	MEAGHER	ACTIVE	LODE	2019	10/27/2014	20 0110N 0060E 011	SE
MMC231121	MMC230946	AKZ 176	MEAGHER	ACTIVE	LODE	2019	10/27/2014	20 0110N 0060E 011	SE
								20 0110N 0060E 014	NE
MMC231122	MMC230946	AKZ 177	MEAGHER	ACTIVE	LODE	2019	10/27/2014	20 0110N 0060E 011	SE
MMC231123	MMC230946	AKZ 178	MEAGHER	ACTIVE	LODE	2019	10/27/2014	20 0110N 0060E 011	SE
								20 0110N 0060E 014	NE
MMC231124	MMC230946	AKZ 179	MEAGHER	ACTIVE	LODE	2019	10/27/2014	20 0110N 0060E 011	SE
MMC231125	MMC230946	AKZ 180	MEAGHER	ACTIVE	LODE	2019	10/27/2014	20 0110N 0060E 011	SE
								20 0110N 0060E 014	NE
MMC231126	MMC230946	AKZ 181	MEAGHER	ACTIVE	LODE	2019	10/27/2014	20 0110N 0060E 011	SE
MMC231127	MMC230946	AKZ 182	MEAGHER	ACTIVE	LODE	2019	10/31/2014	20 0110N 0060E 007	SE
MMC231128	MMC230946	AKZ 183	MEAGHER	ACTIVE	LODE	2019	10/31/2014	20 0110N 0060E 007	SE
MMC231129	MMC230946	AKZ 184	MEAGHER	ACTIVE	LODE	2019	10/31/2014	20 0110N 0060E 007	SE
								20 0110N 0060E 008	SW
MMC231130	MMC230946	AKZ 185	MEAGHER	ACTIVE	LODE	2019	10/31/2014	20 0110N 0060E 008	SW
MMC231131	MMC230946	AKZ 186	MEAGHER	ACTIVE	LODE	2019	10/31/2014	20 0110N 0060E 008	SW,SE
MMC231132	MMC230946	AKZ 187	MEAGHER	ACTIVE	LODE	2019	10/31/2014	20 0110N 0060E 008	SE
								20 0110N 0060E 009	SW
MMC231133	MMC230946	AKZ 188	MEAGHER	ACTIVE	LODE	2019	10/29/2014	20 0110N 0060E 009	SW
MMC231134	MMC230946	AKZ 189	MEAGHER	ACTIVE	LODE	2019	10/29/2014	20 0110N 0060E 009	SW,SE
MMC231135	MMC230946	AKZ 190	MEAGHER	ACTIVE	LODE	2019	10/29/2014	20 0110N 0060E 009	SE
MMC231136	MMC230946	AKZ 191	MEAGHER	ACTIVE	LODE	2019	10/29/2014	20 0110N 0060E 009	SE
								20 0110N 0060E 010	SW
MMC231137	MMC230946	AKZ 192	MEAGHER	ACTIVE	LODE	2019	10/29/2014	20 0110N 0060E 010	SW
MMC231138	MMC230946	AKZ 193	MEAGHER	ACTIVE	LODE	2019	10/29/2014	20 0110N 0060E 010	SW,SE
MMC231139	MMC230946	AKZ 194	MEAGHER	ACTIVE	LODE	2019	10/29/2014	20 0110N 0060E 010	SE
MMC231140	MMC230946	CSZ 1	MEAGHER	ACTIVE	LODE	2019	11/01/2014	20 0120N 0060E 030	NW
MMC231141	MMC230946	CSZ 2	MEAGHER	ACTIVE	LODE	2019	11/01/2014	20 0120N 0060E 030	NW,SW
MMC231142	MMC230946	CSZ 3	MEAGHER	ACTIVE	LODE	2019	11/01/2014	20 0120N 0060E 030	NW
MMC231143	MMC230946	CSZ 4	MEAGHER	ACTIVE	LODE	2019	11/01/2014	20 0120N 0060E 030	NW,SW
MMC231144	MMC230946	CSZ 5	MEAGHER	ACTIVE	LODE	2019	11/01/2014	20 0120N 0060E 030	NW
MMC231145	MMC230946	CSZ 6	MEAGHER	ACTIVE	LODE	2019	11/01/2014	20 0120N 0060E 030	NW,SW
MMC231146	MMC230946	CSZ 7	MEAGHER	ACTIVE	LODE	2019	11/01/2014	20 0120N 0060E 030	NW
MMC231147	MMC230946	CSZ 8	MEAGHER	ACTIVE	LODE	2019	11/01/2014	20 0120N 0060E 030	NW,SW
MMC231148	MMC230946	CSZ 9	MEAGHER	ACTIVE	LODE	2019	11/01/2014	20 0120N 0060E 030	NE,NW

Serial Number	Lead Serial Number	Claim Name	County	Disposition	Case Type	Last Assessment Year	Location Date	Meridian Township Range Section	
MMC231149	MMC230946	CSZ 10	MEAGHER	ACTIVE	LODE	2019	11/01/2014	20 0120N 0060E 030	NE,NW,SW,SE
MMC231150	MMC230946	CSZ 11	MEAGHER	ACTIVE	LODE	2019	11/01/2014	20 0120N 0060E 030	NE
MMC231151	MMC230946	CSZ 12	MEAGHER	ACTIVE	LODE	2019	11/01/2014	20 0120N 0060E 030	NE,SE
MMC231152	MMC230946	CSZ 13	MEAGHER	ACTIVE	LODE	2019	11/01/2014	20 0120N 0060E 030	NE
MMC231153	MMC230946	CSZ 14	MEAGHER	ACTIVE	LODE	2019	11/01/2014	20 0120N 0060E 030	NE,SE
MMC231154	MMC230946	CSZ 15	MEAGHER	ACTIVE	LODE	2019	11/01/2014	20 0120N 0060E 030	NE
MMC231155	MMC230946	CSZ 16	MEAGHER	ACTIVE	LODE	2019	11/01/2014	20 0120N 0060E 030	NE,SE
MMC231156	MMC230946	CSZ 17	MEAGHER	ACTIVE	LODE	2019	11/01/2014	20 0120N 0060E 030	NE
MMC231157	MMC230946	CSZ 18	MEAGHER	ACTIVE	LODE	2019	11/01/2014	20 0120N 0060E 030	NE,SE
MMC231158	MMC230946	CSZ 19	MEAGHER	ACTIVE	LODE	2019	11/01/2014	20 0120N 0060E 030	SW
MMC231159	MMC230946	CSZ 20	MEAGHER	ACTIVE	LODE	2019	11/01/2014	20 0120N 0060E 030	SW
MMC231160	MMC230946	CSZ 21	MEAGHER	ACTIVE	LODE	2019	11/01/2014	20 0120N 0060E 030	SW
MMC231161	MMC230946	CSZ 22	MEAGHER	ACTIVE	LODE	2019	11/01/2014	20 0120N 0060E 030	SW
MMC231162	MMC230946	CSZ 23	MEAGHER	ACTIVE	LODE	2019	11/01/2014	20 0120N 0060E 030	SW
MMC231163	MMC230946	CSZ 24	MEAGHER	ACTIVE	LODE	2019	11/01/2014	20 0120N 0060E 030	SW
MMC231164	MMC230946	CSZ 25	MEAGHER	ACTIVE	LODE	2019	11/01/2014	20 0120N 0060E 030	SW
MMC231165	MMC230946	CSZ 26	MEAGHER	ACTIVE	LODE	2019	11/01/2014	20 0120N 0060E 030	SW
MMC231166	MMC230946	CSZ 27	MEAGHER	ACTIVE	LODE	2019	11/01/2014	20 0120N 0060E 030	SW,SE
MMC231167	MMC230946	CSZ 28	MEAGHER	ACTIVE	LODE	2019	11/01/2014	20 0120N 0060E 030	SW,SE
MMC231168	MMC230946	CSZ 29	MEAGHER	ACTIVE	LODE	2019	11/01/2014	20 0120N 0060E 030	SE
MMC231169	MMC230946	CSZ 30	MEAGHER	ACTIVE	LODE	2019	11/01/2014	20 0120N 0060E 030	SE
MMC231170	MMC230946	CSZ 31	MEAGHER	ACTIVE	LODE	2019	11/01/2014	20 0120N 0060E 030	SE
MMC231171	MMC230946	CSZ 32	MEAGHER	ACTIVE	LODE	2019	11/01/2014	20 0120N 0060E 030	SE
MMC231172	MMC230946	CSZ 33	MEAGHER	ACTIVE	LODE	2019	11/01/2014	20 0120N 0060E 030	SE
MMC231173	MMC230946	CSZ 34	MEAGHER	ACTIVE	LODE	2019	11/01/2014	20 0120N 0060E 030	SE
MMC231174	MMC230946	CSZ 35	MEAGHER	ACTIVE	LODE	2019	11/01/2014	20 0120N 0060E 030	SE
MMC231175	MMC230946	CSZ 36	MEAGHER	ACTIVE	LODE	2019	11/01/2014	20 0120N 0060E 030	SE
MMC231176	MMC230946	CSZ 37	MEAGHER	ACTIVE	LODE	2019	11/01/2014	20 0120N 0060E 030	NW
MMC231177	MMC230946	CSZ 38	MEAGHER	ACTIVE	LODE	2019	11/01/2014	20 0120N 0060E 030	NW,SW
MMC231178	MMC230946	CSZ 39	MEAGHER	ACTIVE	LODE	2019	11/01/2014	20 0120N 0060E 030	SW
MMC231179	MMC230946	CSZ 40	MEAGHER	ACTIVE	LODE	2019	11/01/2014	20 0120N 0060E 030	SW

Serial Number	Lead Serial Number	Claim Name	County	Disposition	Case Type	Last Assessment Year	Location Date	Meridian	Township	Range	Section	Quarter Section
MT106311073	MT106311073	NEW 1	Meagher	ACTIVE	LODE	2025	6/2/2023	20	11N	7E	22	NW
												NE
MT106311074	MT106311073	NEW 2	Meagher	ACTIVE	LODE	2025	6/1/2023	20	11N	7E	22	NE
MT106311075	MT106311073	NEW 3	Meagher	ACTIVE	LODE	2025	6/1/2023	20	11N	7E	22	NE
												NW
MT106311076	MT106311073	NEW 4	Meagher	ACTIVE	LODE	2025	5/30/2023	20	11N	7E	23	NW
MT106311077	MT106311073	NEW 5	Meagher	ACTIVE	LODE	2025	5/30/2023	20	11N	7E	23	NW
												NE
MT106311078	MT106311073	NEW 6	Meagher	ACTIVE	LODE	2025	5/29/2023	20	11N	7E	23	NE
											24	NW
MT106311079	MT106311073	NEW 7	Meagher	ACTIVE	LODE	2025	5/29/2023	20	11N	7E	24	NW
MT106311080	MT106311073	NEW 8	Meagher	ACTIVE	LODE	2025	5/28/2023	20	11N	7E	24	NW
												NE
MT106311081	MT106311073	NEW 9	Meagher	ACTIVE	LODE	2025	5/8/2023	20	11N	7E	24	NE
MT106311082	MT106311073	NEW 10	Meagher	ACTIVE	LODE	2025	6/2/2023	20	11N	7E	22	NW
												NE
MT106311083	MT106311073	NEW 11	Meagher	ACTIVE	LODE	2025	6/1/2023	20	11N	7E	22	NE
MT106311084	MT106311073	NEW 12	Meagher	ACTIVE	LODE	2025	6/1/2023	20	11N	7E	23	NW
MT106311085	MT106311073	NEW 13	Meagher	ACTIVE	LODE	2025	5/30/2023	20	11N	7E	22	NE
											23	NW
MT106311086	MT106311073	NEW 14	Meagher	ACTIVE	LODE	2025	5/30/2023	20	11N	7E	23	NW
												NE
MT106311087	MT106311073	NEW 15	Meagher	ACTIVE	LODE	2025	5/29/2023	20	11N	7E	23	NE
											24	NW
MT106311088	MT106311073	NEW 16	Meagher	ACTIVE	LODE	2025	5/29/2023	20	11N	7E	24	NW
MT106311089	MT106311073	NEW 17	Meagher	ACTIVE	LODE	2025	5/28/2023	20	11N	7E	24	NW
												NE
MT106311090	MT106311073	NEW 18	Meagher	ACTIVE	LODE	2025	5/28/2023	20	11N	7E	24	NE
MT106311091	MT106311073	NEW 19	Meagher	ACTIVE	LODE	2025	6/2/2023	20	11N	7E	22	NW
												NE

MT106311092	MT106311073	NEW 20	Meagher	ACTIVE	LODE	2025	6/1/2023	20	11N	7E	22	NE
MT106311093	MT106311073	NEW 21	Meagher	ACTIVE	LODE	2025	6/1/2023	20	11N	7E	22	NE
											23	NW
MT106311094	MT106311073	NEW 22	Meagher	ACTIVE	LODE	2025	5/30/2023	20	11N	7E	23	NW
MT106311095	MT106311073	NEW 23	Meagher	ACTIVE	LODE	2025	5/30/2023	20	11N	7E	23	NW
												NE
MT106311096	MT106311073	NEW 24	Meagher	ACTIVE	LODE	2025	5/29/2023	20	11N	7E	23	NE
											24	NW
MT106311097	MT106311073	NEW 25	Meagher	ACTIVE	LODE	2025	5/29/2023	20	11N	7E	24	NW
MT106311098	MT106311073	NEW 26	Meagher	ACTIVE	LODE	2025	5/28/2023	20	11N	7E	24	NW
												NE
MT106311099	MT106311073	NEW 27	Meagher	ACTIVE	LODE	2025	5/28/2023	20	11N	7E	24	NE
MT106311100	MT106311073	NEW 28	Meagher	ACTIVE	LODE	2025	6/2/2023	20	11N	7E	22	NW
												NE
MT106311101	MT106311073	NEW 29	Meagher	ACTIVE	LODE	2025	6/1/2023	20	11N	7E	22	NE
MT106311102	MT106311073	NEW 30	Meagher	ACTIVE	LODE	2025	6/1/2023	20	11N	7E	22	NE
											23	NW
MT106311103	MT106311073	NEW 31	Meagher	ACTIVE	LODE	2025	5/30/2023	20	11N	7E	23	NW
MT106311104	MT106311073	NEW 32	Meagher	ACTIVE	LODE	2025	5/30/2023	20	11N	7E	23	NW
												NE
MT106311105	MT106311073	NEW 33	Meagher	ACTIVE	LODE	2025	5/29/2023	20	11N	7E	23	NE
											24	NW
MT106311106	MT106311073	NEW 34	Meagher	ACTIVE	LODE	2025	5/29/2023	20	11N	7E	24	NW
MT106311107	MT106311073	NEW 35	Meagher	ACTIVE	LODE	2025	5/28/2023	20	11N	7E	24	NE
MT106311108	MT106311073	NEW 36	Meagher	ACTIVE	LODE	2025	5/28/2023	20	11N	7E	24	NW
												NE
MT106311109	MT106311073	NEW 37	Meagher	ACTIVE	LODE	2025	6/2/2023	20	11N	7E	15	SW
												SE
											22	NW
												NE
MT106311110	MT106311073	NEW 38	Meagher	ACTIVE	LODE	2025	6/1/2023	20	11N	7E	15	SE
											22	NE
MT106311111	MT106311073	NEW 39	Meagher	ACTIVE	LODE	2025	5/30/2023	20	11N	7E	15	SE

											14	SW
											22	NE
											23	NW
MT106311112	MT106311073	NEW 40	Meagher	ACTIVE	LODE	2025	5/30/2023	20	11N	7E	14	SW
												SE
											23	NW
MT106311113	MT106311073	NEW 41	Meagher	ACTIVE	LODE	2025	6/2/2023	20	11N	7E	14	SW
												SE
											23	NW
												NE
MT106311114	MT106311073	NEW 42	Meagher	ACTIVE	LODE	2025	5/29/2023	20	11N	7E	13	SW
											14	SE
											23	NE
											24	NW
MT106311115	MT106311073	NEW 43	Meagher	ACTIVE	LODE	2025	5/29/2023	20	11N	7E	13	SW
											24	NW
MT106311116	MT106311073	NEW 44	Meagher	ACTIVE	LODE	2025	5/28/2023	20	11N	7E	13	SW
												SE
											24	NW
												NE
MT106311117	MT106311073	NEW 45	Meagher	ACTIVE	LODE	2025	5/28/2023	20	11N	7E	13	SE
											24	NE
MT106311118	MT106311073	NEW 46	Meagher	ACTIVE	LODE	2025	6/2/2023	20	11N	7E	15	SW
												SE
MT106311119	MT106311073	NEW 47	Meagher	ACTIVE	LODE	2025	6/1/2023	20	11N	7E	15	SE
MT106311120	MT106311073	NEW 48	Meagher	ACTIVE	LODE	2025	5/30/2023	20	11N	7E	15	SE
											14	SW
MT106311121	MT106311073	NEW 49	Meagher	ACTIVE	LODE	2025	5/30/2023	20	11N	7E	14	SW
												SE
MT106311122	MT106311073	NEW 50	Meagher	ACTIVE	LODE	2025	6/2/2023	20	11N	7E	14	SW
												SE
MT106311123	MT106311073	NEW 51	Meagher	ACTIVE	LODE	2025	5/29/2023	20	11N	7E	14	SE
											13	SW

MT106311124	MT106311073	NEW 52	Meagher	ACTIVE	LODE	2025	5/29/2023	20	11N	7E	13	SW
MT106311125	MT106311073	NEW 53	Meagher	ACTIVE	LODE	2025	5/28/2023	20	11N	7E	13	SW
												SE
MT106311126	MT106311073	NEW 54	Meagher	ACTIVE	LODE	2025	5/28/2023	20	11N	7E	13	SE
MT106311127	MT106311073	NEW 55	Meagher	ACTIVE	LODE	2025	6/2/2023	20	11N	7E	15	SW
												SE
MT106311128	MT106311073	NEW 56	Meagher	ACTIVE	LODE	2025	6/1/2023	20	11N	7E	15	SE
MT106311129	MT106311073	NEW 57	Meagher	ACTIVE	LODE	2025	5/30/2023	20	11N	7E	15	SE
											14	SW
MT106311130	MT106311073	NEW 58	Meagher	ACTIVE	LODE	2025	5/30/2023	20	11N	7E	14	SW
												SE
MT106311131	MT106311073	NEW 59	Meagher	ACTIVE	LODE	2025	6/2/2023	20	11N	7E	14	SW
												SE
MT106311132	MT106311073	NEW 60	Meagher	ACTIVE	LODE	2025	5/29/2023	20	11N	7E	14	SE
											13	SW
MT106311133	MT106311073	NEW 61	Meagher	ACTIVE	LODE	2025	5/29/2023	20	11N	7E	13	SW
MT106311134	MT106311073	NEW 62	Meagher	ACTIVE	LODE	2025	5/28/2023	20	11N	7E	13	SW
												SE
MT106311135	MT106311073	NEW 63	Meagher	ACTIVE	LODE	2025	5/28/2023	20	11N	7E	13	SE
MT106311136	MT106311073	NEW 64	Meagher	ACTIVE	LODE	2025	6/2/2023	20	11N	7E	15	SW
												SE
MT106311137	MT106311073	NEW 65	Meagher	ACTIVE	LODE	2025	6/1/2023	20	11N	7E	15	SE
MT106311138	MT106311073	NEW 66	Meagher	ACTIVE	LODE	2025	5/30/2023	20	11N	7E	15	SE
											14	SW
MT106311139	MT106311073	NEW 67	Meagher	ACTIVE	LODE	2025	5/30/2023	20	11N	7E	14	SW
												SE
MT106311140	MT106311073	NEW 68	Meagher	ACTIVE	LODE	2025	6/2/2023	20	11N	7E	14	SW
												SE
MT106311141	MT106311073	NEW 69	Meagher	ACTIVE	LODE	2025	5/29/2023	20	11N	7E	13	SW
											14	SE
MT106311142	MT106311073	NEW 70	Meagher	ACTIVE	LODE	2025	5/29/2023	20	11N	7E	13	SW
MT106311143	MT106311073	NEW 71	Meagher	ACTIVE	LODE	2025	5/28/2023	20	11N	7E	13	SW
												SE

MT106311144	MT106311073	NEW 72	Meagher	ACTIVE	LODE	2025	5/28/2023	20	11N	7E	13	SE
MT106311145	MT106311073	NEW 73	Meagher	ACTIVE	LODE	2025	6/2/2023	20	11N	7E	16	SE
											15	SW
MT106311146	MT106311073	NEW 74	Meagher	ACTIVE	LODE	2025	6/2/2023	20	11N	7E	15	SW
MT106311147	MT106311073	NEW 75	Meagher	ACTIVE	LODE	2025	6/2/2023	20	11N	7E	15	SW
												SE
MT106311148	MT106311073	NEW 76	Meagher	ACTIVE	LODE	2025	6/1/2023	20	11N	7E	15	SE
MT106311149	MT106311073	NEW 77	Meagher	ACTIVE	LODE	2025	5/30/2023	20	11N	7E	15	SE
											14	SW
MT106311150	MT106311073	NEW 78	Meagher	ACTIVE	LODE	2025	5/30/2023	20	11N	7E	14	SW
												SE
MT106311151	MT106311073	NEW 79	Meagher	ACTIVE	LODE	2025	6/2/2023	20	11N	7E	14	SW
												SE
MT106311152	MT106311073	NEW 80	Meagher	ACTIVE	LODE	2025	5/29/2023	20	11N	7E	13	SW
MT106311153	MT106311073	NEW 81	Meagher	ACTIVE	LODE	2025	5/29/2023	20	11N	7E	13	SW
MT106311154	MT106311073	NEW 82	Meagher	ACTIVE	LODE	2025	5/28/2023	20	11N	7E	13	SW
												SE
MT106311155	MT106311073	NEW 83	Meagher	ACTIVE	LODE	2025	5/28/2023	20	11N	7E	13	SE
MT106311156	MT106311073	NEW 84	Meagher	ACTIVE	LODE	2025	6/7/2023	20	11N	7E	15	SW
												NW
											16	NE
												SE
MT106311157	MT106311073	NEW 85	Meagher	ACTIVE	LODE	2025	6/2/2023	20	11N	7E	15	SW
												NW
MT106311158	MT106311073	NEW 86	Meagher	ACTIVE	LODE	2025	6/2/2023	20	11N	7E	15	SW
												NW
									11N	7E	15	SE
												NE
MT106311159	MT106311073	NEW 87	Meagher	ACTIVE	LODE	2025	6/1/2023	20	11N	7E	15	SE
												NE
MT106311160	MT106311073	NEW 88	Meagher	ACTIVE	LODE	2025	5/30/2023	20	11N	7E	14	SW
												NW
											15	SE

												NE
MT106311161	MT106311073	NEW 89	Meagher	ACTIVE	LODE	2025	5/30/2023	20	11N	7E	14	SW
												NW
MT106311162	MT106311073	NEW 90	Meagher	ACTIVE	LODE	2025	6/4/2023	20	11N	7E	14	SW
												NW
												SE
												NE
MT106311163	MT106311073	NEW 91	Meagher	ACTIVE	LODE	2025	5/29/2023	20	11N	7E	13	NW
											14	SE
												NE
MT106311164	MT106311073	NEW 92	Meagher	ACTIVE	LODE	2025	5/29/2023	20	11N	7E	13	NW
MT106311165	MT106311073	NEW 93	Meagher	ACTIVE	LODE	2025	5/28/2023	20	11N	7E	13	NW
												SE
												NE
MT106311166	MT106311073	NEW 94	Meagher	ACTIVE	LODE	2025	5/28/2023	20	11N	7E	13	SE
												NE
MT106311167	MT106311073	NEW 95	Meagher	ACTIVE	LODE	2025	6/7/2023	20	11N	7E	16	NE
											15	NW
MT106311168	MT106311073	NEW 96	Meagher	ACTIVE	LODE	2025	6/2/2023	20	11N	7E	15	NW
MT106311169	MT106311073	NEW 97	Meagher	ACTIVE	LODE	2025	6/2/2023	20	11N	7E	15	NW
												NE
MT106311170	MT106311073	NEW 98	Meagher	ACTIVE	LODE	2025	6/1/2023	20	11N	7E	15	NE
MT106311171	MT106311073	NEW 99	Meagher	ACTIVE	LODE	2025	5/30/2023	20	11N	7E	14	NW
											15	NE
MT106311172	MT106311073	NEW 100	Meagher	ACTIVE	LODE	2025	5/30/2023	20	11N	7E	14	NW
MT106311173	MT106311073	NEW 101	Meagher	ACTIVE	LODE	2025	6/2/2023	20	11N	7E	14	NW
												NE
MT106311174	MT106311073	NEW 102	Meagher	ACTIVE	LODE	2025	5/29/2023	20	11N	7E	13	NW
											14	NE
MT106311175	MT106311073	NEW 103	Meagher	ACTIVE	LODE	2025	5/29/2023	20	11N	7E	13	NW
MT106311176	MT106311073	NEW 104	Meagher	ACTIVE	LODE	2025	5/28/2023	20	11N	7E	13	NW
												NE
MT106311177	MT106311073	NEW 105	Meagher	ACTIVE	LODE	2025	5/28/2023	20	11N	7E	13	NE

MT106311178	MT106311073	NEW 106	Meagher	ACTIVE	LODE	2025	6/7/2023	20	11N	7E	15	NW
											16	NE
MT106311179	MT106311073	NEW 107	Meagher	ACTIVE	LODE	2025	6/2/2023	20	11N	7E	15	NW
MT106311180	MT106311073	NEW 108	Meagher	ACTIVE	LODE	2025	6/2/2023	20	11N	7E	15	NW
												NE
MT106311181	MT106311073	NEW 109	Meagher	ACTIVE	LODE	2025	6/1/2023	20	11N	7E	15	NE
MT106311182	MT106311073	NEW 110	Meagher	ACTIVE	LODE	2025	5/30/2023	20	11N	7E	14	NW
											15	NE
MT106311183	MT106311073	NEW 111	Meagher	ACTIVE	LODE	2025	5/30/2023	20	11N	7E	14	NW
MT106311184	MT106311073	NEW 112	Meagher	ACTIVE	LODE	2025	6/2/2023	20	11N	7E	14	NW
												NE
MT106311185	MT106311073	NEW 113	Meagher	ACTIVE	LODE	2025	5/29/2023	20	11N	7E	13	NW
											14	NE
MT106311186	MT106311073	NEW 114	Meagher	ACTIVE	LODE	2025	5/29/2023	20	11N	7E	13	NW
MT106311187	MT106311073	NEW 115	Meagher	ACTIVE	LODE	2025	5/28/2023	20	11N	7E	13	NW
												NE
MT106311188	MT106311073	NEW 116	Meagher	ACTIVE	LODE	2025	5/28/2023	20	11N	7E	13	NE
MT106311189	MT106311073	NEW 117	Meagher	ACTIVE	LODE	2025	5/26/2023	20	11N	7E	17	NW
											18	NE
MT106311190	MT106311073	NEW 118	Meagher	ACTIVE	LODE	2025	5/26/2023	20	11N	7E	17	NW
MT106311191	MT106311073	NEW 119	Meagher	ACTIVE	LODE	2025	5/26/2023	20	11N	7E	17	NW
												NE
MT106311192	MT106311073	NEW 120	Meagher	ACTIVE	LODE	2025	5/26/2023	20	11N	7E	17	NE
MT106311193	MT106311073	NEW 121	Meagher	ACTIVE	LODE	2025	5/26/2023	20	11N	7E	16	NW
MT106311194	MT106311073	NEW 122	Meagher	ACTIVE	LODE	2025	5/26/2023	20	11N	7E	16	NW
												NE
MT106311195	MT106311073	NEW 123	Meagher	ACTIVE	LODE	2025	6/7/2023	20	11N	7E	16	NE
												NE
MT106311196	MT106311073	NEW 124	Meagher	ACTIVE	LODE	2025	6/7/2023	20	11N	7E	15	NW
											16	NE
MT106311197	MT106311073	NEW 125	Meagher	ACTIVE	LODE	2025	6/2/2023	20	11N	7E	15	NW
MT106311198	MT106311073	NEW 126	Meagher	ACTIVE	LODE	2025	6/2/2023	20	11N	7E	15	NW
												NE

MT106311199	MT106311073	NEW 127	Meagher	ACTIVE	LODE	2025	6/1/2023	20	11N	7E	15	NE
MT106311200	MT106311073	NEW 128	Meagher	ACTIVE	LODE	2025	5/30/2023	20	11N	7E	14	NW
											15	NE
MT106311201	MT106311073	NEW 129	Meagher	ACTIVE	LODE	2025	5/30/2023	20	11N	7E	14	NW
MT106311202	MT106311073	NEW 130	Meagher	ACTIVE	LODE	2025	6/2/2023	20	11N	7E	14	NW
												NE
MT106311203	MT106311073	NEW 131	Meagher	ACTIVE	LODE	2025	5/29/2023	20	11N	7E	13	NW
											14	NE
MT106311204	MT106311073	NEW 132	Meagher	ACTIVE	LODE	2025	5/29/2023	20	11N	7E	13	NW
MT106311205	MT106311073	NEW 133	Meagher	ACTIVE	LODE	2025	5/28/2023	20	11N	7E	13	NW
												NE
MT106311206	MT106311073	NEW 134	Meagher	ACTIVE	LODE	2025	5/28/2023	20	11N	7E	13	NE
MT106311207	MT106311073	NEW 135	Meagher	ACTIVE	LODE	2025	5/26/2023	20	11N	7E	7	SE
											8	SW
											17	NW
											18	NE
MT106311208	MT106311073	NEW 136	Meagher	ACTIVE	LODE	2025	5/26/2023	20	11N	7E	8	SW
											17	NW
MT106311209	MT106311073	NEW 137	Meagher	ACTIVE	LODE	2025	5/26/2023	20	11N	7E	8	SW
												SE
											17	NW
												NE
MT106311210	MT106311073	NEW 138	Meagher	ACTIVE	LODE	2025	5/26/2023	20	11N	7E	8	SE
											17	NE
MT106311211	MT106311073	NEW 139	Meagher	ACTIVE	LODE	2025	5/26/2023	20	11N	7E	9	SW
											16	NW
MT106311212	MT106311073	NEW 140	Meagher	ACTIVE	LODE	2025	5/26/2023	20	11N	7E	9	SW
												SE
											16	NW
												NE
MT106311213	MT106311073	NEW 141	Meagher	ACTIVE	LODE	2025	6/7/2023	20	11N	7E	9	SE
											16	NE
												NE

MT106311214	MT106311073	NEW 142	Meagher	ACTIVE	LODE	2025	6/7/2023	20	11N	7E	9	SE
											10	SW
											15	NW
											16	NE
MT106311215	MT106311073	NEW 143	Meagher	ACTIVE	LODE	2025	6/2/2023	20	11N	7E	10	SW
											15	NW
MT106311216	MT106311073	NEW 144	Meagher	ACTIVE	LODE	2025	6/2/2023	20	11N	7E	10	SW
											15	NW
												NE
MT106311217	MT106311073	NEW 145	Meagher	ACTIVE	LODE	2025	6/1/2023	20	11N	7E	15	NE
MT106311218	MT106311073	NEW 146	Meagher	ACTIVE	LODE	2025	5/30/2023	20	11N	7E	11	SW
											14	NW
											15	NE
MT106311219	MT106311073	NEW 147	Meagher	ACTIVE	LODE	2025	5/30/2023	20	11N	7E	11	SW
											14	NW
MT106311220	MT106311073	NEW 148	Meagher	ACTIVE	LODE	2025	6/2/2023	20	11N	7E	11	SW
												SE
											14	NW
												NE
MT106311221	MT106311073	NEW 149	Meagher	ACTIVE	LODE	2025	5/29/2023	20	11N	7E	11	SE
											12	SW
											13	NW
											14	NE
MT106311222	MT106311073	NEW 150	Meagher	ACTIVE	LODE	2025	5/29/2023	20	11N	7E	12	SW
											13	NW
MT106311223	MT106311073	NEW 151	Meagher	ACTIVE	LODE	2025	5/28/2023	20	11N	7E	12	SW
												SE
											13	NW
												NE
MT106311224	MT106311073	NEW 152	Meagher	ACTIVE	LODE	2025	5/28/2023	20	11N	7E	12	SE
											13	NE
MT106311225	MT106311073	NEW 153	Meagher	ACTIVE	LODE	2025	5/26/2023	20	11N	7E	7	SE
											8	SW

MT106311226	MT106311073	NEW 154	Meagher	ACTIVE	LODE	2025	5/26/2023	20	11N	7E	8	SW
MT106311227	MT106311073	NEW 155	Meagher	ACTIVE	LODE	2025	5/26/2023	20	11N	7E	8	SW
												SE
MT106311228	MT106311073	NEW 156	Meagher	ACTIVE	LODE	2025	5/26/2023	20	11N	7E	8	SE
MT106311229	MT106311073	NEW 157	Meagher	ACTIVE	LODE	2025	6/7/2023	20	11N	7E	9	SE
MT106311230	MT106311073	NEW 158	Meagher	ACTIVE	LODE	2025	6/7/2023	20	11N	7E	9	SE
											10	SW
												SW
MT106311231	MT106311073	NEW 159	Meagher	ACTIVE	LODE	2025	5/3/2023	20	11N	7E	10	SW
MT106311232	MT106311073	NEW 160	Meagher	ACTIVE	LODE	2025	6/2/2023	20	11N	7E	10	SE
MT106311233	MT106311073	NEW 161	Meagher	ACTIVE	LODE	2025	6/1/2023	20	11N	7E	10	SE
MT106311234	MT106311073	NEW 162	Meagher	ACTIVE	LODE	2025	5/30/2023	20	11N	7E	10	SE
											11	SW
MT106311235	MT106311073	NEW 163	Meagher	ACTIVE	LODE	2025	5/30/2023	20	11N	7E	11	SW
MT106311236	MT106311073	NEW 164	Meagher	ACTIVE	LODE	2025	6/2/2023	20	11N	7E	11	SW
												SE
MT106311237	MT106311073	NEW 165	Meagher	ACTIVE	LODE	2025	5/29/2023	20	11N	7E	11	SE
											12	SW
MT106311238	MT106311073	NEW 166	Meagher	ACTIVE	LODE	2025	5/29/2023	20	11N	7E	12	SW
MT106311239	MT106311073	NEW 167	Meagher	ACTIVE	LODE	2025	5/28/2023	20	11N	7E	12	SW
												SE
MT106311240	MT106311073	NEW 168	Meagher	ACTIVE	LODE	2025	5/28/2023	20	11N	7E	12	SE
MT106311241	MT106311073	NEW 169	Meagher	ACTIVE	LODE	2025	5/26/2023	20	11N	7E	9	SW
MT106311242	MT106311073	NEW 170	Meagher	ACTIVE	LODE	2025	6/7/2023	20	11N	7E	9	SW
												SE
MT106311243	MT106311073	NEW 171	Meagher	ACTIVE	LODE	2025	5/26/2023	20	11N	7E	7	SE
											8	SW
MT106311244	MT106311073	NEW 172	Meagher	ACTIVE	LODE	2025	5/26/2023	20	11N	7E	8	SW
MT106311245	MT106311073	NEW 173	Meagher	ACTIVE	LODE	2025	5/26/2023	20	11N	7E	8	SE
MT106311246	MT106311073	NEW 174	Meagher	ACTIVE	LODE	2025	5/26/2023	20	11N	7E	8	SE
MT106311247	MT106311073	NEW 175	Meagher	ACTIVE	LODE	2025	6/7/2023	20	11N	7E	9	SE
MT106311248	MT106311073	NEW 176	Meagher	ACTIVE	LODE	2025	6/7/2023	20	11N	7E	9	SE
											10	SW

MT106311249	MT106311073	NEW 177	Meagher	ACTIVE	LODE	2025	6/2/2023	20	11N	7E	10	SW
MT106311250	MT106311073	NEW 178	Meagher	ACTIVE	LODE	2025	6/2/2023	20	11N	7E	10	SW
												SE
MT106311251	MT106311073	NEW 179	Meagher	ACTIVE	LODE	2025	6/1/2023	20	11N	7E	10	SE
MT106311252	MT106311073	NEW 180	Meagher	ACTIVE	LODE	2025	6/1/2023	20	11N	7E	10	SE
											11	SW
MT106311253	MT106311073	NEW 181	Meagher	ACTIVE	LODE	2025	5/30/2023	20	11N	7E	11	SW
MT106311254	MT106311073	NEW 182	Meagher	ACTIVE	LODE	2025	5/30/2023	20	11N	7E	11	SW
												SE
MT106311255	MT106311073	NEW 183	Meagher	ACTIVE	LODE	2025	5/29/2023	20	11N	7E	11	SE
											12	SW
MT106311256	MT106311073	NEW 184	Meagher	ACTIVE	LODE	2025	5/29/2023	20	11N	7E	12	SW
MT106311257	MT106311073	NEW 185	Meagher	ACTIVE	LODE	2025	5/28/2023	20	11N	7E	12	SW
												SE
MT106311258	MT106311073	NEW 186	Meagher	ACTIVE	LODE	2025	5/28/2023	20	11N	7E	12	SE
MT106311259	MT106311073	NEW 187	Meagher	ACTIVE	LODE	2025	6/7/2023	20	11N	7E	9	SW
												SE
MT106311260	MT106311073	NEW 188	Meagher	ACTIVE	LODE	2025	5/26/2023	20	11N	7E	9	SW
MT106311261	MT106311073	NEW 189	Meagher	ACTIVE	LODE	2025	6/7/2023	20	11N	7E	10	SE
MT106311262	MT106311073	NEW 190	Meagher	ACTIVE	LODE	2025	5/25/2023	20	11N	7E	7	SE
											8	SW
MT106311263	MT106311073	NEW 191	Meagher	ACTIVE	LODE	2025	5/25/2023	20	11N	7E	8	SW
MT106311264	MT106311073	NEW 192	Meagher	ACTIVE	LODE	2025	5/25/2023	20	11N	7E	8	SE
MT106311265	MT106311073	NEW 193	Meagher	ACTIVE	LODE	2025	5/25/2023	20	11N	7E	8	SE
MT106311266	MT106311073	NEW 194	Meagher	ACTIVE	LODE	2025	6/7/2023	20	11N	7E	9	SE
MT106311267	MT106311073	NEW 195	Meagher	ACTIVE	LODE	2025	6/7/2023	20	11N	7E	9	SE
											10	SW
MT106311268	MT106311073	NEW 196	Meagher	ACTIVE	LODE	2025	6/2/2023	20	11N	7E	10	SW
MT106311269	MT106311073	NEW 197	Meagher	ACTIVE	LODE	2025	6/2/2023	20	11N	7E	9	SE
											10	SW
												SE
MT106311270	MT106311073	NEW 198	Meagher	ACTIVE	LODE	2025	5/29/2023	20	11N	7E	11	SW
MT106311271	MT106311073	NEW 199	Meagher	ACTIVE	LODE	2025	5/29/2023	20	11N	7E	11	SW

												SE
MT106311272	MT106311073	NEW 200	Meagher	ACTIVE	LODE	2025	5/29/2023	20	11N	7E	11	SE
											12	SW
MT106311273	MT106311073	NEW 201	Meagher	ACTIVE	LODE	2025	5/29/2023	20	11N	7E	12	SW
MT106311274	MT106311073	NEW 202	Meagher	ACTIVE	LODE	2025	5/28/2023	20	11N	7E	12	SW
												SE
MT106311275	MT106311073	NEW 203	Meagher	ACTIVE	LODE	2025	5/28/2023	20	11N	7E	12	SE
MT106311276	MT106311073	NEW 204	Meagher	ACTIVE	LODE	2025	5/26/2023	20	11N	7E	9	SW
MT106311277	MT106311073	NEW 205	Meagher	ACTIVE	LODE	2025	5/30/2023	20	11N	7E	10	SE
											11	SW
MT106311278	MT106311073	NEW 206	Meagher	ACTIVE	LODE	2025	5/30/2023	20	11N	7E	10	SE
											11	SW
MT106311279	MT106311073	NEW 207	Meagher	ACTIVE	LODE	2025	5/25/2023	20	11N	7E	7	SE
											8	SW
MT106311280	MT106311073	NEW 208	Meagher	ACTIVE	LODE	2025	5/25/2023	20	11N	7E	8	SW
MT106311281	MT106311073	NEW 209	Meagher	ACTIVE	LODE	2025	5/25/2023	20	11N	7E	8	SE
MT106311282	MT106311073	NEW 210	Meagher	ACTIVE	LODE	2025	5/25/2023	20	11N	7E	8	SE
MT106311283	MT106311073	NEW 211	Meagher	ACTIVE	LODE	2025	6/7/2023	20	11N	7E	9	SE
												SE
MT106311284	MT106311073	NEW 212	Meagher	ACTIVE	LODE	2025	6/7/2023	20	11N	7E	9	SE
												SE
											10	SW
MT106311285	MT106311073	NEW 213	Meagher	ACTIVE	LODE	2025	6/2/2023	20	11N	7E	10	SW
MT106311286	MT106311073	NEW 214	Meagher	ACTIVE	LODE	2025	6/2/2023	20	11N	7E	10	SW
												SE
MT106311287	MT106311073	NEW 215	Meagher	ACTIVE	LODE	2025	5/29/2023	20	11N	7E	11	SW
MT106311288	MT106311073	NEW 216	Meagher	ACTIVE	LODE	2025	5/29/2023	20	11N	7E	11	SW
												SE
MT106311289	MT106311073	NEW 217	Meagher	ACTIVE	LODE	2025	5/29/2023	20	11N	7E	11	SE
											12	SW
MT106311290	MT106311073	NEW 218	Meagher	ACTIVE	LODE	2025	5/29/2023	20	11N	7E	12	SW
MT106311291	MT106311073	NEW 219	Meagher	ACTIVE	LODE	2025	5/28/2023	20	11N	7E	12	SW
												SE

MT106311292	MT106311073	NEW 220	Meagher	ACTIVE	LODE	2025	5/28/2023	20	11N	7E	12	SE
MT106311293	MT106311073	NEW 221	Meagher	ACTIVE	LODE	2025	6/7/2023	20	11N	7E	10	SE
MT106311294	MT106311073	NEW 222	Meagher	ACTIVE	LODE	2025	5/26/2023	20	11N	7E	9	SW
MT106311295	MT106311073	NEW 223	Meagher	ACTIVE	LODE	2025	5/25/2023	20	11N	7E	7	SE
												NE
											8	NW
MT106311296	MT106311073	NEW 224	Meagher	ACTIVE	LODE	2025	5/25/2023	20	11N	7E	8	NW
MT106311297	MT106311073	NEW 225	Meagher	ACTIVE	LODE	2025	5/25/2023	20	11N	7E	8	NE
MT106311298	MT106311073	NEW 226	Meagher	ACTIVE	LODE	2025	5/25/2023	20	11N	7E	8	NE
MT106311299	MT106311073	NEW 227	Meagher	ACTIVE	LODE	2025	5/26/2023	20	11N	7E	9	NW
MT106311300	MT106311073	NEW 228	Meagher	ACTIVE	LODE	2025	5/26/2023	20	11N	7E	9	NW
												NE
MT106311301	MT106311073	NEW 229	Meagher	ACTIVE	LODE	2025	6/7/2023	20	11N	7E	9	NE
MT106311302	MT106311073	NEW 230	Meagher	ACTIVE	LODE	2025	6/7/2023	20	11N	7E	9	NE
											10	SW
												NW
MT106311303	MT106311073	NEW 231	Meagher	ACTIVE	LODE	2025	6/7/2023	20	11N	7E	10	SW
												NW
MT106311304	MT106311073	NEW 232	Meagher	ACTIVE	LODE	2025	6/7/2023	20	11N	7E	10	SW
												NW
												SE
												NE
MT106311305	MT106311073	NEW 233	Meagher	ACTIVE	LODE	2025	5/30/2023	20	11N	7E	10	SE
												NE
MT106311306	MT106311073	NEW 234	Meagher	ACTIVE	LODE	2025	5/30/2023	20	11N	7E	10	SE
												NE
											11	SW
												NW
MT106311307	MT106311073	NEW 235	Meagher	ACTIVE	LODE	2025	5/29/2023	20	11N	7E	11	SW
												NW
MT106311308	MT106311073	NEW 236	Meagher	ACTIVE	LODE	2025	5/29/2023	20	11N	7E	11	SW
												NW
												SE

												NE
MT106311309	MT106311073	NEW 237	Meagher	ACTIVE	LODE	2025	5/29/2023	20	11N	7E	11	SE
												NE
											12	SW
												NW
MT106311310	MT106311073	NEW 238	Meagher	ACTIVE	LODE	2025	5/29/2023	20	11N	7E	12	SW
												NW
MT106311311	MT106311073	NEW 239	Meagher	ACTIVE	LODE	2025	5/28/2023	20	11N	7E	12	SW
												NW
									11N	7E	12	NW
												SE
												NE
MT106311312	MT106311073	NEW 240	Meagher	ACTIVE	LODE	2025	5/28/2023	20	11N	7E	12	SE
												NE
MT106311313	MT106311073	NEW 241	Meagher	ACTIVE	LODE	2025	5/23/2023	20	11N	7E	7	NE
											8	NW
MT106311314	MT106311073	NEW 242	Meagher	ACTIVE	LODE	2025	5/25/2023	20	11N	7E	8	NW
MT106311315	MT106311073	NEW 243	Meagher	ACTIVE	LODE	2025	5/25/2023	20	11N	7E	8	NE
MT106311316	MT106311073	NEW 244	Meagher	ACTIVE	LODE	2025	5/25/2023	20	11N	7E	8	NE
MT106311317	MT106311073	NEW 245	Meagher	ACTIVE	LODE	2025	5/26/2023	20	11N	7E	9	NW
MT106311318	MT106311073	NEW 246	Meagher	ACTIVE	LODE	2025	5/26/2023	20	11N	7E	9	NW
												NE
MT106311319	MT106311073	NEW 247	Meagher	ACTIVE	LODE	2025	5/31/2023	20	11N	7E	9	NE
MT106311320	MT106311073	NEW 248	Meagher	ACTIVE	LODE	2025	5/31/2023	20	11N	7E	9	NE
											10	NW
MT106311321	MT106311073	NEW 249	Meagher	ACTIVE	LODE	2025	5/31/2023	20	11N	7E	10	NW
MT106311322	MT106311073	NEW 250	Meagher	ACTIVE	LODE	2025	5/31/2023	20	11N	7E	10	NW
												NE
MT106311323	MT106311073	NEW 251	Meagher	ACTIVE	LODE	2025	5/30/2023	20	11N	7E	10	NE
MT106311324	MT106311073	NEW 252	Meagher	ACTIVE	LODE	2025	5/30/2023	20	11N	7E	10	NE
											11	NW
MT106311325	MT106311073	NEW 253	Meagher	ACTIVE	LODE	2025	5/29/2023	20	11N	7E	11	NW
MT106311326	MT106311073	NEW 254	Meagher	ACTIVE	LODE	2025	5/29/2023	20	11N	7E	11	NW

												NE
MT106311327	MT106311073	NEW 255	Meagher	ACTIVE	LODE	2025	5/29/2023	20	11N	7E	11	NE
											12	NW
MT106311328	MT106311073	NEW 256	Meagher	ACTIVE	LODE	2025	5/29/2023	20	11N	7E	12	NW
MT106311329	MT106311073	NEW 257	Meagher	ACTIVE	LODE	2025	5/25/2023	20	11N	7E	7	NE
											8	NW
MT106311330	MT106311073	NEW 258	Meagher	ACTIVE	LODE	2025	5/25/2023	20	11N	7E	8	NW
MT106311331	MT106311073	NEW 259	Meagher	ACTIVE	LODE	2025	5/25/2023	20	11N	7E	8	NE
MT106311332	MT106311073	NEW 260	Meagher	ACTIVE	LODE	2025	5/25/2023	20	11N	7E	8	NE
MT106311333	MT106311073	NEW 261	Meagher	ACTIVE	LODE	2025	5/26/2023	20	11N	7E	9	NW
MT106311334	MT106311073	NEW 262	Meagher	ACTIVE	LODE	2025	5/26/2023	20	11N	7E	9	NW
												NE
MT106311335	MT106311073	NEW 263	Meagher	ACTIVE	LODE	2025	5/31/2023	20	11N	7E	9	NE
MT106311336	MT106311073	NEW 264	Meagher	ACTIVE	LODE	2025	5/31/2023	20	11N	7E	9	NE
											10	NW
MT106311337	MT106311073	NEW 265	Meagher	ACTIVE	LODE	2025	5/31/2023	20	11N	7E	10	NW
MT106311338	MT106311073	NEW 266	Meagher	ACTIVE	LODE	2025	5/31/2023	20	11N	7E	10	NW
												NE
MT106311339	MT106311073	NEW 267	Meagher	ACTIVE	LODE	2025	5/30/2023	20	11N	7E	10	NE
MT106311340	MT106311073	NEW 268	Meagher	ACTIVE	LODE	2025	5/30/2023	20	11N	7E	10	NE
											11	NW
MT106311341	MT106311073	NEW 269	Meagher	ACTIVE	LODE	2025	5/29/2023	20	11N	7E	11	NW
MT106311342	MT106311073	NEW 270	Meagher	ACTIVE	LODE	2025	5/29/2023	20	11N	7E	11	NW
												NE
MT106311343	MT106311073	NEW 271	Meagher	ACTIVE	LODE	2025	5/29/2023	20	11N	7E	11	NE
											12	NW
MT106311344	MT106311073	NEW 272	Meagher	ACTIVE	LODE	2025	5/25/2023	20	11N	7E	7	NE
											8	NW
MT106311345	MT106311073	NEW 273	Meagher	ACTIVE	LODE	2025	5/25/2023	20	11N	7E	8	NW
MT106311346	MT106311073	NEW 274	Meagher	ACTIVE	LODE	2025	5/25/2023	20	11N	7E	8	NE
MT106311347	MT106311073	NEW 275	Meagher	ACTIVE	LODE	2025	5/25/2023	20	11N	7E	8	NE
MT106311348	MT106311073	NEW 276	Meagher	ACTIVE	LODE	2025	5/26/2023	20	11N	7E	9	NW
MT106311349	MT106311073	NEW 277	Meagher	ACTIVE	LODE	2025	5/26/2023	20	11N	7E	9	NW

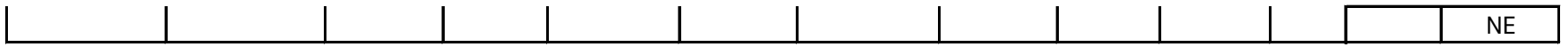
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MT106311350	MT106311073	NEW 278	Meagher	ACTIVE	LODE	2025	5/31/2023	20	11N	7E	9	NE
MT106311351	MT106311073	NEW 279	Meagher	ACTIVE	LODE	2025	5/31/2023	20	11N	7E	9	NE
											10	NW
MT106311352	MT106311073	NEW 280	Meagher	ACTIVE	LODE	2025	5/31/2023	20	11N	7E	10	NW
MT106311353	MT106311073	NEW 281	Meagher	ACTIVE	LODE	2025	5/31/2023	20	11N	7E	10	NW
												NE
MT106311354	MT106311073	NEW 282	Meagher	ACTIVE	LODE	2025	5/30/2023	20	11N	7E	10	NE
MT106311355	MT106311073	NEW 283	Meagher	ACTIVE	LODE	2025	5/30/2023	20	11N	7E	10	NE
											11	NW
MT106311356	MT106311073	NEW 284	Meagher	ACTIVE	LODE	2025	5/25/2023	20	11N	7E	5	SW
											6	SE
											7	NE
											8	NW
MT106311357	MT106311073	NEW 285	Meagher	ACTIVE	LODE	2025	5/25/2023	20	11N	7E	5	SW
											8	NW
MT106311358	MT106311073	NEW 286	Meagher	ACTIVE	LODE	2025	5/23/2023	20	11N	7E	5	SW
												SE
											8	NE
MT106311359	MT106311073	NEW 287	Meagher	ACTIVE	LODE	2025	5/25/2023	20	11N	7E	5	SE
											8	NE
MT106311360	MT106311073	NEW 288	Meagher	ACTIVE	LODE	2025	5/26/2023	20	11N	7E	4	SW
											9	NW
MT106311361	MT106311073	NEW 289	Meagher	ACTIVE	LODE	2025	5/26/2023	20	11N	7E	4	SW
												SE
											9	NW
												NE
MT106311362	MT106311073	NEW 290	Meagher	ACTIVE	LODE	2025	5/31/2023	20	11N	7E	4	SE
											9	NE
MT106311363	MT106311073	NEW 291	Meagher	ACTIVE	LODE	2025	5/31/2023	20	11N	7E	3	SW
											4	SE
											9	NE
											10	NW

MT106311364	MT106311073	NEW 292	Meagher	ACTIVE	LODE	2025	5/31/2023	20	11N	7E	3	SW
											10	NW
MT106311365	MT106311073	NEW 293	Meagher	ACTIVE	LODE	2025	5/31/2023	20	11N	7E	3	SW
												SE
											10	NW
												NE
MT106311366	MT106311073	NEW 294	Meagher	ACTIVE	LODE	2025	5/30/2023	20	11N	7E	3	SE
											10	NE
MT106311367	MT106311073	NEW 295	Meagher	ACTIVE	LODE	2025	5/30/2023	20	11N	7E	2	SW
											3	SE
											10	NE
											11	NW
MT106311368	MT106311073	NEW 296	Meagher	ACTIVE	LODE	2025	6/7/2023	20	11N	7E	4	SW
												SE
MT106311369	MT106311073	NEW 297	Meagher	ACTIVE	LODE	2025	6/7/2023	20	11N	7E	4	SW
MT106311370	MT106311073	NEW 298	Meagher	ACTIVE	LODE	2025	5/25/2023	20	11N	7E	5	SW
											6	SE
MT106311371	MT106311073	NEW 299	Meagher	ACTIVE	LODE	2025	5/25/2023	20	11N	7E	5	SW
MT106311372	MT106311073	NEW 300	Meagher	ACTIVE	LODE	2025	5/25/2023	20	11N	7E	5	SW
												SE
MT106311373	MT106311073	NEW 301	Meagher	ACTIVE	LODE	2025	5/25/2023	20	11N	7E	5	SE
MT106311374	MT106311073	NEW 302	Meagher	ACTIVE	LODE	2025	5/31/2023	20	11N	7E	4	SE
MT106311375	MT106311073	NEW 303	Meagher	ACTIVE	LODE	2025	5/31/2023	20	11N	7E	3	SW
											4	SE
MT106311376	MT106311073	NEW 304	Meagher	ACTIVE	LODE	2025	5/31/2023	20	11N	7E	3	SW
MT106311377	MT106311073	NEW 305	Meagher	ACTIVE	LODE	2025	5/31/2023	20	11N	7E	3	SW
												SE
MT106311378	MT106311073	NEW 306	Meagher	ACTIVE	LODE	2025	5/30/2023	20	11N	7E	3	SE
MT106311379	MT106311073	NEW 307	Meagher	ACTIVE	LODE	2025	5/30/2023	20	11N	7E	2	SW
											3	SE
MT106311380	MT106311073	NEW 308	Meagher	ACTIVE	LODE	2025	5/25/2023	20	11N	7E	5	SW
											6	SE
MT106311381	MT106311073	NEW 309	Meagher	ACTIVE	LODE	2025	5/25/2023	20	11N	7E	5	SW

MT106311382	MT106311073	NEW 310	Meagher	ACTIVE	LODE	2025	5/25/2023	20	11N	7E	5	SW
												SE
MT106311383	MT106311073	NEW 311	Meagher	ACTIVE	LODE	2025	5/25/2023	20	11N	7E	5	SE
MT106311384	MT106311073	NEW 312	Meagher	ACTIVE	LODE	2025	5/31/2023	20	11N	7E	4	SE
MT106311385	MT106311073	NEW 313	Meagher	ACTIVE	LODE	2025	5/31/2023	20	11N	7E	3	SW
											4	SE
MT106311386	MT106311073	NEW 314	Meagher	ACTIVE	LODE	2025	5/31/2023	20	11N	7E	3	SW
MT106311387	MT106311073	NEW 315	Meagher	ACTIVE	LODE	2025	5/31/2023	20	11N	7E	3	SW
												SE
MT106311388	MT106311073	NEW 316	Meagher	ACTIVE	LODE	2025	5/25/2023	20	11N	7E	4	SW
MT106311389	MT106311073	NEW 317	Meagher	ACTIVE	LODE	2025	5/25/2023	20	11N	7E	4	SW
MT106311390	MT106311073	NEW 318	Meagher	ACTIVE	LODE	2025	5/25/2023	20	11N	7E	5	SW
												SE
MT106311391	MT106311073	NEW 319	Meagher	ACTIVE	LODE	2025	5/25/2023	20	11N	7E	4	SW
											5	SE
MT106311392	MT106311073	NEW 320	Meagher	ACTIVE	LODE	2025	5/25/2023	20	11N	7E	4	SE
MT106311393	MT106311073	NEW 321	Meagher	ACTIVE	LODE	2025	5/25/2023	20	11N	7E	3	SW
											4	SE
MT106311394	MT106311073	NEW 322	Meagher	ACTIVE	LODE	2025	5/25/2023	20	11N	7E	3	SW
MT106311395	MT106311073	NEW 323	Meagher	ACTIVE	LODE	2025	5/25/2023	20	11N	7E	3	SW
												SE
MT106311396	MT106311073	NEW 324	Meagher	ACTIVE	LODE	2025	5/24/2023	20	11N	7E	5	SW
MT106311397	MT106311073	NEW 325	Meagher	ACTIVE	LODE	2025	5/25/2023	20	11N	7E	5	SW
												SE
MT106311398	MT106311073	NEW 326	Meagher	ACTIVE	LODE	2025	5/25/2023	20	11N	7E	4	SW
											5	SE
MT106311399	MT106311073	NEW 327	Meagher	ACTIVE	LODE	2025	5/31/2023	20	11N	7E	4	SE
MT106311400	MT106311073	NEW 328	Meagher	ACTIVE	LODE	2025	5/31/2023	20	11N	7E	3	SW
											4	SE
MT106311401	MT106311073	NEW 329	Meagher	ACTIVE	LODE	2025	5/31/2023	20	11N	7E	3	SW
MT106311402	MT106311073	NEW 330	Meagher	ACTIVE	LODE	2025	5/31/2023	20	11N	7E	3	SW
												SE
MT106311403	MT106311073	NEW 331	Meagher	ACTIVE	LODE	2025	5/25/2023	20	11N	7E	4	SW

MT106311404	MT106311073	NEW 332	Meagher	ACTIVE	LODE	2025	5/24/2023	20	11N	7E	5	SW
MT106311405	MT106311073	NEW 333	Meagher	ACTIVE	LODE	2025	5/25/2023	20	11N	7E	5	SW
												NW
												SE
												NE
MT106311406	MT106311073	NEW 334	Meagher	ACTIVE	LODE	2025	5/25/2023	20	11N	7E	4	SW
												NW
											5	SE
												NE
MT106311407	MT106311073	NEW 335	Meagher	ACTIVE	LODE	2025	5/31/2023	20	11N	7E	4	SE
												NE
MT106311408	MT106311073	NEW 336	Meagher	ACTIVE	LODE	2025	5/31/2023	20	11N	7E	3	SW
												NW
											4	SE
												NE
MT106311409	MT106311073	NEW 337	Meagher	ACTIVE	LODE	2025	5/31/2023	20	11N	7E	3	SW
												NW
MT106311410	MT106311073	NEW 338	Meagher	ACTIVE	LODE	2025	5/31/2023	20	11N	7E	3	SW
												NW
												SE
												NE
MT106311411	MT106311073	NEW 339	Meagher	ACTIVE	LODE	2025	5/25/2023	20	11N	7E	4	SW
												NW
MT106311412	MT106311073	NEW 340	Meagher	ACTIVE	LODE	2025	5/24/2023	20	11N	7E	5	SW
												NW
MT106311413	MT106311073	NEW 341	Meagher	ACTIVE	LODE	2025	5/24/2023	20	11N	7E	5	NW
MT106311414	MT106311073	NEW 342	Meagher	ACTIVE	LODE	2025	5/31/2023	20	11N	7E	4	NE
MT106311415	MT106311073	NEW 343	Meagher	ACTIVE	LODE	2025	5/31/2023	20	11N	7E	3	NW
											4	NE
MT106311416	MT106311073	NEW 344	Meagher	ACTIVE	LODE	2025	5/31/2023	20	11N	7E	3	NW
MT106311417	MT106311073	NEW 345	Meagher	ACTIVE	LODE	2025	5/31/2023	20	11N	7E	3	NW
												NE
MT106311418	MT106311073	NEW 346	Meagher	ACTIVE	LODE	2025	5/24/2023	20	11N	7E	5	NW

												NE
MT106311419	MT106311073	NEW 347	Meagher	ACTIVE	LODE	2025	5/31/2023	20	11N	7E	4	NE
MT106311420	MT106311073	NEW 348	Meagher	ACTIVE	LODE	2025	5/31/2023	20	11N	7E	3	NW
											4	NE
MT106311421	MT106311073	NEW 349	Meagher	ACTIVE	LODE	2025	5/31/2023	20	11N	7E	3	NW
MT106311422	MT106311073	NEW 350	Meagher	ACTIVE	LODE	2025	5/31/2023	20	11N	7E	3	NW
												NE
MT106311423	MT106311073	NEW 351	Meagher	ACTIVE	LODE	2025	5/24/2023	20	11N	7E	5	NW
MT106311424	MT106311073	NEW 352	Meagher	ACTIVE	LODE	2025	5/24/2023	20	11N	7E	5	NW
												NE
MT106311425	MT106311073	NEW 353	Meagher	ACTIVE	LODE	2025	5/24/2023	20	11N	7E	5	NW
											6	NE
MT106311426	MT106311073	NEW 354	Meagher	ACTIVE	LODE	2025	5/24/2023	20	11N	7E	5	NW
											6	NE
MT106311427	MT106311073	NEW 355	Meagher	ACTIVE	LODE	2025	5/31/2023	20	11N	7E	4	NE
MT106311428	MT106311073	NEW 356	Meagher	ACTIVE	LODE	2025	5/31/2023	20	11N	7E	3	NW
											4	NE
MT106311429	MT106311073	NEW 357	Meagher	ACTIVE	LODE	2025	5/31/2023	20	11N	7E	3	NW
MT106311430	MT106311073	NEW 358	Meagher	ACTIVE	LODE	2025	5/31/2023	20	11N	7E	3	NW
												NE
MT106311431	MT106311073	NEW 359	Meagher	ACTIVE	LODE	2025	5/24/2023	20	11N	7E	5	NW
MT106311432	MT106311073	NEW 360	Meagher	ACTIVE	LODE	2025	5/24/2023	20	11N	7E	5	NW
												NE
MT106311433	MT106311073	NEW 361	Meagher	ACTIVE	LODE	2025	5/24/2023	20	11N	7E	5	NW
												NE
MT106311434	MT106311073	NEW 362	Meagher	ACTIVE	LODE	2025	5/24/2023	20	11N	7E	5	NW
MT106311435	MT106311073	NEW 363	Meagher	ACTIVE	LODE	2025	5/24/2023	20	11N	7E	5	NW
											6	NE
MT106311436	MT106311073	NEW 364	Meagher	ACTIVE	LODE	2025	5/31/2023	20	11N	7E	4	NE
MT106311437	MT106311073	NEW 365	Meagher	ACTIVE	LODE	2025	5/31/2023	20	11N	7E	3	NW
											4	NE
MT106311438	MT106311073	NEW 366	Meagher	ACTIVE	LODE	2025	5/31/2023	20	11N	7E	3	NW
MT106311439	MT106311073	NEW 367	Meagher	ACTIVE	LODE	2025	5/31/2023	20	11N	7E	3	NW


NE