TECHNICAL REPORT ON THE MOUNT MILLIGAN MINE

NORTH-CENTRAL BRITISH COLUMBIA

CENTERRA GOLD INC.

NI 43-101 Technical Report

Qualified Persons:

John Fitzgerald, B.Sc. Mine Engineering, P.Eng.

Christopher Paul Jago, M.Sc. Geological Sciences, P. Geo.

Slobodan Jankovic, B.Sc. Geology, P.Geo.

Berge Simonian, B.A.Sc. Mining Engineering, P.Eng.

Catherine A. Taylor, B.Eng. & Mgmt. Civil Engineering, P.Eng.

Bruno Borntraeger, B.A.Sc. Geological Engineering P. Eng.

Date: March 26, 2020

Effective Date: December 31, 2019

CERTIFICATE OF QUALIFIED PERSON JOHN FITZGERALD

I, John Fitzgerald, state that:

(a) I am the Vice President Projects & Technical Services at:
Centerra Gold Inc.
1 University Avenue
Toronto, Ontario, M5J 2P1

(b) This certificate applies to the technical report titled Technical Report on the Mount Milligan Mine with an effective date of: December 31, 2019.

(c) I am a “qualified person” for the purposes of National Instrument 43-101 (the “Instrument”). My qualifications as a qualified person are as follows. I am a graduate of the University of Nottingham with an Honours degree in Mining Engineering, awarded July 1989. I am a Professional Engineer with Professional Engineers Ontario, license 100179653. My relevant experience after graduation and over 30 years for the purpose of the Technical Report includes mine operations, projects and planning.

(d) My most recent personal inspection of each property described in the Technical Report occurred during April 2019 and was for a duration of four days.

(e) I am responsible for Items 1.1, 1.11, 1.12, 1.15-1.21, 2-5, 15, 16, 18.1, 18.3, 19-24, 25.2, 25.4, 27.3 and 27.4 of the Technical Report.

(f) I am not independent of the issuer as described in section 1.5 of the Instrument.

(g) My prior involvement with the property that is the subject of the Technical Report is as follows. In my role with Centerra, I have responsibility to ensure that life-of-mine plans are updated on a regular basis to accurately represent the current understanding of ongoing and future operations.

(h) I have read National Instrument 43-101 guidelines. The parts of the Technical Report for which I am responsible have been prepared in compliance with this Instrument; and

(i) At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the parts of Technical Report] for which I am responsible, contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated at Burlington, Ontario this 26th day of March 2020.

(original signed and sealed) John Fitzgerald

John Fitzgerald, PEO License Number 100179653

CERTIFICATE OF QUALIFIED PERSON C. PAUL JAGO

I, Christopher Paul Jago. P. Geo., as author of this report entitled "Technical Report on the Mount Milligan Mine" prepared for Centerra Gold Inc. with an effective date of December 31, 2019, do hereby certify that:

(a) I am currently employed as Exploration Manager by Centerra Gold Services Inc., 299 Victoria Street, Suite 200, Prince George, BC, V2L 5B8.

(b) I am a Registered Professional Geologist with Engineers & Geoscientists British Columbia. Registration # 41112

(c) I have worked full-time as an Exploration Geologist in both industry and government for 12 years since graduation from university, and as an Exploration Manager for over two years.

(d) I worked as the Northcentral-Northeast Regional Geologist for the BC Ministry of Energy and Mines from January 2012 to May 2018; and worked as a Geologist for Freeport-McMoRan Inc. (Oro Valley, Tucson AZ) from May 2008 to December 2011.

(e) I graduated from the Mineral Deposit Research Unit (MDRU), University of British Columbia with a M.Sc. of Geological Sciences in June 2008; and graduated from the University of Toronto with a Bachelor of Science (Geology-Physical Geography) in 2005.

(f) I have read the definition of "qualified person" as set out in the National Instrument 43-101 (NI 43-101) and certify that because of my education, association with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

(g) I most recently visited the Mount Milligan mine from January 27-30th, 2020 and visit the property on a regular basis during exploration programs.

(h) I am responsible for Sections 1.2-1.8, 1.20, 1.21, 4.1, 6-12, 25.1 and 27.1.

(i) I have read NI 43-101 and the technical report has been prepared in compliance with NI 43-101 and Form 101F1.

(j) At the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical report not misleading.

Dated this 26th day of March 2020

(original signed and sealed) C. Paul Jago

Christopher Paul Jago, P. Geo.

CERTIFICATE OF QUALIFIED PERSON SLOBODAN JANKOVIC

I, Slobodan Jankovic, state that:

(a) I am a Senior Director Technical Services at:
Centerra Gold Inc.
1 University Avenue
Toronto, ON, M5J 2P1

(b) This certificate applies to the technical report titled be “Technical Report on the Mount Milligan Mine” with an effective date of December 31, 2019.

(c) I am a “qualified person” for the purposes of National Instrument 43-101 (the “Instrument”). My qualifications as a qualified person are as follows. I am a graduate of the University of Belgrade, The Faculty of Mining and Geology with the bachelor’s degree in Geology. I graduated in 1986 with the specialization in Mineral Exploration and Mining Geology upon which I have completed the post graduate study with the same specialization. My relevant experience after graduation and over 35 years for the purpose of the Technical Report includes working on the resource estimates at one of the world largest copper – gold porphyry deposits in the Timok region. I have developed further relevant experience working for Vale-Inco, HudBay Minerals, Wardrop Engineering and Detour Gold as the chief geologist. The most recent experience before joining Centerra Gold in January 2019 included work with Scotia Bank Investment banking during which I was responsible in many resource evaluations and due diligence evaluations worldwide.

(d) My most recent personal inspection of each property described in the Technical Report occurred on April 8, 2019 and was for a duration of 3 days.

(e) I am responsible for Item (s) 1.10, 1.20, 14, 25.1, 27.4 of the Technical Report.

(f) I am not independent of the issuer as described in section 1.5 of the Instrument.

(g) My prior involvement with the property that is the subject of the Technical Report is as follows. I have been actively working in collaboration with the mine geologist and exploration team of the Mt. Milligan operation on the proper geological, structural and geochemical understanding and interpretation. I have review and guided to some extent the grade control model and the relevant production data. In addition, I have been involved in implementing the proper geo-metallurgical program including samples collection which would be supplemented by the updated recovery model.

(h) I have read National Instrument 43-101 Technical Report on the Mount Milligan Mine has been prepared in compliance with this Instrument; and

(i) At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the Technical Report on the Mount Milligan Mine and/or the sections 1.10, 14, 1.20, 25.1 27.4 of 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 at Toronto, this 26th of March 2020.

(original signed and sealed) Slobodan Jankovic

Slobodan Jankovic

Association of Professional Geoscientists of Ontario, APGO, license number 1388

CERTIFICATE OF QUALIFIED PERSON BERGE SIMONIAN

I, Berge Simonian, state that:

(a) I am a Chief Metallurgist at:
Mount Milligan
Thompson Creek Metals, Inc
299 Victoria Street, Suite 200
Prince George, BC V2L 5B8

(b) This certificate applies to the technical report titled Technical Report on the Mount Milligan Mine with an effective date of: December 31, 2019.

(c) I am a “qualified person” for the purposes of National Instrument 43-101 (the “Instrument”). My qualifications as a qualified person are as follows. I am a graduate of the University of British Columbia with a Bachelor of Applied Science in Mining Engineering (2005 graduation) and an active, practicing Professional Engineer registered, and in good standing, under Engineers and Geoscientists of British Columbia (EGBC). My relevant experience after graduation and over the last fifteen (15) years for the purpose of the Technical Report includes work in operations, engineering design, process control, and consulting ranging from industrial minerals, base metals, and precious metals. This work includes data analysis, modelling, and advanced control projects as well as site work involving operator training, commissioning, process analysis, process improvement, and process optimization.

(d) The requirement for a site visit is not applicable to me as I am an active member of Mount Milligan’s management team.

(e) I am responsible for Items 1.9, 1.13, 1.20, 1.21, 13, 17, 25.3, and 27.5 of the Technical Report.

(f) I am not independent of the issuer as described in section 1.5 of the Instrument.

(f) My prior involvement with the property that is the subject of the Technical Report is as follows. I have been the Chief Metallurgist at Mount Milligan since September 2017.

(g) I have read National Instrument 43-101. The parts of the Technical Report for which I am responsible have been prepared in compliance with this Instrument; and

(h) At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the parts of 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 at West Kelowna, British Columbia this 26th of March 2020.

(original signed and sealed) Berge Simonian

Berge Simonian, P.Eng. (41276)

CERTIFICATE OF QUALIFIED PERSON CATHERINE A. TAYLOR

I, Catherine A. Taylor, state that:

(a) I am the Vice President Risk & Insurance at:
Centerra Gold Inc.
1 University Avenue
Toronto, ON, M5J 2P1

(b) This certificate applies to the technical report entitled “Technical Report on the Mount Milligan Mine” with an effective date of December 31, 2019.

(c) I am a “qualified person” for the purposes of National Instrument 43-101 (the “Instrument”). My qualifications as a qualified person are as follows. In 1982 I graduated from McMaster University, Hamilton, Ontario, Canada with a Bachelor of Engineering & Management (Civil) degree. My relevant experience after graduation and for the purpose of the Technical Report includes working in technical and risk management positions for the past 38 years in a variety of industries including oil & gas, chemicals, construction, public safety, financial services and mining. Prior to joining Centerra, I held senior risk management roles at mining companies Barrick Gold Corporation and Kinross Gold Corporation.

(d) My most recent personal inspection of each property described in the Technical Report occurred on October 7, 2019 and was for a duration of 4 days.

(e) I am responsible for Sections 1.20, 25.5 and 26 of the Technical Report.

(f) I am not independent of the issuer as described in section 1.5 of the Instrument.

(g) My prior involvement with the property that is the subject of the Technical Report is as follows. I am responsible for the Enterprise Risk Management program of Centerra Gold Inc. As such, I am responsible for ensuring that risks from all sources which threaten our properties and operations are identified, assessed, mitigated and monitored.

(h) I have read National Instrument 43-101 Technical Report on the Mount Milligan Mine which has been prepared in compliance with this Instrument; and

(i) At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the sections of the Technical Report on the Mount Milligan Mine 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 at Toronto, this 26th day of March 2020.

(original signed and sealed) Catherine A. Taylor

Catherine A. Taylor, P.Eng.
Professional Engineers Ontario License # 90291873

CERTIFICATE OF QUALIFIED PERSON BRUNO BORNTRAEGER

I, Bruno Borntraeger, state that:

(a) I am a Specialist Geotechnical Engineer | Associate at:
Knight Piesold Ltd.
Suite 1400 750 West Pender St.
Vancouver, BC., Canada, V6C 2T8

(b) This certificate applies to the technical report titled Technical Report on the Mount Milligan Mine with an effective date of: December 31, 2019

(c) I am a “qualified person” for the purposes of National Instrument 43-101 (the “Instrument”). My qualifications as a qualified person are as follows. I am a graduate from the University of British Columbia with a Bachelor of Applied Science in Geological Engineering,1990. I am a member in good standing of the Engineers and Geoscientists of British Columbia (License #20926]. My relevant experience after graduation for the purpose of the Technical Report includes 30 years geotechnical engineering, mine waste and water management, environmental compliance, mine development with practical experience in feasibility studies, detailed engineering, permitting, construction, operations and closure.

(d) My most recent personal inspection of each property described in the Technical Report occurred on October 8 to 10, 2019 and was for a duration of 3 days.

(e) I am responsible for sections 1.14 and 18.2 of the Technical Report.

(f) I am independent of the issuer as described in section 1.5 of the Instrument.

(g) My prior involvement with the property that is the subject of the Technical Report is as follows. Tailings Dam Engineer of Record (2010-present).

(h) I have read National Instrument 43-101. The part of the Technical Report for which I am responsible has been prepared in compliance with this Instrument; and

(i) At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the part of 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 at Vancouver, BC this 26th of March 2020.

(original signed and sealed) Bruno Borntraeger

Bruno Borntraeger, P. Eng. (BC, 20926)

Cautionary Note Regarding Forward-Looking Information

Information contained in this Technical Report and the documents referred to herein which are not statements of historical facts, may be “forward-looking information” for the purposes of Canadian securities laws. Such forward looking information involves risks, uncertainties and other factors that could cause actual results, performance, prospects and opportunities to differ materially from those expressed or implied by such forward looking information. The words “expect”, “target”, “estimate”, “may”, “will”, and similar expressions identify forward-looking information. These forward-looking statements relate to, among other things, mineral reserve and mineral resource estimates; grades and recoveries; development plans; mining methods and metrics including strip ratio; recovery process; production expectations including expected cash flows, capital cost estimates and expected life of mine operating costs; and expected outcomes of continuous improvement projects and opportunities. .

Forward-looking information is necessarily based upon a number of estimates and assumptions that, while considered reasonable by Centerra Gold Inc. (“Centerra”) are inherently subject to significant political, business, economic and competitive uncertainties and contingencies. There may be factors that cause results, assumptions, performance, achievements, prospects or opportunities in future periods not to be as anticipated, estimated or intended. These factors include the following risks relating to the Mount Milligan project, Centerra and/or TCM: (A) strategic, legal, political and regulatory risks, including delays or refusals to grant required permits and licenses; the status of relationships with local communities; Indigenous claims and consultative issues, increases in contributory demands; management of external stakeholder expectations; litigation; potential defects of title not known as of the date hereof; the impact of changes in, or to the more aggressive enforcement of laws, regulations and government practices; the inability of Centerra or TCM to enforce its respective legal rights in certain circumstances; risks related to anti-corruption legislation; and potential risks related to kidnapping or acts of terrorism; (B) financial risks, including sensitivity of the business to the volatility of metal prices; the imprecision of mineral reserves and mineral resources estimates, and the assumptions they rely on; the accuracy of the production and cost estimates; reliance on a few key customers for the gold-copper concentrate at Mount Milligan; the impact of currency fluctuations; continued compliance with financial covenants in Centerra’s credit agreement that is secured by certain assets used at the Mount Milligan Mine, Centerra’s access to cash flow from its subsidiaries; and changes to taxation laws; and (C) operational and geotechnical risks, including the adequacy of insurance to mitigate operational risks; unanticipated ground and water conditions; shortages of water for processing activities; mechanical breakdowns; the occurrence of any labour unrest or disturbance; the ability to accurately predict decommissioning and reclamation costs; the ability to attract and retain qualified personnel; geological problems, including earthquakes and other natural disasters; metallurgical and other processing problems; unusual or unexpected mineralogy or rock formations; tailings design or operational issues, including dam breaches or failures; delays in transportation; and exposure to epidemics and pandemics which may impact supply chain or employees.

There can be no assurances that forward-looking information and statements will prove to be accurate, as many factors and future events, both known and unknown could cause actual results, performance or achievements to vary or differ materially, from the results, performance or achievements that are or may be expressed or implied by such forward-looking statements contained herein or incorporated by reference. Accordingly, all such factors should be considered carefully when making decisions with respect to Centerra and prospective investors should not place undue reliance on forward-looking information. Forward-looking information in this technical report is as of the issue date, March 26, 2020. Centerra and the Qualified Persons who authored this Technical Report assume no obligation to update or revise forward-looking information to reflect changes in assumptions, changes in circumstances or any other events affecting such forward-looking information, except as required by applicable law.

Non-GAAP Measures

This document contains the following non-GAAP financial measures: all-in sustaining costs per ounce sold on a by-product basis and free cash flow. These financial measures do not have any standardized meaning prescribed by GAAP and are therefore unlikely to be comparable to similar measures presented by other issuers, even as compared to other issuers who may be applying the World Gold Council ("WGC") guidelines, which can be found at http://www.gold.org.

All-in sustaining costs on a by-product basis per ounce sold include adjusted operating costs, the cash component of capitalized stripping costs, corporate general and administrative expenses, accretion expenses, and sustaining capital, net of copper and silver credits. The measure incorporates costs related to sustaining production. Copper and silver credits represent the expected revenue from the sale of these metals. Free cash flow is calculated as cash provided by operations less additions to property, plant and equipment.

Management believes that the use of these non-GAAP measures will assist analysts, investors and other stakeholders of the Company in understanding the costs associated with producing gold, understanding the economics of gold mining, assessing our operating performance, our ability to generate free cash flow from current operations and to generate free cash flow on an overall Company basis, and for planning and forecasting of future periods. However, the measures do have limitations as analytical tools as they may be influenced by the point in the life cycle of a specific mine and the level of additional exploration or expenditures a company has to make to fully develop its properties. Accordingly, these non-GAAP measures should not be considered in isolation, or as a substitute for, analysis of our results as reported under GAAP.

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1. SUMMARY

This Technical Report summarizes the current and planned operations, Mineral Resources and Mineral Reserves for the operating Mount Milligan copper-gold mine (the Mount Milligan Mine, the Project or the Property) located between Fort St. James and Mackenzie, British Columbia, Canada. The Technical Report was prepared by and for Centerra Gold Inc. (Centerra) by qualified persons as listed in Section 2. This Technical Report conforms to National Instrument 43-101 *Standards of Disclosure for Mineral Projects* (NI 43-101) and follows the format set out in Form 43-101F1 for Technical Reports.

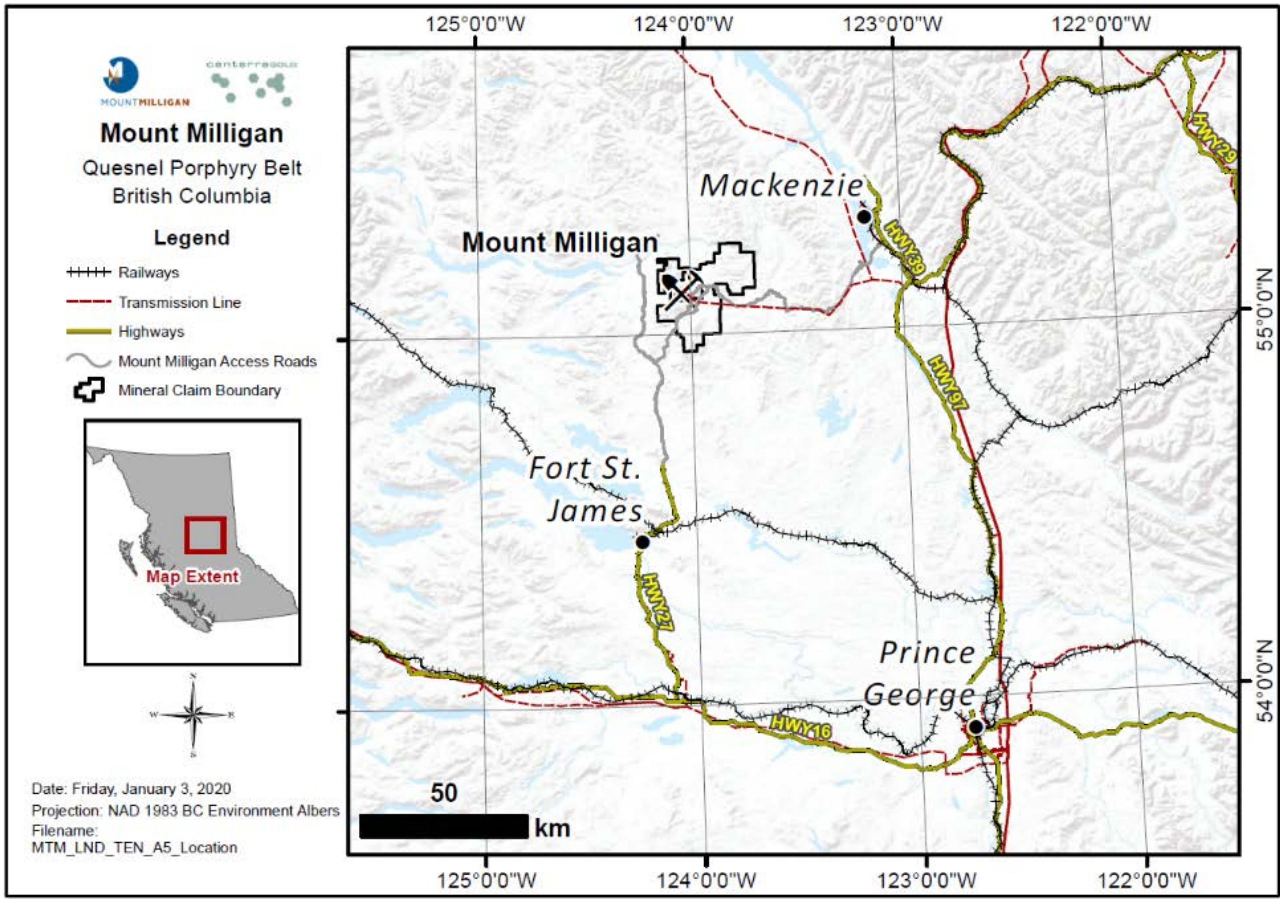
The previous technical report for the Mount Milligan Mine, “Technical Report on the Mount Milligan Mine”, published March 22, 2017 with an effective date December 31, 2016, is referred to in this technical report as the 2017 Technical Report.

All dollar figures in this Technical Report refer to US dollars, unless otherwise noted.

1.1. PROJECT DESCRIPTION AND LOCATION

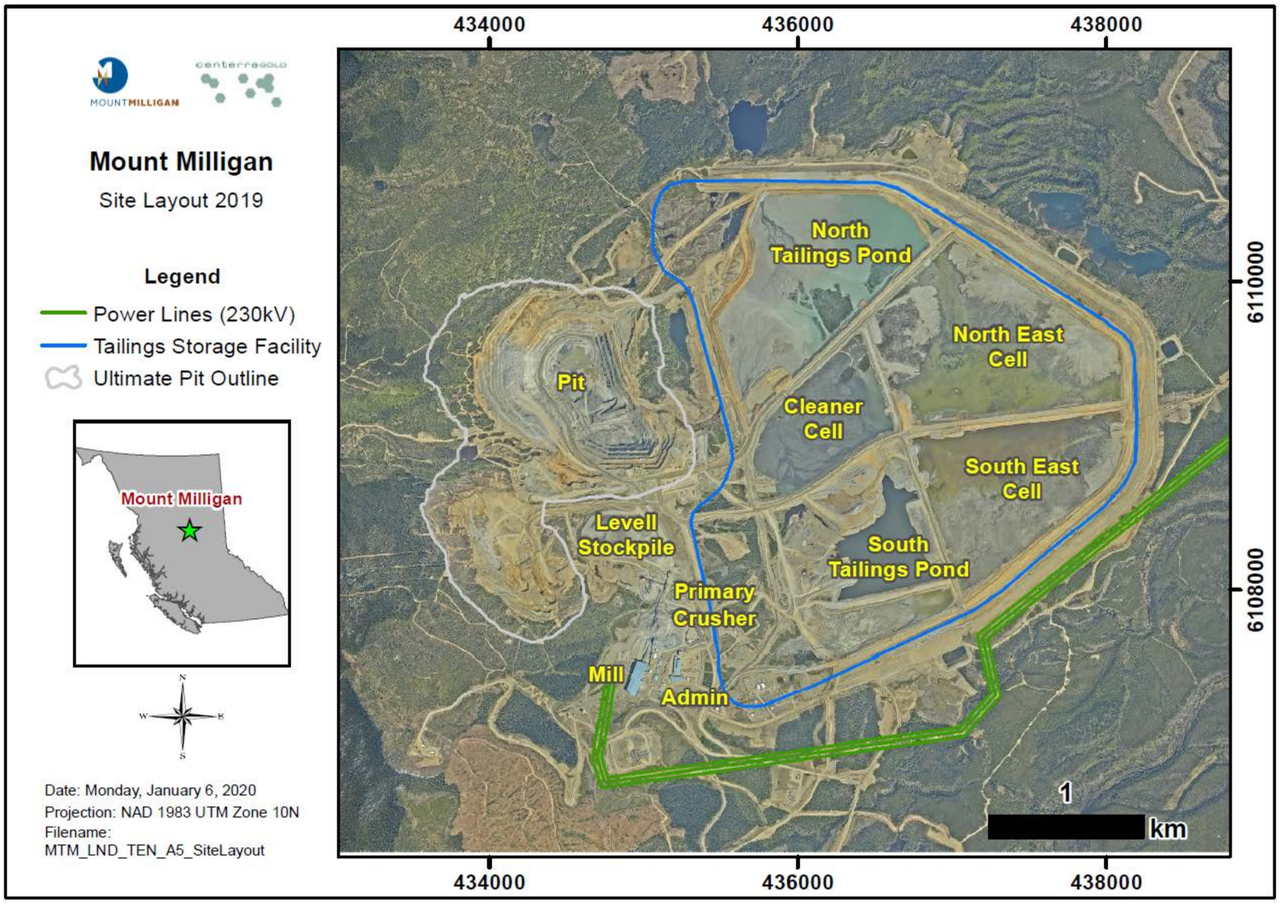
The Mine is located 155 kilometres (km) northwest of Prince George (population approximately 79,000) in north-central British Columbia. Forestry–based communities Mackenzie (population approximately 3,200) and Fort St. James (population approximately 1,600) are within daily commuting distance of the Project site. Both communities are serviced by rail. Figure 1-1 shows the location of the mine and local communities.

Figure 1-1: Mount Milligan Mine Location



The mine site comprises an open pit mine, tailings storage facility (TSF), mineralized stockpiles, a processing plant, workshop, warehouse, administration buildings and camp. Figure 1-2 provides a plan view of the Project.

Figure 1-2: Plan View of Mount Milligan Mine Site



1.2. PROJECT HISTORY AND OWNERSHIP

Limited exploration activity was first recorded in 1937. In 1984, prospector Richard Haslinger (Haslinger) and BP Resources Canada Limited (BP Resources) located claims on the current site. In 1986, Lincoln Resources Inc. (Lincoln) optioned the claims and in 1987 completed a diamond drilling program that led to the discovery of significant copper-gold mineralization. In the late 1980s, Lincoln reorganized, amalgamated with Continental Gold Corp. (Continental Gold) and continued ongoing drilling in a joint venture with BP Resources.

In 1991, Placer Dome Inc. (Placer Dome) acquired the Project from the joint-venture partners, resumed exploration drilling and completed a pre-feasibility study for the development of a 60,000 t/d open pit mine and flotation process plant.

Barrick Gold Corporation (Barrick) purchased Placer Dome in 2006 and sold its Canadian assets to Goldcorp Inc. (Goldcorp), who then in turn sold the Project to Atlas Cromwell Ltd. (Atlas Cromwell).

Atlas Cromwell changed its name to Terrane Metals Corp. (Terrane) and initiated a comprehensive work program.

In October 2010, Thompson Creek Metals Company Inc. (TCM) acquired the Mount Milligan development project through its acquisition of Terrane, entered a streaming agreement with Royal Gold and subsequently constructed the Mount Milligan Mine, which commenced commercial production in February 2014.

In October 2016, TCM was acquired by a subsidiary of Centerra (the Acquisition) and, in connection with that acquisition, Terrane and certain other subsidiary entities of TCM were amalgamated into TCM. The Mount Milligan Mine is now fully owned by TCM, an indirect subsidiary of Centerra.

Table 1-1: Historical Production to December 31, 2019

Years	Milled Ore Tonnage ('000 t)	Head Grade		Metal Recovery		Concentrate Production			Waste Tonnage ('000 t)
		Cu (%)	Au (g/t)	Cu Rec (%)	Au Rec (%)	Concentrate ('000 dmt)	Cu (M lb)	Au ('000 oz)	
2013	2,055	0.29	0.56	79.2%	54.3%	18.7	10.4	20.1	24,753
2014	14,290	0.27	0.63	80.4%	63.1%	125.4	68.0	184.0	11,224
2015	16,138	0.26	0.64	80.2%	68.6%	140.7	75.2	226.0	14,413
2016	19,277	0.19	0.58	74.7%	58.9%	125.6	61.6	212.0	20,363
2017	17,743	0.18	0.64	78.9%	62.4%	121.5	56.4	228.1	20,557
2018	13,556	0.20	0.71	81.4%	64.5%	106.0	49.6	199.5	19,764
2019	16,350	0.26	0.53	81.3%	67.4%	159.5	75.0	187.8	23,730
Total	**99,410**	**0.23**	**0.62**	**79.5%**	**63.8%**	**797.3**	**396.2**	**1,257.4**	**134,804**

Figures are shown on a 100% produced basis.

Royalties and metals streams associated with the Project are discussed in Section 4.

1.3. GEOLOGY

The Mount Milligan deposit is located within the Quesnel terrane of the North American Cordillera. The Quesnel terrane is part of the Intermontane Belt: a composite belt of volcanic arc and oceanic terranes that evolved outboard of the western margin of North America. The terranes of the Intermontane Belt are interpreted to have accreted to North America during the Middle Jurassic.

The Mount Milligan property is mostly underlain by Upper Triassic volcanic rocks of the Witch Lake succession. The Witch Lake succession is moderately-to-steeply east-northeast dipping and characterized by augite-phyric volcaniclastic and lesser coherent basaltic andesite to andesite, with subordinate epiclastic beds. In the northwestern part of the Mount Milligan property, volcanic rocks are intruded by Early Jurassic to Cretaceous rocks of the Mount Milligan intrusive complex located about 5 to 9 km north of the Mount Milligan porphyry deposit. The Early Jurassic component of the intrusive complex comprises monzonitic rocks with minor dioritic-monzodioritic and gabbroic-monzogabbroic rocks.

Mineralization at the Mount Milligan deposit consists of two styles, early-stage porphyry gold-copper (Au-Cu) and late-stage high-gold low-copper (HGLC). The early-stage porphyry Au-Cu mineralization comprises mainly chalcopyrite and pyrite, occurs with potassic alteration and early-stage vein types, and is spatially associated with composite monzonite porphyry stocks (especially at their hanging-wall and footwall margins), hydrothermal breccia, and narrow dyke and breccia complexes. Late-stage, structurally controlled pyritic HGLC style mineralization is associated with carbonate-phyllic alteration and intermediate- to late-stage vein types, and is spatially associated with faults, fault breccias and faulted lithological contacts (i.e. faulted monzonite porphyry dyke margins). It crosscuts and overprints the earlier stage porphyry Au-Cu mineralization.

Porphyry style Au-Cu mineralization occurs in the hanging-wall and footwall zones of the MBX, Saddle, Southern Star, and Goldmark stocks. Disseminated and vein/veinlet-hosted mineralization is associated with the composite monzonite stocks, their brecciated margins and variably altered volcanic host rocks. Core zones of auriferous chalcopyrite-pyrite mineralization with magnetite rich potassic alteration transition laterally and vertically to pyrite rich HGLC zones within the inner propylitic (albitic) and carbonate-phyllic alteration shells; the latter appear to be late stage and exhibit strong structural control.

Chalcopyrite, the main copper-bearing mineral, is associated with potassic alteration at the contact margin between volcanic and intrusive rocks. It occurs most commonly as fine-grained disseminations and fracture fillings, and less commonly as veinlets and in veinlet selvages. Adjacent to the MBX stock, chalcopyrite may be accompanied by pyrite to form coarse sulphide aggregates. Chalcopyrite-bearing veins contain pyrite and magnetite in a gangue of potassium feldspar, quartz, and calcite.

Pyrite content increases with distance from the MBX and Southern Star stocks and is most abundant in propylitically altered rocks. Pyrite occurs as disseminations, veinlets, large clots, patches, and as replacements of mafic minerals. Gold mineralization in the 66 zone is associated with 10-20% pyrite. Cross-cutting vein relationships indicate several generations of pyrite mineralization.

Gold occurs as grains from 1 to 100 µm in size, as observed in process samples. Grains occur as microfracture fillings and are attached to pyrite, chalcopyrite, or bornite (Ditson, 1997). Gold also forms inclusions within pyrite, chalcopyrite, and magnetite grains. SEM work indicates electrum throughout the deposit with varying gold to silver ratios.

1.4. DEPOSITS

The Mount Milligan deposits are categorized as silica-saturated alkalic Cu-Au porphyry deposits associated with alkaline monzodioritic-to-syenitic igneous rocks and are recognized in only a few mineral provinces worldwide. Porphyry copper ± gold deposits commonly consist of vein stockworks, vein sets, veinlets, and disseminations of pyrite, chalcopyrite ± bornite that occur in large zones of economic bulk-mineable mineralization within porphyritic igneous intrusions, their contact margins, and adjoining host rocks. The mineralization is spatially, temporally, and genetically associated with hydrothermal alteration of the intrusive bodies and host rocks.

Examples of alkalic Cu-Au porphyry deposits in British Columbia include Galore Creek, Mount Polley, Copper Mountain, New Afton, Mount Milligan and Lorraine. British Columbia deposits occur in both the Quesnel and Stikine island arc terranes and range in age from Late Triassic to Early Jurassic. Global examples include Ok Tedi in Papua New Guinea as well as Northparkes and Cadia in Australia.

1.5. EXPLORATION

Since the Acquisition, TCM has focussed on compiling all historical geologic and exploration data, building an Exploration department and conducting near-field, brownfield and greenfield exploration programs. The program to compile and review historical exploration reports, documents and data yielded 227,000 files and 404 GB of data. Exploration has included ground and airborne geophysics surveys, trenching and diamond core drilling.

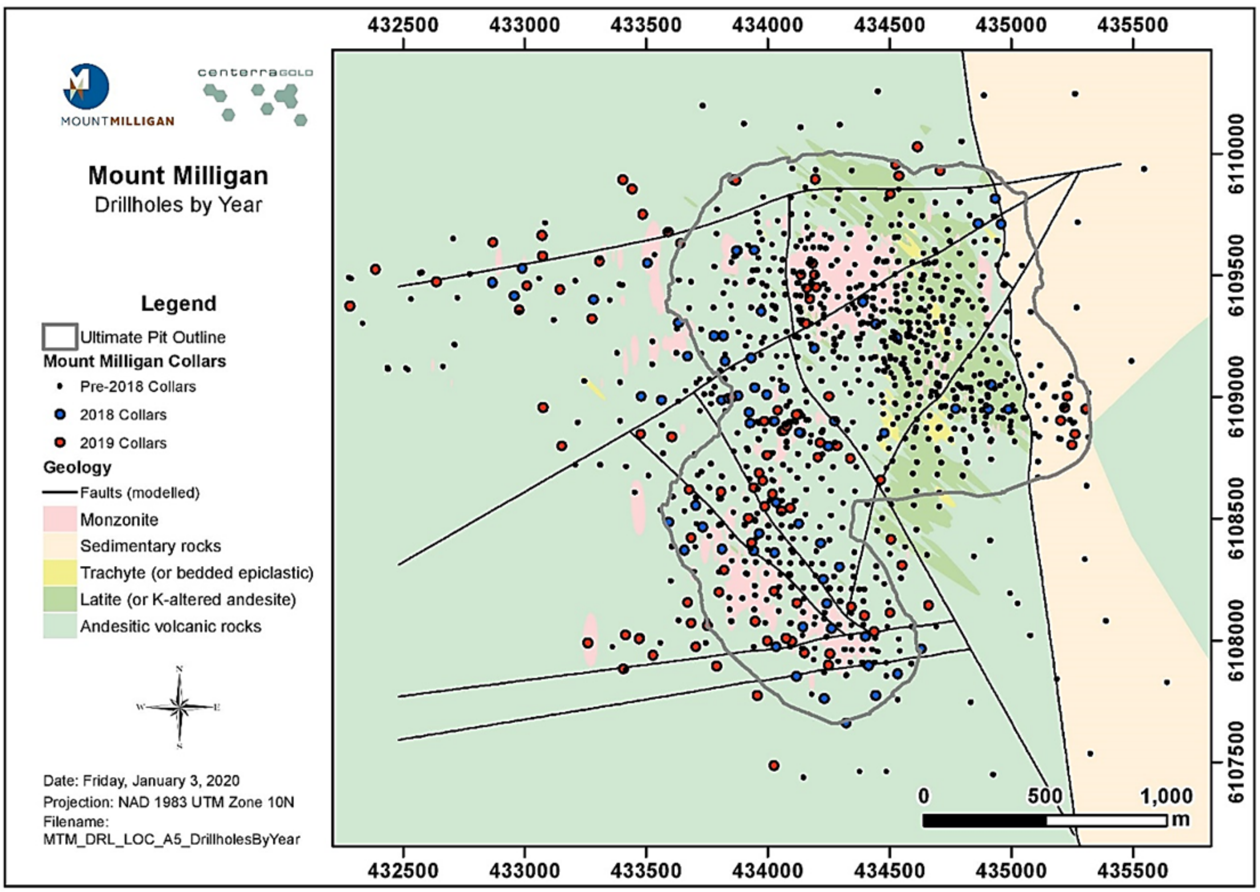
Historically, five exploration target zones were identified in the resource area (DWBX, WBX, MBX, 66 and Southern Star); three in the brownfield area within the mine lease (North Slope, Goldmark and South Boundary); and three in the greenfield area outside the mine lease (Heidi, Mitzi and Snell). Exploration since 2017 has continued to test most of these zones and refine understanding of their geological relationships and mineral potential. In addition, new target zones have been developed and continue to be tested.

Current exploration efforts target the two predominant mineralization types – early stage porphyry Au-Cu mineralization and late-stage, structurally controlled high-gold low-copper (HGLC) mineralization.

1.6. DRILLING

Numerous drilling programs have been conducted since the deposit was first drilled in 1987. Except for early programs, the majority of core drilled has been of NQ size. In total, there have been 1,218 holes drilled at Mount Milligan, recovering over 320 km of core. Figure 1-3 is a map showing collar locations of drill holes of the various campaigns.

Figure 1-3: Drill hole locations at Mount Milligan, 1987 - 2019



Geotechnical information has been routinely recorded for all diamond drilling programs including core recovery, rock quality (RQD), hardness or compressive strength (CS), degree of breakage, degree of weathering or oxidation, fracture and joint frequency, and specific gravity (SG). Core recovery routinely exceeds 90% and averages 96%.

1.7. SAMPLES

All Mount Milligan Assay Laboratory procedures are accompanied by appropriate, industry standard instrument calibration and QA/QC (Quality Assurance/Quality Control) measures, including quarterly third-party analysis checks. Ore and acid-base accounting analyses Standard Operating Procedure includes steps to confirm on-site laboratory method accuracy, precision, contamination control, sample tracking, and recordkeeping. The assay laboratory also receives blind duplicate samples from the Ore Control Geologist/Technician which are compared against daily sample analysis. This is managed as part of the MTM Assay Laboratory Quality Management System.

Most samples analyzed for the Mount Milligan deposits have been collected from NQ-sized core. Cores were either split (early programs) or sawn along the long axis with one-half sampled for assayed and the other half retained in core boxes and the core library.

A formal QA/QC program, including the insertion of standard, blank and duplicate samples for assay, was introduced after 1992. Prior to that date, external check assays were commissioned from independent laboratories.

1.8. DATA VERIFICATION

Slobodan Jankovic, QP for the Mineral Resource estimate, conducted a site visit at Mount Milligan from April 8th to 11th, 2019. The site visit included a review of site facilities, logging and sampling procedures, and the lithology and alteration domain controls used in resource estimation. No significant issues were identified with respect to the assay sampling procedures, chain of custody or the geological data collection.

Validation of the mapping co-ordinates, elevations, assay quality control/quality assurance program and the DDH database has been completed by TCM and predecessor owners of Mount Milligan.

Throughout 2019, additional validations and verifications of the database were conducted during the migration to the acQuire data systems management software. These included:

- Review of the 2007 Allnorth transformation to confirm pre-2007 drill holes originally surveyed in the local mine grid were transformed to NAD83 UTM Zone 10 consistently,
- Verification of downhole survey data from raw data files where available for 2004 to 2019 drill holes,

- Correction of downhole survey data to NAD83 UTM Zone 10 north for 2006 to 2019 (previous compilations recorded downhole survey data to True North and the UTM convergence at Mount Milligan is approximately -0.85°),
- Verification of all copper and gold assay values from the previous database compared to original assay certificates for drill holes from 2004 to 2019,
- Compilation of missing 2004, 2006-2007, and 2011-2016 QAQC data to the database, and
- Compilation of 2004-2019 laboratory QAQC data to the database from original assay certificates.

The data reviews found the assay data acceptable and any errors or omissions were minor. Centerra considers the final 2019 database to be robust and verified. The QPs of this report believe the database is adequate for the estimation of Mineral Resources according to CIM Estimation of Mineral Resources and Mineral Reserves best practice guidelines.

1.9. MINERAL PROCESSING AND METALLURGICAL TESTWORK

Metallurgical investigations conducted by various research laboratories prior to commencement of operations conclusively showed that froth flotation is the optimum process for the recovery of copper and gold; with this processing approach being adopted. These investigations were the basis of the performance models used in previous resource modelling. The 2017 Technical Report addressed previous assumptions in the copper and gold recovery models together with identified issues in the plant to produce new performance equations.

Since disclosure of the 2017 Technical Report, further investigations and projects have been undertaken to improve the recovery process and update the accuracy of the copper and gold recovery models. Using these new performance models, the LOM average recoveries are estimated at 80.6% for copper and 61.8% for gold, targeting a concentrate grade with a LOM average of 21.5% copper. Test results indicated that impurity element contents in the concentrate were below the penalty levels normally imposed by most smelters; therefore, no significant penalties are expected.

Further improvements to metallurgical recovery are being assessed including the use of alternative flotation equipment such as Staged Flotation Reactors or Direct Flotation Reactors. An initial assessment for the Mt Milligan flowsheet and ore has shown potential to increase both gold and copper recoveries using this flotation equipment with on-site piloting in progress at the time of writing this report.

1.10. MINERAL RESOURCE ESTIMATE

The Centerra Gold (CG) Project and Technical Services (P&TS) geology team completed an update of the mineral resource estimate in the fourth quarter of 2019 for the Project.

The Mount Milligan Au-Cu porphyry deposit contains a combined Measured and Indicated Mineral Resource of 125.4 million tonnes (Mt) at 0.19% Cu and 0.35g/t Au containing 517 million pounds (lbs) of copper and 1.4 million ounces (oz) of gold and an Inferred Mineral Resource of 4 Mt at 0.12% Cu and 0.46g/t Au.

The mineral resource within the 2019 ($1,500/oz gold and $3.50/lb copper) resource pit shell was based on a cut-off grade of 0.2% copper-equivalent (CuEq). The Mineral Resource was defined to the December 31, 2019 mining surface and is reported exclusive of the Mineral Reserve in Table 1-2.

Table 1-2: Mineral Resource Statement, Effective Date December 31, 2019

Category	Cut-off CuEq (%)	Tonnes (000s)	Copper Grade (%)	Gold Grade (g/t)	Contained Copper (Mlb)	Contained Gold (koz)
Measured (M)	0.2	50,582	0.16	0.44	182	713
Indicated (I)	0.2	74,788	0.20	0.29	336	695
Total M+I	**0.2**	**125,370**	**0.19**	**0.35**	**517**	**1,408**
Inferred	0.2	3,736	0.12	0.46	10	55

Notes:
(1) CIM definitions were followed for Mineral Resources.
(2) Mineral Resources are reported at a 0.2% CuEq cut-off value using metal prices of $3.50 per pound copper and $1,500 per ounce gold, and a US$/C$ exchange rate of US$1.00/C$1.25.
(3) All figures have been rounded to reflect the relative accuracy of the estimates.
(4) Mineral resources that are not mineral reserves do not have a demonstrated economic viability.
(5) Mineral Resources reported exclusive of Mineral Reserves.

The resource modelling work was completed by Mr. Slobodan Jankovic, P. Geo. (APGO#1388). Mr. Jankovic is Senior Director for Centerra Gold Project and Technical Services and is a Qualified Person as defined by National Instrument (NI) 43-101.

The effective date of the Mineral Resource Statement is December 31, 2019. The mineral resources have been estimated in conformity with CIM Estimation of Mineral Resource and Mineral Reserves Best Practices Guidelines (Nov 2019) and are reported in accordance with the Canadian Securities Administrators' National Instrument 43-101.

The Mineral Resource is based upon a geologic block model that incorporated over 133,869 individual assays from 295,961m of core drilling in 1,134 drill holes. Assay data density with drill hole spacing of 25 m to 50 m is deemed to be suitable for this type of deposit to be classified within the

Measured and Indicated category and within each of the interpreted mineralized zones. The drill hole database is supported by quality assurance/quality control (QA/QC) check assays.

In the process of completing the end-2019 resource estimate, the CG technical team validated and verified the database, and updated the geological interpretations based on available data with the following conclusions:

- Based on verification of the drill hole database, it is the QP's opinion that the information is suitable to support the end-2019 Mineral Resource estimate. Measurements obtained from the density samples taken as a part of data verification program appear to be consistent with values used for the December 2016 resource estimates.

- The resource estimate was interpolated by inverse distance square (ID2) and ordinary kriging (OK) methodologies. No significant discrepancies exist between the methods, and values obtained from ID2 estimation have been used for the respective domains and/or elements based on the final validation and compiled into the resource tabulation table.

A material resource has been delineated and the more recent drilling included in this estimate outlines the potential of the deposit to continue down dip to the west.

1.11. MINERAL RESERVE ESTIMATE

The Mineral Reserve estimate was derived through the development of an ultimate open pit design using the Mineral Resource model and pit optimization parameters as a basis for mine design.

The open pit was optimized using long-term metal price estimates of $3.00/lb Cu and $1,250/oz Au, an exchange rate of US$1.00/C$1.25, and costs related to mining, processing and G&A (including site services), and sustaining capital costs. Other factors considered include metallurgical recoveries, concentrate grades, transportation costs, smelter treatment charges, the H.R.S. Resources royalty and the Royal Gold stream in determining economic viability. The Mineral Reserve estimate reported in Table 1-3 has been classified as 60% Proven and 40% Probable on a tonnage basis.

A Net Smelter return (NSR) cut-off comprised of the costs for processing and G&A operating costs (opex) and sustaining capital unit costs (capex) was calculated to be $7.64/t or C$9.55/t. Mining opex is excluded from this calculation as the definition of ore (and waste) is made at the pit rim; with mining opex having been considered in definition of the optimized pit shell. One-time processing or G&A sustaining capex items were also excluded from the NSR cut-off.

Note that the 2017 Technical Report for the Mount Milligan Mine calculated an NSR cut-off of $6.25/t or C$8.12/t.

The Proven and Probable Mineral Reserve totals 191.0Mt at 0.23% Cu and 0.39g/t Au containing 959 million pounds of copper and 2.41 million ounces of gold (Table 1-3).

Table 1-3: Mineral Reserve Statement, Effective Date December 31, 2019

Mineral Reserve Category	Tonnes (000)	Copper Grade (%)	Gold Grade (g/t)	Contained Copper (Mlb)	Contained Gold (000 oz)
Proven	114,753	0.23	0.41	571	1,525
Probable	76,275	0.23	0.36	389	882
Proven + Probable	191,028	0.23	0.39	959	2,407

Notes:
(1) CIM definitions were followed for Mineral Reserves estimation.
(2) Mineral Reserves are estimated at $7.64/t (C$9.55/t) NSR cut-off value using metal prices of $3.00 per pound copper and $1,250 per ounce gold, and a US$/C$ exchange rate of US$1.00/C$1.25.
(3) Figures may not total exactly due to rounding.

The proven reserve estimate includes stockpiled material totalling 6,197kt with grades of 0.14% copper and 0.43g/t gold and contained metal of 19Mlb copper and 86koz gold. As such, the proven and probable reserve within the ultimate pit is 184.8Mt with grades of 0.23% copper and 0.39g/t gold and contained metal of 940Mlb copper and 2,321koz gold.

The end-2018 reserve estimate for Mount Milligan Mine was estimated to be 447.6Mt comprising 211.6Mt proven and 235.9Mt probable reserves. Adjusting this estimate for end-2019 open pit topography results in a total 426.1Mt proven and probable reserves. The following factors are estimated to account for the reduction from this figure to the end-2019 reserve estimate of approximately 185Mt within the ultimate pit:

- 87Mt decrease due to applying the December 31, 2019 resource model against the end-2018 ultimate pit design.
- 107Mt decrease due to applying costs per Table 15-1 to the December 31, 2019 resource model.
- 47Mt decrease due to applying metallurgical recoveries per Section 13 to the December 31, 2019 resource model.

The aforementioned decreases are cumulative to give the total 241Mt proven and probable reserve decrease from the end-2018 reserve estimate (adjusted to end-2019 open pit topography) to the end-2019 reserve estimate, with the latter based on the December 31, 2019 resource model. It should be noted that the order in which these factors are applied could affect the decrease estimated for each factor.

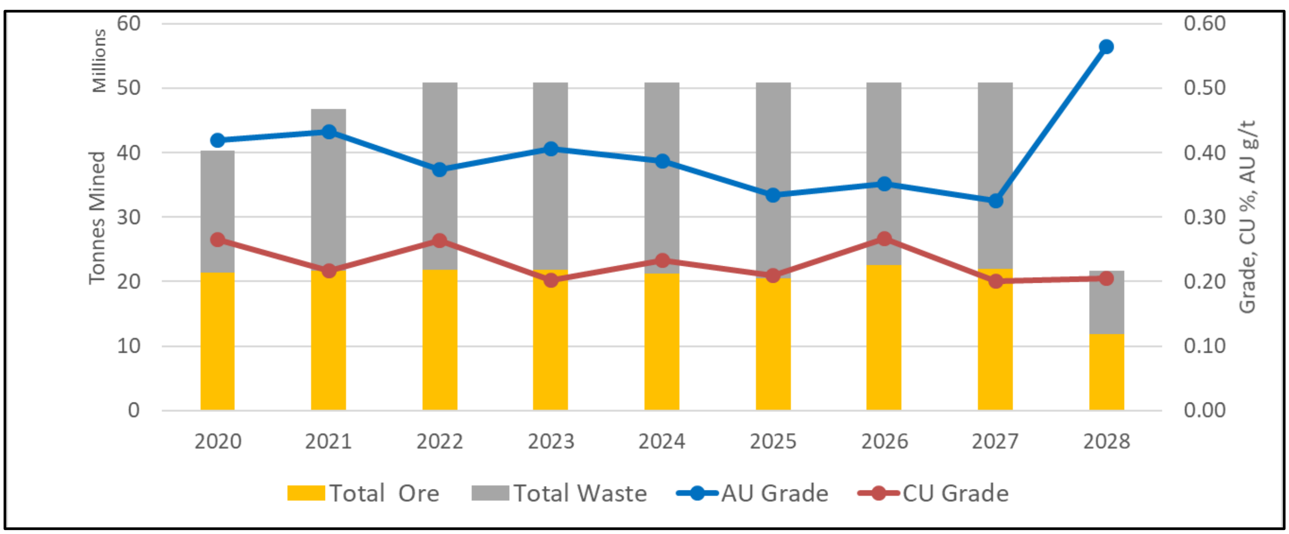
The Mineral Reserve estimate was prepared by Mount Milligan Mine and Centerra Corporate Technical Services personnel under the supervision of John Fitzgerald P.Eng., Vice President Projects & Technical Services for Centerra. The reserve estimate is effective December 31, 2019.

For clarity, the reserve estimate and resultant life-of-mine plan presented in this document is materially different to that presented in the 2017 Technical Report, with both significant reserves and mine life reductions. This is a result of re-evaluating all aspects of the resource model, updating cost estimates for all operating and sustaining capital cost items, updating metallurgical recoveries, and re-estimating reserves based on changes to modifying factors.

1.12. MINING

The mining operation is a conventional shovel and truck open pit mine feeding a 60,000 t/d (permitted throughput on an annualised basis) processing plant. The planned mine life is 9 years with a Proven and Probable Reserve of 191.0 million tonnes @ 0.23% copper and 0.39 g/t gold. The pit has been planned as a series of discrete pushbacks and scheduled to maximize the production of ore. Total ore and waste will be mined at an average rate of 40.2 Mt/a in 2020, 46.8 Mt/a in 2021 and 50.8 Mt/a in 2023 through 2027, decreasing to 21.6 Mt/a in 2028 yielding an overall LOM waste:ore ratio of 1.24:1.0. The mining sequence has been developed to allow for provision of suitable waste material for annual TSF construction requirements. Figure 1-4 provides the LOM mining schedule for ore and waste rock with the average estimated copper and gold grades on an annual basis.

Figure 1-4: Annual Material Mined



The mine currently employs 45 pieces of mobile production equipment comprised of three blasthole drills, two rope shovels, two rubber-tired front-end loaders, 15 haul trucks and various other dozers, loaders, graders and excavators. Over the remaining mine life, it is estimated that the peak haul truck fleet will need to increase to 20 units.

1.13.RECOVERY METHODS

The average process plant feed grade of 0.23% Cu is delivered throughout the LOM period at an average daily rate of 60,000 tonnes to yield a marketable 21.5% Cu concentrate. Process plant ore feed quality is maintained to honour metallurgical constraints such as ORE/HGLC ratio, Py:Cpy ratio and mercury (Hg) content. Average recovery to concentrate projected to be achieved during the LOM period is 80.6% for copper and 61.8% for gold.

The Mount Milligan process plant was originally designed to process ore at a nominal rate of 60,000 t/d, producing a marketable concentrate containing copper, gold, and silver. A secondary crushing circuit, installed in 2016, together with process plant optimization projects, increased the potential throughput to a nominal rate of 62,500 t/d. Key process equipment consists of:

- Primary crushing plant with a 1.525 m x 2.794 m gyratory crusher;
- Secondary crushing plant with two cone crushers prior to the grinding circuit, each powered by one 1,000 kW motor;
- SAG/ball mill/pebble crusher grinding circuit comprised of one SAG mill, two ball mills and two cone crushers;
- A flotation circuit comprised of a total of 19 rougher, scavenger and cleaner cells; and
- Regrinding and gravity concentration circuits comprised of one tower mill, two IsaMills™ and one centrifugal gold concentrator.

1.14. TAILINGS STORAGE FACILITY

The TSF at the Mount Milligan Mine is designed to store tailings solids and potentially acid generating (PAG) and oxide/weathered waste rock materials in designated areas. The TSF embankment is constructed as a centreline dam using open pit overburden and non-acid generating (NAG) waste rock materials. Construction of each of the embankment stages is scheduled to correspond with material availability from the Open Pit and the projected rate of rise. There will be sufficient volume of waste material produced over the LOM to raise the tailings dam to the required final elevation of 1,101m.

From the process plant, two tailing streams — the rougher/scavenger tailings and the first cleaner/scavenger tailings — are deposited and stored in separate tailing storage areas within the TSF. The rougher-scavenger tailings contain mostly non-sulphide gangue minerals, while the cleaner scavenger tailings contain most of the sulphide gangue minerals. The latter is kept in a lined pond and underwater to prevent acid generation from the oxidation of the sulphide minerals.

1.15. WATER MANAGEMENT

Mount Milligan experienced low water storage volumes in 2016 that continued into the following year, resulting in processing operations shutdown in December 2017 through to early-2018. Since then, the mine has received authorization for surface water pumping from Philip Lake 1 and Rainbow Creek until 2021 when it is anticipated that long term surface water and groundwater sources will be determined and authorized. In addition to surface water resources, groundwater sources have been located and authorized to augment TSF water volumes.

The economic analysis includes capex and opex totaling $41.5 millions related to external water supply costs.

1.16. ENVIRONMENT AND PERMITTING

Mount Milligan Mine currently holds or is in the process of obtaining all permits required for the operation of its business for the defined life-of-mine.

1.17.COMMUNITY SUSTAINABILITY

In 2006, Terrane initiated a consultation program with local communities and Indigenous groups. In May 2008, Terrane convened a Community Sustainability Committee whose membership is comprised of regional community stakeholders and impacted Indigenous groups. The Committee acts as TCM’s primary mechanism for community engagement concerning the mine’s activities and investments into the region and continues to meet on a quarterly basis. A Sustainability Management Plan for the mine, which was reviewed and approved by the Committee, remains in place.

TCM is also party to a Socio-Economic Agreement with the McLeod Lake Indian Band and an Impact Benefit Agreement (IBA) with Nak’azdli Whut’en. Both agreements commit the Company to the provision of financial payments and these amounts have been incorporated into the economic analysis in this Technical Report.

1.18. CAPITAL AND OPERATING COSTS

Total operating and capital costs over Mount Milligan's 9-year LOM are estimated at $2,839 million, including $828 million for mining, $1,029 million for processing, $333 million for administration (G&A), $140 million for transportation costs, selling and marketing costs of $88 million, treatment and refining charges of $199 million and capital expenditures of $222 million, as shown in Table 1-4.

The LOM capital expenditures required to exploit the Mineral Reserves in the LOM plan is estimated at $222 million, which includes capital equipment and component replacements, planned improvements to crushing equipment, the tailings pumping system, and site facilities, as well as water management, but excludes $125 million TSF construction costs (included in mine opex). Waste mined at Mount Milligan is used for routine TSF raises, the cost of which is capitalized to the TSF rather than as capitalized stripping. The current mine plan does not contemplate any growth capital.

Table 1-4: Operating and Capital Cost Summary

Costs Summary (Total LOM)	Total $M
Mining	828
Processing	1,029
Admin	333
Transportation	140
Selling and Marketing	88
Treatment and Refining	199
Capital	222
Total	**2,839**

The all-in sustaining cost per ounce sold, on a by-product basis, which includes sustaining capital and takes into account copper revenue as a credit, averages $704/oz of gold for the period from 2020 to the end of the LOM. All-in sustaining cost per ounce sold, on a by-product basis, is a non-GAAP financial performance measure. For further information please see the Non-GAAP Measures section of this document.

1.19. FINANCIAL EVALUATION

Using a copper price of $3.00/lb, gold price of $1,250/oz and C$:US$ exchange rate of 1.25:1.00, as assumed for the Mineral Reserve estimation process, over its 9-year LOM Mount Milligan is expected to generate $3.09 billion of revenue, net of streaming obligations and smelter fees. Net free cash flow on an undiscounted basis for the Mount Milligan Mine from 2020 to the end of its mine life in

2028, but accounting for the cost of closure, is estimated to be $398M; this is a non-GAAP financial performance measure. The net present value (“NPV”) is $342M after tax at a 5% discount rate. Table 1-5 provides a summary of economic analysis results.

Table 1-5: Economic Analysis Summary

Item	Total ($000)
Net Gold Revenue	1,410,388
Net Copper Revenue	1,683,351
Total Revenue	3,093,739
Cash Flow from Operating Activities	748,587
Capital	-346,277
Free Cash Flow (before financing)	402,310

1.20.INTERPRETATION AND CONCLUSIONS

Based on the information contained herein, the Qualified Persons, as authors of this Technical Report, offer the following interpretations and conclusions.

Geology and Mineral Resources

Mount Milligan is a roughly tabular, near-surface, alkalic copper-gold porphyry deposit currently being mined.

The procedures for drilling, sampling, sample preparation and analyses are appropriate for the type of mineralization and estimation of Mineral Resources.

Combined Measured and Indicated Mineral Resources, exclusive of Mineral Reserves, total 125.4Mt at 0.19% Cu and 0.35g/t Au containing 517Mlb copper and 1.4Moz gold. Inferred Mineral Resources total 3.7Mt at 0.12% Cu and 0.46g/t Au containing 10Mlb copper and 0.05Moz gold.

The classification of Mineral Resources conforms to CIM Definition Standards.

Mineral Resources were estimated as of December 31, 2019 within a conceptual open pit shell using metal prices of $3.50/lb copper and $1,500/oz gold, and are reported exclusive of the Mineral Reserve. The estimate takes into consideration factors such as metallurgical recoveries, concentrate grades, transportation costs, smelter treatment charges, and royalty and streaming arrangements in determining economic viability.

Mining and Mineral Reserves

Proven and Probable Mineral Reserves total 191.0Mt at 0.23% Cu and 0.39g/t Au containing 959Mlb copper and 2.4Moz gold, effective 31st December 2019. This estimate was derived through the design of an ultimate pit, conforming to practical mining requirements, based on an optimized pit shell using metal prices of US$3.00/lb Cu and US$1,250/oz Au and an exchange rate of C$:US$ of 1.25:1.00.

The Mineral Reserve estimate has been prepared using industry standard best practise methodologies with the classification of Proven and Probable Mineral Reserves conforming to CIM definitions and NI 43-101 requirements.

Mining is carried out using a conventional drill-blast, load and haul approach. All waste rock is either used for TSF dam construction or stored within the TSF to comply with environmental approvals requirements.

The aforementioned Mineral Reserve estimate is significantly lower than that reported at end-2018 which was 447.6Mt at 0.19% Cu and 0.33g/t Au containing 1,836Mlb copper and 4.7Moz Au. The causes of this change are summarised in section 1.11.

Mineral Processing

Metallurgical recoveries are derived from operating results for the current process plant flowsheet configuration.

This LOM update assumes that an average 60kt/d process plant throughput will be achieved from 2021 onwards. While this has not yet been achieved over a calendar year, various operational and maintenance improvement initiatives are in progress with this throughput goal as the ultimate objective; including improvements to the secondary crushing circuit and SAG mill liner performance.

In late-2017, low water inventories resulted in a temporary shutdown of the process plant. During winter 2018-2019, a shutdown was avoided but throughput was decreased to align with available water supply. The risk to throughput due to low water inventories in the TSF is being mitigated by additional wells in the TSF and the newly commissioned Lower Rainbow Valley Groundwater Well Field.

Economic Analysis

Capital costs have been estimated for a nine-year mine life and operating costs have been estimated for an average 60,000t/d process plant feed rate. Operating cost estimates were developed from first principles using site historic costs as a basis for calibrating the estimates.

The base case economic assumptions align with parameters used for Mineral Reserve estimation including metal prices of $1,250/oz Au and $3.00/lb Cu, and a CAD:USD exchange rate of 1.25.

The life-of-mine all-in sustaining cost per ounce sold, on a by-product basis, before taxes is estimated at $704/oz gold sold. This is a non-GAAP financial performance measure.

The NPV for the base case model is $342M after tax at a 5% discount rate.

Risks and Mitigation

The Board of Directors for Centerra has created a Risk Committee whose mandate is Enterprise Risk Management (ERM) governance and oversight. The VP Risk & Insurance at Centerra prepares and presents a quarterly report for the Risk Committee on the key strategic, operational, project and exploration risks, as well as emerging risks.

A risk register has been created for Mount Milligan Mine which includes the following subjects:

- Health and safety
- Water management
- Mine and process plant production
- Infrastructure
- Community and Indigenous relations
- Capital and operating costs
- Metallurgy
- Site environmental
- Mining
- Human Resources
- Surrounding environmental
- Transportation

Section 26 provides a discussion on risks, risk mitigation and the current Risk Register.

1.21. RECOMMENDATIONS

The following recommendations are provided by the various Qualified Persons.

Mineral Exploration

It is recommended that mineral exploration continue targeting both of the mineralization styles that have been identified in the Mount Milligan deposit; early stage porphyry Au-Cu and late stage structurally controlled high-gold low-copper (HGLC). This should be done through continued development of a comprehensive 3D exploration model developed and updated from information in the drilling database which compiles lithology, alteration, mineralization, structural and geophysical data. The 3D model will improve understanding of the geometry of the deposit from fault block to fault block and better determine controls on both styles of mineralization and where to find extensions of known mineralization that could potentially be added to resources and reserves.

Ongoing exploration programs should continue to identify, delineate and test target zones within the three strategic mineral exploration domains: Near-pit infill/expansion, Brownfield (outside the ultimate pit boundary but inside the mine lease) and Greenfield (outside the mine lease). Budgets for programs over the next three years should be on par with Centerra programs in recent years (2018-2020). Annually, these have been $5.5-6.5 million comprised of core drilling (30,000 to 43,000 m), geophysics surveys (15 to 60 line-km IP surveys and other surveys) and in-house team building.

Mineral Resources and Grade Control

The following are recommendations related to the resource estimate and grade control:

- Reconciliation of the head grade obtained from the recommended test bench to the 31st December 2019 resource model.
- Additional work to improve local grade estimation (Cu:Au correlation) within the resource model, including remodeling the current domain interpretation.
- Grade control testing and short-term block model development.
- Additional drilling to determine the extension of mineralization along the western ultimate pit wall.
- Progress the geometallurgical program with compilation of a block model.

Water Management

Continue to identify water sources for LOM operations and progress related approvals in consultation with Indigenous partners.

Continue to monitor water supply, storage and use very closely to minimize potential process plant downtime due to lack of water.

Mine Planning and Operations

Evaluate using an optimized pit with a lower revenue factor as the basis for designing the ultimate pit, which is expected to reduce the strip ratio and possibly provide a more manageable balance between ore and waste tonnes. However, this is also likely to decrease the mined ore tonnes and may reduce the mine life.

Re-work the mining schedule, including considerations for additional waste rock stockpiles, to provide adequate TSF dam building material for each construction season.

Evaluate opportunities to further improve mining efficiencies and decrease opex, such as through the application of automation.

Evaluate opportunities for in-pit waste and tailings storage.

Mineral Processing and Metallurgy

Evaluate use of alternative flotation equipment such as Staged Flotation Reactors or Direct Flotation Reactors. An initial assessment for the Mt Milligan flowsheet and ore has shown potential to increase both gold and copper metallurgical recoveries.

Evaluate alternative flowsheets for the treatment of high-gold low-copper (HGLC) and high Py:Cpy ratio ore to improve metal recoveries. This includes the use of flowsheets using cyanide to enhance the metallurgical recovery of gold.

Evaluate tailings re-processing, an opportunity that was previously assessed at a high level.

Operation in General

Evaluate increasing the mining rate and process plant throughput to reduce unit costs which could increase the portion of the mineral resource which can be extracted profitably.

2. INTRODUCTION

This Technical Report, which was prepared by and for Centerra, summarizes the current and planned operations and the Mineral Reserves and Mineral Resources for the Mount Milligan Mine located between Fort St James and Mackenzie, British Columbia, Canada.

Centerra acquired its indirect 100% interest in the Mount Milligan Mine on October 20, 2016 when it acquired (through a subsidiary) TCM (the Acquisition). Prior to the Acquisition, TCM held the Property through its wholly owned subsidiary Terrane, which it acquired in 2010. As part of the Acquisition, TCM and Terrane amalgamated. The Project was constructed by TCM prior to the Acquisition and commenced commercial production in February 2014.

When used in this Technical Report, “TCM” means the company existing prior to the Acquisition, unless otherwise noted.

Centerra, a global mining company organized under the laws of Canada, is engaged in the acquisition, exploration, development, and operation of mineral properties. Centerra’s shares are listed on the Toronto Stock Exchange under the trading symbol “CG”.

Items of significant change from the 2017 Technical Report include copper and gold price assumptions, metallurgical recovery estimates, capital and operating costs, increased NSR cut-off value and geologic model updates, which have resulted in an updated Mineral Resource and Mineral Reserve estimate and a new ultimate pit design and mining-processing schedule. As per the 2017 Technical Report, the stream arrangement with Royal Gold and H.R.S. Resources royalty are incorporated into this Technical Report, which impacts on the Project’s expected cash flows over the life-of-mine (LOM).

All dollar figures in this Technical Report refer to US dollars, unless otherwise noted.

2.1. SOURCES OF INFORMATION

This Technical Report is based on published material and data, professional opinions, and unpublished materials available to Centerra or prepared by its employees. In addition, certain information used to support this Technical Report was derived from previous technical reports on the Project and from reports and documents listed in the References section. Other sources of data include geologic and block model reports, drill hole assay data, the block model, mine plans, cost estimates and economic models which were prepared by employees of Centerra.

This Technical Report has been prepared in compliance with NI 43-101 and follows the format set out in Form 43-101F1 for Technical Reports.

2.2. CONTRIBUTING PERSONS AND SITE INSPECTIONS

This Technical Report has been prepared by the persons listed in Table 2-1, each of whom is a Qualified Person (QP) as defined in NI 43-101 and has provided a QP certificate. All QPs have visited the Mount Milligan Mine.

Other Centerra employees compiled certain sections of this Technical Report under the supervision of those identified in Table 2-1. These Centerra employees are experienced technical and accounting/finance professionals in their respective areas of expertise.

Standard professional procedures have been followed in preparing the contents of this Technical Report. Data used in this Technical Report have been verified, where possible, and all data is considered to have been collected in a professional manner.

Table 2-1: Qualified Persons and Responsibilities

Qualified Person	Title	Primary Area(s) of Responsibility	Technical Report Sections Authored
John Fitzgerald (P.Eng), Centerra	Vice President, Capital Projects & Technical Services	Mineral reserves, mining, cost estimates, economic analysis, water management, environment, sustainability.	1.1, 1.11, 1.12, 1.15-1.21, 2-5, 15, 16, 18.1, 18.3, 19-24, 25.2, 25.4, 27.3 and 27.4.
Paul Jago (P.Geo), Centerra Gold Services Inc	Exploration Manager	Geology, Exploration, Drilling, and Data Verification.	1.2-1.8, 1.20, 1.21, 4.1, 6-12 and 27.1.
Berge Simonian (P.Eng), Thompson Creek Metals Inc	Chief Metallurgist	Mineral processing and metallurgy.	1.9, 1.13, 1.20, 1.21, 13, 17, 25.3 and 27.5.
Slobodan Jankovic (P.Geo), Centerra	Sr. Director, Technical Services	Mineral Resource Estimate.	1.10, 1.20, 1.21, 14, 25.1 and 27.2.
Cathy Taylor (P.Eng. ARM), Centerra	Vice President, Risk & Insurance	Risk and Mitigation.	1.20, 25.5 and 26.
Bruno Borntraeger (P.Eng), Knight Piesold Consulting	Specialist Geotechnical Engineer	Tailings Storage Facility dam.	1.14 and 18.2.

2.3. UNITS

This Technical Report utilizes metric units throughout as set forth in the Glossary included in Section 28.1. Grades are in percent of copper metal by weight and grams per tonne (g/t) for gold. Tonnages are metric tonnes of 2,204.6 pounds. Gold sales in units of troy ounces with a conversion of 31.1 grams per troy ounce.

3. RELIANCE ON OTHER EXPERTS

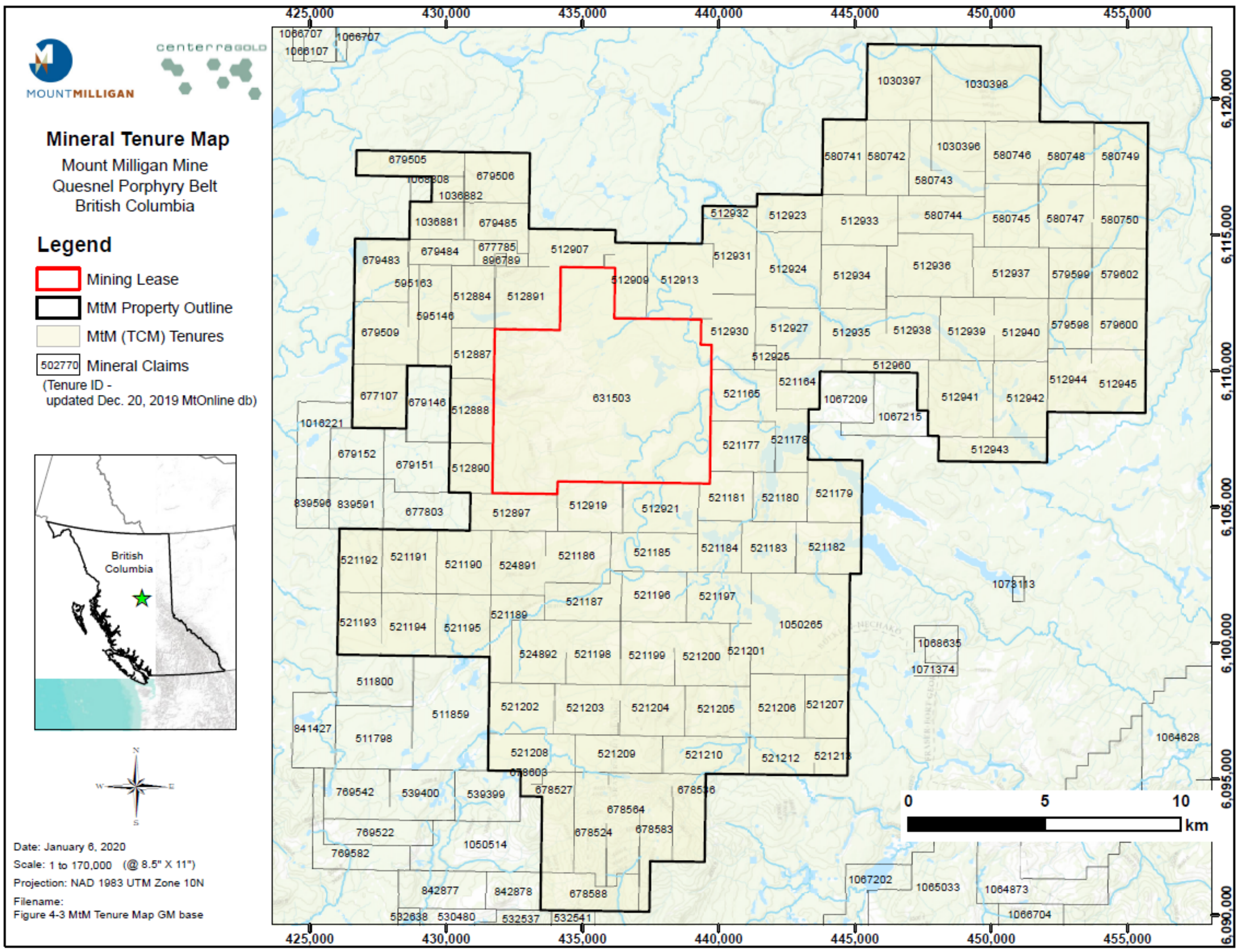
Mount Milligan Mine relies on various specialist consultants to provide advice on many aspects of the operation and support services. One such consultant, Bruno Borntraeger, Knight Piesold Consulting, has provided content for this technical report relating to the Tailings Storage Facility (TSF). No other external consultants have provided content for this technical report.

4. PROPERTY DESCRIPTION AND LOCATION

4.1. PROPERTY DESCRIPTION

The Mount Milligan mine is situated in Mining Lease 631503 completely surrounded by many mineral claims which form the Mount Milligan Property. Figure 4-1 is a plan view of Mineral Tenure Map.

Figure 4-1: Mount Milligan Mineral Tenure Map



A complete list of mineral tenures is depicted on Table 4-1, which is shown on the following two pages.

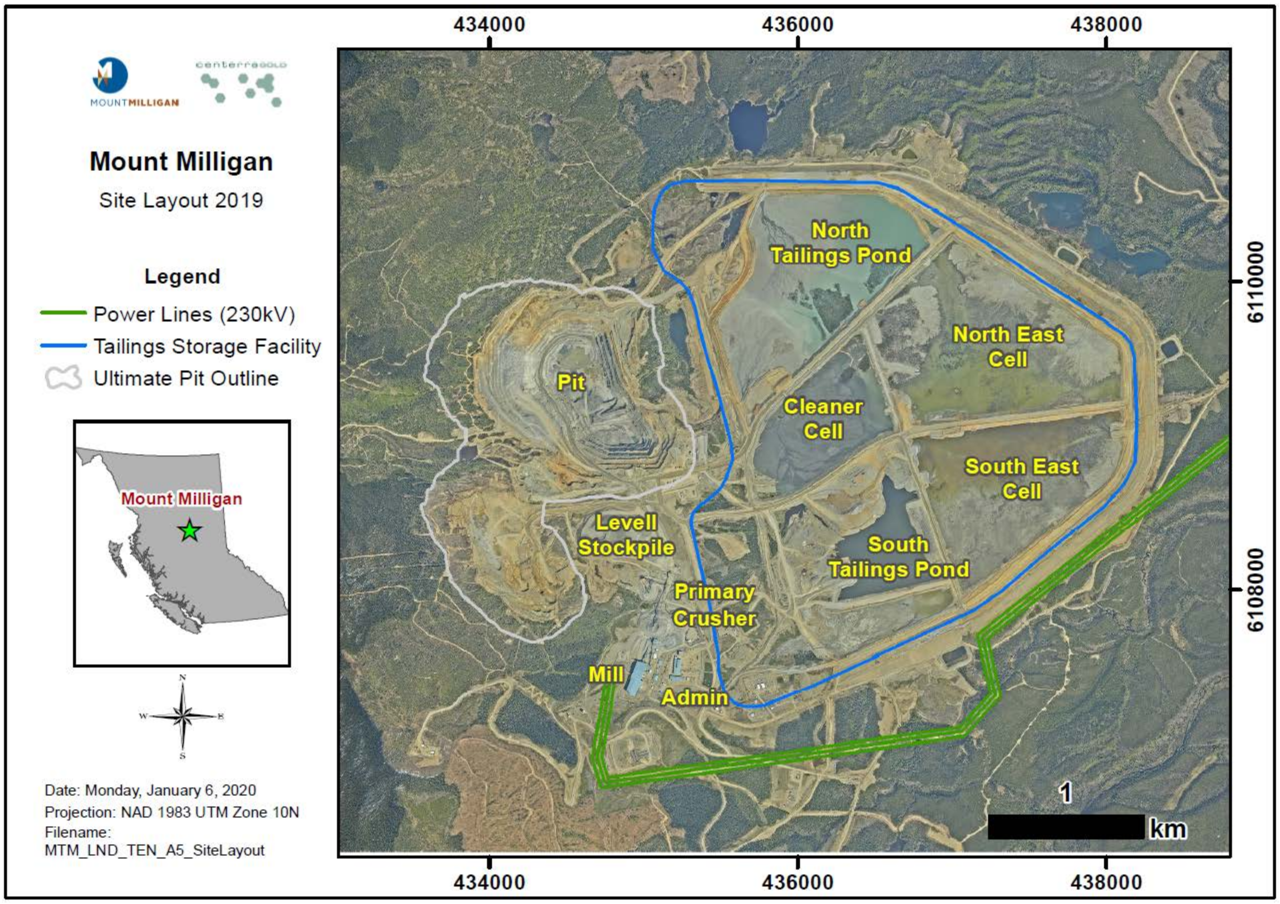
Table 4-1: List of Mineral Tenures

Title Number	Claim Name	Owner	Title Type	Title Sub Type	Map Number	Issue Date	Good To Date	Status	Area (ha)
512884		283374 (100%)	Mineral	Claim	093N	2005/MAY/18	2024/MAR/14	GOOD	369.6
512887		283374 (100%)	Mineral	Claim	093N	2005/MAY/18	2024/MAR/14	GOOD	295.8
512888		283374 (100%)	Mineral	Claim	093N	2005/MAY/18	2024/MAR/14	GOOD	370.0
512890		283374 (100%)	Mineral	Claim	093N	2005/MAY/18	2024/MAR/14	GOOD	296.1
512891		283374 (100%)	Mineral	Claim	093N	2005/MAY/18	2023/MAR/14	GOOD	554.4
512897		283374 (100%)	Mineral	Claim	093N	2005/MAY/18	2024/MAR/14	GOOD	444.3
512907		283374 (100%)	Mineral	Claim	093N	2005/MAY/18	2023/MAR/14	GOOD	424.9
512909		283374 (100%)	Mineral	Claim	093N	2005/MAY/18	2023/MAR/14	GOOD	351.1
512913		283374 (100%)	Mineral	Claim	093O	2005/MAY/18	2023/MAR/14	GOOD	665.2
512919		283374 (100%)	Mineral	Claim	093N	2005/MAY/18	2023/MAR/14	GOOD	444.3
512921		283374 (100%)	Mineral	Claim	093O	2005/MAY/18	2023/MAR/14	GOOD	518.4
512923		283374 (100%)	Mineral	Claim	093O	2005/MAY/18	2023/MAR/14	GOOD	332.4
512924		283374 (100%)	Mineral	Claim	093O	2005/MAY/18	2023/MAR/14	GOOD	665.2
512925		283374 (100%)	Mineral	Claim	093O	2005/MAY/18	2023/MAR/14	GOOD	74.0
512927		283374 (100%)	Mineral	Claim	093O	2005/MAY/18	2023/MAR/14	GOOD	406.7
512930		283374 (100%)	Mineral	Claim	093O	2005/MAY/18	2023/MAR/14	GOOD	480.6
512931		283374 (100%)	Mineral	Claim	093O	2005/MAY/18	2023/MAR/14	GOOD	480.3
512932		283374 (100%)	Mineral	Claim	093O	2005/MAY/18	2023/MAR/14	GOOD	92.3
512933		283374 (100%)	Mineral	Claim	093O	2005/MAY/18	2023/MAR/14	GOOD	517.1
512934		283374 (100%)	Mineral	Claim	093O	2005/MAY/18	2024/MAR/14	GOOD	554.3
512935		283374 (100%)	Mineral	Claim	093O	2005/MAY/18	2024/MAR/14	GOOD	443.7
512936		283374 (100%)	Mineral	Claim	093O	2005/MAY/18	2024/MAR/14	GOOD	720.6
512937		283374 (100%)	Mineral	Claim	093O	2005/MAY/18	2023/MAR/14	GOOD	517.3
512938		283374 (100%)	Mineral	Claim	093O	2005/MAY/18	2023/MAR/14	GOOD	462.1
512939		283374 (100%)	Mineral	Claim	093O	2005/MAY/18	2023/MAR/14	GOOD	462.1
512940		283374 (100%)	Mineral	Claim	093O	2005/MAY/18	2023/MAR/14	GOOD	462.1
512941		283374 (100%)	Mineral	Claim	093O	2005/MAY/18	2023/MAR/14	GOOD	665.9
512942		283374 (100%)	Mineral	Claim	093O	2005/MAY/18	2023/MAR/14	GOOD	554.9
512943		283374 (100%)	Mineral	Claim	093O	2005/MAY/18	2023/MAR/14	GOOD	370.1
512944		283374 (100%)	Mineral	Claim	093O	2005/MAY/18	2023/MAR/14	GOOD	369.9
512945		283374 (100%)	Mineral	Claim	093O	2005/MAY/18	2023/MAR/14	GOOD	462.3
512960		283374 (100%)	Mineral	Claim	093O	2005/MAY/18	2023/MAR/14	GOOD	203.4
521164	MILL 1	283374 (100%)	Mineral	Claim	093O	2005/OCT/14	2024/MAR/14	GOOD	332.9
521165	MILL 2	283374 (100%)	Mineral	Claim	093O	2005/OCT/14	2023/MAR/14	GOOD	443.9
521177	MILL 3	283374 (100%)	Mineral	Claim	093O	2005/OCT/14	2024/MAR/14	GOOD	444.1
521178	MILL 4	283374 (100%)	Mineral	Claim	093O	2005/OCT/14	2023/MAR/14	GOOD	277.5
521179	MILL 5	283374 (100%)	Mineral	Claim	093O	2005/OCT/14	2023/MAR/14	GOOD	462.8
521180	MILL 6	283374 (100%)	Mineral	Claim	093O	2005/OCT/14	2023/MAR/14	GOOD	370.2
521181	MILL 7	283374 (100%)	Mineral	Claim	093O	2005/OCT/14	2024/MAR/14	GOOD	351.7
521182	MILL 8	283374 (100%)	Mineral	Claim	093O	2005/OCT/14	2023/MAR/14	GOOD	444.4
521183	MILL 9	283374 (100%)	Mineral	Claim	093O	2005/OCT/14	2023/MAR/14	GOOD	370.4
521184	MILL10	283374 (100%)	Mineral	Claim	093O	2005/OCT/14	2024/MAR/14	GOOD	296.3
521185	MILL 11	283374 (100%)	Mineral	Claim	093O	2005/OCT/14	2023/MAR/14	GOOD	444.5
521186	MILL 12	283374 (100%)	Mineral	Claim	093N	2005/OCT/14	2023/MAR/14	GOOD	444.5
521187	MILL 13	283374 (100%)	Mineral	Claim	093N	2005/OCT/14	2023/MAR/14	GOOD	407.6
521189	MILL 14	283374 (100%)	Mineral	Claim	093N	2005/OCT/14	2023/MAR/14	GOOD	370.6
521190	MILL 15	283374 (100%)	Mineral	Claim	093N	2005/OCT/14	2023/MAR/14	GOOD	463.0
521191	MILL 16	283374 (100%)	Mineral	Claim	093N	2005/OCT/14	2023/MAR/14	GOOD	463.0
521192	MILL 17	283374 (100%)	Mineral	Claim	093N	2005/OCT/14	2023/MAR/14	GOOD	370.4
521193	MILL 18	283374 (100%)	Mineral	Claim	093N	2005/OCT/14	2023/MAR/14	GOOD	370.6
521194	MILL 19	283374 (100%)	Mineral	Claim	093N	2005/OCT/14	2023/MAR/14	GOOD	463.3
521195	MILL 20	283374 (100%)	Mineral	Claim	093N	2005/OCT/14	2023/MAR/14	GOOD	463.3
521196	MILL 21	283374 (100%)	Mineral	Claim	093O	2005/OCT/14	2023/MAR/14	GOOD	444.6
521197	MILL 22	283374 (100%)	Mineral	Claim	093O	2005/OCT/14	2023/MAR/14	GOOD	444.6
521198	MILL 23	283374 (100%)	Mineral	Claim	093N	2005/OCT/14	2023/MAR/14	GOOD	463.4
521199	MILL 24	283374 (100%)	Mineral	Claim	093O	2005/OCT/14	2023/MAR/14	GOOD	463.4
521200	MILL 25	283374 (100%)	Mineral	Claim	093O	2005/OCT/14	2023/MAR/14	GOOD	463.4
521201	MILL 26	283374 (100%)	Mineral	Claim	093O	2005/OCT/14	2023/MAR/14	GOOD	185.4

Title Number	Claim Name	Owner	Title Type	Title Sub Type	Map Number	Issue Date	Good To Date	Status	Area (ha)
521202	MILL 27	283374 (100%)	Mineral	Claim	093N	2005/OCT/14	2023/MAR/14	GOOD	445.0
521203	MILL 28	283374 (100%)	Mineral	Claim	093N	2005/OCT/14	2023/MAR/14	GOOD	445.0
521204	MILL 29	283374 (100%)	Mineral	Claim	093O	2005/OCT/14	2023/MAR/14	GOOD	445.0
521205	MILL 30	283374 (100%)	Mineral	Claim	093O	2005/OCT/14	2023/MAR/14	GOOD	445.0
521206	MILL 31	283374 (100%)	Mineral	Claim	093O	2005/OCT/14	2023/MAR/14	GOOD	463.6
521207	MILL 32	283374 (100%)	Mineral	Claim	093O	2005/OCT/14	2023/MAR/14	GOOD	370.9
521208	MILL 33	283374 (100%)	Mineral	Claim	093N	2005/OCT/14	2023/MAR/14	GOOD	445.2
521209	MILL 34	283374 (100%)	Mineral	Claim	093N	2005/OCT/14	2023/MAR/14	GOOD	445.2
521210	MILL 35	283374 (100%)	Mineral	Claim	093O	2005/OCT/14	2023/MAR/14	GOOD	445.2
521212	MILL 36	283374 (100%)	Mineral	Claim	093O	2005/OCT/14	2023/MAR/14	GOOD	333.9
521213	MILL 37	283374 (100%)	Mineral	Claim	093O	2005/OCT/14	2023/MAR/14	GOOD	167.0
524891	ARM	283374 (100%)	Mineral	Claim	093N	2006/JAN/08	2023/MAR/14	GOOD	463.0
524892	STRONG	283374 (100%)	Mineral	Claim	093N	2006/JAN/08	2023/MAR/14	GOOD	463.4
579598		283374 (100%)	Mineral	Claim	093O	2008/MAR/28	2023/MAR/14	GOOD	295.8
579599		283374 (100%)	Mineral	Claim	093O	2008/MAR/28	2023/MAR/14	GOOD	295.6
579600		283374 (100%)	Mineral	Claim	093O	2008/MAR/28	2023/MAR/14	GOOD	369.7
579602		283374 (100%)	Mineral	Claim	093O	2008/MAR/28	2023/MAR/14	GOOD	369.5
580741		283374 (100%)	Mineral	Claim	093O	2008/APR/08	2023/MAR/14	GOOD	443.0
580742		283374 (100%)	Mineral	Claim	093O	2008/APR/08	2023/MAR/14	GOOD	443.0
580743		283374 (100%)	Mineral	Claim	093O	2008/APR/08	2023/MAR/14	GOOD	406.1
580744		283374 (100%)	Mineral	Claim	093O	2008/APR/08	2023/MAR/14	GOOD	461.7
580745		283374 (100%)	Mineral	Claim	093O	2008/APR/08	2023/MAR/14	GOOD	461.7
580746		283374 (100%)	Mineral	Claim	093O	2008/APR/08	2023/MAR/14	GOOD	461.5
580747		283374 (100%)	Mineral	Claim	093O	2008/APR/08	2023/MAR/14	GOOD	461.7
580748		283374 (100%)	Mineral	Claim	093O	2008/APR/08	2023/MAR/14	GOOD	461.5
580749		283374 (100%)	Mineral	Claim	093O	2008/APR/08	2023/MAR/14	GOOD	461.5
580750		283374 (100%)	Mineral	Claim	093O	2008/APR/08	2023/MAR/14	GOOD	461.7
595146		283374 (100%)	Mineral	Claim	093N	2008/DEC/01	2023/MAR/14	GOOD	443.6
595163		283374 (100%)	Mineral	Claim	093N	2008/DEC/01	2023/MAR/14	GOOD	147.9
631503		283374 (100%)	Mineral	Lease	093N	2009/SEP/09	2020/SEP/09	GOOD	5,138.0
677107	FURB	283374 (100%)	Mineral	Claim	093N	2009/DEC/01	2023/MAR/14	GOOD	462.4
677785		283374 (100%)	Mineral	Claim	093N	2009/DEC/02	2024/MAR/14	GOOD	147.8
678524		283374 (100%)	Mineral	Claim	093K	2009/DEC/03	2023/MAR/14	GOOD	464.0
678527		283374 (100%)	Mineral	Claim	093K	2009/DEC/03	2023/MAR/14	GOOD	464.0
678536		283374 (100%)	Mineral	Claim	093J	2009/DEC/03	2023/MAR/14	GOOD	389.7
678564		283374 (100%)	Mineral	Claim	093J	2009/DEC/03	2023/MAR/14	GOOD	464.0
678583		283374 (100%)	Mineral	Claim	093J	2009/DEC/03	2023/MAR/14	GOOD	464.0
678588		283374 (100%)	Mineral	Claim	093J	2009/DEC/03	2023/MAR/14	GOOD	464.3
678603		283374 (100%)	Mineral	Claim	093K	2009/DEC/03	2023/MAR/14	GOOD	55.7
679483		283374 (100%)	Mineral	Claim	093N	2009/DEC/05	2023/MAR/14	GOOD	461.9
679484		283374 (100%)	Mineral	Claim	093N	2009/DEC/05	2023/MAR/14	GOOD	221.7
679485		283374 (100%)	Mineral	Claim	093N	2009/DEC/05	2023/MAR/14	GOOD	350.9
679505		283374 (100%)	Mineral	Claim	093N	2009/DEC/05	2023/MAR/14	GOOD	369.2
679506		283374 (100%)	Mineral	Claim	093N	2009/DEC/05	2023/MAR/14	GOOD	443.1
679509		283374 (100%)	Mineral	Claim	093N	2009/DEC/05	2023/MAR/14	GOOD	462.2
896789	MILL 9	283374 (100%)	Mineral	Claim	093N	2011/SEP/13	2023/MAR/14	GOOD	18.5
1030396	GD1	283374 (100%)	Mineral	Claim	093O	2014/AUG/19	2023/MAR/14	GOOD	369.2
1030397	GD2	283374 (100%)	Mineral	Claim	093O	2014/AUG/19	2023/MAR/14	GOOD	664.1
1030398	GD3	283374 (100%)	Mineral	Claim	093O	2014/AUG/19	2023/MAR/14	GOOD	1,106.9
1036881	DB1	283374 (100%)	Mineral	Claim	093N	2015/JUN/23	2023/MAR/14	GOOD	277.1
1036882	DB2	283374 (100%)	Mineral	Claim	093N	2015/JUN/23	2023/MAR/14	GOOD	110.8
1050265		283374 (100%)	Mineral	Claim	093O	2017/FEB/24	2023/MAR/14	GOOD	1,334.2
110	**Total mineral tenures**							**Total ha.**	**51,078.3**

The mine site comprises an open pit mine, tailings storage facility (TSF), mineralized stockpiles, a processing plant, workshop, warehouse, administration buildings and camp. Figure 4-2 provides a plan view of the Project.

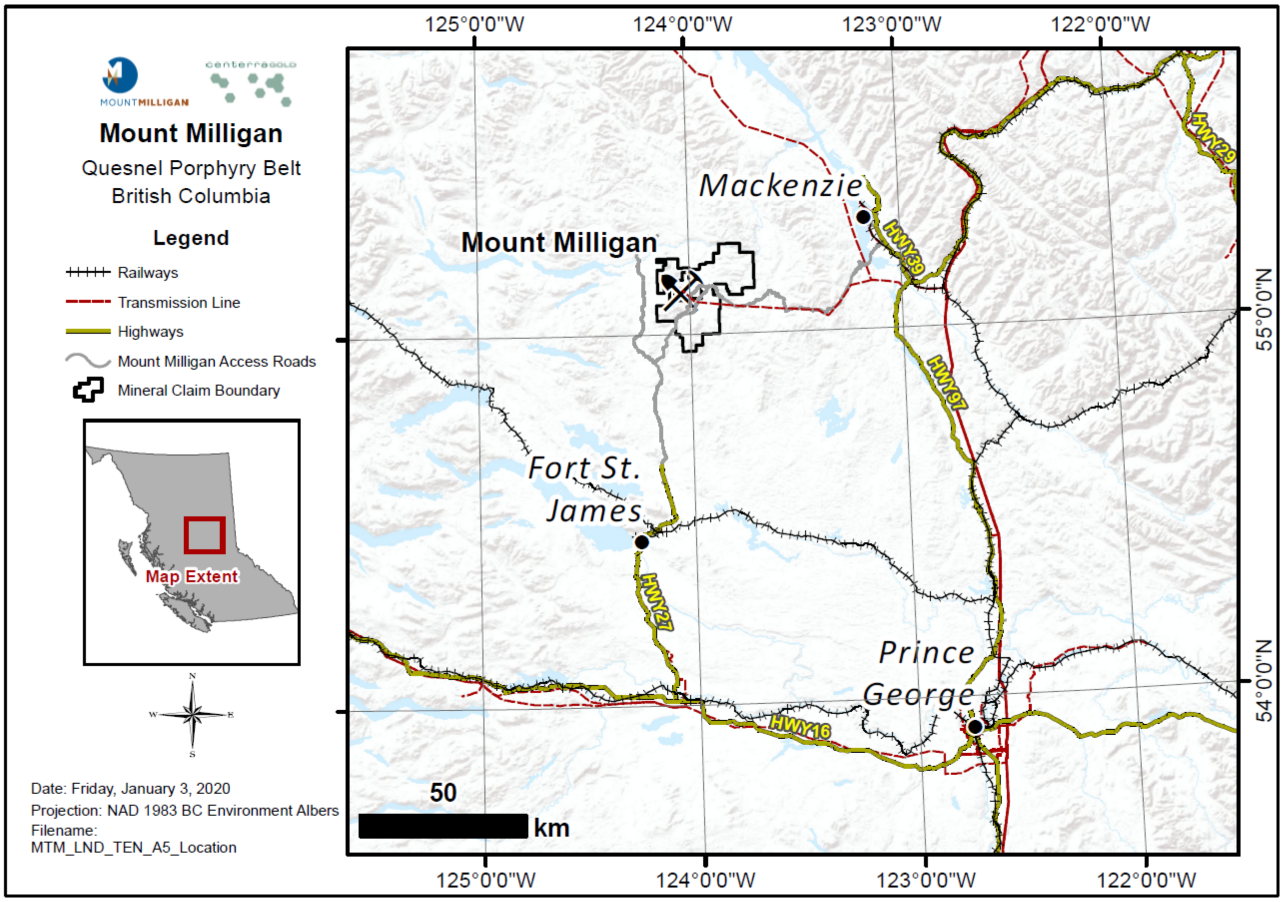
Figure 4-2: Plan View of Mount Milligan Mine Site



4.2. LOCATION

The Mount Milligan Mine is located in north-central British Columbia, Canada as shown in Figure 4-3.

Figure 4-3: Mount Milligan Mine Location



The mine is accessible from the east via Mackenzie on the Finlay Philip Forest Service Road and the North Philip Forest Service Road. There is active logging in the area, and therefore, the road is maintained in good condition by the various user groups. A western route, completed in 2005, provides shorter access from Fort St. James via the North Germansen Road. This route includes 26.7 km of forest service roads, with the balance on public roads. Road travel to the site of the Mount Milligan Mine is 254 km from Prince George (population approximately 79,000).

4.3. ROYALTIES AND STREAMING AGREEMENTS

H.R.S. Resources Royalty

In accordance with an option agreement dated July 16, 1986, as amended, between H.R.S. Resources Corp., successor in interest to Richard Haslinger, and TCM as the successor in interest to Goldcorp Canada Ltd., H.R.S. Resources Corp. is entitled to a royalty equivalent to a 2% NSR on production from four mineral claims collectively called the HEIDI claims. The HEIDI claims form a portion of the mining lease. In accordance with the terms of the royalty agreement, the royalty has been payable on concentrate produced since June 25, 2016 (of the third year of commercial production). TCM has the right of first refusal on any proposed sale of the royalty by H.R.S. Resources Corp.

Stream Agreement with Royal Gold

Pursuant to an agreement dated October 2010, as subsequently amended in December 2011, August 2012, December 2014 and October 20, 2016 (the Stream Agreement), with RGLD AG and Royal Gold, Inc. (collectively Royal Gold), TCM has agreed to sell to Royal Gold 35% of the gold produced and 18.75% of the copper production at the Mount Milligan Mine. Royal Gold pays $435 per ounce of gold delivered and pays 15% of the spot price per metric tonne of copper delivered. When the Stream Agreement was originally entered into by TCM (prior to the Acquisition by Centerra), TCM received an upfront payment of $781.5 million for the rights to receive future gold production. The Stream Agreement was amended by TCM as part of the Acquisition to reduce the percentage of gold production allocated to Royal Gold (from 52.25% to 35%) and to include 18.75% of the copper production from the Mount Milligan mine.

The Stream Agreement has an initial 50-year term, with automatic successive 10-year renewal periods.

TCM sells copper and gold concentrate from Mount Milligan Mine to customers and in connection with such sales, TCM purchases gold ounces and copper warrants in the market for delivery to Royal Gold in an amount based on a portion of the gold and copper content contained in the copper and gold concentrate sold to customers.

The Stream Agreement covers substantially the entire Property.

The Stream Agreement includes certain restrictions on assignment or transfer of the respective rights of both parties to the Stream Agreement.

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

5.1. ACCESS

The Mount Milligan Mine is accessible from the east via Mackenzie on the Finlay Philip Forest Service Road and the North Philip Forest Service Road. These roads are maintained in good condition by the various user groups. A western route, completed in 2005, provides shorter access from Fort St. James via the North Germansen Road. This route includes 26.7 km of forest service roads, with the balance on public roads. Road travel to the site of the Mount Milligan Mine is 254 km from Prince George (population approximately 79,000).

5.2. CLIMATE

The area has short cool summers and cold winters, which are summarized Table 5-1.

Table 5-1: Mount Milligan Climate Statistics

	Month												
	Jan	**Feb**	**Mar**	**Apr**	**May**	**Jun**	**Jul**	**Aug**	**Sep**	**Oct**	**Nov**	**Dec**	**Year**
Average Precipitation (mm)													
Rainfall	0.3	1.9	4.0	15.9	54.8	58.3	66.3	50.1	67.1	52.4	12.6	1.0	385.0
Snowfall	78.5	64.0	50.5	19.6	6.4	0.8	0.8	0.8	2.8	16.3	68.6	76.0	385.0
Total Precipitation	78.8	65.9	54.5	35.4	61.2	59.2	67.2	50.9	69.9	68.7	81.1	77.0	770.0
Lake Evaporation	-	-	-	42.2	69.2	81.5	84.7	68.4	37.0	7.1	-	-	390.0
Average Monthly Temperature (°C)													
Mean	-10.9	-8.1	-3.7	1.6	7.0	11.3	13.4	12.8	8.0	2.3	-4.7	-9.2	1.7

Source: 2009 technical report for the Property (“2009 Report”), verified Centerra, 2017

5.3. LOCAL RESOURCES

Labour and services are readily available from the surrounding towns of Prince George, Fort St. James, Mackenzie, Vanderhoof, Smithers and Fraser Lake.

5.4. INFRASTRUCTURE

Infrastructure available to the Project consists of the main forest service roads accessing the Property from the east and west. Electric power is accessed from the BC Hydro Kennedy Substation south of Mackenzie. Canadian National Railway service is available from Fort St. James and Mackenzie which connects to the major western and eastern rail routes. Please see Section 18 for additional information regarding infrastructure.

5.5. PHYSIOGRAPHY

The Property lies near the northern boundary of the Southern Plateau and Mountain Region of the Canadian Cordilleran Interior System. More specifically, the Property is within the Nechako Plateau near the southern limits of the Swannell Range of the Omineca Mountains. The Property is dominated by a chain of peaks aligned in a north-south direction. Mount Milligan, which is 8 km north of the Project, is the highest of these peaks. It rises to the elevation of 1,508 m and is rounded and symmetrical in shape. The Project deposit is to the south of Mount Milligan on the eastern slopes of the chain, at an elevation of 1,100 m in an area of gentle relief.

The Nechako Plateau was covered by the Cordilleran ice cap, which moved eastward and northward from the Coast Ranges towards the Rocky Mountains near McLeod Lake, over-riding the mountains, coating the landscape with a blanket of glacial till, and altering the pre-glacial drainage patterns. Drumlins, flutings, eskers, and melt-water channels of various dimensions are noticeable features of the plateau surface. The Property is well-drained except for depressions where natural vegetation succession has filled in ponds to form bog-like fens. Drainage from the area is to the northeast via Nation River into Williston Lake, which forms part of the Peace-Mackenzie River basin.

6. HISTORY

6.1. PROPERTY OWNERSHIP

Table 6-1 below summarizes historical ownership, mineral exploration and development activities of the area that has become the Mount Milligan mine and property in northcentral British Columbia.

Table 6-1: Mount Milligan Historical Ownership and Development Timeline

1929	Placer gold and platinum was discovered on Rainbow Creek by George Snell
1931	Whole length of Rainbow Creek (over 40 km) staked and worked by nearly 100 men, gold was flat, well-worn and not of local origin (Galloway, 1931)
1937	Prospecting in the geographical Mount Milligan area, George Snell
1972	Pechiney Development Ltd. drilled 5 holes to test geophysical and geochemical anomalies
1984-1986	R. Haslinger staked mineral claims; optioned claims to Selco-BP (in 1986 signed option agreement with Lincoln Resources Inc.)
1987	MBX Main deposit discovery; Lincoln Resources Inc. #12 drill hole
1989	Southern Star deposit discovery; Lincoln Resources-Continental Gold Corp and JV with BP Resources
1990	Placer Dome purchased Mount Milligan
1991	Pre-Feasibility Study completed
1992	Intended to develop the mine in 1993 but project was determined to be sub-economic due to low metal prices
1993	Obtained permits required for commercial production at 60,000 t/d (permits expired in 2003)
1996/1998	Re-evaluations completed
2004/2005	Project re-evaluation; drilling for metallurgical test work; updated resource block model; laboratory studies identified potential for improved metallurgy
2006	Barrick Gold Corp. purchased Placer Dome and sold Canadian assets to Goldcorp Inc.; Goldcorp sold Mount Milligan to Terrane Metals Corp. (Atlas Cromwell Ltd.); feasibility study and permitting process commenced
2008	Terrane completes Mount Milligan feasibility study and prepares to file Environmental Assessment (EA) application
2009	EA approved (March); Mines Act permit received (September)
2010	Thompson Creek Metals Company Inc. (TCMC) acquired the development project through acquisition of Terrane; construction began mid-2010
2013	Mine commissioned (October)
2014	Commercial production achieved
2016	Completed ramp-up (January); secondary crusher construction and commissioning; TCMC acquired by Centerra Gold Inc.

6.2. HISTORICAL EXPLORATION AND DEVELOPMENT ACTIVITIES (1937-2011)

1937-1972

In 1937, prospector George Snell found gold bearing float on the southwest flank of the geographical Mount Milligan (~8 km NNW of the Mount Milligan deposit). In 1945, Mr. Snell returned to the area and staked 10 two-post claims west of Mitzi Lake. Five pyritic andesite float samples returned assays ranging from trace to 150.2 g/t Au (4.38 ounces/ton; Nelson and Bellefontaine, 1996). The source of the float was not located, and no other gold-bearing mineralization was found in place.

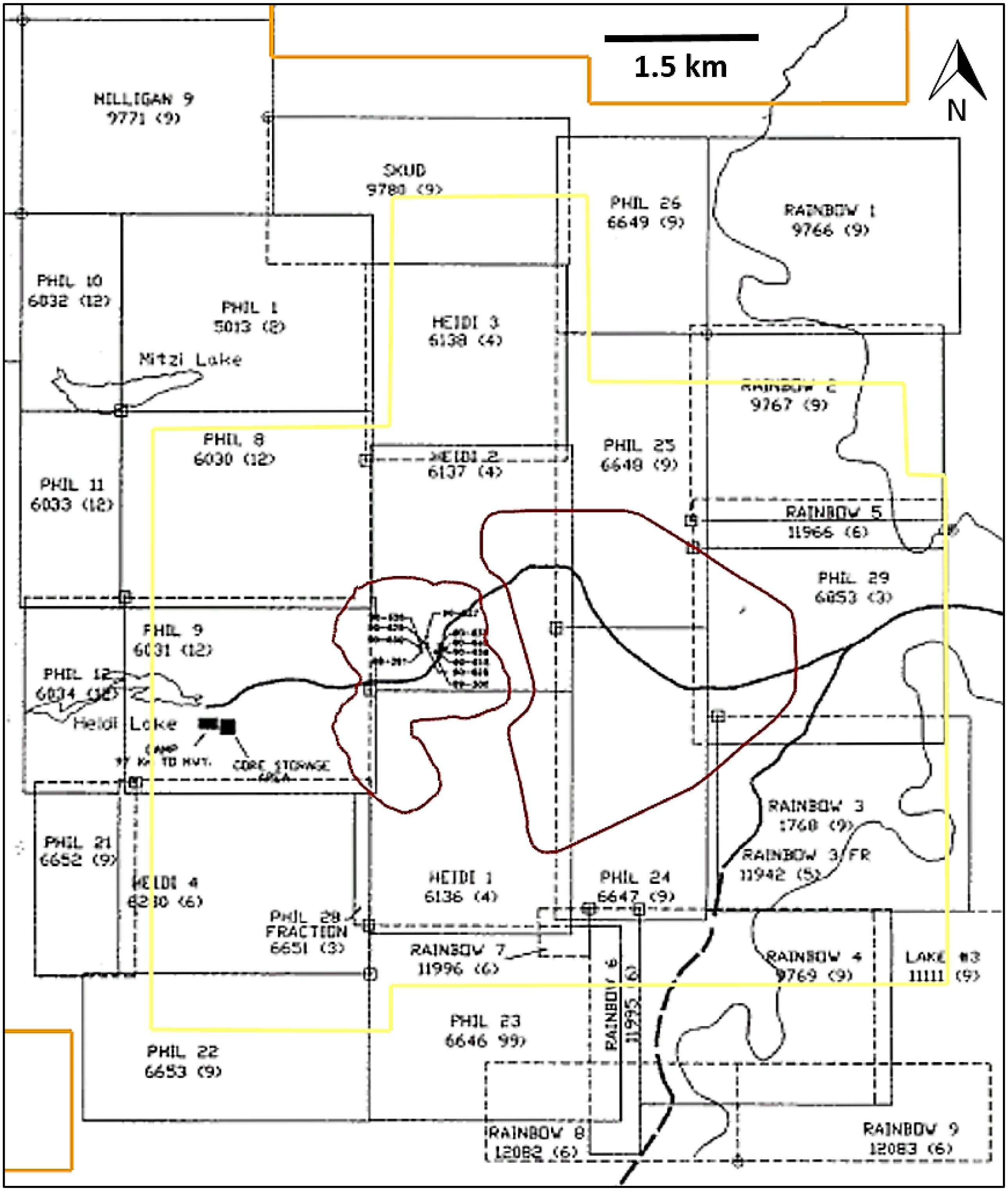
The first recorded claims in the Mount Milligan region were the Mosquito 1-10 two-post claims staked on August 4th, 1972 by Pechiney Development Ltd. (“Pechiney”). Subsequent exploration work identified induced polarization (IP) and soil geochemical anomalies. Pechiney drilled five drill holes to evaluate the anomalies but identified no significant mineralization and allowed the claims to lapse.

1983-1989

In 1983, Selco Inc. (“Selco”) staked the Phil 1 through 12 claims (Figure 6-1) in what is now the Brownfield-Greenfield exploration area west of the mine on the Mount Milligan property (also the ground covered by the original Mosquito claims). Preliminary surveys were completed. Also during that summer, prospector Richard Haslinger discovered copper-gold mineralization in bedrock exposed in a creek in what is now called Saddle zone (formerly Creek zone). In early 1984, Selco amalgamated with BP Resources Canada Ltd. (“BP Resources”). In April 1984, Richard Haslinger staked the Heidi claims to the east and adjacent with the Phil claims (Figure 6-1). Most of the current open pit is located on historic Heidi claims 1 and 2. BP Resources optioned the Heidi claims from Richard Haslinger in July 1984. In late 1984 and early 1985, BP Resources staked the Phil 21 through 29 claims south and east of the Heidi claims (Figure 6-1).

Systematic exploration near the Mount Milligan porphyry Au-Cu deposit area began in 1983. This region has sparse outcrop with most of the bedrock covered by a blanket of glacial till generally varying between <1 to 50 m in thickness with a maximum of about 100 m thickness; nonetheless, several geologic mapping programs in the project area were completed. Between 1983 and 1985, BP Resources, through their Selco Division, completed initial reconnaissance surface geochemical surveys and revealed an extensive area of anomalous copper and gold in the deposit area.

Figure 6-1: Historical Claims of the Mount Milligan Property



Note: (ARIS report 252299) with 2017 ultimate pit and TSF boundaries (dark red), current mine lease boundary (pale yellow) and property boundary (orange)

Follow up detailed mapping was completed in areas of anomalous gold-in-soil (Blanchflower, 1986). Around this time trenching, totalling 1,400 m, was completed by BP Resources focusing on the Creek, Boundary, and Esker zones where pervasive sericite, epidote, and K-feldspar alteration was encountered with anomalous gold and copper values. The work identified polymetallic auriferous vein systems and weak Au-Cu porphyry mineralization near Heidi Lake. Geophysical programs completed by BP Resources over the deposit area included initial ground magnetic surveys in 1984 to 1985, very low frequency (VLF) electromagnetic surveys in 1985, and induced polarization (IP) surveys in 1985 and 1987. This work identified a broad coincident geophysical anomaly interpreted to be resulting from disseminated sulphides and associated alteration minerals.

On April 21, 1986, Lincoln Resources Inc. ("Lincoln") entered into an agreement with BP Resources to continue exploration of the claims. The agreement allowed Lincoln to earn a 51% interest in the property, which was subsequently increased to 69.84% through the operation of dilution provisions. In July 1986, Lincoln entered into a new option agreement with Richard Haslinger on the Heidi claims. Further trenching was completed by Lincoln in 1986 and 1987, concentrating on the Esker, Creek, South Boundary, and North Slope areas.

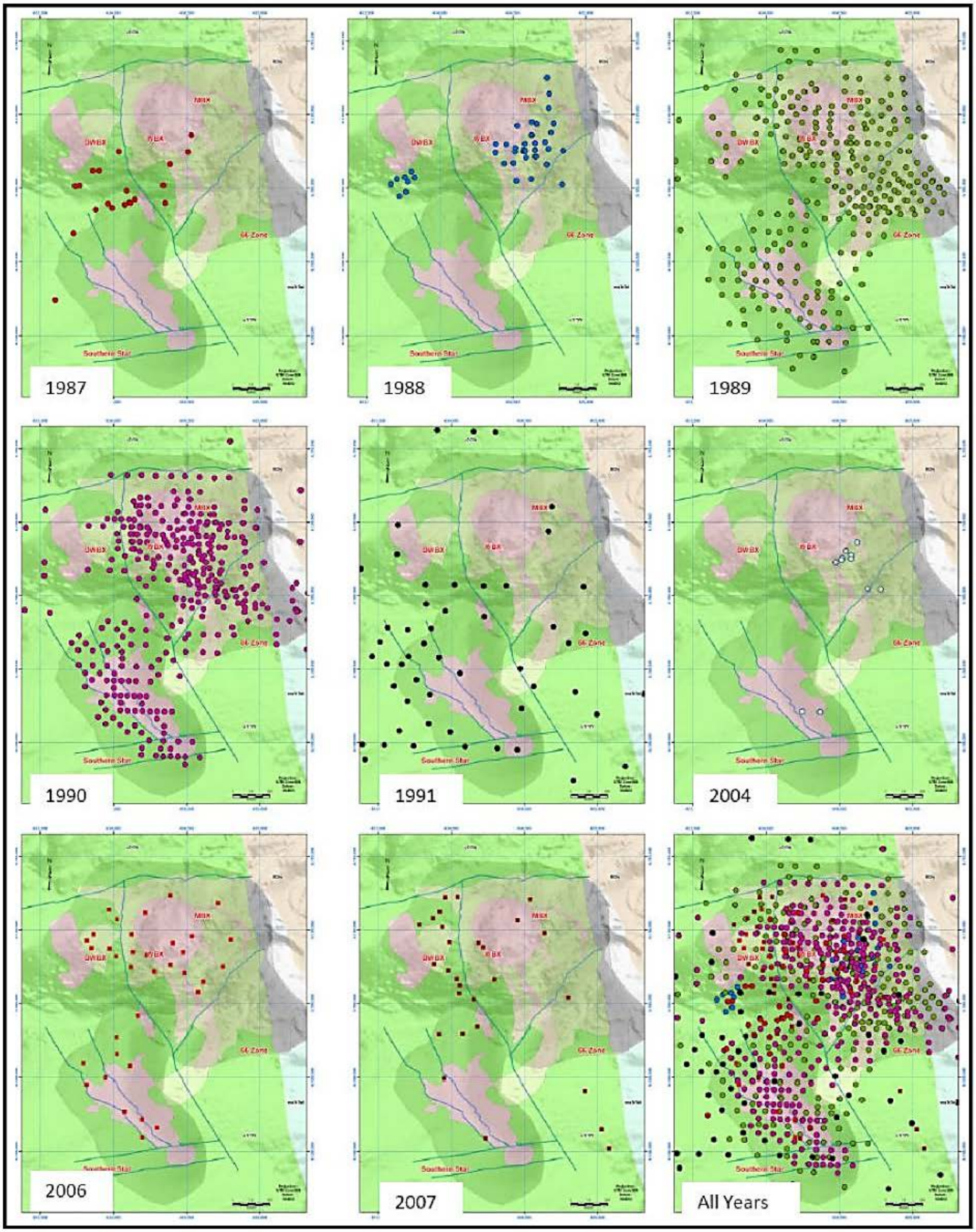
In 1987, Lincoln commenced drilling (Figure 6-2), following up on targets identified during the BP Resources and Lincoln geophysical, geochemical, and trenching programs. The Esker Zone was the first area drilled. This program's test results indicated moderate to high (1.33 to 42.7 g/t) gold grades over narrow widths. The Creek Zone similarly intersected low to moderate gold grades, generally in the range of 1 to 2 g/t over widths of 1 to 2 m.

The MBX zone was initially targeted due to the presence of a magnetic anomaly located by reconnaissance ground magnetic surveys conducted by BP Resources and later by Lincoln (Rebagliati, 1987). The 1987 Lincoln survey was a detailed 31.1 km (50 m x 10 m) ground magnetic survey that outlined the MBX zone. There were also coincident IP and soil geochemical anomalies associated with the magnetic features. In September 1987, Lincoln undertook a drilling campaign to test targets identified by BP Resources. This work resulted in the first discovery of significant Au-Cu mineralization in the MBX zone (Rebagliati, 1988). Diamond drill hole 87-12 intersected intensely potassic altered monzonite dykes and volcanic rocks with disseminated pyrite and chalcopyrite, returning 50.51 metres grading 0.6 g/t gold and 0.24% copper. Hole 87-13 intersected 97 metres grading 0.62 g/t gold and 0.27% copper.

Exploration conducted by United Lincoln in 1988 included additional diamond drilling in the Esker and Creek zones but focused primarily on the porphyry Au-Cu style mineralization intersected in the MBX zone. On July 31st, Lincoln reorganized to become United Lincoln Resources Inc. ("United

Lincoln"). In September, United Lincoln staked the Milligan, Rainbow 1 through 4, the SKUD mineral claims (Figure 6-1). In August 1988, Continental Gold Corp. ("Continental") acquired 64% of the shares of United Lincoln. On March 15th, 1989, Continental and United Lincoln amalgamated and concurrently transferred the amalgamated undertaking to their subsidiary and successor company, DASS No. 39 Holdings Ltd., which changed its name to Continental Gold Corp. ("Continental") on the same date.

Figure 6-2: Drill Hole Location by Year (1987-2007)



Source: 2009 technical report for the Property ("2009 Report"), verified Centerra, 2017

A major drilling program was launched in 1989 that included 87,662 m of diamond drilling in 336 holes. Hole 89-200 intersected mineralization of the Southern Star deposit to the south of the MBX Main deposit. The 1989 program essentially outlined all the principle zones of the project, including the high-gold low-copper 66 zone (named after drill hole 88-66), the zone along the western margin of the MBX stock (WBX zone), and a deeper downthrown portion of the WBX, the DWBX zone, in the footwall block of the Harris fault. An airborne geophysical survey was also flown by Aerodat Limited in 1989 covering the area north and west of the deposit area; it consisted of magnetics and VLF-EM, flown on east-west oriented flight lines at 100 m spacing.

1990-1998

In 1990, Continental continued staking and acquiring claims in the region including the RAINBOW 5 through 9, RAINBOW 3 Fraction, BEE and SEE mineral claims, MBX 14 through 29, and RAIN placer claims. It also acquired the BONANZA, MARTIN, and TRNAVA mineral claims. An infill diamond drilling program was carried out from January to September (386 drill holes totalling 82,924 m). Of these, 352 were drilled by Continental and the remaining 34 were drilled by Placer Dome who acquired control of the property in late 1990. In September, Placer Dome Inc. ("Placer Dome") had purchased BP Resources' share of the Phil and Heidi mineral claims. Placer Dome, and their wholly owned subsidiary PDI Subco, then acquired approximately 98% of the shares of Continental by takeover bid, and in November resumed exploration drilling.

In January 1991, PDI Subco acquired the balance of the outstanding Continental shares. With these acquisitions, Placer Dome became the primary proponent of the project and continued the process of seeking regulatory approval. In April 1991, a Pre-Feasibility Study was produced for the development of a 60,000 t/d open pit mine and flotation process plant. Placer Dome completed a 90-hole (17,969 m) drill program focused on the Southern Star deposit with additional exploration drilling to the west of Southern Star and north of Heidi Lake. Geotechnical, metallurgical, and condemnation holes were also completed during this period.

Other exploration activities in 1991 included a regional mapping program completed by consultant Atholl Sutherland-Brown and a trenching program conducted by Placer Dome geologists in the North Slope area, north of Heidi Lake. Geophysical work included IP surveys north of the BP Resources survey area and VLF surveys north of the deposit and in the Phillips Lake area. The IP surveys primarily outlined the pyritic haloes surrounding the intrusions and mineralized zones. Ground magnetic surveys were also completed in 1991 north of the deposit area and at Phillips Lake. Results showed both the MBX and Southern Star stocks are positive magnetic anomalies, although the MBX stock lies immediately south of a broad (5.5 km2) magnetic high anomaly that is interpreted to be related to a southern extension of the Mount Milligan Intrusive Complex, mapped about 5 km to the north of the Main deposit area (Nelson and Bellefontaine,1996). Also, around this time, a small helicopter-borne DIGHEM survey was flown. The survey collected both magnetic and EM data; the EM data exhibits a pattern very similar to the IP ground surveys.

In 1992, it was determined the project was sub-economic and Placer Dome wrote-off the carried value of the property. Drilling that year was completed in the Phillips Lake area in the eastern part of the property to test some IP and magnetic targets. No mineralization was intersected.

In 1993, Placer Dome received provincial and federal approvals to develop the project (these approvals expired in 2003).

In 1996, Placer Dome re-evaluated the project and developed an updated geological model that included new domains and hard boundaries. Test pits were excavated into the bedrock surface to obtain additional geotechnical information.

In 1998, an economic re-evaluation was completed using the 1996 model for the MBX Main deposit and the 1991 model for Southern Star. A variety of alternate mining and processing scenarios were investigated.

2004-2011

Exploration resumed in 2004 when Placer Dome compiled and reprocessed historical data (geological, geochemical, geophysical) into a GIS database. A 3-D geological model was developed and enhanced interpretation studies were completed. Pulps from previous drilling programs were analysed with an ASD reflectance spectrometer to obtain alteration mineral spectra to aid in geological modelling (Lustig, 2007). A 14-hole (2,184 m) diamond drilling program was conducted for additional metallurgical testing.

In 2005, geochemical studies were carried out including a regional stream sediment sampling program to assess the downstream dispersion from the Mount Milligan deposit using several analytical and sampling techniques including Bulk Leach Extractable Gold (BLEG). Additionally, an M.Sc. study was initiated through UBC/MDRU to investigate alteration zoning and geochemical dispersion with the objective of developing a 3-D alteration model.

In February 2006, Placer Dome provided a NI 43-101 compliant mineral resource estimate (Placer Dome Inc. News Release dated February 20, 2006; Lustig, 2006). In May 2006, Barrick Gold Corporation purchased Placer Dome and sold Placer Dome’s Canadian assets to Goldcorp Inc.

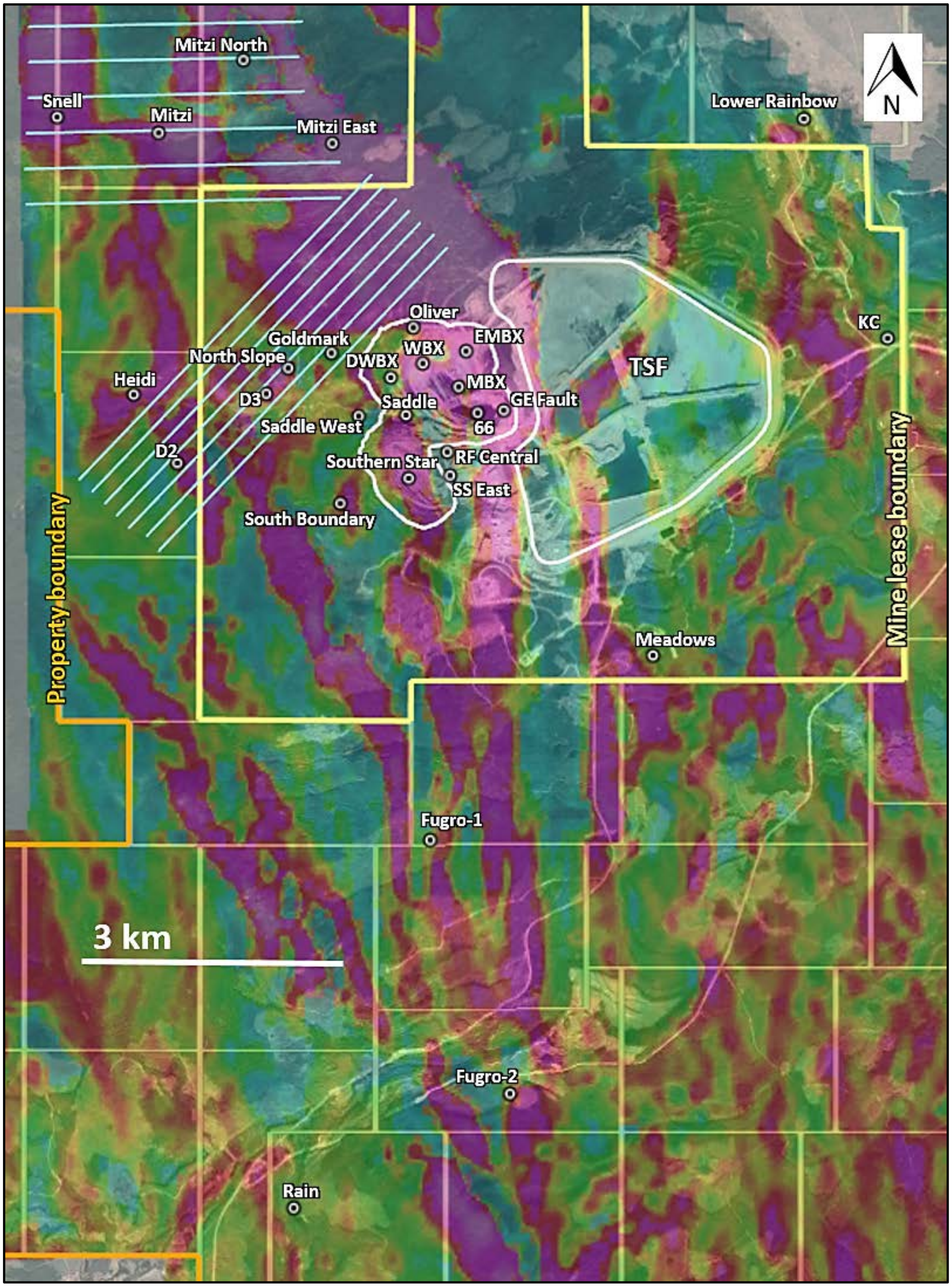
(“Goldcorp”), including the Mount Milligan property. Goldcorp then sold certain assets (including Mount Milligan) to Atlas Cromwell Ltd. (“Atlas Cromwell”). In July 2006, Atlas Cromwell was renamed Terrane Metals Corporation (“Terrane”). In September, Terrane commenced a 36-hole (9,557 m) drilling program to collect more material for metallurgical test work, to collect geological and geotechnical information along the margin of the MBX Main deposit, and infill drilling of target areas identified by previous operators.

In 2007, an additional 33-hole (10,515 m) drilling program was completed for geotechnical purposes along the margin of the Southern Star deposit and to collect geological information in the planned mine infrastructure area.

In the summer seasons of 2008 and 2009, Terrane completed a mapping program, a regional geochemical stream survey and geophysical programs. In the mapping program, a total of 175 rock samples were collected from across the property, with 40 of these sent for thin section analysis and 42 sent for assays. Additionally, in May-June 2008, an airborne magnetics and HeliGEOTEM II electromagnetic survey consisting of 1,458 line-km was flown with coverage over 264 km2 (66%) of the Mount Milligan property (Figure 6-3; Fugro Airborne Surveys Corp., 2008). A total of 101 traverse lines were flown with a line spacing of 200 m; and an additional 10 tie lines were flown with a line spacing of 2,000 m. This survey was designed to identify geophysical signatures characteristic of the project deposits in areas with thick overburden cover. The survey was successful in identifying numerous geophysical anomalies. In October 2008, Geotech Ltd. flew a ZTEM (Z-Tipper Axis Electromagnetic) survey over the Mt-Milligan Test Block on behalf of Geoscience BC. The survey consisted of 25 approximately 8.0 km long, east-west oriented flight lines, totaling 200 line-km, that were obtained at nominal 250 m line spacings over an approximately 6 x 8 km area. The survey comprised airborne Tipper AFMAG (audio frequency electromagnetics) measurements, as well as aeromagnetics using a caesium magnetometer. ZTEM inversion results were produced in 2009 (Geotech Limited, 2009).

In 2009, ground-based IP surveys (53.6 line-km) were completed on two grids to investigate 12 of the HeliGEOTEM geophysical anomalies (Figure 6-3). The North Grid (20 line-km) comprised six east-west oriented lines at 400 m spacing; the South Grid (33.6 line-km) comprised seven SW-NE oriented lines at 200 m spacing. The two- and three-dimensional IP surveys (2-DIP and 3-DIP) were conducted by SJ Geophysics Ltd. correspondingly on the North and South grids (Hermiston, 2009). Survey results indicated that five of the HeliGEOTEM anomalies, including the Mitzi and Snell targets, had coincident aeromagnetic and IP chargeability signatures comparable to those associated with the MBX Main and Southern Star deposits. In October 2009, Terrane provided a revised NI 43-101 compliant mineral resource estimate (Wardrop, 2009).

Figure 6-3: Mount Milligan Exploration Targets



Google Earth (2019) image of Mount Milligan mine property showing Resource (NPI) and Exploration drilling target areas. Transparency overlay is airborne magnetic survey (1VD; Fugro Airborne Surveys, 2008). Pale blue grid lines are the North Grid (2DIP) and South Grid (3DIP) survey lines. The 2019 ultimate pit and TSF boundaries are shown in white lines. Property, mine lease and mineral claim boundaries are also shown.

In 2009 and 2010, Terrane completed a soil geochemical orientation survey over the Mount Milligan deposit, a soil geochemical survey over the North Grid to supplement the 2DIP survey, and a regional geochemical stream survey. The soil orientation survey comprised 97 sample sites, at which Soil Gas Hydrocarbon (SGH), Mobile Metal Ion (MMI) and conventional B-horizon soil samples were taken. A total of 46 sites were sampled in the stream sampling survey, with water, silt and BLEG samples collected at each site. A prominent copper-gold anomaly was outlined in the areas of the known mineralization, but also extended up-ice and up-slope from the deposits. The North Slope area is highly anomalous, but no significant near surface source has been discovered to explain the full extent of the Cu-Au soil anomaly. A multi-element soil anomaly on Snell Hill, west of Mitzi Lake, exhibits a geochemical signature suggestive of a high-level intrusive source (Heberlein, 2010).

In 2010, Quantec Geoscience Ltd. completed a Titan-24 Direct Current resistivity (DC) and IP chargeability survey (9 lines; 35.8 line-km) and a 21.3 line-km Audio Magnetotelluric resistivity (MT) survey over the deposit area. Results identified 27 geophysical anomalies with potential for porphyry copper mineralization from near surface to >1,500 m depth, with nine high priority targets and seven secondary targets (Martinez Del Pino and Eadie, 2010). Nine diamond drill holes (4,944 m) were completed in the South Grid area (North Slope and D3 targets) during a September to November exploration drilling program. Narrow intervals of high-grade porphyry Au-Cu mineralization associated with potassic alteration (biotite-magnetite) and broader Au-rich, Cu-poor intersections with ’66 zone’ style alteration-mineralization (quartz-sericite-pyrite-carbonate; QSPC) were identified.

On October 20th, 2010, Thompson Creek Metals Company Inc. (TCMC) acquired the issued and outstanding equity of Terrane, including the Mount Milligan deposit and large land holdings. From November 2010 to March 2011, eight diamond drill holes (5,591 m) were completed to follow up previous drilling results in the footwall of the down-dropped MBX (DWBX) stock and other porphyry stocks (Goldmark and Unnamed stock). Results of the Titan-24 survey were used for targeting, and drilling tested for an extension of the DWBX zone to the west (Figure 6-3).

Table 6-2: Historical (1987-2011) Drilling Programs Summarized by Year

Year	Metres	Holes	Company
1987	2,304	23	Lincoln
1988	6,645	47	United Lincoln
1989	87,662	336	United Lincoln, Continental
1990	82,924	386	Continental, Placer Dome
1991	17,969	90	Placer Dome
1992	526	4	Placer Dome
2004	2,184	14	Placer Dome
2006	9,557	36	Terrane
2007	10,515	33	Terrane
2010-11	10,535	17	Terrane
Total	**230,821**	**986**	

The phased start-up of the mine commenced on August 15th, 2013 and commercial production began on February 18th, 2014. For details on exploration programs from 2014 onwards, see the Exploration section (Sections 9 and 10) of this report.

On October 20th, 2016, Centerra Gold Inc. ("Centerra") acquired the issued and outstanding common shares of Thompson Creek and the large Mount Milligan land holding.

6.3. HISTORICAL PRODUCTION

Mine waste stripping activities began in 2012, while mill commissioning began in Q3 2013. Commercial production was achieved in February 2014, defined as operation of the mill at 60% design capacity mill throughput for 30 days. Table 6-3 presents historical production tonnes, grade, recoveries and concentrate production for calendar years 2013 through 2019.

Table 6-3: Historical Production as of December 31, 2019

Years	Milled Ore Tonnage ('000 t)	Head Grade		Metal Recovery		Concentrate Production			Waste Tonnage ('000 t)
		Cu (%)	Au (g/t)	Cu Rec (%)	Au Rec (%)	Concentrate ('000 dmt)	Cu (M lb)	Au ('000 oz)	
2013	2,055	0.29	0.56	79.2%	54.3%	18.7	10.4	20.1	24,753
2014	14,290	0.27	0.63	80.4%	63.1%	125.4	68.0	184.0	11,224
2015	16,138	0.26	0.64	80.2%	68.6%	140.7	75.2	226.0	14,413
2016	19,277	0.19	0.58	74.7%	58.9%	125.6	61.6	212.0	20,363
2017	17,743	0.18	0.64	78.9%	62.4%	121.5	56.4	228.1	20,557
2018	13,556	0.20	0.71	81.4%	64.5%	106.0	49.6	199.5	19,764
2019	16,350	0.26	0.53	81.3%	67.4%	159.5	75.0	187.8	23,730
Total	**99,410**	**0.23**	**0.62**	**79.5%**	**63.8%**	**797.3**	**396.2**	**1,257.4**	**134,804**

Figures are shown on a 100% basis.

7. GEOLOGICAL SETTING AND MINERALIZATION

7.1. REGIONAL GEOLOGY

The Mount Milligan deposit is located within the Quesnel terrane of the North American Cordillera. The Quesnel terrane is part of the Intermontane Belt: a composite belt of volcanic arc and oceanic terranes that evolved outboard of the western margin of North America (Nelson et al., 2013). The terranes of the Intermontane Belt are interpreted to have accreted to North America during the Middle Jurassic (Mihalynuk et al., 2004).

The Quesnel terrane (Figure 7-1) represents a series of late Paleozoic to Jurassic island arcs. The oldest rocks assigned to the Quesnel terrane are Upper Paleozoic volcanic and sedimentary rocks of the Lay Range assemblage (Nelson and Bellefontaine, 1996). These are overlain by the Middle to Upper Triassic Takla Group which is subdivided into a lower unit of basinal fine clastic sedimentary rocks, the Slate Creek succession, and an upper unit of predominantly mafic to intermediate volcanic rocks that are referred to by a variety of names reflecting the presence of local volcanic centres; in the Mount Milligan deposit area it is referred to as the Witch Lake succession (Nelson and Bellefontaine, 1996). The Takla Group in this area is overlain by intermediate volcanic rocks of the Lower Jurassic Chuchi Lake succession. The Triassic and Early Jurassic volcanic packages both have mildly alkaline, or shoshonitic, geochemical signatures (Barrie, 1993). The 180-km long Hogem intrusive complex represents the batholith core of the Quesnel terrane in northcentral BC. It evolved from more mafic peripheral to more felsic central phases generally, and from weakly alkaline to sub-alkaline compositions from the Late Triassic to Early Cretaceous; except for a more strongly alkaline Early Jurassic phase that includes the Chuchi syenite and Duckling Creek syenite complexes, known for being copper-gold prospective (Garnett, 1978; Devine et al., 2014). Mount Milligan is on trend with the Chuchi syenite complex at the southern end of the Hogem intrusive complex.

At the latitude of Mount Milligan, the Quesnel terrane is juxtaposed against Cassiar platform and Slide Mountain terrane to the east and the Cache Creek and Stikine terranes to the west (Figure 7-1). On the east, the parautochthonous Cassiar platform includes Proterozoic to Paleozoic carbonate and siliciclastic strata that formed along the passive margin of Ancestral North America (Laurentia). Slide Mountain terrane is comprised of late Paleozoic oceanic rocks that formed outboard of Ancestral North America. On the west, Stikine terrane is a late Paleozoic to Mesozoic volcanic arc terrane with many similarities to Quesnel terrane. Cache Creek terrane is a Late Paleozoic to Mesozoic accretionary complex that formed in an ocean basin outboard of the allochthonous Quesnel and Stikine terranes and became trapped between them during Mesozoic accretion and oroclinal closure (Mihalynuk et al., 1994).

On its western side, the Quesnel terrane is bounded by the Pinchi and Ingenika strike-slip faults while on the eastern side, it is bounded by northwest-trending thrust and strike-slip faults that include the Swannell fault, Manson-McLeod fault system, and Eureka thrust. The boundary between the Quesnel terrane and Cassiar platform is a complex structural zone that includes Early Jurassic east-directed thrust faults that juxtapose Quesnel terrane above the Cassiar platform. These faults and related folds are locally overprinted by somewhat younger west-directed structures that reverse this stacking order, as well as by dextral strike-slip and normal faults that were active in Cretaceous and early Tertiary time (Schiarizza, 2005). The Quesnel terrane is juxtaposed against the obducted Slide Mountain terrane by a series of west-directed thrusts. Subsequent Cretaceous to early Tertiary dextral faulting has dismembered the Slide Mountain terrane within fault zones including the Manson-McLeod system east of Mount Milligan.

Younger rocks commonly found in the region include Cretaceous granitic stocks and batholiths; Eocene volcanic and sedimentary rocks; flat-lying basalt of both Neogene and Quaternary age; and extensive blankets and veneers of Quaternary glacial till, glaciofluvial, and glaciolacustrine deposits.

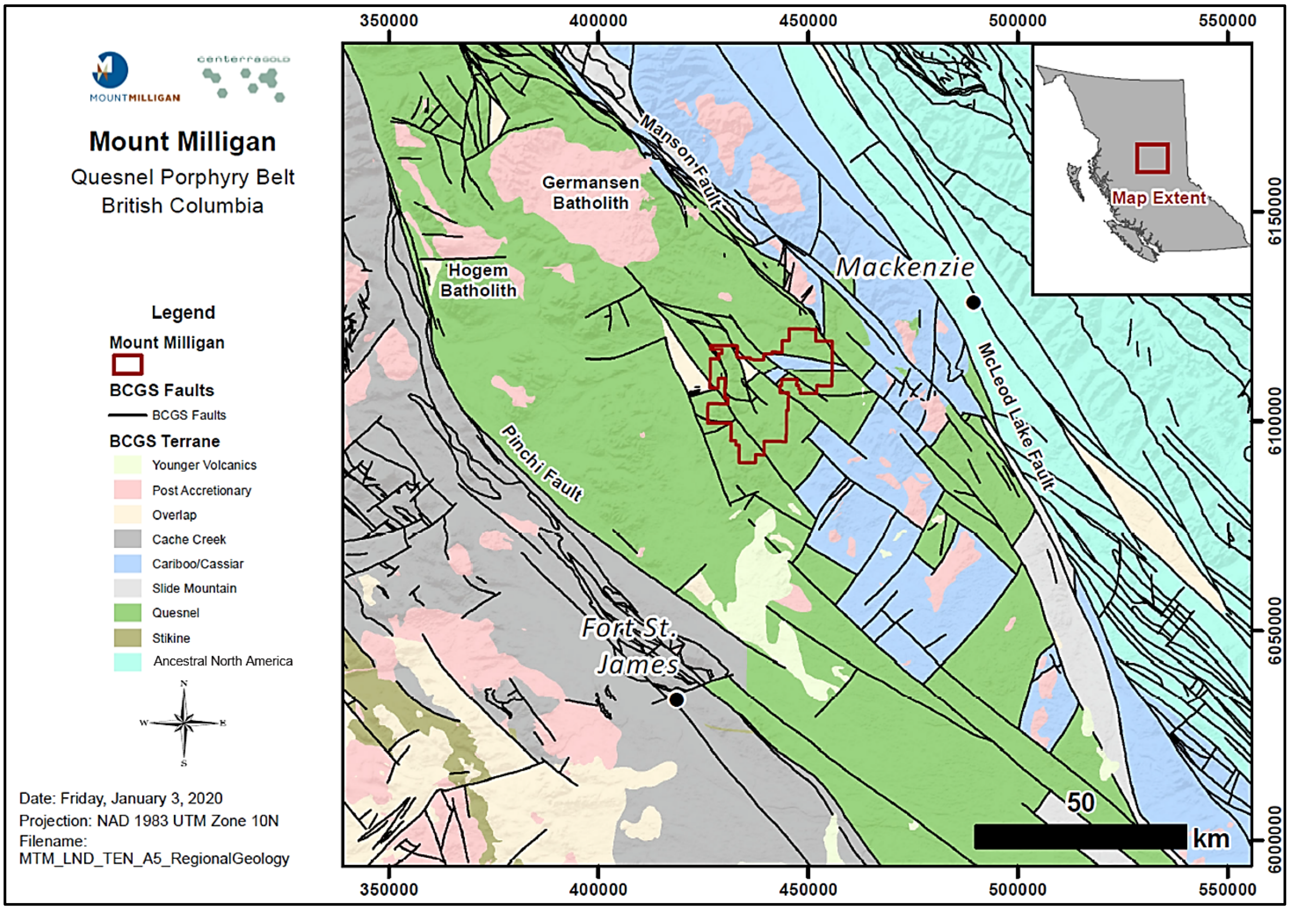
7.2. LOCAL AND PROPERTY GEOLOGY

The Mount Milligan property is mostly underlain by Upper Triassic volcanic rocks of the Witch Lake succession (Figure 7-2). The Witch Lake succession is moderately-to-steeply east-northeast dipping and characterized by augite-phyric volcaniclastic and lesser coherent basaltic andesite to andesite, with subordinate epiclastic beds. In the northwestern part of the Mount Milligan property, volcanic rocks are intruded by Early Jurassic to Cretaceous rocks of the Mount Milligan intrusive complex (Figure 7-2) located about 5 to 9 km north of the Mount Milligan porphyry deposit. The Early Jurassic component of the intrusive complex comprises monzonitic rocks with minor dioritic-monzodioritic and gabbroic-monzogabbroic rocks.

In the deposit area to the south, the MBX (Magnetite breccia), Southern Star, Saddle, Goldmark, North Slope, and Heidi stocks and dyke complexes comprise the Heidi Lake stock cluster and are composed of monzonite porphyry rocks and various hydrothermal breccia phases. The Early Jurassic intrusions are coeval with the Chuchi Lake succession, which is not currently recognized on the property but lies approximately 15 km to the west. Younger intrusions in the area include

Cretaceous granites of the Mount Milligan intrusive complex, and Wolverine Metamorphic Complex (WMC). The latter includes schistose to gneissic amphibolite-grade Windermere Group basement rocks of Ancestral North America that were rapidly exhumed in the Paleogene (Ferri et al., 1994; Staples, 2009). The WMC rocks are locally exposed in the northeastern corner of the property (Figure 7-1). Tertiary sedimentary rocks locally overlie older rocks in wedge-shaped half-graben features evident on the east side of the deposit across the Great Eastern Fault (Figure 7-2).

Figure 7-1: Regional Geological Setting

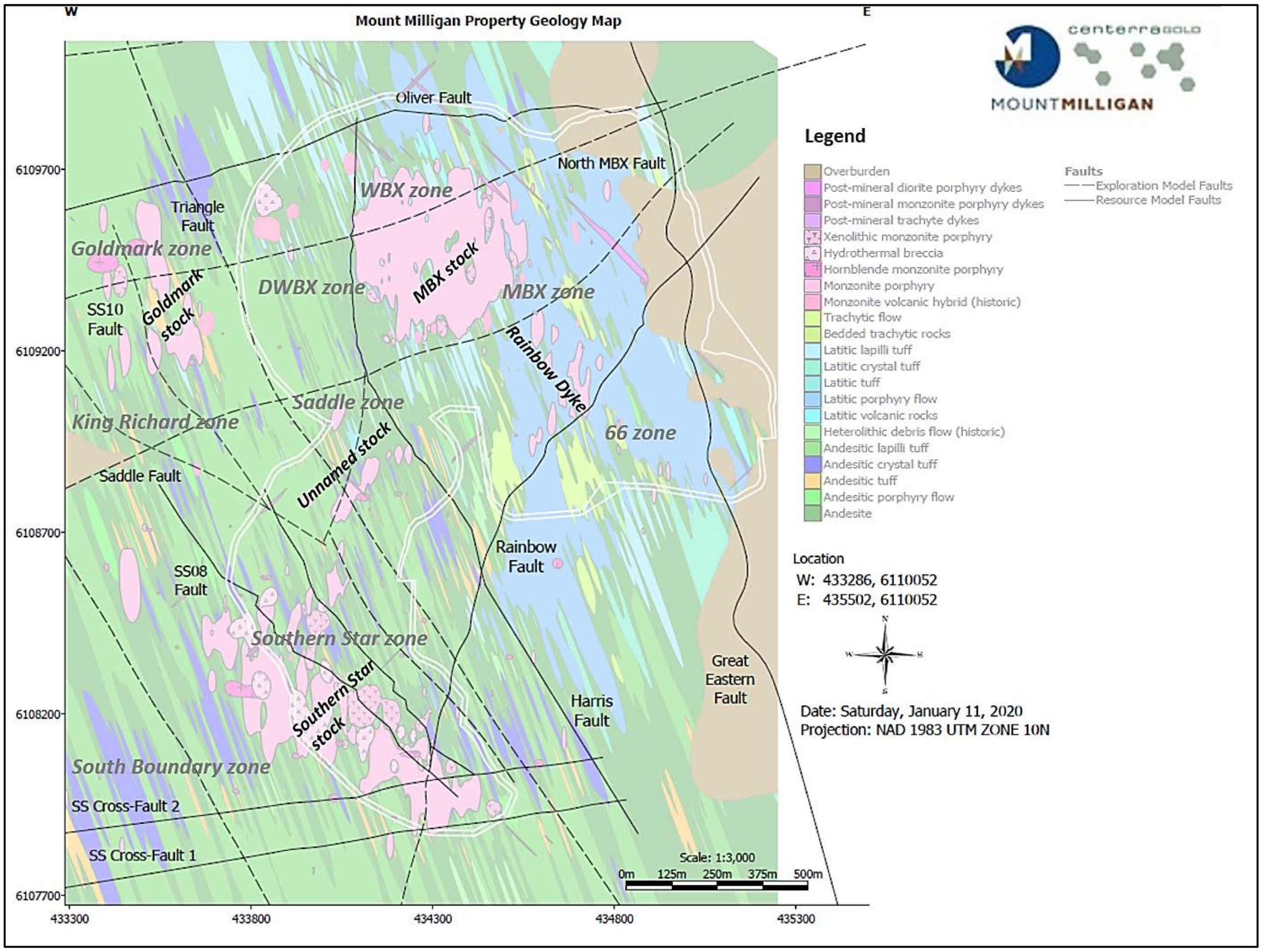


7.3. LITHOLOGY

Witch Lake Succession

Volcanic and volcaniclastic rocks of the Late Triassic Witch Lake succession are host rocks of the Mount Milligan deposit (Figure 7-2). Monolithic fragmental varieties of andesite (augite lapilli tuff, crystal tuff, and tuff), form most of the unit. Minor augite porphyritic flows are present in the west side of the deposit. Small, discontinuous heterolithic debris flows and polymictic breccias are found scattered throughout the deposit, and interbedded with the monolithic fragmental rocks. Plagioclase and/or hornblende phenocrysts are locally present within flows, individual lapilli, and crystal tuff.

Figure 7-2: Geological Map of the Mount Milligan Deposits



Much of the volcanic rocks to the east of the MBX and Southern Star stocks have historically been classified as latite (Figure 7-2). They were distinguished from andesitic volcanic rocks by a darker colour, a general absence of visible hornblende, and the presence of alteration minerals associated with the potassic assemblage including biotite, magnetite, and potassium feldspar. It is common to see only one or two of the potassic alteration minerals and magnetite alteration is more common than biotite or potassium feldspar. These rocks are more appropriately interpreted as potassically altered andesite equivalents, but for consistency in drill sections and models, drilling campaigns have continued to use the historical latite lithology codes (Table 7-1).

Rocks classified as trachyte are inter-bedded with 'latite' to the east of the MBX stock (Figure 7-2). They are characterized by high potassium feldspar content and a lack of mafic minerals. Minor fine-grained plagioclase is also present. Massive and bedded varieties of trachytic rocks are recognized in the volcanic stratigraphy. Bedded varieties are generally discontinuous and locally exhibit cross-bedding and graded bedding. Curvilinear pyrite-chlorite partings or bands of pyrite-chlorite are common along bedding planes. These rocks are now considered to be felsic volcanic and epiclastic units that have undergone potassic alteration. They are the only stratigraphic markers in the deposit area providing evidence that the geology is moderately tilted to the east-northeast.

Nelson et al. (1991) reported that the abundance of potassium feldspar in the volcanic rocks led past authors to a field classification of augite-porphyritic latites and banded trachytes. Microscopic examination indicated that the potassium-rich nature of the rocks is due to infiltration of secondary potassium feldspar in veinlets and microfractures, as clumps with pyrite and epidote, as seams in plagioclase phenocrysts, and as fine-grained aggregates along bedding planes in the sediments.

Intrusive Rocks – Synmineral

The Main and Southern Star deposits are centred on two principal intrusive bodies, the MBX (Magnetite breccia) and Southern Star stocks. Several smaller intrusive bodies are recognized as being contemporaneous with the mineralizing events, including the Unnamed stock in the Saddle zone (formerly Creek zone) and the Goldmark stock (Figure 7-2).

The MBX stock is a moderately west-southwest plunging monzonite body, approximately 400 m in diameter. Potassically altered hydrothermal breccia occurs along its margins. In the southeastern portion of the Main deposit, the Rainbow Dyke (a large sheet intrusion up to 50 m wide) protrudes from the footwall of the MBX stock (Figure 7-2). The dyke changes morphology from a sill-like body near the stock to a shallowly east-dipping curviplanar dyke, 200 m east of the stock, suggestive of a cone sheet (high level igneous intrusion). The Southern Star stock is a moderately west-dipping composite monzonite porphyry complex, approximately 800 x 300 m in area. Its margins are more irregular and undulated than those of the MBX stock. Hydrothermal breccia is extensive throughout the Southern Star stock. Breccia bodies are typically graded from their margins inward by relatively unaltered monzonite to crackle brecciated monzonite, to well-developed mosaic breccia with strongly potassium feldspar altered matrix.

The MBX and Southern Star stocks generally contain up to 30% sub-parallel plagioclase feldspar phenocrysts, 1-10 millimeters (mm) in length. These phenocrysts occur within a fine-grained greyish pink groundmass of potassium feldspar with lesser plagioclase feldspar, and accessory quartz, hornblende, biotite and magnetite. The Southern Star stock has coarser plagioclase phenocrysts than the core of the MBX stock. Porphyritic monzonite units vary both texturally and compositionally within the composite stocks. Late synmineral plagioclase hornblende monzonite porphyry dykes are common throughout the Southern Star stock.

Intrusive Rocks – Post-Mineral

Three major types of post-mineral dykes cut the MBX Main and Southern Star deposits: trachytic, monzonitic, and dioritic varieties. These dykes are generally fresh-looking and lack sulphide mineralization.

Trachytic dykes are the earliest and most common in the southwestern portion of the MBX, northern portion of the Southern Star deposit, and throughout the Saddle zone (Figure 7-2). They are up to 15 m wide and dip moderately to the northwest. They are grey, aphanitic to fine-grained, and may contain accessory magnetite. They often have late carbonate veins (calcite/dolomite) with sericite selvages.

Monzonitic dykes are recognized throughout both deposits. They are up to 10 m wide and dip moderately to the northwest. The dykes are characterized by 15 to 35% fresh euhedral plagioclase phenocrysts up to 5 mm across in a dark aphanitic matrix, and may contain augite phenocrysts up to 5 mm. Groundmass predominantly comprises fine-grained potassium feldspar as indicated by staining. Accessory magnetite is always present. Some monzonitic dykes are weakly propylitic altered.

Dioritic dykes are the youngest intrusive rocks. Although they are recognized in both deposits, they are most common in the northern (MBX Main) deposit area. They are up to 5 m wide and dip steeply to the northeast. The dioritic dykes are characterized by abundant plagioclase phenocrysts up to 10 mm (often zoned) in a fine-grained groundmass, and generally contain 2-4 mm long hornblende phenocrysts and minor quartz eyes up to 1 mm. Some dioritic dykes are weakly propylitic or carbonate altered.

The rock codes used in Mount Milligan drilling programs are indicated in Table 7-1. The historic rock names have been simplified due to the variability in naming by different geologists logging drill core over time and multiple programs.

Table 7-1: Rock Units of the Mount Milligan Deposits Used in Drill Programs Descriptions

Lithology		
Code	**Name**	**Description**
DRPD	Post-Mineral Dykes	Plag diorite porphyry
MZPD	Post-Mineral Dykes	Plag monzonite porphyry
TRD	Post-Mineral Dykes	Trachyte
HMZP	MBX and SS Stocks	Plag-Hbl monzonite porphyry
HYBX	MBX and SS Stocks	Hydrothermal breccia
MVHD	MBX and SS Stocks	Monzonite volcanic hybrid
MZPP	MBX and SS Stocks	Plag monzonite porphyry
XNMZ	MBX and SS Stocks	Xenolithic monzonite porphyry
TRBT	Trachytic Volcanics	Bedded trachytic flow
TRFW	Trachytic Volcanics	Trachytic flow
LAT	Latitic Volcanics	Latitic rocks - undifferentiated
LATF	Latitic Volcanics	Latitic Tuff
LNLT	Latitic Volcanics	Px latitic lapilli tuff
LPFW	Latitic Volcanics	Px latitic porphyry flow
LPXT	Latitic Volcanics	Px latitic crystal tuff
ANDS	Andesitic Volcanics	Andesitic rocks - undifferentiated
ANLT	Andesitic Volcanics	Px andesite lapilli tuff
ANTF	Andesitic Volcanics	Andesitic tuff
APFW	Andesitic Volcanics	Px andesite porphyry flow
APXT	Andesitic Volcanics	Px andesite crystal tuff
ARGL	Clastic Sedimentary Rocks	Argillite
SED	Undifferentiated Epiclastics	Sediments
HBHL	Undifferentiated Volcaniclastics	Lithic tuff breccia
HTDF	Undifferentiated Volcaniclastics	Heterolithic debris flow
LTTF	Undifferentiated Volcaniclastics	Lithic tuff
PBX	Undifferentiated Volcaniclastics	Polymictic volcaniclastic breccia
FALT	Fault	
GRDR	Felsic Intrusive Rocks	Granodiorite quartz porphyry
MNDR	Intermediate Intrusive Rocks	Plagioclase monzodiorite (+/-) porphyry
GABR	Mafic Intrusive Rocks	Gabbro
BSLT	Mafic Volcanic Rocks	Basalt
OVB	Soil Overburden	Undivided surficial materials
CASE	Casing	Everything above casing block

Structure

The volcanic stratigraphy within and around the Mount Milligan deposits generally dips moderately to steeply to the east-northeast. This is congruent with the geometry of the MBX and Southern Star stocks as moderately west dipping bodies and suggests that the entire Mount Milligan hydrothermal system has been tilted to the east-northeast. Exactly how and when this tilting occurred remains unclear, however there are at least two episodes (Early-Middle Jurassic and Cretaceous-Paleogene) of deformation recognized on the property.

The structural style of the Mount Milligan area regionally and at the scale of the deposit is characterized by tilted fault blocks of differing stratigraphic levels bounded by steeply dipping east-northeast and northwest trending faults, as well as by north and northeast trending faults. At the regional scale this has been interpreted to reflect a releasing bend between transcurrent northwest-trending dextral faults during Late Cretaceous to Eocene time (Nelson and Bellefontaine, 1996). A similar scenario is observed at the deposit scale where adjacent fault blocks exhibit alteration and mineralization characteristic of deeper (e.g. MBX) and shallower or peripheral (e.g. 66, DWBX) porphyry environments. The timing relationship between northeast- and northwest-trending faults remains ambiguous, likely due in part to multiple reactivations of structures.

A recent detailed structural investigation in the Mount Milligan open pit has recognized opposing shear sense indicators along northeast-trending structures, lending support for interpreted reactivation; these structures record an early dextral-reverse shear followed by later sinistral shear (Shaban and Barnett, 2019). Examples of faults in this orientation are the Alpine, North MBX, Saddle, Oliver and SS cross faults (Figure 7-2). Dextral reverse shear on these faults is interpreted to be Cretaceous in age, accommodating regional east-vergent shortening that was widespread in the Northern Cordillera at this time. The subsequent sinistral shear may have occurred during regional dextral transcurrent faulting along northwest-trending faults (Shaban and Barnett, 2019); the northeast-trending faults would be in the antithetic R’ orientation with respect to the main northwest-trending dextral faults and therefore would have accommodated minor sinistral shear.

Shallow to moderately southeast- and east-dipping faults are documented to be the youngest structures in the immediate vicinity of the open pit (Shaban and Barnett, 2019). These include the Great Eastern fault which juxtaposes hydrothermally altered Witch Lake succession rocks in its footwall against Tertiary sedimentary and volcanic rocks in its hanging-wall to the east, and the Rainbow fault which is a well-developed southeast-side-down normal fault (Figure 7-2). They are interpreted to have formed as part of the Late Cretaceous-Paleogene dilational jog in the regional dextral fault system.

Other northwest- and north-trending steeply-dipping faults occur outside of the pit and were not addressed in the recent structural investigation. These faults have been identified through drilling and geophysical interpretation and include the North Slope, KR, Golo, SS10, Triangle and Harris faults (Figure 7-2). They are interpreted as dextral-oblique normal faults predominantly, although some, such as Harris fault, have apparent reverse sense of shear.

There is evidence that at least some of these structures are long-lived, with a component of deformation that predates or is synchronous with Early Jurassic mineralization and accretion of the Quesnel terrane onto Ancestral North America. Several of the recognized east-northeast- and northwest-trending structures have been successfully targeted for high-gold low-copper (HGLC) mineralization during recent exploration programs and exhibit enhanced phyllic alteration with late stage pyrite-dominant veins. Similarly, at a regional scale, Nelson and Bellefontaine (1996) note that preferential emplacement of Early Jurassic intrusions along northeast and northwest trends suggests localization along pre-existing crustal scale structures. These would correspond to arc-parallel and arc-transverse structures globally recognized to localize magmatism and porphyry mineralization (Richards et al., 2001).

7.4. ALTERATION

Alteration of host rocks at Mount Milligan is well developed. The alteration assemblages include potassic, sodic-calcic, inner- and outer-propylitic, and carbonate-phyllic (Jago et al., 2014). Overprint of the assemblages is common and can be locally additive or destructive of mineralization grade (Jago et al., 2014). Porphyry gold-copper mineralization broadly coincides with zones of initial potassic alteration (DeLong, 1996), and high-gold low-copper mineralization with structurally controlled carbonate-phyllic (quartz-sericite-pyrite-carbonate; QSPC) alteration. In the following descriptions, vein classifications follow the convention of Jago et al., (2014), which was also adapted for geological logging.

Potassic Alteration

Regional mapping and petrographic studies in the Mount Milligan area have demonstrated that Witch Lake volcanic rocks and derived epiclastic sedimentary rocks are affected by strong potassic alteration as far as 4 km from the deposits (Nelson et al., 1991). Potassic alteration is best developed along the contact margins of the MBX and Southern Star stocks as well as the Rainbow Dyke and decreases in intensity both towards the core of the stocks, and outward into the host volcanic rocks. The alteration assemblage within the stocks is characterized by potassium feldspar, biotite, and magnetite with minor quartz and apatite (the potassic assemblage of Jago, 2008). The alteration assemblage in the surrounding volcanic rocks is characterized by the same assemblage with the addition of actinolite as well as minor albite, calcite and chlorite (the calc-potassic assemblage of Jago, 2008). Potassium feldspar is the predominant alteration mineral within the intrusions, whereas biotite is more prevalent within the host andesites (DeLong, 1996; Jago and Tosdal, 2009; Jago et al., 2014). Secondary biotite forms up to 30% of wall rocks near intrusive contacts, replacing the andesite protolith but also occurs as envelopes to potassium feldspar veinlets (DeLong et al., 1991). Associated vein types include all early-stage veins, including potassium feldspar (E1), quartz ± potassium feldspar (E2), chalcopyrite (E3), chalcopyrite-pyrite (E4), and magnetite (E5) veins as well as transitional pyrite-magnetite (T1) veins.

Sodic-Calcic Alteration

Sodic-calcic alteration is spatially restricted to the immediate periphery of the preserved potassic alteration zone within volcanic rocks. It comprises albite, actinolite, chlorite and pyrite with minor epidote, calcite, magnetite, apatite, titanite, quartz and possibly riebeckite (Jago et al., 2014). Albite typically replaces potassium feldspar in groundmass along with uralitization of augite phenocrysts. Sodic-calcic alteration represents an intermediate alteration between the high temperature potassic alteration and the relatively low temperature propylitic alteration (Jago et al., 2014). This spatial association of sodic-calcic alteration is recognized in other BC alkalic porphyry systems including Copper Mountain (Lang et al. 1995) and Galore Creek (Micko et al., 2014). Associated vein types include chalcopyrite-pyrite (E4), actinolite-chlorite (T2), and pyrite (T3) veins. It is also well-developed in the Heidi stock (Heidi zone), a copper target located outside the Mount Milligan mine lease, 3.5 km west of the open pit.

Inner Propylitic Alteration

The inner propylitic alteration assemblage occurs outboard of the sodic-calcic shell and it overprints earlier potassic alteration (Jago et al., 2014). It contains epidote, albite, calcite, actinolite, and chlorite with minor potassium feldspar, biotite, apatite, titanite, and quartz (Jago et al., 2014). This assemblage is relatively narrow (~20 m wide) and discontinuous in the MBX zone, marking the spatial transition between sodic-calcic and outer propylitic alteration assemblages. Associated veins include transitional pyrite (T3) and peripheral epidote-albite-pyrite (P1) veins.

Outer Propylitic Alteration

The outer propylitic alteration assemblage is widespread and peripheral to all other alteration zones but also crosscuts older alteration assemblages along minor faults (DeLong et al., 1991; Jago et al., 2014). Epidote is the most common mineral and is associated with chlorite, calcite, actinolite, pyrite and hematite with minor quartz, titanite, apatite, and montmorillonite (Jago et al., 2014). It is associated with peripheral epidote-albite-pyrite (P1) and pyrite-epidote (P2) veins as well as late-stage pyrite-calcite (L3/L4) veins.

Carbonate-phyllic Alteration

Carbonate-phyllic (quartz-sericite-pyrite-carbonate; QSPC) alteration is well-developed along stratigraphic horizons in the MBX and 66 zones, and along major east-northeast and north-northwest trending faults. It consists of dolomite, ankerite, potassium feldspar, adularia, albite, muscovite/sericite, chlorite, pyrite and hematite with minor illite, quartz, epidote, riebeckite, titanite, and rutile. In the 66 zone, the best gold grades (4-5 g/t Au) occur at the interface between potassic and carbonate-phyllic alteration where a quartz-sericite-pyrite assemblage predominates (Jago et al., 2014). This zone of carbonate-phyllic alteration is interpreted as representing a shallower level of the Mount Milligan porphyry system (such as the phyllic alteration shell in typical calc-alkalic porphyry systems) that has been downthrown approximately 100 m by faulting and juxtaposed on the MBX zone (Jago et al., 2014). High-gold low-copper mineralization along major east-northeast and north-northwest trending faults is associated with a sericite-carbonate-chlorite-hematite assemblage hosting late-stage vein types including dolomite-sericite-quartz-pyrite (L1), dolomite-ankerite (L2), pyrite-calcite (L3/L4), chlorite-calcite (L5), and hematite (L6) veins.

7.5. MINERALIZATION

Mineralization at the Mount Milligan deposit consists of two styles, early-stage porphyry gold-copper (Au-Cu) and late-stage high-gold low-copper (HGLC). The early-stage porphyry Au-Cu mineralization comprises mainly chalcopyrite and pyrite, occurs with potassic alteration and early-stage vein types and is spatially associated with composite monzonite porphyry stocks (especially at their hanging-wall and footwall margins), hydrothermal breccia, and narrow dyke and breccia complexes. Late-stage, structurally controlled pyritic HGLC style mineralization is associated with carbonate-phyllic alteration and intermediate- to late-stage vein types, and is spatially associated with faults, fault breccias and faulted lithological contacts (i.e. faulted monzonite porphyry dyke margins). It crosscuts and overprints the earlier stage porphyry Au-Cu mineralization.

Porphyry style Au-Cu mineralization occurs in the hanging-wall and footwall zones of the MBX, Saddle, Southern Star, and Goldmark stocks (Figure 7-2). Disseminated and vein/veinlet-hosted mineralization is associated with the composite monzonite stocks, their brecciated margins and variably altered volcanic host rocks. Core zones of auriferous chalcopyrite-pyrite mineralization with magnetite rich potassic alteration transition laterally and vertically to pyrite rich HGLC zones within the inner propylitic (albitic) and carbonate-phyllic alteration shells; the latter appear to be late stage and exhibit strong structural control.

Brownfield target areas west of the mine have mainly peripheral alteration and mineralization signatures; these include near surface HGLC zones proximal to structures. Narrow intrusive units, dykes or pencil porphyries, are common and are locally Au-Cu mineralized. The Heidi target farther west and outside the mine lease is a shallow copper prospect with low gold.

Hypogene Mineralization

The bulk of the mineralization consists of disseminated and vein-hosted chalcopyrite (in potassic and sodic-calcic alteration assemblages) and disseminated and vein-hosted pyrite (in all alteration assemblages). Late stage polymetallic (subepithermal) veins are rare, but most pronounced in the DWBX zone. These veins contain sphalerite, galena and sulfosalt in addition to chalcopyrite and pyrite.

Chalcopyrite

Chalcopyrite is associated with potassic alteration at the contact margin between volcanic and intrusive rocks. It occurs most commonly as fine-grained disseminations and fracture fillings, and less commonly as veinlets and in veinlet selvages. Adjacent to the MBX stock, chalcopyrite may be accompanied by pyrite to form coarse sulphide aggregates. Chalcopyrite-bearing veins contain pyrite and magnetite in a gangue of potassium feldspar, quartz, and calcite. In massive trachytic rocks, chalcopyrite occurs with pyrite along curviplanar partings and as disseminations. Chalcopyrite also occurs in gold-bearing early quartz veins in the Southern Star, WBX and DWBX zones.

Pyrite

Pyrite content increases with distance from the MBX and Southern Star stocks and is most abundant in propylitically altered rocks. Pyrite occurs as disseminations, veinlets, large clots, patches, and as replacements of mafic minerals. Gold mineralization in the 66 zone is associated with 10-20% pyrite. Cross-cutting vein relationships indicate several generations of pyrite mineralization.

Gold

Gold occurs as grains from 1 to 100 μm in size, as observed in process samples. Grains occur as microfracture fillings and are attached to pyrite, chalcopyrite, or bornite (Ditson, 1997). Gold also forms inclusions within pyrite, chalcopyrite, and magnetite grains. SEM work indicates electrum throughout the deposit with varying gold to silver ratios.

Petrographic work completed in 2019 on 22 drill core samples from the 66, Saddle and DWBX zones shows the settings of electrum (~3-20 µm) can be summarized as; a) isolated inclusions in cores of pyrite crystals; b) associated with microclusters of chalcopyrite and arsenopyrite in larger pyrite crystals; c) in or near microfractures filled with chalcopyrite cutting pyrite; d) at the margins of sulphide minerals such as between tetrahedrite and pyrite, or between pyrite and chalcopyrite; and e) along microfractures through pyrite. The mean Au/Ag ratio of the entire suite of 74 electrum particles was 67.0% Au, or roughly 2:1 Au/Ag.

Silver

Silver is a minor metal found in the Mount Milligan deposits. Polymetallic gold-silver bearing sulphide and sulfosalt rich veins are present in volcanic rocks peripheral to the MBX and Southern Star stocks. Tetrahedrite is the most widespread Ag-bearing sulfosalt mineral at Mount Milligan. Silver is also known to occur throughout the deposits in conjunction with copper and gold mineralization, from micron-scale electrum grains and inclusions.

Polymetallic Veins

Gold- and silver-bearing sulphide, sulfosalt and carbonate-rich veins are present in andesitic volcanic rocks peripheral to the MBX and Southern Star stocks on the western margin of the deposit, in the DWBX and Saddle zones. They comprise pyrite with lesser chalcopyrite, sphalerite, galena, molybdenite, arsenopyrite, tetrahedrite-tennantite, and minor quartz and carbonate. Alteration envelopes are not always present but are recognized to consist of chlorite-sericite-carbonate in propylitically altered andesitic volcanic rocks. Polymetallic veins intersected in the DWBX zone exhibit subepithermal textures (Sillitoe, 2010).

Supergene Mineralization

Supergene enrichment is poorly developed at Mount Milligan. It is recognized in the MBX, WBX, and Southern Star zones, and is deeper and more extensive in the MBX and WBX zones than in Southern Star (Placer Dome, 1991). Supergene enrichment is restricted to the sporadic occurrence of secondary copper minerals including sulphides (covellite, chalcocite, and djurleite), oxides (cuprite and tenorite), carbonates (malachite and azurite) and native copper. The secondary sulphides occur as rims around chalcopyrite. Oxides, especially cuprite, occur as surface coatings on native copper.

Secondary copper minerals commonly occur with iron oxides (goethite, magnetite, and hematite) and iron carbonate (siderite), particularly where malachite and azurite are present. Hydrous iron oxides, which are generally referred to as limonite and include goethite, commonly replace chalcopyrite and pyrite. Limonite either completely replaces sulphide minerals or occurs as coatings on surfaces of fractures and hairline cracks. Limonitic coatings commonly occur at depths greater than those at which pyrite or chalcopyrite are completely replaced. Supergene enrichment is best represented in the MBX and WBX zones between 6109150N to 6109800N, and between 434000E to 434850E, where it is partially developed over a 20 m thickness. Locally, the supergene mineralization reaches 50-60 m in thickness, particularly along the north and eastern margins of the MBX stock.

Length Width Continuity

Mount Milligan is a roughly tabular, near-surface silica-saturated alkalic gold-copper porphyry deposit that measures approximately 2,500 m north-south, 1,500 m east-west, and extends to a vertical depth greater than 500 m. There is 1 to 50 m depth of till overburden in the deposit area. Within this system, the overall shapes of the mineralized bodies are irregular and gradational. The two principal deposits, MBX Main and Southern Star, are centred on composite monzonite porphyry stocks. The MBX Main deposit has been subdivided into zones based on structural boundaries and/or changes in alteration-mineralization. These zones include the MBX, WBX, DWBX and 66. The Southern Star deposit has not been subdivided into zones but may be considered to include the Saddle zone at its northern end. The limits of these mineralized zones can vary greatly in size and shape depending on which grade cut-offs are used. The known deposit extents are limited by the lack of drill hole information and could potentially be extended down-dip of monzonite porphyry stocks to the west, down-dip of stratigraphy to the east, and to the north and south along the linear trend of the stock cluster.

8. DEPOSIT TYPES

The Mount Milligan deposits are categorized as silica-saturated alkalic Cu-Au porphyry deposits (Lang et al, 1995; Panteleyev, 1995) associated with alkaline monzodioritic-to-syenitic igneous rocks and are recognized in only a few mineral provinces worldwide (Deyell and Tosdal, 2004). Alkalic Cu-Au porphyry deposits form one endmember of a continuum of porphyry deposits (the other endmember being calc-alkalic Cu-Mo deposits) and are thought to form towards the back arc when associated with convergent plate boundaries (Westra and Keith, 1981; Logan and Mihalynuk, 2014) and are syn-collisional.

Porphyry copper ± gold deposits commonly consist of vein stockworks, vein sets, veinlets, and disseminations of pyrite, chalcopyrite ± bornite that occur in large zones of economic bulk-mineable mineralization within porphyritic igneous intrusions, their contact margins, and adjoining host rocks. The mineralization is spatially, temporally, and genetically associated with hydrothermal alteration of the intrusive bodies and host rocks.

While alkalic and calc-alkalic porphyry deposits share many similarities, there are significant differences that distinguish them. As the name suggests, mineralization in these systems is temporally and spatially associated with variably alkaline intrusions. Due to the lower overall silica content relative to alkalis, alkalic deposits typically have no (silica-undersaturated), or few (silica-saturated) quartz veins. Instead, mineralization occurs as sulfide veinlets and sulfide replacement of iron ($Fe2+$) bearing minerals. Alkalic porphyry deposits are commonly smaller than the calc-alkaline type and occur in clusters with variable intensity and character of mineralized zones within the cluster. Mineralization can be centred on narrow (<200m diameter), vertically elongated pipes (pencil porphyries), dyke complexes or may be hosted in larger composite plutonic complexes (Holliday and Cooke, 2007). Metal ratios in alkalic systems can have considerably higher gold tenor.

Alkalic systems also have distinct alteration assemblages, which typically have a narrower footprint than in calc-alkaline deposits (Bissig et al, 2010). Commonly, there is a magnetite-rich potassic alteration assemblage at the core of the deposit (that may contain hydrothermal breccia) which laterally transitions through calc-potassic alteration comprising calcium-bearing minerals and/or an albite rich sodic-calcic assemblage, and then an outer shell of propylitic alteration that may be reddened by hematite dusting. Intervening phyllic or argillic assemblages, as recognized in calc-alkalic systems, are not well-developed. Instead, high-level stratabound alteration zones characterized by albite-sericite alteration assemblages may be the alkalic equivalent of lithocap root zones (Holliday and Cooke, 2007). The presence of abundant late carbonate alteration is also common to alkalic systems.

Examples of alkalic Cu-Au porphyry deposits in British Columbia include Galore Creek, Mount Polley, Copper Mountain, New Afton, Mount Milligan and Lorraine. British Columbia deposits occur in both the Quesnel and Stikine island arc terranes and range in age from Late Triassic to Early Jurassic (Logan and Mihalynuk, 2014). Global examples include Ok Tedi in Papua New Guinea as well as Northparkes and Cadia in Australia.

9. EXPLORATION

This section focuses on components of the 2014 to 2019 exploration programs, other than drilling, specifically the compilation of historical reports and data, reinterpretation of historic data, development of a three-dimensional (3D) exploration model, surface geological mapping, geochemistry and geophysics programs. Detailed information on exploration drilling programs from 2014 to 2019 is included in Section 10 (Drilling). Information on exploration programs prior to 2014 is covered in Section 6 (History).

9.1. RECENT EXPLORATION ACTIVITIES (2014-2019)

Exploration, 2014-2016

In 2014, TCM conducted a two-phase exploration program from August-October on the northeastern side of the property, an area which had not been well explored previously. TCM engaged Equity Exploration Consultants Ltd. ("Equity") to conduct the program. Phase-1 comprised geological mapping, prospecting and soil sampling. A total of 147 line-kilometres were mapped and 1,527 samples assayed. Ten samples were sent to a specialized laboratory for thin section studies. Phase-2 included ground geophysical surveys (IP and magnetics) and a more detailed mapping, prospecting and soil sampling program that focused on target areas. This work identified a new target 17 km northeast of the deposit, Prospect 26, with strong propylitic alteration and anomalous Cu, Zn and Sb in soil values. The target is associated with a resistivity low ringed by a zone of high chargeability (Hughes and Perk, 2014). Several additional zones with anomalous As, Sb, Mo, Pb and Zn values in soil were identified but remain unexplained.

Exploration, 2017-2019

In January 2017, Centerra-TCM conducted a thorough review and compilation of historic exploration reports, documents and databases relating to the initial prospecting, discovery and development of the Mount Milligan property and surrounding land held by the TCM land tenure. Sources were located from TCM archive servers, from Centerra-TCM's lead consultants and contractors, and from government publications. The first compilation report was delivered by Equity on February 15th, 2017, summarizing approximately 227,000 files comprising 404 GB of data.

In July 2017, Centerra-TCM began the process of building an in-house Exploration department including the hiring of an Exploration Manager. The objective of the new team was to identify exploration targets and additional high-quality resources on the property, to design and safely conduct systematic exploration programs to test these, to use enhanced and improved exploration methodologies, to develop a comprehensive 3D exploration model and well-managed database, and to support improved resource block modeling and mine planning. Spatially, the Mount Milligan exploration strategy was divided into three principal domains that define exploration programs and budgeting:

1. Near-pit/Within-pit - within the 2017 ultimate pit shell and boundary (as defined by Centerra Gold Inc., March 22, 2017 NI 43-101 Technical Report on Mount Milligan Mine authored by Andrews, Berthelsen, and Lipiec).
2. Brownfield – outside the 2017 ultimate pit shell but within the mine lease boundary;
3. Greenfield – outside the mine lease boundary but within the Mount Milligan property mineral tenure block.

These domains also roughly coincide with three scales of measurement within a clustered system of mineralized and potentially mineralized monzonite porphyry stocks. Boundaries between the three domains in this context may have some overlap.

1. Near-pit/Within-pit (~1.5 km diameter area) – porphyry core scale, known centre of porphyry system;
2. Brownfield (5-8 km diameter area) – porphyry cluster scale, linear trends of monzonite porphyry stocks related to a single cluster (Heidi Lake stock cluster);
3. Greenfield (up to 30 km diameter area) – stand-alone deposit scale, additional deposit that is unrelated to the Mount Milligan deposit and is not within the Heidi Lake stock cluster.

The 2017 Centerra-TCM exploration program included a ground-based geophysical survey conducted from August to November by Peter E. Walcott & Associates Limited ("Walcott"). A geophysical program totalling 67.6 line-kilometres (23 lines) of IP survey was completed over two target areas, Fugro-Rain (south of the mine lease in the greenfield area) and KC (east of the mine lease overlapping both brownfield and greenfield areas; Figs. 6-3, 10-3). A ground-based magnetic survey was also completed and comprised 309 line-kilometres at 100 m line spacing over the same areas. Results include a 1.3 x 1.7 km sub-circular magnetic high anomaly with coincident chargeability high at the Fugro-Lip target, one kilometre south of the mine lease; a WNW trending 1.9 x 0.8 km chargeability high with coincident magnetic low anomaly at the Fugro-2 target, 3.5 km south of the mine lease; and an arcuate 1.2 x 0.3 km magnetic high anomaly at the KC target, 500 m east of the TSF within the mine lease where two surface grab samples collected in August 2017 by Centerra-TCM geologists returned anomalous values of 0.128% Cu, 0.157 g/t Au and 0.146% Cu, 0.123 g/t Au.

Also in 2017, two memos were provided in June and October by an external geophysicist (M. Nosyrev) which reinterpreted historical Mount Milligan geophysical data including the 2010 Titan-24 DCIP geophysical survey (Quantec Geoscience Ltd.) and the 2009 South Grid 3DIP survey (SJ Geophysics Ltd.). The memos attempted to identify the geophysical features of known mineralization to help guide exploration drill targeting. Target geophysical signatures included the marginal gradient zones of steep or shallow dipping chargeability high anomalies, and marginal gradient zones of west-dipping moderate chargeability and resistivity high features interpreted to be porphyry intrusions. The first memo also provided a map of faults across the Mount Milligan deposit based on geophysical signatures; several of these were unmapped in historic geological models for the deposit. Both memos were used by Equity and Centerra-TCM geologists to develop interpretations of geological relationships in all three exploration domains: near-pit (NPI), brownfield and greenfield (Figure 9-2).

In 2018, a 15.5 line-kilometre IP geophysical survey (six lines at 100 m spacing; five east-west lines and one north-south line) was completed in May-June by Walcott in the KC target area east of the TSF (Figure 6-3, Figure 10-3). Several anomalous ring-shaped features identified in the 2017 ground magnetic survey and a northeast trending airborne resistivity high feature had lines run across them. One 400 m wide moderate chargeability and moderate-high resistivity target was identified near Rainbow Creek, one kilometre southeast of the TSF near the Meadows groundwater target area within the mine lease.

In 2019, geophysics programs completed by Walcott included those listed below. The detailed airborne magnetic surveys were conducted using a stinger mounted airborne magnetic system utilizing an Astar B2 helicopter provided by Silver King Helicopters Inc. of Smithers, B.C.

1) 631 line-km (38.9 km^2 area) low altitude aeromagnetic survey at 75 m line spacing flown on May 31st over the western side of the Mount Milligan property in both brownfield (49.1%) and greenfield (50.9%) exploration areas (Figure 10-3);
2) 1,437 line-km (129.0 km^2 area) low altitude aeromagnetic survey at 100 m line spacing covering the groundwater exploration area and surrounding areas north and east of the mine and TSF. Within lease (brownfield) coverage was 19.4^2 km (15.0%) and the rest was greenfield coverage (within property claim block) and beyond (Figure 10-3).
3) First phase of an IP survey (12 lines, 16.7 line-km, 200 m line spacing) was conducted from September 18th to October 17th across the North Slope, Goldmark and Oliver zones across a series of ENE trending faults (Figure 10-2).

4) Joint ZTEM-MT inversion was run using the ground MT (magnetotelluric resistivity) data collected in the 2010 Titan-24 DCIP geophysical survey to help guide the 2009 3D ZTEM inversion (Geotech Limited, 2009). The purpose of the joint inversion was to start developing a deeper view of the Mount Milligan porphyry system.

9.2. SUMMARY AND INTERPRETATION OF EXPLORATION ACTIVITIES

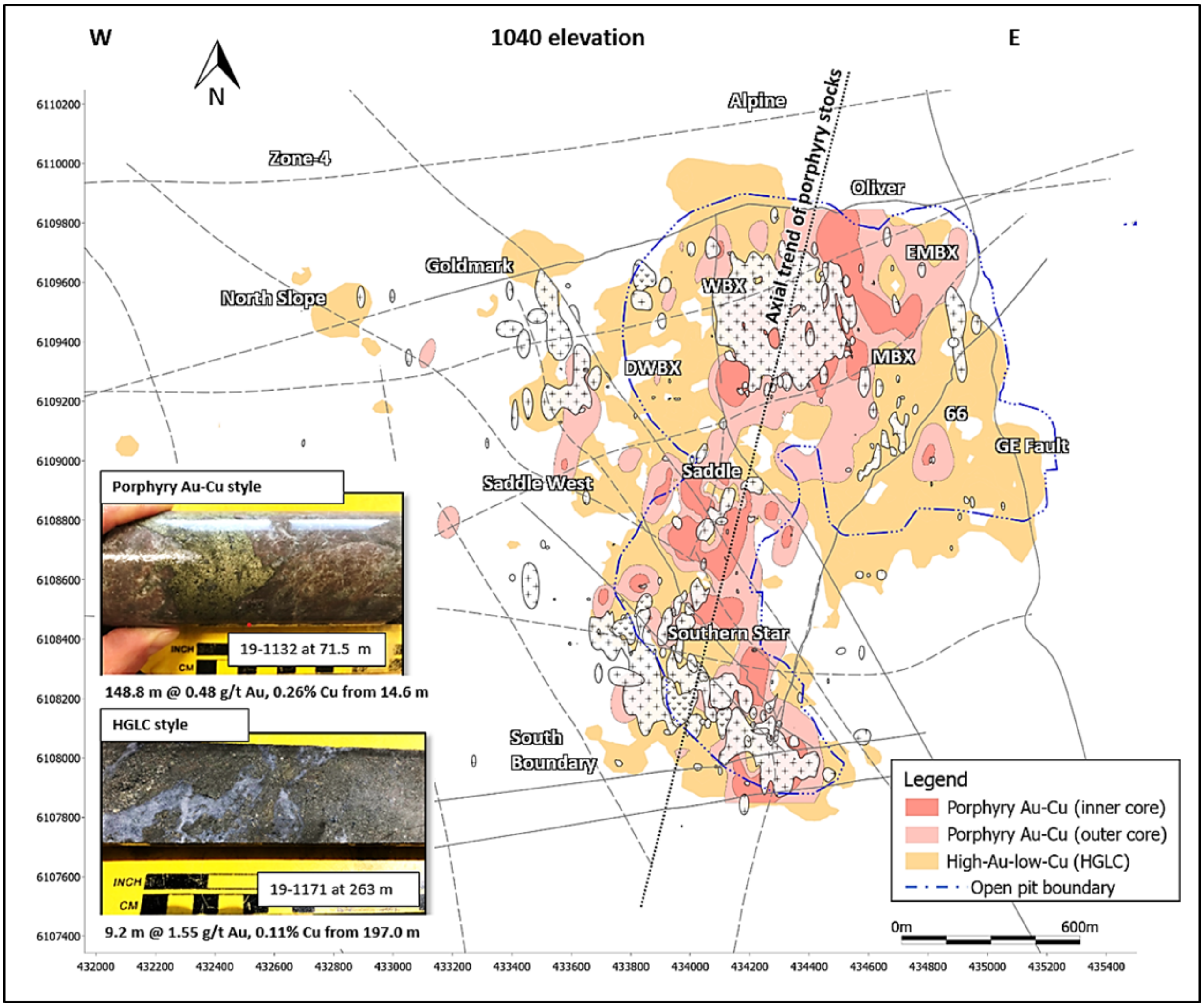
Two Styles of Mineralization

Since mid-2017, several conceptual exploration targets have been generated through the development of a 3D exploration model and complete drilling database, reinterpretation of historical geophysical data, enhancement of structural model for the deposit, ground-based and airborne geophysics programs, and exploration diamond drilling (Section 10). These activities have resulted in the identification of two distinct styles of mineralization at the Mount Milligan deposit.

1. Early-stage porphyry Au-Cu mineralization (and early-stage vein types) associated with composite monzonite porphyry stocks and related hydrothermal breccia, and narrower dyke and breccia complexes.
2. Late-stage, structurally controlled high-gold low-copper (HGLC) mineralization (and intermediate- to late-stage vein types) that is associated with faults and fault breccias, crosscuts/overprints the earlier stage porphyry mineralization and is more spatially widespread.

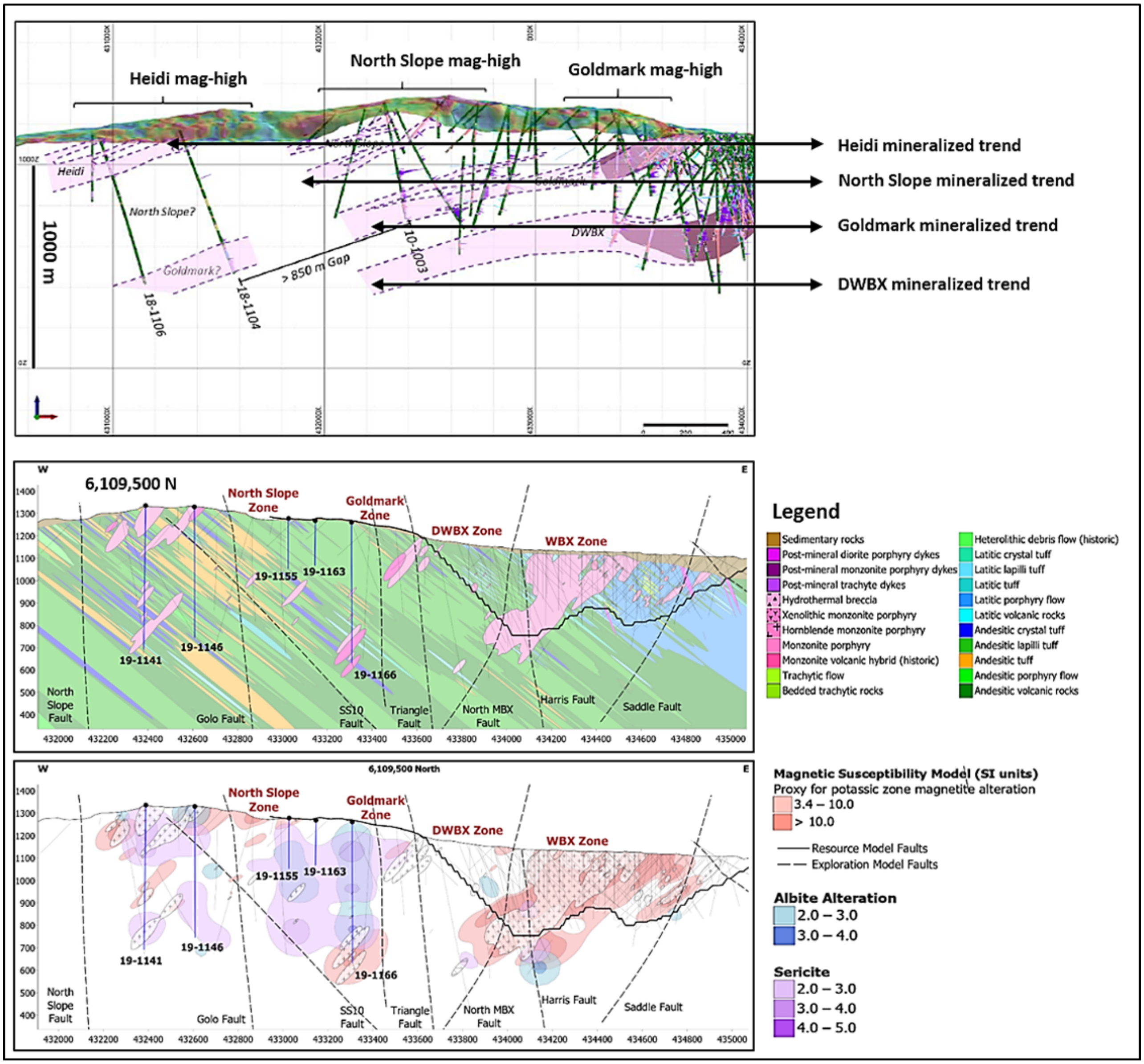
These two mineralization styles are zonal, overlapping, have different predominant sulfide species (chalcopyrite-pyrite versus pyrite only), and require different strategies in exploration, mining and metallurgy. Combined, these may expand economically viable mineralization in several directions including to the NNE-SSW along the axis of the main porphyry trend (the geometry of which may include a series of offsets on subparallel cross-faults; Figure 9-1); and west of existing resources following the ENE-WSW structural fabric and down-dip orientation of the porphyry stocks and dyke/breccia zones. There is an apparent shallow dip of mineralization to the west of the existing resource which is interpreted to be in part due to the moderate eastward tilt (westward dip) of the porphyry system as well as by repeated east-side-down offsets on northwest-trending subparallel normal or listric faults (Figure 9-2).

Figure 9-1: Mineralization of Mount Milligan Porphyries



Schematic zonal and overprinting distribution of porphyry Au-Cu and high-gold low-copper (HGLC) mineralization. Inset photos are representative of the two styles of mineralization. Interpreted NNE-SSW axial trend of porphyry stocks is also shown as are zone names. Faults (solid and dashed lines) are from the exploration model. Open pit boundary is the 2017 pit shell boundary at 1040m elevation above mean sea level.

Figure 9-2: Sectional Views of Mount Milligan Mineralization



View is to the north on all sections. *Top:* Schematic interpretation provided by Equity of a series of shallowly west-dipping mineralized trends associated with the DWBX, Goldmark, North Slope and Heidi zones from east to west near section 6,109,500 N. Magnetics (1VD) draped over topography. *Middle:* Interpreted moderate dip of monzonite intrusive units in the Centerra-TCM exploration model on section 6,109,500 N. The difference in attitude with the trends shown in top image may be due to a series of steeply dipping east-side-down normal faults and related offsets. *Bottom:* Alteration model showing magnetic susceptibility shells representing potassic alteration, and albite and sericite shells which are a proxy for the extent of the mineralizing paleo-hydrothermal system. Open pit is the 2017 pit shell boundary.

10. DRILLING

Diamond drilling at the Mount Milligan property was designed to test and delineate mineralized material, to obtain metallurgical samples, to condemn areas planned for infrastructure and to gather geotechnical and environmental information. A total of 238,851 m in 1,014 drill holes were drilled by Lincoln, United Lincoln, Continental Gold, Placer Dome, Terrane and TCMC (pre-Centerra Gold acquisition) between 1987 to 2016. Most of the drilling samples were collected from NQ (47.6 mm) diamond drill core. Since 2017, Centerra-TCM has completed a total of 81,445 m of drilling in 204 drill holes in drilling programs with various purposes including; metallurgical, near-pit infill and resource expansion (NPI), brownfield (within mine lease) exploration and greenfield (within property outside mine lease) exploration (Figure 10-1, Table 10-1).

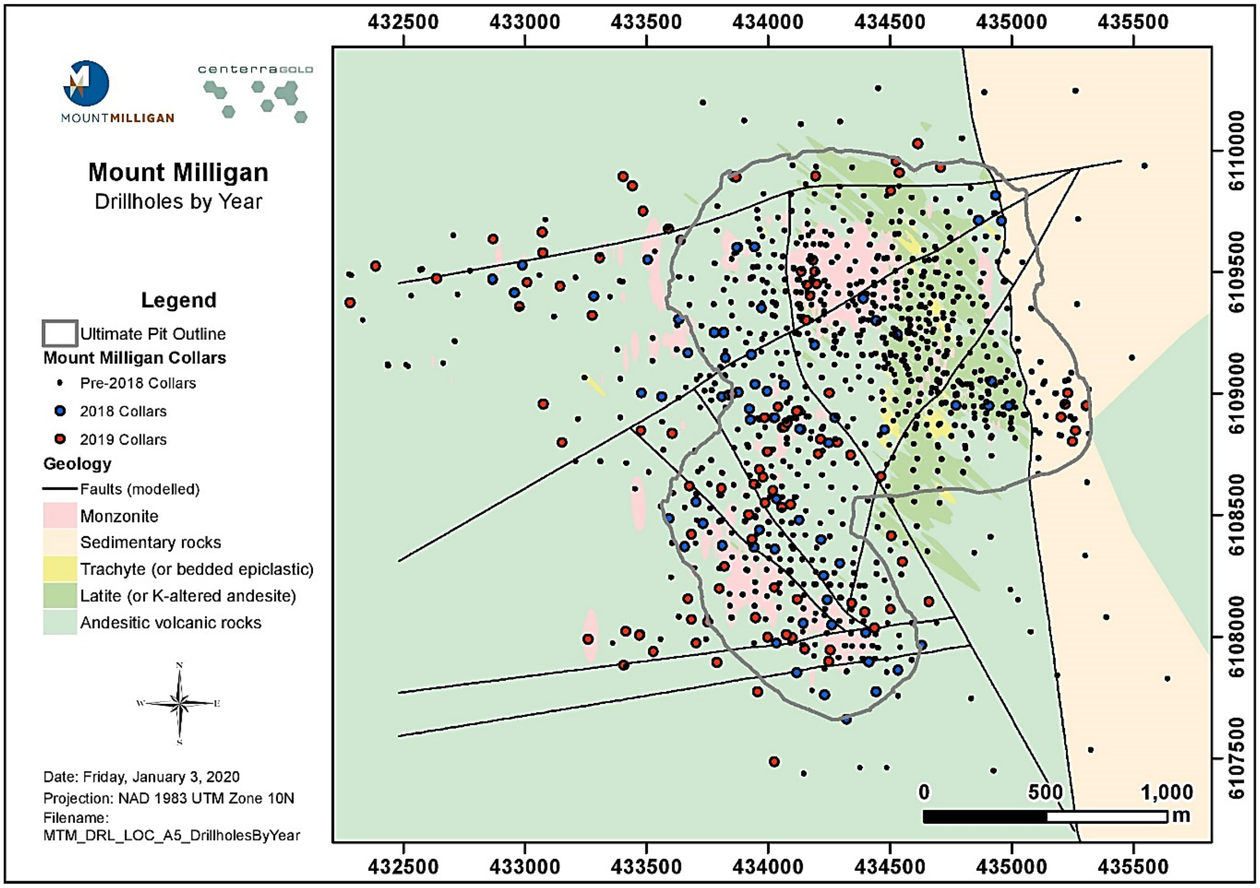
The complete historic drilling campaigns are summarized in Table 10-1. Only values for mineral-focused exploration and resource infill/expansion drilling are included (excludes groundwater exploration drilling and near surface till drilling for construction material).

Table 10-1: Drilling Programs Summarized by Year

Year	Metres	Holes	Company
1987	2,304	23	Lincoln
1988	6,645	47	United Lincoln
1989	87,662	336	United Lincoln, Continental
1990	82,924	386	Continental, Placer Dome
1991	17,969	90	Placer Dome
1992	526	4	Placer Dome
2004	2,184	14	Placer Dome
2006	9,557	36	Terrane
2007	10,515	33	Terrane
2010-11	14,984	36	TCM
2015	1,786	5	TCM
2016	1,795	4	TCM
2017	7,692	21	Centerra-TCM
2018	30,942	76	Centerra-TCM
2019	42,821	107	Centerra-TCM
Total	**320,306**	**1,218**	

Figure 10-1 shows all drill hole collar locations for the property, including historical and recent drilling programs.

Figure 10-1: Drill Hole Collar Map (all drilling)



10.1.DATA COLLECTION

Pre-2004 Drilling

Geotechnical information was collected for all drill holes after drill hole 87-70, and geological information was collected for all drilling conducted at the property. The information was collected on drill log forms that varied depending on the operator conducting the work and the type of deposit being targeted. Some drill holes were assayed for gold and not for copper, others for copper and not for gold. Lincoln, United Lincoln, and Continental Gold used a written form, with a header and descriptive format. The header included information on drill hole number, location, orientation, drilling company, geologist and dates. The descriptive section was broken down to intervals based on primary lithology, with descriptive reference to alteration, structure and mineralization.

Placer Dome used a coding system known as GEOLOG, with specific information captured on coded geologic logs, which were then entered into a computer and verified using GEOLOG software. The header section included drilling metadata similar to that recorded by previous operators. Geological information included lithology, alteration, and mineralization including both intensity/amount and mode of occurrence. GEOLOG forms were supplemented by graphical logs as well as descriptive remarks.

Geotechnical information was recorded on forms and captured in spreadsheets. Recorded information included core recovery, rock quality (RQD), hardness or compressive strength (CS), degree of breakage, degree of weathering or oxidation, fracture and joint frequency, and specific gravity (SG).

Geochemical analysis of silver was routinely conducted in early drilling with 14,896 samples analyzed from holes 88-61 to 89-212. Samples from later drilling were not analyzed for silver.

2004 Drilling (Placer Dome)

Placer Dome’s 2004 drill program was conducted to obtain core for metallurgical test work, with drill holes located within the MBX, 66, and Southern Star zones (Figure 9-1, Figure 10-1). Holes were twinned with or drilled near existing holes, with a total of 2,184 m drilled in 14 holes. Aggressive Diamond Drilling Ltd. of Kelowna, British Columbia conducted the drilling using a Boyles 56 drill rig.

Drilling information was captured directly to an acQuire database. All samples from the program were also analyzed for silver.

2006-2007 Drilling (Terrane)

In early September 2006, Terrane commenced a four-phase diamond drilling program. The first phase was designed to acquire samples for metallurgical test work. Initiated in late 2006, the second phase targeted mineralization on the west side of the MBX stock. The third phase commenced in March 2007 and was designed to gather geotechnical information across the MBX zone. The fourth phase commenced in June 2007 and was designed to gather geological and geotechnical information on the margins of the Southern Star zone, and in areas of proposed infrastructure.

Cyr Drilling International Ltd. (Cyr Drilling) of Manitoba conducted all four phases using a skid-mounted machine and HQ diameter drill rods. An ACE core orientation tool was used to provide oriented core for subsequent measurements.

Geological information was compiled on Excel spreadsheets for all drill holes completed by Terrane. The geological spreadsheets included drill hole numbers, locations, orientation, drilling dates and geologist name. The detailed geological information included lithology, alteration and mineralization, including both intensity/amount and mode of occurrence. Magnetic susceptibility information was collected at two metre intervals. Geotechnical information was recorded on a separate set of Excel spreadsheets and captured core recovery, RQD and fracture intensity. Specific gravity was also measured. As part of the sampling protocol developed by Terrane for the 2006-2007 drill program, every second sample was analyzed by a multi-element ICP package which included silver.

More detailed description of the four-phase diamond drilling program is as follows.

- **Phase I** (32 drill holes totalling 7,690 m) was designed to acquire fresh material for metallurgical test work. The program tested a range of lithologies and alteration types across the mineralized domains of the MBX and Southern Star deposits. The holes were collared in fences across the strike length of deposits, and targeted material that had been well-defined from historical drilling.
- **Phase II** (23 drill holes of HQ diameter core totaling 9,292 m) was designed to target areas of mineralization that had been less densely drilled by previous operators. Drilling targeted the DWBX which was intersected as part of the Phase I drill program. In addition, several holes were located between the Southern Star and MBX Main deposits.
- **Phase III** (5 drill holes of HQ diameter core totaling 1,525 m) was designed to gather further geological and geotechnical information along the perimeters of the MBX Main deposit, including on the western limits of the MBX stock through the Harris fault, on the northern margin of the MBX stock into altered host andesite volcanic rocks, and to the SSE from the MBX zone into the 66 zone. Twelve samples (20 cm each; 2.4 m total) were collected for unconfined compressive strength (UCS) testing; and 130 samples (20 cm each; 26 m total) for point load testing (PLT).
- **Phase IV** (9 drill holes of HQ diameter core totaling 1,565 m) included three geotechnical drill holes on the perimeter of the Southern Star deposit, and six condemnation drill holes designed to gather subsurface information in areas of the proposed infrastructure. The program followed completion of resource modeling for the 2008 report and results were included in resource modelling for the 2009 report. Condemnation drilling was located outside of the resource model limit and did not encounter significant mineralization.

2010-2016 Drilling (TCM)

In 2010-2011, 17 drill holes (10-1003 to 10-1012 and 11-1013 to 11-1019), totalling 10,535 m, were drilled for exploration potential. The program targeted an extension of the DWBX zone towards the western part of the Main deposit and north of the King Richard Creek.

In 2011, an additional 10 holes were drilled for metallurgical characterization of material to be mined within the Phase-3 pit envelope defined in 2009. These holes were named MET 11-01 to MET 11-10. A total of 867.5 m was drilled, logged, sampled and assayed.

Apex Diamond Drilling Ltd. (Apex) of Smithers, B.C. completed the 2010-2011 drilling using a skid-mounted machine and HQ and NQ diameter drill rods. No oriented core data was collected. Detailed core logging and sampling was done by Equity Exploration Consultants Ltd. (“Equity”) of Vancouver, B.C. and included lithology, alteration and mineralization. Geotechnical information captured included core recovery, RQD and fracture intensity. Data was captured using Geospark software, an Access-based relational database system that stores drill hole and geological data in individual data tables. Upon completion of logging, data was validated and exported as Excel files for each data table.

Exploration drilling resumed in 2015. TCMC contracted Equity to manage a five-hole (1,786 m) exploration drilling program focused on the Mitzi and Snell target areas in the northwest part of the property claim package, five kilometres northwest of the open pit (Figure 6-3). The program ran from October to November and one of the five drill holes was abandoned at 41.5 m depth. The aim of the program was to test the Mitzi and Snell coincident magnetic, chargeability and geochemical anomalies for porphyry-style Au-Cu mineralization. Previous work in the Snell target area included 10 drill holes completed in 1990 that failed to identify significant mineralization; these were drilled to relatively shallow depths below surface (45 m to 188 m). In 2008, the area was included in a property-wide HeliGEOTEM II airborne geophysical survey; and in 2009 was covered by the North Grid 2DIP survey (Figure 6-3) and grid soil sampling in 2009-10. This work rekindled interest in the Snell and Mitzi target areas as prospective for porphyry Au-Cu deposits. Drilling at Mitzi (588 m) tested the southwest edge of a ring-shaped chargeability high anomaly and intersected albite and phyllic (sericitic) alteration and anomalous copper values (301 ppm Cu over 9.52 m; DDH 15-1024). Drilling results at Snell (1,198 m) suggested the high chargeability and multi-element geochemical anomalies are related to epiclastic argillaceous rocks (Branson and Voordouw, 2016) in a narrow northwest trending fault-related basin.

In 2016, TCMC completed a four-hole (1795 m) exploration drilling program with two holes at Mitzi target (1,065 m) and two holes at Snell target (730 m). The first two drill holes tested the Mitzi aeromagnetic and surrounding ring-shaped chargeability high anomalies. At the Snell target, two drill holes were drilled to the north of the 2015 drill holes in the coincident aeromagnetic and chargeability high anomaly with anomalous multi-element geochemistry. Intervals of biotite-chlorite and sericite-chlorite alteration were intersected, and local weak pyrite and pyrrhotite mineralization with some chalcopyrite noted. Assays returned narrow intervals anomalous to weak Au-Cu mineralization.

L.D.S Diamond Drilling Ltd. of Kamloops, B.C. completed the 2015-2016 drilling programs using a skid-mounted machine and NQ diameter drill rods. No oriented core data was collected. Detailed core logging and sampling was done by Equity. Data was captured directly in a Geospark access database and included lithology, alteration, mineralization and structure. Magnetic susceptibility was measured every three metres using a Terraplus KT-10 magnetic susceptibility meter. Geotechnical information captured included core recovery and RQD. At completion of logging, data was validated and exported as a Microsoft Access database file (*.mdb).

2017-2019 Drilling (Centerra-TCM)

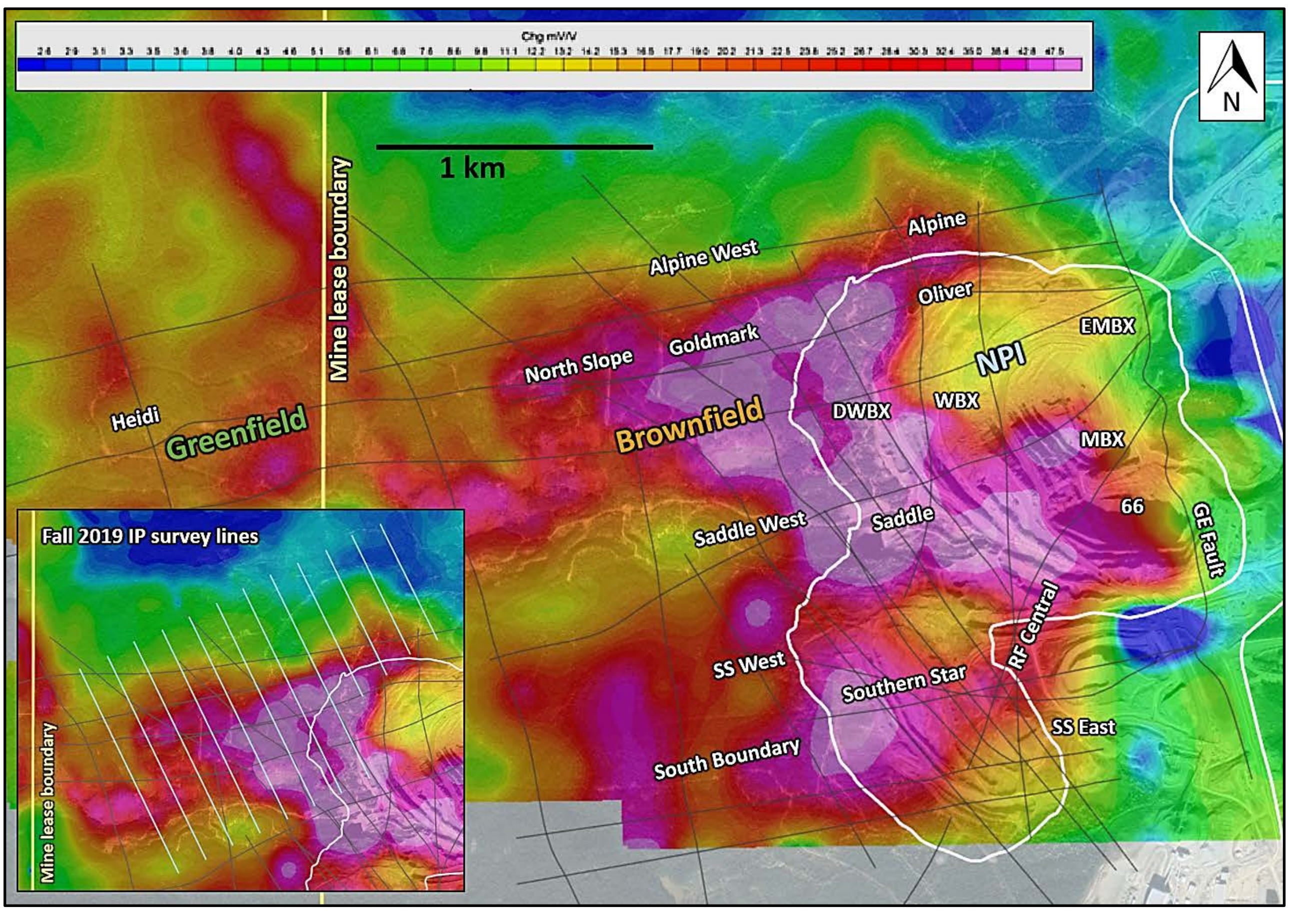
In 2017, an eight-hole (1,082 m) geometallurgical drilling program was completed in May in the MBX and 66 zones. This was followed in August to December by a 13-hole (6611 m) near-pit infill and expansion drilling program in the WBX, DWBX, MBX and Saddle zones within the 2017 ultimate pit boundary (as defined by the Centerra Gold Inc., March 22, 2017 NI 43-101 Technical Report on Mount Milligan Mine authored by Andrews, Berthelsen, and Lipiec). All eight of the geometallurgical drill holes (66 Zone) and four of the 13 exploration drill holes were included in the year-end resource update. Only four of the exploration holes were included because of delays with commercial labs and required QA/QC re-assays. The remaining nine holes were used in the 2018 year-end resource update.

In addition, a total of 32 till samples were collected for gold grain counts and indicator mineral grain analyses during a till drilling program in the second quarter from the Phase 5 and Phase 8 mining areas. Annual till drilling programs by TCMC were designed to test for construction material for dam building purposes. Samples (~20 kg wet each with average sample length of 3.3 m) were collected from eight shallow drill holes over a 915 m east-west trend at the northern margin of the Southern Star zone (Phase-8), and in 10 drill holes over an 890 m trend at the north-northeast corner of the MBX pit, outside the EMBX zone (Phase-6; Figure 9-1). Average drilling depth was 30 m. For the first batch of 15 samples analyzed by Overburden Drilling Management Limited (ODM) of Ottawa, Ontario, the total calculated visible gold grain count in heavy mineral concentrate (HMC) ranged from

43 - 1,653 ppb Au, averaging 511 ppb Au (0.51 g/t Au). A second batch of 17 samples ranged between 35 - 62,755 ppb Au, averaging 4,667 ppb Au (4.67 g/t Au). Excluding the two highest grade samples (62,755 ppb and 9,344 ppb) the average of the remaining 15 samples was 483 ppb Au (0.48 g/t Au).

In 2018, Centerra-TCM completed a 33-hole (12,167 m) Phase-1 near-pit infill and expansion (NPI) drilling program that ran from February to July in multiple zones within the 2017 ultimate pit boundary, including the DWBX, WBX, MBX, Saddle, Southern Star, EMBX and 66 zones (Figure 9-1, Figure 10-2). An 18-hole (6490 m) Phase-2 NPI program ran from September to November in the Saddle and Southern Star zones. Results continued to demonstrate continuity of significant mineralization both within and below the 2017 ultimate pit shell.

Figure 10-2: Mount Milligan Exploration Zones



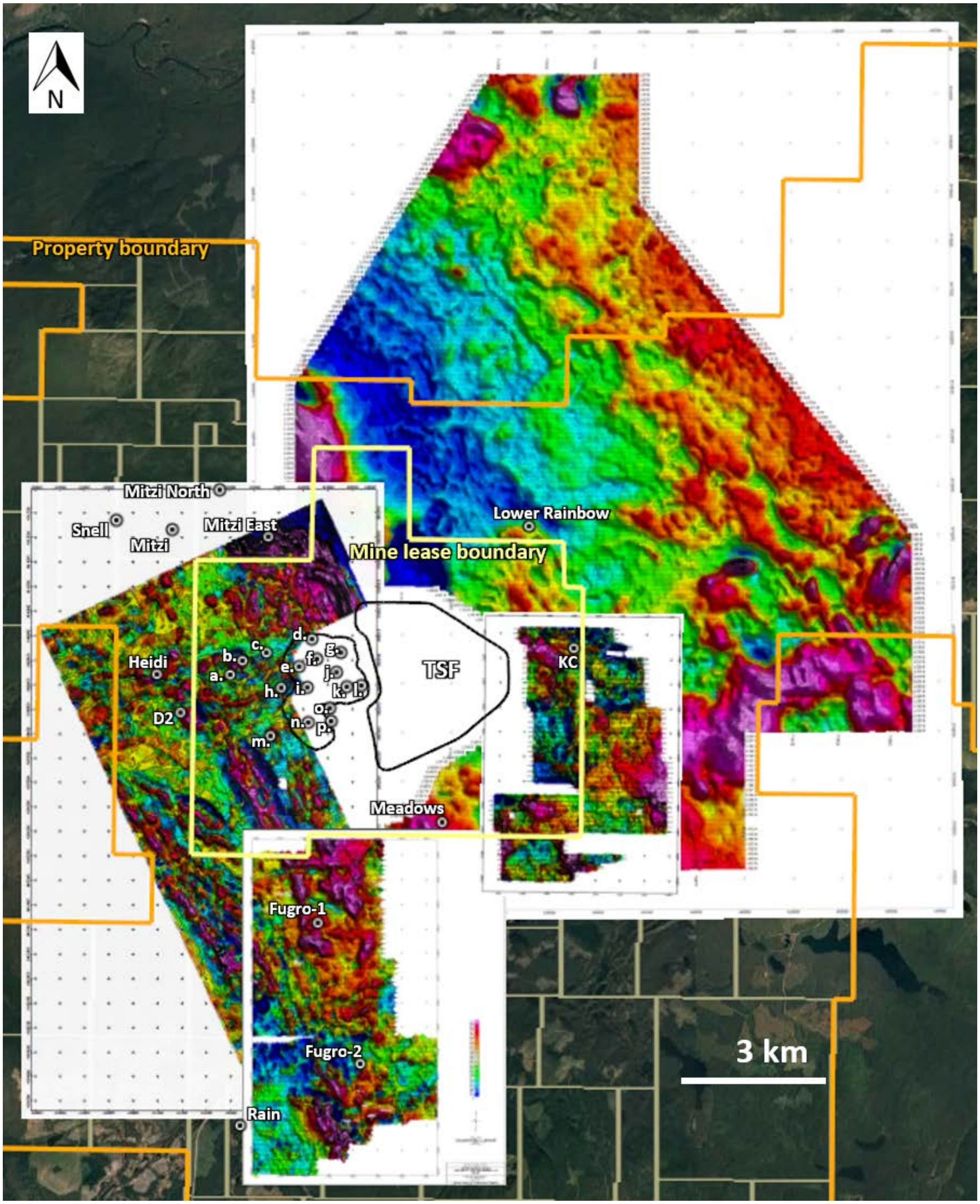
Exploration zones for NPI, Brownfield and Greenfield areas of the Mt. Milligan deposit and Mount Milligan property. Grey lines include both modelled and conceptual faults. White lines are 2017 ultimate pit boundary and TSF. Background is IP chargeability and LiDAR. The chargeability scale ranges from 0-47.5 mV/V. Inset map shows the September-October 2019 ground-based IP survey lines (18 line-km).

Centerra-TCM exploration programs in 2018 included a ground-based 12-hole (6,669 m) brownfield (within mine lease) drilling program that ran from June to August in the Goldmark, North Slope and Saddle West (King Richard) zones west of the 2017 ultimate pit boundary (Figure 9-1, Figure 10-2). Results indicated two domains of shallow high-gold low-copper (HGLC) mineralization in the combined Goldmark-North Slope zone and Saddle West zones, as well as indications of deep porphyry Au-Cu mineralization, particularly in the North Slope zone.

A total of 46 drill holes from the NPI and brownfield programs were included in the year-end resource update, expanding the 2018 Measured and Indicated resource shell (at CAD $8.12 NSR cut-off) to the west and at depth.

A helicopter-supported 13-hole (5,616 m) greenfield (outside mine lease) drilling program ran from June to August in the Mitzi East, Heidi, D2 and Fugro zones located north, west and south of the mine lease (Figure 6-3, Figure 10-3). The drilling at Mitzi East completed a 4,000 x 500 m wide fence with 2015-2016 Mitzi drilling outside the northwest corner of the mine lease and was used to delineate the margins and far-field signature of the porphyry system to the north. There were no significant assay results. A shallow copper target was identified in the Heidi zone, 800 m west of the mine lease.

Figure 10-3: Geophysics Maps Plotted on Google Earth Image (2019)



Google Earth image (2019) with compilation of ground-based (Fugro and KC) magnetic surveys completed in 2017 and low altitude aeromagnetic surveys completed in 2019 by Centerra-TCM and Peter E. Walcott & Associates Limited. The maps show contours of total magnetic field intensity (TMI; nT). Scales of nT values vary between map, Fugro (55844-56480), KC (55886-56256), Western block (55709-59342), Eastern block (55544-56298). Lettered target zones are: a. D3, b. North Slope, c. Goldmark, d. Oliver, e. DWBX, f. WBX, g. EMBX, h. Saddle West (King Richard), i. Saddle (Creek), j. MBX, k. 66, l.GE Fault, m. South Boundary, n. Southern Star, o. RF Central, p. SS East. Ultimate pit (2017) and TSF boundaries shown in black line.

In April 2018, gold grain analysis results from HMC were received for an additional 20 till samples taken from two drill holes in the 2018 till drilling program (S18-073 and S18-076). Material (averaging 7.7 wet kg per sample) was collected at 1.5 m to 3.0 m intervals through the complete till column of two interpreted east-west trending paleo-channels at the north and south end of the Southern Star zone, the channels separated by 900 m. Calculated total visible gold in HMC ranged from 0-893 ppb, with median value of 290 ppb. The median value for all 52 till samples collected in the near-pit area since 2017 is 296 ppb Au.

In December 2018, Centerra-TCM completed a five-hole (567 m) Phase-1 scout drilling program for groundwater exploration east and north of the TSF. This program tested results of Nuclear Magnetic Resonance (NMR) and Transient Electromagnetic (TEM) geophysical surveys completed between August and November and was designed to support water well targeting efforts for process plant operations. Three of the five drill holes in two target areas (Alpine Lake and Lower Rainbow) produced low flow water and three holes reportedly reached top of bedrock below the surficial geology. No significant assay values were returned from bedrock samples.

In 2019, Centerra-TCM completed a 53-hole (19,832 m) Phase-1 near-pit infill and expansion (NPI) drilling program that ran from February 22nd to September 14th in multiple zones within the 2017 ultimate pit boundary or marginal to it, including Southern Star (and SS Satellite Pit appendage), Saddle, WBX, GE Fault and Oliver zones (Figure 10-2). A 19-hole (7,455 m) Phase-2 NPI program ran from September 15th to November 30th in the WBX, Oliver, GE Fault, RF Central and SS East zones. Significant mineralization was intersected in each zone (less so for RF Central). Assay results continue to demonstrate significant intervals of additional porphyry Au-Cu mineralization both within, below and outside the margins of the 2017 ultimate pit shell.

Exploration drilling programs in 2019 included a ground-based 19-hole (8,672 m) brownfield program that ran from April 30th to August 26th in the North Slope, Goldmark, Saddle West (King Richard), SS West and South Boundary zones west of the 2017 ultimate pit boundary. A 12-hole (5,634 m) Phase-2 brownfield drilling program ran from August 26th to December 16th in the same zones. Results further defined shallow high-gold low-copper mineralization as being spatially associated with parallel ENE-WSW (and possibly NNW-SSE) trending fault structures and related breccias; verified low-grade porphyry Au-Cu mineralization in the North Slope zone between ~340-620 m depth related to a dyke and breccia complex that remains open in several directions; and intersected low-grade porphyry Au-Cu mineralization at comparable depth in the SS West (~560-670 m) and Saddle West zones (~400-430 m). In addition, a ground-based four-hole (1,228 m) greenfield (outside mine lease) drilling program ran from July 5th-28th in the Heidi zone that further defined the extent and style of shallow low-grade copper mineralization.

In February 2019, Centerra-TCM completed a five-hole (611 m) Phase-2 scout drilling program for groundwater exploration east and northeast of the TSF. This program continued following up results of NMR and TEM geophysical surveys completed in 2018 and was designed to support water well targeting efforts for process plant operations. One hole in the Upper Rainbow (19-WE08) target area showed indications of groundwater and one hole in the Lower Rainbow (19-WE09) produced flowing artesian conditions. No holes reached bedrock.

Project management of mineral-focused drilling programs in 2017-2018 was conducted jointly by Centerra-TCM geologists and Equity consultants, and solely by Centerra-TCM geologists in 2019. L.D.S Diamond Drilling Ltd. of Kamloops, B.C. completed the 2017-2019 drilling using Longyear 38 skid-mounted core drills with mainly NQ and lesser HQ diameter drill rods. Apex Diamond Drilling Ltd. of Smithers, B.C. completed the helicopter supported greenfield exploration program in 2018 using two Hydracore 2000 helicopter-portable core drills and NQ diameter drill rods. Oriented core data was collected in 2018-2019 using a Boart Longyear Trucore tool, marked on the bottom of each run. The azimuth of the drill was aligned using a hand-held magnetic compass or the Reflex TN14 north-seeking gyrocompass aided by foresights and back sights laid out by the mine survey team using a Trimble differential GPS (DGPS) unit. All drill core was logged, photographed, sampled and assayed.

At the core logging facility, geotechnicians would reassemble the core and draw a blue orientation line along the bottom of the core. The angular difference between runs, the quality of core reassembly, and the quality of the orientation mark were recorded as measures of orientation QA/QC. Alpha and Beta angles of structural features were recorded by a structural geologist using a Reflex IQ-logger tool and plotting software. The IQ-logger can capture structural measurements and plot them in real time on a stereonet. Structural data measurements are automatically organized by depth and corrected to true-north azimuth and dip.

Detailed core logging and sampling was done by Equity in 2017, both Equity and Centerra-TCM in 2018, and by Centerra-TCM in 2019. Data was captured directly in a Geospark access database and included lithology, alteration, mineralization, detailed vein data using a modified vein classification scheme after Jago et al. 2014, and structure. Starting with drill hole 19-1215 onwards, data was captured directly into an acQuire database.

Geotechnical information collected included core recovery, RQD, specific gravity and point load testing. Magnetic susceptibility was measured every metre using a Terraplus KT-10 magnetic susceptibility meter. Specific gravity measurements were collected every 10 m on all resource and Brownfield exploration drill holes. On average, samples selected for SG measurements were 10 cm long. Specific gravity values were calculated by dividing the dry mass of a sample by its wet mass. The wet mass was measured by measuring the mass of the rock sample suspended in water. Point load measurements were conducted by geotechnicians every 10 m on all resource drill holes using a Roctest PIL-7 point-load tester.

10.2. SUMMARY AND INTERPRETATION OF 2017-2019 DRILLING (CENTERRA-TCM)

This sub-section summarizes the current understanding of exploration target zones resulting from drilling programs in 2017 to 2019. The summary of zones goes from north to south generally, and each zone is indicated as being within the scope of NPI, Brownfield, or Greenfield drilling programs. For each zone, selected significant composite assay intervals are provided. To be considered significant for exploration results, composite assay intervals must be longer than 2.0 m, have grade greater than 0.1 g/t Au or 0.1% Cu and include maximum internal waste of 4.0 m where it exists. Intervals less than 2.0 m but with grade above 1.0 g/t Au are also reported.

North Slope Zone (and Zone-4) – Brownfield

Porphyry Au-Cu mineralization in the North Slope zone is associated with a variably faulted, monzonite porphyry dyke and breccia/fracture zone complex. The dyke and breccia complex controls the apparent shallowly west-dipping ‘North Slope’ mineralized trend which is suprajacent to the deeper western extension of the ‘Goldmark’ trend (Figure 9-2). Significant porphyry Au-Cu style mineralization ranges from about 300 m to over 600 m depth in the west part of the zone within a NNW trending moderate chargeability feature. Farther east towards the Goldmark zone, porphyry style mineralization appears to shallow slightly and ranges from about 250-400 m depth. Overprinting HGLC mineralization becomes more prevalent in the east part of the zone towards the chargeability high associated with the Oliver fault (Figure 10-2); this can also be seen in the alteration model where there is a higher intensity of sericitic ± albitic alteration between the North Slope and Goldmark zones (Figure 9-2).

Selected North Slope zone intercepts from 2018 drilling include:

- 18-1097: 99.00 metres @ 0.346 g/t Au, 0.175% Cu from 283.50 metres;
- *including 2.86 metres @ 2.580 g/t Au, 0.682% Cu from 305.08 metres;*
- 18-1107: 18.34 metres @ 1.408 g/t Au, 0.076% Cu from 66.00 metres;
- *including 1.34 metres @ 16.200 g/t Au, 0.327% Cu from 83.00 metres.*

There is potential for additional discovery on the northern part of the North Slope zone, north of the Oliver fault, where it adjoins Zone-4 (Figure 9-1). This area remains untested by drilling and comprises several shallow and deep geophysical targets. These are proximal to a modelled fault intersection and are associated with a steep chargeability gradient anomaly which intersects west-dipping coincident moderate resistivity-chargeability anomalies and shallow magnetic highs.

Goldmark Zone and Goldmark-Oliver Trend – Brownfield

The Goldmark zone is centered on the N-S trending, west-dipping Goldmark monzonite porphyry stock and dyke complex. It is situated in the center of the chargeability halo that surrounds the MBX Main zone (Figure 10-2). The Goldmark mineralized trend is suprajacent to the western extension of the composite DWBX stock and associated mineralization (Figure 9-2). Low grade porphyry Au-Cu style mineralization is concentrated at the hanging-wall/footwall margins of dykes and at the footwall margin of the Goldmark stock; and in the deeper zone at the margins of the DWBX stock. HGLC style mineralization occurs throughout the zone and at shallow levels. It is spatially associated with minor fault and fracture zones, and with lithological contacts including the faulted footwall margin of the Goldmark stock and the margins of some hornblende monzonite porphyry dykes. It also appears to follow the ENE fault trend and associated high chargeability zone (Figure 10-2).

Selected Goldmark zone intercepts from 2018-2019 drilling include:

- 18-1086: 23.12 metres @ 6.210 g/t Au, 0.043% Cu from 20.19 metres;
- *including 3.67 metres @ 38.052 g/t Au, 0.081% Cu from 38.33 metres;*
- 19-1166: 12.00 metres @ 2.498 g/t Au, 0.169% Cu from 250.00 metres;
- *including 0.93 metres @ 30.000 g/t Au, 0.254% Cu from 252.07 metres;*
- 19-1234: 13.31 metres @ 5.828 g/t Au, 0.032% Cu from 152.20 metres;
- *including 6.80 metres @ 11.070 g/t Au, 0.043% Cu from 152.20 metres.*

There is potential for adding near surface HGLC resource close to the northern margin of the 2017 ultimate pit boundary following the ENE trending Oliver fault if continuity of mineralization can be demonstrated. The 2019 Goldmark-Oliver IP survey was completed across this trend from the North Slope zone in the west to Oliver zone in the east and will guide further exploration for shallow HGLC mineralization close to the pit. Drill targeting in this area generally tests a shallow east dipping chargeability feature and gradient zone. There is less potential for additional economic low-grade porphyry style mineralization in the Goldmark zone except in the footwall of the Goldmark stock, and at greater depth (~450 m and deeper) in the hanging-wall and footwall margins of the DWBX stock.

Oliver and EMBX Zones – Brownfield and NPI

The Oliver zone covers the northern margin of the 2017 ultimate pit. This is an area where potassic altered andesite has been identified historically, suggesting the potassic alteration shell of the MBX Main deposit is open to the north where drilling is sparse. The target area also includes the northern portion of the axial trend of porphyry stocks, the eastern portion of the Goldmark-Oliver high gold trend, and their intersection (Figure 9-1). Drill targeting in this area generally tests the same shallow east dipping chargeability feature as in the Goldmark zone, but the feature is deeper at Oliver.

Significant mineralized intercepts from 2018-2019 drilling are spatially related to faults and faulted margins of monzonite porphyry dykes, and have more of an HGLC signature. These include:

- 19-1221: 32.00 metres @ 0.372 g/t Au, 0.049% Cu from 362.00 metres;
- *including 1.45 metres @ 2.295 g/t Au, 0.048% Cu from 367.00 metres;*
- 19-1225: 24.70 metres @ 0.624 g/t Au, 0.011% Cu from 245.80 metres;
- *including 8.00 metres @ 1.564 g/t Au, 0.007% Cu from 256.50 metres.*

The EMBX zone (East MBX zone) also lies within the northern extent of the potassic alteration shell associated with the MBX Main deposit but is inside the 2017 ultimate pit boundary. Significant intercepts from the 2018-2019 drilling are spatially related to faults and faulted lithological contacts within the host volcanic rocks and two stages of alteration (potassic and an albite bearing propylitic overprint).

- 18-1078: 19.15 metres @ 1.574 g/t Au, 0.076% Cu from 98.05 metres.
- *including 5.20 metres @ 3.431 g/t Au, 0.093% Cu from 112.00 metres.*

MBX, WBX and DWBX Zones – NPI

The MBX, WBX, and DWBX zones are mostly within the 2017 ultimate pit shell except at depth. Drill holes from 2018-2019 programs are dual purpose as they are collared in the pit for infill drilling (resource classification upgrade) and extended below the ultimate pit shell for resource expansion. Drilling has shown continuity of significant mineralization both within and below the ultimate pit shell. Programs in the MBX and WBX zones have targeted the footwall of the MBX stock, provided more pierce points through the Harris Fault, tested the extent of an intrusive sill feature subjacent to the Rainbow Dyke, and infilled low density drilling where needed to support mineral resource classification. Deep geophysical targets include a steeply dipping chargeability gradient zone that may be related to a north-south trending fault that is not recognized in surface mapping. Both porphyry Au-Cu and HGLC style mineralization have been identified in drilling. Significant intercepts in the MBX and WBX zones from 2018-2019 drilling include:

- 18-1069: 100.75 metres @ 0.417 g/t Au, 0.364% Cu from 23.00 metres;
- 18-1070: 72.34 metres @ 0.519 g/t Au, 0.232% Cu from 3.66 metres;
- *including 3.75 metres @ 3.495 g/t Au, 0.456% Cu from 37.23 metres;*
- *and 3.40 metres @ 1.129 g/t Au, 0.384% Cu from 57.00 metres;*
- 19-1207: 120.00 metres @ 0.250 g/t Au, 0.222% Cu from 164.00 metres.

The DWBX zone lies on the western side of Harris Fault in a downthrown fault block relative to the adjacent block that hosts the WBX and MBX zones. Because of this, the DWBX stock (an interpreted offset root of the MBX stock) is at greater depth than the MBX stock and there is more of a peripheral alteration-mineralization signature near surface (Figure 9-2). Some narrow HGLC intersections have been returned from recent drilling in the shallower DWBX zone. The composite DWBX stock has porphyry Au-Cu mineralization along its hanging-wall and footwall margins. Recent drilling has been designed to infill the west wall of the ultimate pit and extend mineralization related to the DWBX stock westward and deeper. The DWBX stock and associated breadth of mineralization appears to taper towards the north, however a moderate west-dipping hydrothermal breccia body has been identified at shallow depth in host volcanic rocks in the northern part of the DWBX zone. This feature is believed to be associated with emplacement of porphyry stocks and remains to be tested down-dip to the west and into the Goldmark stock footwall volcanic rocks.

Significant mineralized intercepts in the DWBX zone from 2017 drilling include:

- 17-1049: 15.00 metres @ 13.488 g/t Au, 0.031% Cu from 290.00 metres;
- *including 7.12 metres @ 28.068 g/t Au, 0.026% Cu from 296.58 metres;*
- 17-1049: 57.34 metres @ 1.074 g/t Au, 0.076% Cu from 328.16 metres;
- *including 13.45 metres @ 3.908 g/t Au, 0.068% Cu from 351.90 metres;*
- 17-1051: 45.20 metres @ 1.780 g/t Au, 0.044% Cu from 60.20 metres;
- *including 7.50 metres @ 5.510 g/t Au, 0.059% Cu from 60.20 metres;*
- *and 20.60 metres @ 1.569 g/t Au, 0.058% Cu from 75.30 metres.*

66 and GE Fault Zones – NPI

The 66 zone lies on the southeast side of Rainbow Fault, predominantly in the hanging-wall block. It is interpreted as a downthrown fault block (~100 m of throw) and a preserved portion of the sericitic (quartz-sericite-pyrite-carbonate; QSPC) alteration shell overprinting potassic alteration, from a higher level of the porphyry deposit, likely the upper part of the MBX Main zone. It is the foremost setting for HGLC style mineralization and associated vein types at the Mount Milligan deposit. This mineralization is more widespread in the 66 zone than elsewhere in the deposit where it is more narrowly confined to fault and breccia structures. Recent work has highlighted the importance of narrow porphyry dykes to mineralization in the volcanic stratigraphy. Higher gold grade appears be localized in the hanging-wall of a north-dipping monzonite porphyry unit, possibly an offset southeastern extension of the Rainbow Dyke. If an extension of the Rainbow Dyke, it is conceivable that this mineralizing intrusion represents a cone sheet that formed above the MBX stock during emplacement and spans across the MBX and 66 zones. Bedded/banded and massive trachytic rocks in the volcanic stratigraphy are favourable hosts to mineralization when proximal to, or intersecting with, porphyry dykes. A deeper porphyry unit, historically called the “Lower Monzonite”, appears to be un-mineralized but is not well tested by drilling.

Recent drilling programs have followed up HGLC intersections from the 2017 metallurgical drilling program, and tested for continuity and controlling factors of mineralization. The southeast-dipping Rainbow fault appears to cut off mineralization at depth but there is potential for additional HGLC mineralization down-dip of the fault within its hanging-wall below the 2017 ultimate pit shell. Significant intercepts in the 66 zone from 2018 drilling include:

- 18-1080: 32.99 metres @ 1.100 g/t Au, 0.003% Cu from 3.96 metres;
- 18-1080: 125.00 metres @ 0.914 g/t Au, 0.011% Cu from 47.00 metres;
- *including 7.00 metres @ 7.408 g/t Au, 0.022% Cu from 72.00 metres;*
- 18-1084: 55.95 metres @ 1.990 g/t Au, 0.044% Cu from 3.05 metres;
- *including 7.00 metres @ 1.560 g/t Au, 0.042% Cu from 25.00 metres;*
- *and 12.10 metres @ 6.589 g/t Au, 0.035% Cu from 42.00 metres.*

The GE Fault zone lies east of the 66 zone on the east side of the Great Eastern Fault within its hanging-wall. Historically, the Great Eastern Fault was interpreted to be steeply dipping and truncate the deposit on the east side. However, recent TEM geophysical modelling suggests the fault is moderately east dipping and may represent a faulted disconformity within the volcanic stratigraphy. As such, there is potential for extension of 66 zone mineralization to the east in the footwall block of the Great Eastern Fault and beyond the limits of the ultimate pit. Drilling in 2019 has shown that significant mineralization is spatially associated with faults and fault breccia with monzonite porphyry clasts and early-stage vein fragments, and quartz rich sericitic (QSPC) alteration overprinting early-stage potassic. Significant intercepts from 2019 drilling include:

- 19-1169: 42.88 metres @ 0.215 g/t Au, 0.226% Cu from 111.00 metres;
- 19-1213: 33.40 metres @ 0.662 g/t Au, 0.056% Cu from 152.00 metres;
- *including 3.07 metres @ 4.700 g/t Au, 0.180% Cu from 160.93 metres;*
- 19-1213: 8.50 metres @ 4.547 g/t Au, 0.061% Cu from 283.00 metres;
- *including 5.50 metres @ 6.756 g/t Au, 0.031% Cu from 286.00 metres.*

Saddle and Saddle West Zones – NPI and Brownfield

The Saddle zone (formerly Creek zone) is where prospector R. Haslinger made his creek bed discovery. This area was less densely drilled historically due to accessibility issues related to King Richard Creek. The creek was dammed as part of mine construction and drilling resumed with Centerra-TCM in 2017. The zone is centred on a north-south trending, moderately west-dipping monzonite porphyry stock called the Unnamed stock and subjacent sheeted dyke complex, situated midway (~400 m) between the MBX stock to the NNE and the Southern Star stock complex to the SSW (Figure 9-1). The Saddle zone comprises shallow HGLC style mineralization related to transitional and late-stage veins near faults in the Unnamed stock hanging-wall; a subjacent magnetite hydrothermal breccia and early-stage vein zone with porphyry Au-Cu style mineralization in the footwall of the Unnamed stock; a subjacent wedge of well mineralized andesite, and a sheeted porphyry dyke complex of which the coarse-crystalline crowded plagioclase monzonite units appear to be the best Au-Cu mineralized. In geophysics, the Saddle zone is represented by a west dipping moderate chargeability and resistivity feature that is flanked by domains of high chargeability and low resistivity.

Drilling in 2018-2019 tested for up- and down-dip extension of mineralization in the footwall of the stock; for mineralization at depth below the ultimate pit boundary; for a high-grade shoot of porphyry Au-Cu mineralization projecting southward towards the Southern Star zone; and infill drilling for the resource model. Selected intercepts in the Saddle zone from the 2018-2019 drilling include:

- 18-1056: 57.20 metres @ 0.630 g/t Au, 0.274% Cu from 212.50 metres;
- *including 5.40 metres @ 2.340 g/t Au, 0.986% Cu from 230.60 metres;*
- *and 7.70 metres @ 0.959 g/t Au, 0.477% Cu from 240.00 metres;*
- 18-1057: 149.45 metres @ 0.456 g/t Au, 0.435% Cu from 178.40 metres;

- *including 9.96 metres @ 1.418 g/t Au, 1.048% Cu from 219.18 metres;*
- *and 2.00 metres @ 1.114 g/t Au, 0.788% Cu from 258.52 metres;*
- 18-1058: 89.85 metres @ 0.445 g/t Au, 0.466% Cu from 160.92 metres;
- *including 2.64 metres @ 1.181 g/t Au, 0.517% Cu from 203.38 metres;*
- 19-1158: 76.75 metres @ 0.318 g/t Au, 0.375% Cu from 64.00 metres.
- *including 2.00 metres @ 1.127 g/t Au, 1.150% Cu from 128.00 metres.*

The Saddle West (or King Richard zone) zone is centered on the ENE trending Saddle Fault, south of the Goldmark zone within Heidi Lake valley. Till overburden in the valley ranges from 10-100 m. Drilling in this area has targeted a down-dip extension of the Saddle zone. Results suggest the area is prospective for shallow HGLC style mineralization in a parallel ENE trend to the Goldmark-Oliver trend (~700 m to the north). Significant intercepts show mineralization is associated with faulted/fractured margins of monzonite porphyry dykes, and with transitional to late stage veins with sericitic alteration overprinting early-stage potassic alteration. Significant intercepts from 2018-2019 drilling include:

- 18-1093: 23.04 metres @ 0.982 g/t Au, 0.073% Cu from 242.00 metres;
- *including 2.89 metres @ 6.537 g/t Au, 0.115% Cu from 262.15 metres;*
- 18-1108: 7.89 metres @ 4.687 g/t Au, 0.034% Cu from 112.11 metres;
- *including 3.89 metres @ 9.357 g/t Au, 0.024% Cu from 112.11 metres;*
- 19-1185: 14.92 metres @ 2.135 g/t Au, 0.263% Cu from 97.08 metres;
- *including 7.07 metres @ 4.293 g/t Au, 0.449% Cu from 99.00 metres.*

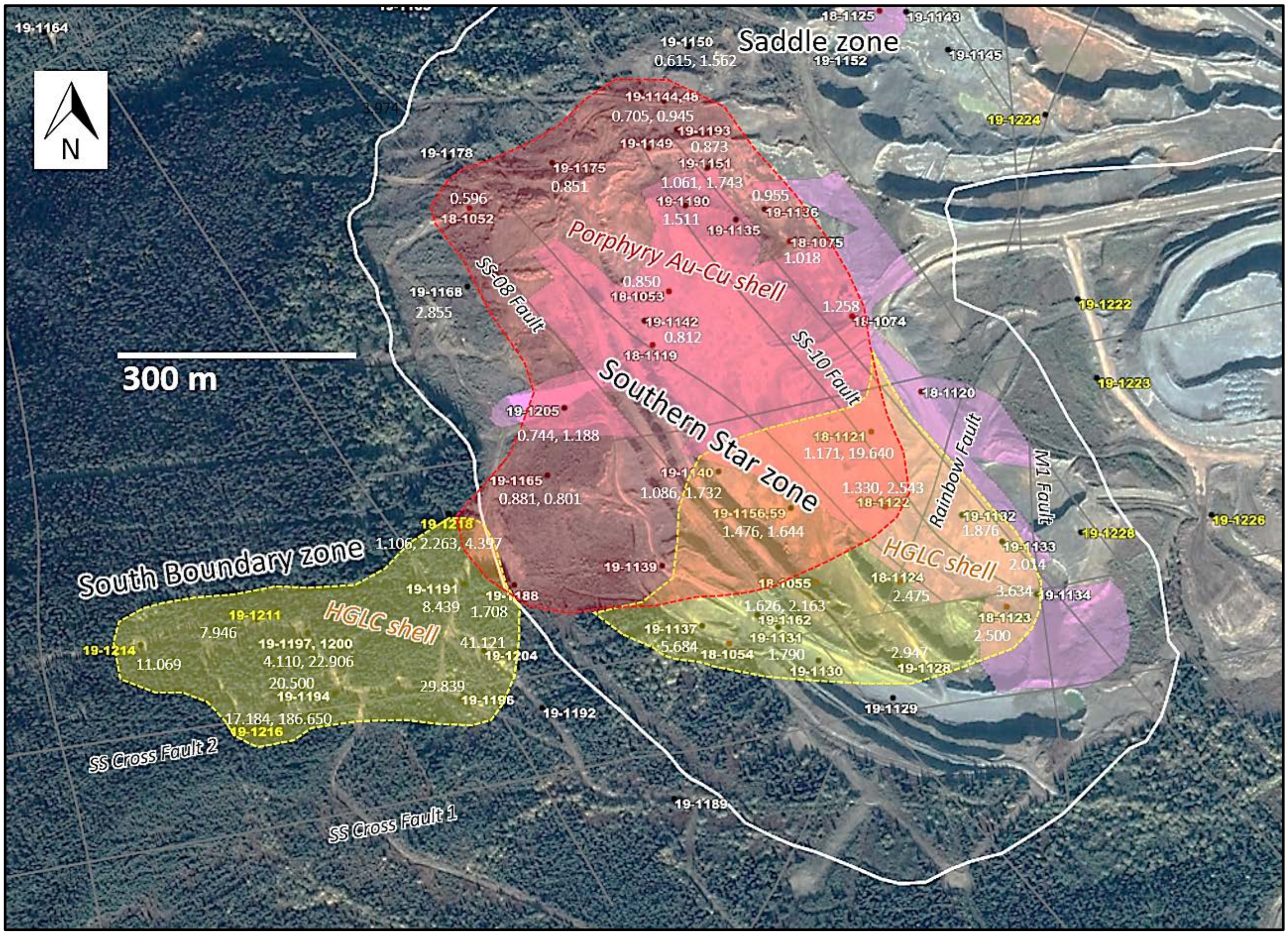
Southern Star and South Boundary Zones – NPI and Brownfield

The west-dipping Southern Star composite monzonite porphyry stock, dyke and breccia complex is crossed by a set northwest trending faults interpreted as east-side-up reverse faults at roughly 100 m spacing; and ENE-trending cross faults near the southern part of the complex which bound a tapering southern monzonite lobe (Figure 9-1). Porphyry Au-Cu style mineralization in the Southern Star deposit is hosted in monzonite porphyries and volcanic host rocks crosscut by abundant variably-mineralized hydrothermal breccias and xenolithic monzonite bodies. Breccias occur at the hanging-wall and footwall margins of stocks; some are gradational from crackled to milled, and they can to be dilutive where un-mineralized. The source of mineralized porphyry clasts in some breccias may be farther west and at greater depth than current drilling. Geophysical targets include deep southwest-dipping coincident chargeability-resistivity gradient zones with high magnetism.

Recent drilling has tested for extension of mineralization along dip in the hanging-wall and footwall of the intrusive complex and below the ultimate pit boundary, and focused on infill for resource estimation and support of Phase-8 mine planning. Significant assay results from 2018-2019 drilling suggest there is a porphyry Au-Cu mineralized shell in the north and west part of the Southern Star deposit which transitions to a HGLC shell in the southern part of the deposit. Ratios of Au:Cu, used as a proxy for the evolving state of a mineralizing fluid, are lowest (~0.60-0.95) in the north and west part of the deposit, increase to (~1.1-1.6) in a middle overlap zone, and are highest (1.6-5.7) in the south part, ignoring highest outliers (Figure 10-4). This suggests that the top of the Southern Star porphyry deposit is to the south, where the high gold shell remains open. This is a similar relationship as seen in the MBX zone (porphyry Au-Cu) and 66 zone (HGLC) across the Rainbow Fault; and a southwest extension of the Rainbow Fault may be related to the juxtaposition of these mineralization styles in the Southern Star zone as well (Figure 9-1). A north-south trending fault (M1 Fault) appears to cut off significant HGLC grades to the east of Southern Star. Selected results from 2018-2019 drilling include:

- 18-1123: 239.27 metres @ 0.480 g/t Au, 0.192% Cu from 12.19 metres;
- *including 8.00 metres @ 1.208 g/t Au, 0.404% Cu from 46.00 metres;*
- *and 4.00 metres @ 1.387 g/t Au, 0.539% Cu from 70.00 metres;*
- *and 8.00 metres @ 0.979 g/t Au, 0.330% Cu from 84.00 metres;*
- *and 11.25 metres @ 1.282 g/t Au, 0.452% Cu from 178.00 metres;*
- *and 6.00 metres @ 1.124 g/t Au, 0.402% Cu from 195.00 metres;*
- 19-1132: 148.75 metres @ 0.484 g/t Au, 0.258% Cu from 14.63 metres;
- *including 3.45 metres @ 1.258 g/t Au, 0.682% Cu from 81.55 metres;*
- *and 11.40 metres @ 1.556 g/t Au, 0.516% Cu from 91.00 metres.*

Figure 10-4: Google Earth Image (2019) of the Southern Star and South Boundary Zones



In the Southern Star, there is a northern porphyry Au-Cu mineralization shell (red) and southern high-gold low-copper (HGLC) mineralization shell (yellow). These shells are based on 2018-2019 drill holes that returned significant mineralized intervals. Drill holes are shown with Au:Cu ratio values (white text) from significant assay composites. The Au:Cu ratio is used as a proxy for evolution of the mineralizing fluid and increases to the south suggesting a high gold cap on the Southern Star porphyry deposit. The extent of the high gold cap remains untested by drilling and is open to the south. In the South Boundary zone is another more evolved HGLC shell (higher Au:Cu) trending ENE following SS Cross Fault 2, but remains open in all directions. This structurally controlled high gold zone may link to the high gold cap of the Southern Star porphyry but a connection has not yet been definitively established though drilling. The pink polygon represents the Southern Star stock. The thick white line is the 2017 ultimate pit boundary. Yellow drill hole labels are from 2019 Phase-2 brownfield drilling.

The South Boundary zone is the southernmost of the three subparallel ENE trending structurally-controlled HGLC targets (Goldmark-Oliver, Saddle West, South Boundary) west of the main porphyry stock trend (Figure 9-1). South Boundary lies west of the Southern Star zone in a similar relationship as the Saddle West to the Saddle zone and Goldmark to the DWBX-WBX zones. The area was trenched by BP Resources in the 1980's and wide-space drilled by United Lincoln/Continental Gold in 1989 and 1991. This exploration work identified a high gold 'polymetallic vein' style target (Sketchley, Rebagliati and Delong, 1995). Drilling resumed with Centerra-TCM in 2019 and was designed to validate historical HGLC drilling results, test for expansion and continuity of mineralization, test for a monzonite porphyry stock or dyke complex, and more generally test the hanging-wall block above the westward dipping Southern Star stock. Geophysical targets include a shallow-to-moderately west dipping chargeability-resistivity gradient feature with high chargeability and low resistivity in its hanging-wall (shallow) and footwall (deep), and a coincident magnetic high feature. This is representative of the Southern Star stock and its westward down-dip extension. In the core of the South Boundary zone is another magnetic high feature that has a ~200 m diameter ring-shape which was also targeted (Figure 10-3).

Results of the 2019 drilling indicate that mineralization is spatially related to narrow faults, breccia and fracture zones in volcanic rocks with quartz-sericite-pyrite-carbonate (QSPC) alteration and associated transitional-to-late stage veins; and to faulted and brecciated monzonite porphyry dykes with early stage alteration-mineralization and associated veins that have been overprinted by the QSPC assemblage and veins. Some chalcopyrite is noted in late stage veins suggesting remobilization of earlier stage mineralization. Selected results from the 2019 drilling include:

- 19-1200: 49.30 metres @ 1.214 g/t Au, 0.053% Cu from 354.00 metres;
- *including 1.68 metres @ 5.883 g/t Au, 0.166% Cu from 375.32 metres;*
- *and 1.10 metres @ 1.100 g/t Au, 0.021% Cu from 388.00 metres;*
- *and 4.30 metres @ 8.796 g/t Au, 0.306% Cu from 399.00 metres;*
- 19-1211: 34.00 metres @ 0.882 g/t Au, 0.111% Cu from 149.00 metres;
- *including 2.00 metres @ 11.400 g/t Au, 0.506% Cu from 150.96 metres.*

Heidi Zone – Greenfield

The Heidi zone is located 2.7 km west of the 2017 ultimate pit boundary and 900 m outside the mine lease boundary (Figure 9-2, Figure 10-2, Figure 10-3). The area was explored and wide-spaced drilled (<300 m depth) by Placer Dome in 1991. A monzonite porphyry stock (Heidi stock) was identified and mapped, and drilling results indicated shallow low-grade copper mineralization. Follow-up geochronological analysis gave a U-Pb date of 189 ± 3.3 Ma (Mortensen et al., 1995) which is the oldest measured monzonite in the Heidi Lake porphyry stock cluster, which averages ~184.8 Ma (Jago, 2008). Centerra-TCM resumed drilling the target in 2018 towards the end of a helicopter supported greenfield drilling program. Geophysically, the target area comprises a 1.6 km long east-west linear magnetic high anomaly that continues eastward into the North Slope zone, as well as a 350 m diameter ring-shaped magnetic high anomaly in the core of the Heidi zone; also a coincident SW-NE trending moderate chargeability and high resistivity feature that may be indicative of the Heidi stock.

Results of drilling in 2018-2019 showed that significant mineralized intervals are related to narrow monzonite porphyry dykes in andesitic rocks. Alteration includes a distinct mix of early-stage potassic and sodic-calcic (albite-actinolite-chlorite) assemblages overprinted by propylitic and late-stage QSPC assemblages. Early to transitional vein types include pyrrhotite-bearing polymetallic veins that can be massive, and disseminated pyrrhotite appears to be part of the sodic-calcic alteration assemblage. Higher gold grades occur at depth in 18-1104 which may be a distal western extension of the shallow-dipping Goldmark mineralization trend (Figure 9-2), and is spatially associated with a faulted hanging-wall of a monzonite porphyry dyke. Selected results from the 2018-2019 drilling include:

- 18-1104: 20.00 metres @ 0.206 g/t Au, 0.132% Cu from 568.00 metres;
- 18-1106: 33.60 metres @ 0.074 g/t Au, 0.205% Cu from 17.00 metres;
- *including 3.52 metres @ 0.176 g/t Au, 0.416% Cu from 18.48 metres;*
- *and 2.00 metres @ 0.116 g/t Au, 0.254% Cu from 33.00 metres;*
- 19-1179: 83.73 metres @ 0.048 g/t Au, 0.154% Cu from 24.00 metres;
- *including 5.51 metres @ 0.302 g/t Au, 0.268% Cu from 24.00 metres;*
- *and 8.00 metres @ 0.087 g/t Au, 0.182% Cu from 56.00 metres.*

10.3.SURVEY CONTROL

Pre-2006 Drilling

The first grids on the property were established in 1984 and 1985 when BP Resources conducted soil geochemical surveys along flagged and tagged lines. The lines were put in using chain and compass and were not corrected for the local magnetic deviation apparently caused by the Mount Milligan Intrusive Complex to the northwest.

Between 1986 and 1988, Lincoln and United Lincoln established a new grid for geophysical surveys. These lines were cut and marked by aluminum tags and pickets. The first grid lines were put in using a compass and were not corrected for the local magnetic deviation. Later lines were turned off a baseline by a transit and EDM and were put in using a backsight and foresight. This grid was used to establish the locations of diamond drill holes 87-1 to 89-120.

In November 1988, McElhanney Surveying established a mine grid whose relative position was set by labeling a point adjacent to the legal corner post of the PHIL 9 and PHIL 12 mineral claims as being 10000 N and 10000 E. The northern and eastern boundaries of the PHIL 9 mineral claim were surveyed to tie the mine grid to the legal corner post, and to establish control points for surveying in drill holes.

Drill holes 87-1 to 88-70 were surveyed to the mine gird as time permitted during the 1988 diamond drill program. The other diamond drill holes were surveyed as they were drilled.

The mine grid and legal survey grids are tied at a point called PCON, which is on a small hill approximately 500 m northwest of the MBX deposit. PCON is tied to a regional Datum (NAD27 UTM Zone 10) through regional geodetic points on Knob Hill northeast of the MBX zone and to a point on the 124th meridian in the area south of the deposit.

For 2004 Placer Dome drilling, McElhanney Surveying Ltd. (McElhanney Surveying) of Prince George were contracted to survey the drill sites.

All pre-2006 drill hole collars were surveyed to the mine grid and elevations have been corrected to the 2008 LiDAR surface.

2006-2007 (Terrane)

Planned drill holes were initially spotted by the Terrane field geologist using a handheld GPS, and markers were placed for drill collar location and field sites.

In April and May of 2007, AllNorth Consultants Ltd. of Prince George was contracted to survey in the 2006–2007 program drill holes. This survey was conducted using a LEICA RTK GNSS base station and rover combination rated for sub-decimeter accuracy. The latest survey was completed on September 11, 2007 and the field data was subsequently post-processed into NAD83 UTM Zone 10 coordinates.

2010-2016 Drilling (TCM)

In 2010-2011, 27 drill holes (10-1003 to 10-1012, 11-1013 to 11-1019, and MET 11-01 to MET 11-10), were surveyed using a Trimble differential GPS capable of sub-decimeter accuracy. The elevation of these collars is based on the 2008 Lidar survey.

During June 2013, 20 holes were drilled to define the thickness of the overburden in the east part of the Main and 66 zones. Collars of vertical holes S13-01 to S13-20 were surveyed by mine personnel using a Trimble DGPS. Elevations were verified using the 2008 Lidar survey.

2015-2016 drill collars were surveyed using a handheld Garmin GPS which was adequate for the exploratory nature of the drill holes. These holes were not used for mineral estimation.

2017-2019 Drilling (Centerra-TCM)

For 2017-2019, all final drill hole locations within the 2017 ultimate pit boundary (resource and near-pit exploration) were surveyed using mine survey Trimble differential GPS equipment (models Trimble TSC7 controller paired to a SPS882 GNSS antenna or a Trimble TSC3 controller paired to a SPS986 GNSS antenna). For greenfield and brownfield exploration, collars were surveyed using Trimble TSC3 capable of sub-decimeter accuracy, a Trimble GeoExp6000 capable of sub-meter accuracy or a handheld GPS (Garmin 62) depending on depending on availability and radio link signal strength of the rover GNSS antenna away from the base station.

10.4. DOWNHOLE SURVEYS

Pre-2004 Drilling

Downhole dips for diamond drill holes 87-1 to 90-758 (pre-Placer Dome) were determined by acid tests. Downhole azimuths were not determined because it was believed that the results obtained by conventional downhole survey methods utilizing a compass would have been inaccurate due to the concentrations of magnetite. Nine holes were surveyed with a Sperry Sun gyro instrument. Data from this study was analyzed and a weighted average azimuth change of +0.02597 degrees per meter was calculated and applied to all holes with bottom of hole dips of ≤85°.

The holes drilled under the management of Placer Dome employed a Sperry Sun magnetic survey instrument, with the location of survey points controlled by magnetic susceptibility readings of the drill core. Downhole surveys during the 2004 drill program were completed with an Icefield Instruments inclinometer tool.

2006–2007 Drilling (Terrane)

Terrane employed a Reflex EZ-Shot magnetic survey instrument to measure azimuth, dip, roll angle, temperature, and magnetic field strength. Magnetic field strength measurements were used to identify measurements that may have been influenced by magnetic interference. Anomalous readings were flagged and excluded from the data used in resource estimation.

2010-2016 Drilling (TCM)

For 2010-2011 & 2015 drill programs, downhole surveys were completed using a Reflex EZ-Shot downhole tool to take azimuth and dip readings approximately every 50 m. Magnetic field strength measurements were used to identify measurements that may have been influenced by magnetic interference. Anomalous readings were flagged and excluded from the data used in resource estimation.

No downhole surveys were carried out on the 20 holes drilled in 2013 to define the thickness of overburden.

In 2016 a Reflex TN-14 Gyrocompass was used for drill hole alignment. A Reflex Gyro was used for downhole surveys with tool with azimuth and dip readings taken approximately every 50 m downhole. The Reflex Gyro tool was used in 2016 to reduce anticipated influence from highly magnetic rock. The 2015 and 2016 drilling programs were exploratory in nature, located 5 km northwest of the Mount Milligan deposit and are not included in any resource estimate.

2017-2019 Drilling (Centerra-TCM)

Down hole surveys for 2017 to 2019 drill holes were completed using both a Reflex EZ-trac downhole tool (greenfield exploration drilling) and Reflex EZ-Gyro (brownfield exploration and resource infill drilling).

The EZ-trac survey tool was used to capture azimuth and dip readings approximately every 50 m downhole. A magnetic declination value of 17.8° was used in for the 2017-2019. Magnetic field strength measurements were used to help identify measurements that may have been influenced by magnetic interference. Anomalous readings were flagged and excluded from the data used in resource estimation.

The EZ-Gyro is a north-seeking, non-magnetic compass unaffected by magnetic interference. At completion of drilling, a single measurement was conducted every 50 feet (~15 m) intervals, and 3 consecutive measurements at a single depth were conducted every 500 feet (EZ-Gyro ‘Optimized’ mode) for quality control purposes.

While drilling vertical holes, EZ-Gyro surveys were used to provide real time indication that the drill hole was within acceptable limits of deviation.

10.5. CORE RECOVERY

All the drill holes are diamond drill core holes with predominantly NQ size core diameter. Some HQ size has also been drilled since 2006, mainly during the 2006-2007 feasibility drilling period. A few six-inch holes were drilled for metallurgical samples. Geotechnical information was routinely recorded for all diamond drilling programs including core recovery, rock quality (RQD), hardness or compressive strength (CS), degree of breakage, degree of weathering or oxidation, fracture and joint frequency, and specific gravity (SG). Core recovery routinely exceeds 90% and averages 96%. Based on the high core recovery there is no known material impact on the reliability of results.

11. SAMPLE PREPARATION, ANALYSES, AND SECURITY

11.1. PRE-2004 SAMPLES

Sampling Method and Approach

Most of the diamond drill hole samples were collected from NQ diameter (47.6 mm) drill core. Core was marked (intervals and split lines) for sampling by the geologist on nominal one metre (Lincoln and United Lincoln) or two metre (Continental Gold) intervals except where veins, alteration or lithologic contacts were encountered. Core was split in half using a core splitter, with one half shipped to the laboratory for analysis and the other half retained on site.

Core was stored in outside core racks or dead-stacked on the ground. All racks were vandalized in 2005, leaving only the dead-stacked core intact. Check assay programs were run by both Continental and Placer Dome and the data passed QA/QC, prior to the core vandalism in 2004. Some of this core was re-organized during the 2004, 2005, 2006, and 2007 field seasons.

Figure 11-1: Heidi Lake Core Storage Yard in 2005 after Occurrence of Vandalism

Photo by Darren O’Brien.

Sample Preparation and Laboratory

Samples for drill holes 87-1 to 88-60 were prepared (and assayed) by Acme Analytical Laboratories Ltd. (Acme); drill holes 88-61 to 90-758 by Mineral Environments Laboratories Ltd. (Min-En); and drill holes 90-759 to 91-862 by the Placer Dome Research Centre (PDRC). All laboratories were in Vancouver, British Columbia.

Lab certification details for Acme Analytical Laboratories Ltd, Mineral Environments Laboratories Ltd, and Placer Dome Research Center are unknown during the time the analysis were performed.

Samples were prepared as follows:

- Samples were first dried at 95°C.
- Samples were jaw crushed to nominal 6 mm.
- Samples were then roll crushed to nominal 0.3 mm.
- Samples were riffle split until a 300 to 400 g sub-sample was generated.
- Sub-samples were then pulverized to 95% passing a 120 mesh screen.

Assaying

Gold was assayed by the laboratory at which they were prepared (Acme, Min-En, and PDRC) by standard fire assay with an atomic absorption finish on a 30 g pulp sample. Copper was assayed by digesting 2 g of sample in aqua regia and determining the assay value by atomic absorption spectrometry. Gold assay batches consisted of 24 samples and copper assay batches of 70 samples. Bondar Clegg and Chemex, both of Vancouver, British Columbia, performed check assays on selected sample pulps using the same protocol.

Min-En performed metallic screen fire assays for gold by weight-averaging the entire +120 mesh fraction with the average of two assays of the -120 mesh fraction.

Trace element geochemical analysis of silver by atomic absorption spectroscopy (Min-En) was routinely conducted on samples from drill holes 88-61 to 89-212.

The assay procedures were performed by methods conforming to industry-standard practices for gold-copper porphyry deposits.

Quality Control

Standards, blanks, and duplicates were not routinely inserted into the sample stream during the historic drill programs from 1987 to 1992. However, there were external check assay programs in place, both by Continental Gold and Placer Dome, and one of these check programs did employ standards as a check on accuracy. Additional check assay programs have been initiated over the years of the pre-2004 assay samples and are described in detail in Thompson Creek Metals Company Inc.'s (TCMC) 2015 Technical Report (refer to TCMC January 21, 2015 NI 43-101 Technical Report check assay programs). The report concluded from the check assay programs that the data was acceptable and free of significant bias.

Security

No specific sample security measures were in place during the pre-2004 drill programs.

11.2.2004 PLACER DOME SAMPLES

Sampling Method and Approach

Diamond drill hole samples were collected from NQ-2 diameter (50.5 mm) drill core. Core was marked (intervals and split lines) for sampling by the geologist on two metre intervals except where pyrite veins or lithologic contacts were encountered. Core was split in half using a core splitter, with one half shipped to the laboratory for analysis and the other half retained on site.

The 2004 drill core was subject to vandalism in 2005 and is no longer available (Figure 11-1).

Sample Preparation and Laboratory

Samples for drill holes 04-920 to 04-933 (2004) were prepared (and assayed) by Eco-Tech Laboratories Ltd. (Eco-Tech) of Kamloops, British Columbia which maintained an ISO 9001:2000 standard for the provision of assay and geochemical analytical services. Samples were prepared as follows:

- Samples were jaw crushed to nominal 5 mesh.
- Samples were crushed to nominal 10 mesh (equipment used is not documented).
- Samples were then riffle split until a nominal 1 kg sub-sample was generated.
- Sub-samples were then pulverized to nominal 140 mesh.

Assaying

Gold was assayed by Eco-Tech by standard fire assay with an atomic absorption finish on a 30 g pulp sample. Copper assays utilized an aqua regia sample decomposition with analysis by inductively coupled plasma atomic emission spectroscopy (ICP-AES). Copper analyses greater than 7,000 ppm were rerun using atomic absorption spectrometry.

The method codes used by Eco-Tech are not documented on the assay certificate but the lab was/is accredited (as per the text above).

Quality Control

Placer Dome introduced a quality assurance and quality control (QA/QC) program through the systematic insertion of one blank, one pulp duplicate and one company standard inserted every 37 core samples sent for analysis to Eco-Tech. Eco-Tech received a flask labeled “Standard A”, which was used as the company standard. “Standard A” consisted of CGS-1 from CDN Resource Laboratories Ltd. of Delta, BC. CGS-1 was prepared from ore supplied by B.C. Metals Corporation from the Red Chris Porphyry deposit in British Columbia. Assay results of the CGS-1 standard were consistently higher than the recommended standard value but within the acceptable ranges defined on the certificate. Analyses of pulp duplicates yielded generally excellent correlation for copper and fair to poor correlation for gold. All blanks produced null to low gold and copper values. No external lab check assays were performed for the 2004 samples.

Security

No specific sample security measures were documented for the 2004 drill program.

11.3. 2006–2007 TERRANE SAMPLES

Sampling Method and Approach

Terrane collected samples from HQ (63.5 mm) diamond drill core. Intervals were nominally two metres of core length but were shortened at the discretion of the geologist at lithological, structural or major alteration contacts. Prior to marking the sample intervals, technicians photographed and geotechnically logged the core. Technicians then marked the sample intervals and assigned sample numbers. After the sample intervals were marked, the geologist logged the core in detail and the core was sent for sampling.

Drill core was split in half using a hydraulic core splitter, with one-half placed in a sample bag for shipping to the sample preparation laboratory and the other half placed back in the core box for future reference. Core boxes were cross-stacked on pallets and remain at the project site.

It is the QP's opinion that two metre sample length is considered appropriate for collecting representative samples from the mineralized zones at Mount Milligan.

Sample Preparation and Laboratory

The initial splitting of drill core at the project site was the only aspect of sample preparation performed by Terrane employees. The half-core samples were then shipped to ALS Chemex (now ALS Minerals) in Vancouver, British Columbia for sample preparation and analysis. ALS Chemex laboratories (ALS) in North America are registered to ISO 9001:2000 for the provision of assay and geochemical analytical services by QM Quality Registrars. In addition, ALS' main North American laboratory in North Vancouver, British Columbia is accredited by the Standards Council of Canada (SCC) for specific tests listed in the Scope of Accreditation No. 579, which is available at https://www.scc.ca/en/system/files/client-scopes/677_e.pdf. This accreditation is based on international standards (ISO 17025) and involves extensive site audits and ongoing performance evaluations.

Samples were prepared using Method Code PREP-31 as follows:

- Samples were logged into the tracking system and barcodes applied.
- Samples were dried and weighed.
- Samples were fine crushed >70% passing 2 mm.
- Samples were split to 250 g and pulverized to >85% passing 75 µm.

Assaying

Drill core samples were analyzed for gold content using ALS method Au-AA25. Gold assays utilized a fire assay fusion sample decomposition of a 30 g pulp with an atomic absorption spectrometry finish. Copper analysis was completed using ALS method Cu-OG46. Copper assays utilized aqua regia sample decomposition with analysis by ICP-AES.

Every second sample was also analyzed for multiple elements using ALS method ME-MS41. This method analyzed trace levels for 50 elements by aqua regia digestion and a combination of ICP-

AES and inductively coupled plasma mass spectrometry (ICP-MS). Silver was analyzed as part of this multi-element package.

Assay methods used by Terrane (Au-AA25, Cu-OG46, and ME-MS41) at ALS are listed on their ISO 17025 accreditation.

Quality Control

In addition to an internal laboratory quality control program utilized by ALS, Terrane maintained an additional quality assurance and quality control (QA/QC) program through the systematic use of standards, blanks, and duplicates. For every 20 samples, one standard and one blank were inserted into the sample stream by core sampling personnel at the project site. Two different copper-gold standards were purchased from CDN Resource Laboratories Ltd. in Delta, British Columbia. Standards were alternated for each batch of 20 samples. In addition, for every 20th sample, the sample preparation laboratory created a duplicate pulp for a comparative analysis.

The results of the QA/QC program were reviewed, and corrective action was taken on sample batches with quality control samples exceeding acceptable limits. A slight high bias of the copper analyses of 2-5% relative to the accepted value of the certified reference material used was noted. A similar bias was noted in the laboratory's internal QC analyses. This bias was within the range of some of the labs participating in the round robin analyses of the standards. There was no appreciable bias of the gold analyses relative to the standards.

External pulp duplicate check analyses at a separate laboratory (Acme) indicated no significant relative bias in the gold analyses and a consistent high bias of the primary analyses of ~5% for copper.

Assessment of precision based on routine analyses of preparation duplicates (split after initial crushing) indicated the precision of gold analyses is slightly less than ideal and copper is slightly greater.

Routine analyses of blank material indicated no systematic contamination during sample preparation. A few isolated higher analyses resulted in batch re-assays with acceptable results.

It is the QP's opinion that the quality control program for the 2006-2007 Mount Milligan drill programs met generally accepted industry standards and confirm the accuracy, precision and lack of contamination.

Security

Samples were sealed in large rice sacks and stored in the core sampling shed to improve the security of the samples while at the project site, and to ensure the validity and integrity of the samples taken. Twice weekly, the sacks were shipped from the project site directly to the ALS preparation laboratory via Russell Transfer, a bonded independent expeditor based in Fort St. James, British Columbia.

11.4. 2010-2011 TCMC SAMPLES

Sampling Method and Approach

TCMC collected samples from predominately NQ diameter (47.6 mm) diamond drill core. Intervals were nominally two metres of core length but were shortened, at the discretion of the geologist, at lithological, structural, or major alteration contacts. The drill core was marked with the sample intervals, assigned sample numbers and photographed before the core was sent for cutting.

Drill core was cut in half using a diamond drill core saw by TCMC staff, with one-half placed in a sample bag for shipping to the sample preparation laboratory and the other half placed back in the core box for future reference. Core boxes were cross-stacked on pallets and remain at the project site.

Sample Preparation and Laboratory

For the 2010-2011 drilling campaigns TCMC continued with the preparation method and analytical protocol utilised by Terrane for the 2006-2007 program. Drill core was cut at the project site and the half-core samples were then shipped to sent to the ALS sample preparation facility in Terrace BC for sample preparation. Splits of the pulps were subsequently shipped to the ALS laboratory in Vancouver for analysis.

Assaying

The 2010-2011 drill core samples were analyzed for gold content using ALS method Au-AA25. Gold assays utilized a fire assay fusion sample decomposition of a 30 g pulp with an atomic absorption spectrometry finish. Copper analysis was completed using ALS Chemex’s method Cu-OG46. Copper assays utilized aqua regia sample decomposition with analysis by ICP-AES.

Every second sample was also analyzed for multiple elements using ALS Chemex's method ME-MS41. This method analyzed trace levels for 50 elements by aqua regia digestion and a combination of ICP-AES and ICP-MS. Silver was analyzed as part of this multi-element package.

Assay methods used by Terrane (Au-AA25, Cu-OG46, and ME-MS41) at ALS are listed on their ISO 17025 accreditation.

Quality Control

All campaigns utilized the same quality control protocol initiated during the 2004 drill program with the insertion of standard reference material, duplicates and blanks into the sample stream to monitor precision, accuracy and contamination of the sampling and analytical process. Results of these analyses were continuously and independently monitored, with a 'failure table' documenting the quality control samples that exceeded acceptable limits and tracking the corrective measures taken. Where necessary, analytical batches were re-assayed to achieve final analyses that met industry standards of quality.

Precision was measured through duplicate samples taken at various stages of sample size reduction; quarter core field duplicates, preparation duplicates taken after coarse crushing and pulp duplicates taken routinely as part of the ALS and BV internal quality control.

Coarse barren limestone or quartzite was routinely added to each batch to monitor possible contamination during the sample preparation stage. No significant contamination was indicated during the assay programs.

In addition to the quality control samples submitted to the primary laboratory, approximately 5% of the sample pulps were submitted to Acme Labs in Vancouver for the 2010-2011 campaigns as an independent check against analytical bias and accuracy.

Through continuous monitoring of the quality control results, significant issues affecting the results were identified and resolved, and it is the QP's opinion that the quality control and check assays completed confirmed that the 2010-2011 Mount Milligan assay data were accurate, precise and free of contamination to industry standards and is of sufficient quality to be used in a resource estimation.

Security

Samples were sealed in large rice sacks and stored in the core sampling shed to improve the security of the samples while at the project site, and to ensure the validity and integrity of the samples taken.

Twice weekly, the sacks were shipped from the project site directly to the ALS preparation laboratory via Russell Transfer, a bonded independent expeditor based in Fort St. James, British Columbia.

11.5. 2015-2016 TCM SAMPLES

Sampling Method and Approach

TCM collected samples from predominately NQ diameter (47.6 mm) diamond drill core. Intervals were nominally two metres of core length but were shortened, at the discretion of the geologist, at lithological, structural, or major alteration contacts. The drill core was marked with the sample intervals, assigned sample numbers and photographed before the core was sent for cutting.

Drill core was cut in half using a diamond drill core saw by TCM staff, with one-half placed in a sample bag for shipping to the sample preparation laboratory and the other half placed back in the core box for future reference. Core boxes were cross-stacked on pallets and remain at the project site.

Sample Preparation and Laboratory

Bureau Veritas Laboratory (BV) in Vancouver, British Columbia was utilized for sample preparation and analysis for the 2015 and 2016 drilling campaigns located approximately 5 km northwest of the current open pit. BV's laboratories in North America are registered to ISO 9001 for the provision of assay and geochemical analytical services. BV's analytical laboratory in Vancouver, British Columbia is accredited by the Standards Council of Canada (SCC) for specific tests listed in the Scope of Accreditation No. 720, which is available https://www.scc.ca/en/system/files/client-scopes/ASB_SOA_15895_Scope_v3_2019-04-11.pdf. This accreditation is based on international standards (ISO 17025) and involves extensive site audits and ongoing performance evaluations.

Drill core was cut at the project site and half-core samples sent to BV in Vancouver, British Columbia for sample preparation and analysis.

Samples were prepared as follows (code PRP80-250):

- Samples were dried and weighed.
- 1 kg sub-samples were crushed to ≥ 80% passing ~2 mm.
- Crushed material was then riffle split and a 250 g sample pulverized to ≥ 85% passing 75 µm.

Assaying

Drill core samples from 2015-2016 were analyzed utilizing BV's 37 multi-element AQ251 method. The method utilizes ICP-MS analysis of a 15 g sample after modified aqua regia digestion for low to ultra-low determination. The analytical flow sheet called for fire assay of samples returning >0.20 g/t Au by ICP but the 2015 program returned no samples exceeding that grade, with the highest value running 0.15 g/t Au.

Assay method ME-AQ251 used by Terrane are listed on ALS's ISO 17025 accreditation.

Quality Control

The campaigns utilized the same quality control protocol initiated during the 2004 drill program with the insertion of standard reference material, duplicates and blanks into the sample stream to monitor precision, accuracy and contamination of the sampling and analytical process. Results of these analyses were continuously and independently monitored, with a 'failure table' documenting the quality control samples that exceeded acceptable limits and tracking the corrective measures taken. Where necessary, analytical batches were re-assayed to achieve final analyses that met industry standards of quality.

For QA/QC, duplicate samples were taken at various stages of sample size reduction. Quarter core field duplicates, preparation duplicates taken after coarse crushing, as well as pulp duplicates were all taken routinely as part of the BV quality control.

Coarse barren limestone or quartzite was routinely added to each batch to monitor possible contamination during the sample preparation stage. No significant contamination was indicated during the assay programs.

The results of the QA/QC program for the 2015 and 2016 drill programs indicated that Cu and Mo analyses were accurate, precise and free of contamination. Gold assays were also uncontaminated but significantly less accurate and precise. The low accuracy in the gold values can be attributed to the imprecision of the analytical method used. The aqua regia digestion with ICP-MS finish procedure is known to be less accurate and precise than fire assay due to the smaller test weight and limited gold solubility in refractory minerals.

The 2015 and 2016 drilling programs were exploratory in nature, located 5 km northwest of the Mount Milligan deposit. The QAQC analysis shows that the data are sufficiently accurate and precise for exploration purposes and were not used in resource modelling.

Security

Samples were sealed in large rice sacks sealed with individually number security straps. Samples were shipped from Mount Milligan Mine to BV's preparation facility in Smithers, British Columbia in 2015 and to BV's analytical facility in Vancouver, British Columbia in 2016. BV reported all bags were received in good condition, with security tags intact and with no evidence of tampering.

11.6. 2017-2019 CENTERRA-TCM SAMPLES

Sampling Method and Approach

Diamond drill core samples were collected from predominately NQ diameter (47.6 mm) diamond drill core. Eight HQ diameter metallurgical drill holes from 2017 (17-1029 to 17-1038) were also sampled. Sample intervals were nominally two metre core length but were shortened at the discretion of the geologist at lithological, structural or major alteration contacts. Prior to detailed logging and marking of the sample intervals, technicians logged the core for geotechnical characteristics. After the geologist logged the core in detail, the drill core was marked with the sample intervals, cut lines and assigned sample numbers. The drill core was photographed prior to the core being sent for cutting.

Most of the NQ drill core was cut in half using a diamond drill core saw, with one-half placed in a sample bag for shipping to the laboratory and the other half placed back in the core box for future reference. For metallurgical test work in 2017, selected core intervals had their remnant HQ drill core halves cut an additional time to create a quarter core sample to be sent to the lab. In 2019, NQ drill core from holes 19-1132 to 19-1134 was cut an additional time so three quarters of the core could be sent to BV for analysis; one quarter was placed back in the box.

Sample Preparation and Laboratory

From 2017 to present, drill samples were cut at the project site and half-core samples were sent for sample preparation and pulverization to BV in Vancouver, British Columbia.

A subset of approximately 1,166 samples from 3 drill holes (17-1049, 17-1050, 17-1051) were sent to Activation Laboratories (ActLabs) in Kamloops, British Columbia. ActLabs laboratory in Kamloops, British Columbia is accredited by the Standards Council of Canada (SCC) for specific tests listed in the Scope of Accreditation No. 790, which is available at https://www.scc.ca/en/system/files/client-scopes/ASB_SOA_15974_v3_2020-01-09.pdf. This accreditation is based on international standards (ISO 17025) and involves extensive site audits and ongoing performance evaluations.

At BV, samples were prepared as follows (code PRP80-250):

- Samples were dried and weighed.
- Entire sample crushed to ≥ 80% passing ~2 mm.
- Crushed material was then riffle split and a 250 g sample pulverized to ≥ 85% passing 75 µm.

At ActLabs, samples were prepared as follows (code RX1):

- Samples were dried and weighed.
- Entire sample crushed to ≥ 80% passing 2 mm.
- Crushed material then riffle split and a 250 g sample pulverized (mild steel) to ≥ 95% passing 105 µm.

Assaying

At BV, samples from 2017-2019 were analyzed for precious and base metals, as well as multi-elements. Gold was assayed using a 30 g fire assay with atomic absorption spectrometry (AAS) finish (BV lab code FA430). Gold results ≥ 10 ppm (over upper detection limit of method) triggered 30 g fire assay with gravimetric finish (FA530). All samples were also analyzed for a 45 element package (including copper and base metals) using a 4-acid digest and inductively coupled plasma mass spectrometry/emission spectroscopy (ICP-MS/ES) on a 0.25 g aliquot (BV lab code MA200). Copper results ≥ 1% triggered analysis using ICP-MS with AAS finish (MA404) on a 0.50 g aliquot. Silver results ≥ 100 ppm triggered 30 g fire assay with gravimetric finish (FA530). Sulfur >10% triggered Leco analysis (TC000). Mercury was analyzed using an Aqua Regia digest with cold vapor atomic absorption spectrometry finish (CV400).

Assay methods used by TCM for gold (FA430, FA530) and over-limit copper and sulfur (MA404, TC000) are listed on BV's ISO17025:2017 accreditation.

At ActLabs, samples from 2017 were analyzed for similar previous metals and base metals as BV. Gold was assayed using a 30 g fire assay with atomic absorption spectrometry (AAS) finish (ActLabs code 1A2). Gold results ≥ 10 ppm (over upper detection limit of method) triggered 30 g fire assay with gravimetric finish (code 1A3). Samples were also analyzed for a 47 element package (including copper and base metals) using a 4-acid digest and inductively coupled plasma mass spectrometry (ICPMS) on a 0.25 g aliquot (Actlabs code UT-4M). Copper, lead and zinc were further analyzed by using a 4-acid digestion with inductively coupled plasma optical emission spectroscopy (ICP-OES) finish. Sulfur >10% triggered Leco analysis (Actlabs code 4F).

Assay methods used by TCM gold, silver and copper are listed under Act Labs ISO17025:2017 accreditation.

Quality Control

For 2017-2019 diamond drilling, certified reference materials (CRM), blanks, and duplicates were used to monitor quality assurance and quality control (QA/QC) of the core sampling, processing, and assaying processes. All samples were marked with a unique sample ID number and sample tag in the core box by a logging geologist and cut using an electric core saw. While sampling drill core, the logging geologists inserted CRMs and coarse blank samples alternately into the sample sequence every 10 samples. Clean coarse marble landscape rock weighed in ~1 kg samples were used for blank material. Two different copper-gold standards were purchased from CDN Resource Laboratories Ltd. in Delta, British Columbia, and two different copper-gold and multi-element certified standards from Ore Research and Exploration Pty Ltd. The standards were selected to match low, medium and high-grade mineralization ranges and are dominantly sourced from copper-gold bearing porphyry intrusive rocks.

In addition to the inserted reference materials, field and coarse reject duplicates were inserted alternately into the sample sequence every 20 samples. Field duplicates were prepared by quartering one half of the core, with one quarter sent for analysis with a unique sample ID, and the other remaining in the core box. Coarse reject duplicates were prepared at BV labs prior to sample pulverization. The QC insertion rates are acceptable according to current CIM best practice standards, with QC samples accounting for ~15% of the 2017-2019 assay database.

After the assay results were received from the lab, gold and copper assays were checked by a Centerra database manager using control charts for the CRMs, blanks and duplicates. Any quality control failures (samples bracketing CRMs with assay values ± three standard deviations of the expected value) were documented and relevant batches of samples were requested for re-assay by BV labs using the primary pulp. After completion of the programs approximately 5% of the sample pulps from BV were submitted to SGS laboratory in Burnaby, British Columbia as an independent check for analytical bias and accuracy.

QA/QC of 2017 to 2019 drill core demonstrates that assay data are accurate, precise and free of contamination to industry standards and in the QP's opinion, of sufficient quality to be used in a resource estimation.

Security

After drill hole samples were cut and bagged in poly-plastic sample bags, they were prepared for sample shipment by a geotechnician. Individual sample bags were collected in rice bags and sealed with zip ties and individually numbered security tags. A copy of the sample submission and chain of custody forms for each batch was included in the last rice bag of samples, and emailed to the exploration office and BV labs.

Production Blast Hole Samples

Blast hole samples are collected daily by the Mine Geology team and delivered to the Mount Milligan Assay Laboratory for analysis. Sample results are used for daily grade control purposes including the determination of ore/waste boundaries, grade differentiation between ore blocks, and ore blending strategies.

Blast hole sample results are not used in annual resource estimate updates, but are reconciled monthly with the Geology block model, and actual mill results.

Sample Delivery, Intake, and Preparation

- Blast hole samples are collected once or twice a day by the Ore Control Geologist/Technician and delivered to the on-site Mount Milligan Assay Laboratory for analysis.
- Samples are delivered in individual heavy-duty plastic bags with unique barcode ID tags.
- Upon delivery at the assay laboratory, a Geology Sample Submission form is submitted to the laboratory stating the date, submitter and the number of samples along with the sample identifications and the analysis required (Cu, Au, Ag, S, Fe, ABA).
- Each sample is transferred from the bag to individual drying pans and is scanned into a batch created in the assay laboratory's Laboratory Information Management System (LIMS) software.
- Samples are dried, crushed, and riffle split down to ~300 g.
- The split material is then pulverized to -200 mesh (75 microns), transferred to a barcoded envelope and staged for analysis in wet chemistry and fire assay.

Wet Chemistry

- Metal analysis is completed via acid digestion and atomic absorption analysis for each sample. Copper, iron and silver values are read on a PerkinElmer PinAAcle atomic absorption spectrometer using internal calibration with quality control standards to include blanks, drift check standards, duplicates and certified reference material standards.

- Acid Base Accounting (ABA) analysis consists of volumetric titration to determine neutralization potential and sulfur analysis to determine acid potential. Sulfur analysis is completed on a LECO CS230SH to include quality control blanks and certified reference material standards.

Fire Assay

- Flux preparation, fusion, cupellation, parting and atomic absorption analysis are completed for each sample, and gold values are read on a PerkinElmer PinAAcle atomic absorption spectrometer using internal calibration with quality control standards to include blanks, drift check standards, duplicates and certified reference material standards.

Quality Control and Remedial Action Procedure

All Mount Milligan Assay Laboratory procedures are accompanied by appropriate, industry standard instrument calibration and QA/QC (Quality Assurance/Quality Control) measures, including quarterly third-party analysis checks. Ore and acid-base accounting analyses SOP includes steps to confirm on-site laboratory method accuracy, precision, contamination control, sample tracking, and recordkeeping. The assay laboratory also receives blind duplicate samples from the Ore Control Geologist/Technician which are compared against daily sample analysis. This is managed as part of the MTM Assay Laboratory Quality Management System.

Adequacy of Sample Preparation, Analysis and Security

In the opinion of the QP for this Item of the Technical Report, sample preparation, security, and analytical procedures utilized during drilling programs were adequate and conducted according to CIM Estimation of Mineral Resources and Mineral Reserves best practice guidelines.

12. DATA VERIFICATION

12.1. SITE VISIT

Slobodan Jankovic, QP for the Mineral Resource estimate, conducted a site visit at Mount Milligan from April 8th to 11th, 2019. The site visit included a review of site facilities, logging and sampling procedures, and the lithology and alteration domain controls used in resource estimation. No significant issues were identified with respect to the assay sampling procedures, chain of custody or the geological data collection.

12.2. DATABASE VERIFICATION

Coordinate System

The geologic modelling and resource estimates were completed in UTM coordinates (NAD83 UTM Zone 10). Historical data (prior to and including 2004) were established in local mine grid, and this is tied to the legal survey grid (NAD27) at a point called PCON. These historical data were first transformed to NAD27 (Table 12-1) before transformation to NAD83 using the Canadian National Transformation Version 2 (NTV2).

Table 12-1: PCON Location to NAD27 UTM Zone 10n Transformation

	Mine Grid	UTM (NAD27)
Northing	10201.26	6109873
Easting	12113.76	433761.9
Scale Factor = 0.9996394444		
Rotation = +1.59882094		

Subsequently, a transformation matrix was established using historical drilling as the control. This transformation provided sub-millimetre accuracy within the project limits.

Refer to Section 10 (Survey Control subsection) for a detailed description of the historical coordinate system established at the Mount Milligan project.

Elevation

In 2008, a comparative study examining the differences between the historic topographic survey and the 2008 LiDAR survey was completed. This study found that the differences were variable across the project area, but generally the old topographic survey is located at a slightly lower elevation than the 2008 survey, with a mean difference of -1.36 m.

The coordinates of the drill hole collars were compared to the 2008 survey. On average, drill holes completed during the period 2004-2007 were found to be within 0.20 metres of the LiDAR survey. These holes were surveyed in UTM coordinates using differential GPS. Drill holes completed prior to 2004 were found to be on average two metres lower in elevation on average than the LiDAR survey. These holes were surveyed using mine grid coordinates. All collar elevations for drill holes completed prior to 2010 have been registered to the 2008 LiDAR surface.

Assay Quality Control

1987 to 2014

Assay control programs have been completed at Mount Milligan on 1987 to 2011 drill holes as summarized in Thompson Creek Metals, January 21, 2015 NI 43-101 Technical Report on Mount Milligan Mine authored by Clifford and Berthelsen.

Check assays were routinely conducted on 1987 and 1988 drilling samples starting in 1989, and several check assay campaigns were subsequently completed. A summary check assay review by Smee and Stanley (1992) concluded that that there did not appear to be any systematic bias in the analytical data for all drill holes from 88-61 to 91-825. Discrepancies noted in previous studies were within the expected error of the analytical precision of assay methods used.

In addition to check assay programs, field-based quality control programs and performance monitoring were introduced during the 2004 metallurgical drilling program with routine insertion of standard reference material, blanks and preparation duplicates. Both field-based QA/QC and check assay programs continued in subsequent drilling programs.

2015 to 2019

QA/QC procedures implemented for the 2015-2019 drilling programs are described under the Quality Control subsections of Section 11.

Further to the above described QA/QC procedures, routine data checks are performed to ensure the assays in the drill hole database are checked against assay certificates received by the lab.

Drill Hole Database

In construction the drill hole database, a series of assay quality control programs have been carried out. These include external check assay programs since 1989 and use of reference materials (blanks, standards, duplicate samples) and external check assay programs since 2004, as described above. As well, several database compilation, verification, and review programs (including third party independent review) have been undertaken.

In 2007, Terrane contracted Maxwell Geoservices from Vancouver, B.C. to compile historical drill hole data into a digital SQL database. In April 2007, Maxwell Geoservices digitized and compiled historical paper logs and original assay certificates into a DataShed database. After the data compilation, 10% of data entry was compared against the original paper copy as an audit of the digitization process.

In 2007, Terrane contracted Independent Mining Consultants (IMC) to complete a review of the drill hole database prior to mineral resource modeling and estimation. The review included analysis of check assay data, nearest-neighbour comparisons, comparison of drill hole assays in the database to original assay certificates, and comparison of primary assay data to metallurgical test results. ICM concluded that there is no strong evidence of systematic bias in the Mount Milligan database.

In 2014, prior to deposit modelling, TCMC completed a review of the database. Corrections were made for missing sample intervals and/or assay results, specific gravity results, and lithological descriptions.

In January 2018, the drill hole database was updated to support development of a 3D exploration model. This required detailed review of historical data for consistency to support modelling, and drill hole and assay data were added for 2015, 2016, and 2017 drilling programs. Historical (pre-2018) assays were not modified and the new assays added to the database for 2018-2019 drill holes followed the analytical, QA/QC and security procedures as described in Section 11.

2019 Review

Throughout 2019, additional validations and verifications of the database were conducted during the migration to acQuire data systems management software. These included:

- Review of the 2007 AllNorth transformation to confirm pre-2007 drill holes originally surveyed in the local mine grid were transformed to NAD83 UTM Zone 10 consistently,
- Verification of downhole survey data from raw data files where available for 2004 to 2019 drill holes,
- Correction of downhole survey data to NAD83 UTM Zone 10 north for 2006 to 2019 (previous compilations recorded downhole survey data to True North and the UTM convergence at Mount Milligan is approximately -0.85°),
- Verification of all copper and gold assay values from the previous database compared to original assay certificates for drill holes from 2004 to 2019,
- Compilation of missing 2004, 2006-2007, and 2011-2016 QAQC data to the database, and
- Compilation of 2004-2019 laboratory QAQC data to the database from original assay certificates.

The data reviews found the assay data acceptable and any errors or omissions were minor. Centerra-TCM considers the final 2019 database to be robust and verified.

Conclusion

The QPs of this report believe the database is adequate for the estimation of Mineral Resources according to CIM Estimation of Mineral Resources and Mineral Reserves best practice guidelines.

13. MINERAL PROCESSING AND METALLURGICAL TESTING

13.1. INTRODUCTION

Mount Milligan is a copper-gold porphyry deposit, consisting of two principal zones, the Main Zone and the Southern Star (SS) Zone. The Main Zone includes four contiguous sub-zones: MBX, WBX, DWBX and 66 (low-copper and high-gold grades, southeast of the MBX sub-zone). These geologic zones are the basis for the metallurgical test work.

The Mount Milligan Mine deposit is being mined using conventional open-pit equipment, with the ore being processed through a gyratory crusher, secondary crushing and a SAG-ball mill-pebble crusher (SABC) combination together with a rougher and cleaner flotation plant, producing a marketable gold-rich copper concentrate.

13.2. SUMMARY

Metallurgical investigations conducted by various research laboratories prior to commencement of operations conclusively showed that froth flotation is the optimum process for the recovery of a copper concentrate containing gold and silver. These investigations were the basis of the performance models used in previous resource modelling. The 2017 Technical Report addressed previous assumptions in the copper and gold recovery models together with identified issues in the plant to produce new performance equations.

Since disclosure of the 2017 Technical Report, further investigations and projects have been undertaken to improve the recovery process and update the accuracy of the copper and gold recovery models. Using these new performance models, the LOM average recoveries are estimated at 80.6% for copper and 61.8% for gold, targeting a concentrate grade with a LOM average of 21.5% copper. Test results indicated that impurity element contents in the concentrate were below the penalty levels normally imposed by most smelters; therefore, no significant penalties are expected.

13.3. PROCESS PERFORMANCE (POST-2017 TECHNICAL REPORT)

Actual copper and gold recovery values achieved since the beginning of 2017 (the cut-off for discussions in the previous NI 43-101 was December 31, 2016) have not met expectations set by the original metal recovery curves developed during the feasibility study.

Upon start-up in the winter of 2016, the secondary crushing circuit experienced blinding of the screen deck due to frozen fines build-up on the screen surfaces, which continued throughout the winter of 2016/2017. Since then, various mechanical and screen configuration modifications were implemented in combination with the use of an anti-freezing agent to maximize the availability of the secondary crushing circuit.

Prior to the installation of the secondary crushing circuit, the SAG mill would receive a top size feed of 150mm solely from the primary gyratory crusher. With the ability to divert up to 100% of the gyratory discharge to the secondary crushing circuit, the SAG mill's feed size was significantly reduced, averaging an 80% passing size (P_{80}) of 64mm for 2019. This led to modifications of the SAG mill's liner configuration, SAG mill grinding media size, SAG mill discharge screen deck configuration, operating philosophy of the pebble crushers, ball mill grinding media sizes, and ball mill cyclone configurations.

As the modifications to the grinding circuit took effect in 2018, the non-winter months when the availability of the secondary crushing circuit was not a concern, led to some record days and weeks of operation, specifically daily throughput. The process plant was able to achieve and maintain greater than 60,000 tpd fresh feed, while maintaining a P_{80} size of sub-212µm to the flotation circuit. On occasions when the fresh feed exceeded 65,000tpd, the P_{80} size to flotation would exceed 212µm, with associated impact on flotation performance.

From September 2018 until mid-April 2019 throughput was curtailed for water inventory management and the secondary crushing circuit was operated minimally during that time. Major maintenance activities were performed during the curtailment to ensure optimal equipment availability once the water restriction was removed. Also, during the curtailment, various continuous improvement (CI) initiatives were installed in the grinding circuit and the cleaner flotation circuit, both of which will be discussed in subsequent sections.

Upon removal of the water restriction in mid-April 2019, records were achieved for the monthly tonnes per operating day (tpod) for May, June, and July, where the average daily throughput attained in July, 61,438tpod, was the highest of the entire history of the operation for any month.

The 2017 Technical Report discussed plans to further investigate circuit performance issues in the flotation circuits. Significant testwork has been completed since the publishing of the 2017 Technical Report. This testwork, coupled with Continuous Improvement initiatives in the operation have justified the development of new copper and gold recovery curves. These curves, and their use in the new LOM plan are discussed in the sections below.

Data used during the feasibility study were reviewed in the 2017 Technical Report and specific reinterpretations of this information is discussed below.

13.4. UPDATED METALLURGICAL INTERPRETATION

History

The design of the flotation circuit is such that sulfides, specifically chalcopyrite and pyrite, were recovered in the rougher/scavenger flotation circuit. Chalcopyrite was then separated from pyrite through regrinding and three stages of cleaner flotation. As throughput ramped up toward design rates, the cleaner circuit was identified as a bottleneck in early 2016. This was primarily attributed to excess pyrite reporting to the cleaner circuit and the inability of the cleaner circuit to efficiently depress pyrite through pH modification.

Reagent testing was undertaken in early 2016 to debottleneck the cleaner circuit. Potassium Amyl Xanthate (PAX) was used as the primary collector for both the rougher/scavenger and cleaner circuits since it is considered a strong bulk sulfide collector, targeting both chalcopyrite and pyrite. A new collector reagent, Aerophine 3409 (A3409), was implemented in the rougher/scavenger circuit, selectively targeting chalcopyrite. As it was fully implemented in the flotation circuit, the sulfur recovery in the rougher/scavenger circuit dropped significantly while the metal recoveries in the cleaner circuit increased. The A3409 reagent was plant tested between November 2016 and March 2017.

Data analysis of plant performance during that time revealed missed opportunities in the flotation circuit where more sulfides could have been recovered in the rougher/scavenger without negatively impacting the cleaner circuit. Further optimization efforts led to moderate amounts of PAX being used in conjunction with the A3409, leading to copper recovery improvements.

Geoflotation Program

A geoflotation (geoflot) test program was undertaken from June 2016 to June 2017. Geoflot tests were bench-scale flotation tests performed on blast-hole cuttings from ore-control polygons. A total of 300 samples were ground to a P_{80} of 200µm with recoveries calculated for cumulative flotation times of 2, 4, 6, 8, and 18 minutes.

Daily plant performance, in conjunction with the test program results, were used to develop two models. The models were compared to the short term and long-term process plant forecast models previously published in the 2017 Technical Report.

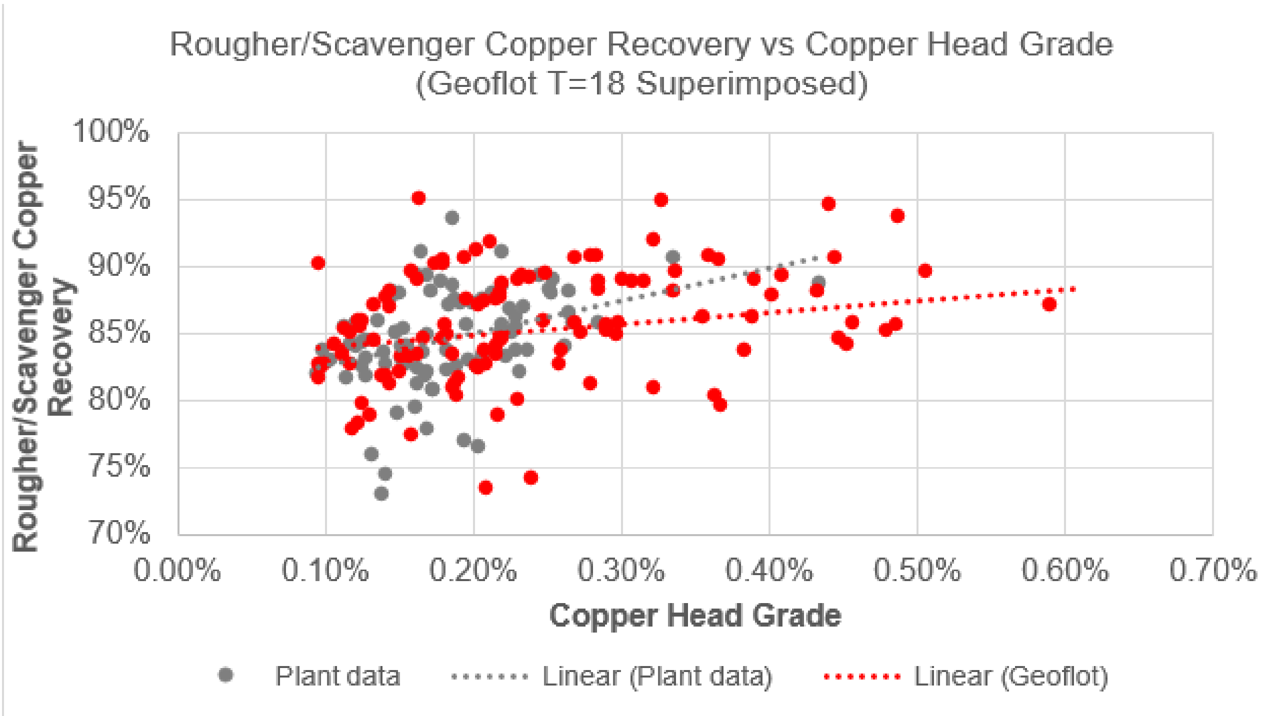
The two geoflot models produced were titled the Base Case Direct Regression (DR) model and the Aspirational model. Plant data from April to July 2017 was considered stable and were the basis of the two models.

13.5. COPPER RECOVERY CURVES

Comparison of Geoflot Data and Plant Performance

As discussed in the previous section, the geoflot tests were performed such that concentrates were collected at the 2, 4, 6, 8, and 18-minute mark of the test. Comparison of plant data and each of those timestamps showed that the 18-minute tests were most reflective of the performance of the plant. Figure 13-1 shows the plant performance superimposed on the 18-minute geoflot data. It shows that there is a good correlation between the plant's performance and the results from the geoflot testwork.

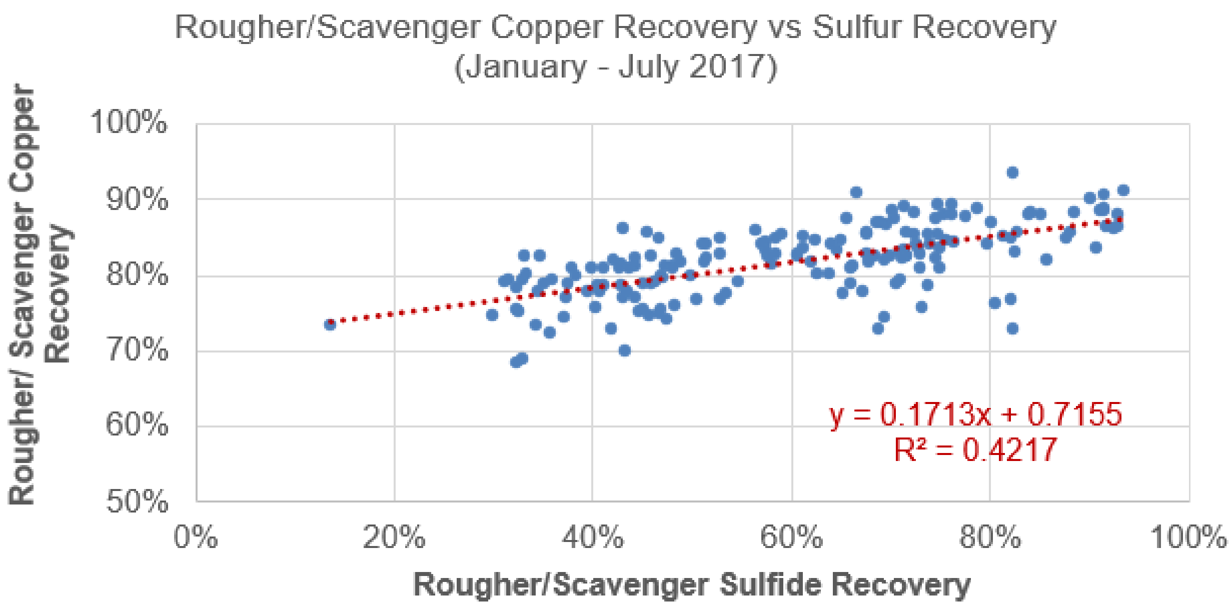
Figure 13-1: Plant Performance and Geoflot Results



Rougher/Scavenger Copper Recovery Model

The copper recovery in the rougher/scavenger flotation circuits is driven mainly by the total sulfur recovery of the rougher/scavenger circuit as shown in Figure 13-2. Two rougher/scavenger models, named Direct Regression and Aspirational, were developed. Plant data from January to July 2017 was used to determine this relationship to incorporate greater data coverage.

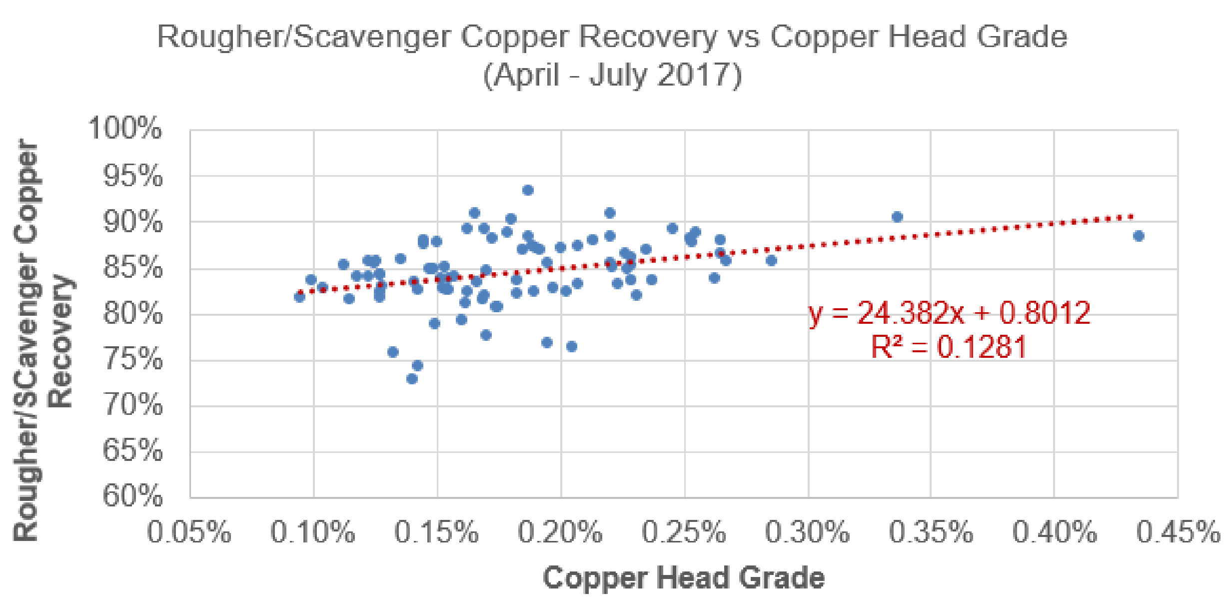
Figure 13-2: Rougher/Scavenger Copper Recovery Vs Rougher/Scavenger Sulfur Recovery



Direct Regression Model

The Direct Regression (DR) model uses copper head grade to predict rougher/scavenger recovery. Actual plant data from April to July 2017 was used to generate the plot in Figure 13-3 and related DR formula. This model agrees with historic plant data which showed higher copper recoveries at higher copper head grades.

Figure 13-3: Rougher/Scavenger Copper Recovery vs Copper Head Grade



Aspirational Model

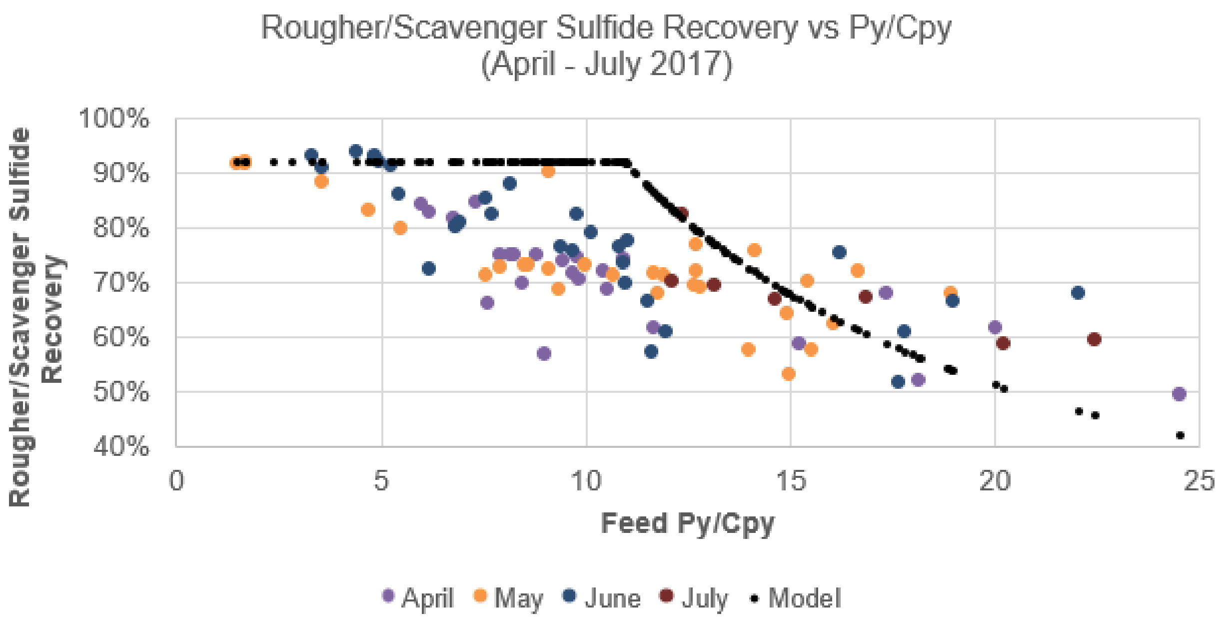
The Aspirational model is a variant of the DR model which focuses on optimizing sulfide recovery. Generally, copper feed grades to the process plant range from 0.10% to 0.35%, with sulfur feed grades ranging from 1.50% to 4%. Additionally, mineral liberation analyses (MLA) show that approximately 100% of sulfides at Mount Milligan are made up of chalcopyrite and pyrite. This poses challenges to the flotation process, particularly when ore with higher sulfur, associated with higher pyrite content is fed to the process. Therefore, the pyrite to chalcopyrite ratio (Py:Cpy) has become an important parameter to monitor.

During the development of this model, the following assumptions were made based on the available information at the time of its development:

- Based on historical data analysis the maximum sulfur recovery from the rougher/scavenger circuit is 92%
- Locked cycle tests indicate cleaner circuit feed should have Py:Cpy ratio ≤ 12 to avoid overloading conditions

Combining the two assumptions, the rougher/scavenger circuit should have throughput reduced when fresh feed Py:Cpy ratio is near 12, resulting in less than ideal sulfur recovery being attained when the feed Py:Cpy ratio is greater than 12. Figure 13-4 shows the relevant data from production with the Aspirational model. The model is designed to keep the Py:Cpy ratio less than 12 for the rougher/scavenger concentrate.

Figure 13-4: Rougher/Scavenger Sulfur Recovery vs Feed Py:Cpy



Investigations into the Aspirational model yielded the relationship of the Py:Cpy ratio. The model is referenced here in brief to describe the importance of maintaining the Py:Cpy ratio below 12. However, the Direct Regression model is the model that is used and will solely be discussed in the remainder of this section.

Predicting Cleaner Copper Recovery

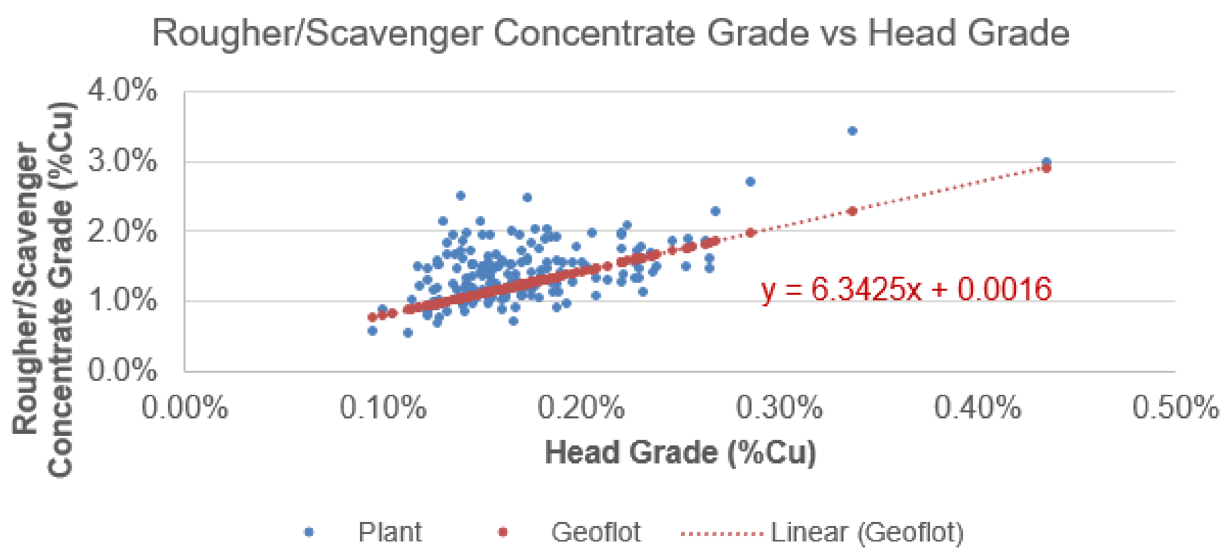
Removing the constraint on excess pyrite, the performance of the cleaner circuit is mostly affected by the upgrade ratio which is defined as:

Upgrade ratio = (Concentrate Grade)/(Feed Grade)

Rougher/Scavenger Concentrate Grade

The first step in estimating the cleaner circuit recovery was to obtain the rougher/scavenger circuit upgrade ratio which generates the cleaner feed grade. Linear regression was used on the geoflot data to obtain the relationship between rougher feed grade and rougher/scavenger concentrate grade. The regression line was subsequently overlaid on plant data as shown in Figure 13-5. The regression line has its y-intercept near zero, therefore, the slope of the line provides a model of the upgrade ratio for the rougher/scavenger circuit.

Figure 13-5: Rougher/Scavenger Concentrate Cu Grade vs Head Grade



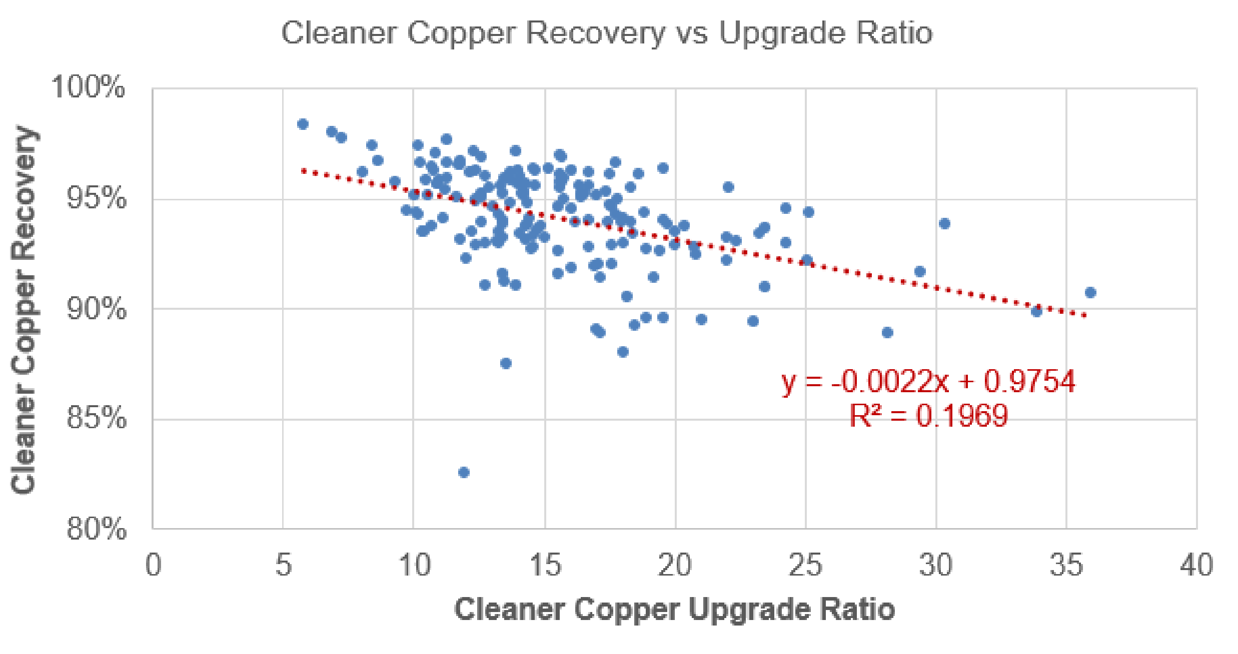
Despite most of the data points lying above the regression line, the expectation is that, with an increase in sulfide recovery per the Aspirational Model discussion, the rougher/scavenger concentrate copper grade and the upgrade ratio will drop slightly.

Cleaner Upgrade Ratio

The cleaner upgrade ratio can be calculated by two independent variables, cleaner concentrate grade and cleaner feed grade. From the previous section, the rougher/scavenger concentrate grade can be calculated by the equation generated in Figure 13-5, while the cleaner concentrate grade can be used as an input variable. Figure 13-6 shows the correlation between the cleaner copper recoveries and cleaner copper upgrade ratio based on plant data.

The significance of maintaining the rougher/scavenger and cleaner upgrade ratios independently is discussed in a subsequent section.

Figure 13-6: Cleaner Copper Recovery vs Cleaner Copper Upgrade Ratio



Overall Copper Recovery

In this DR model, the following equations are applied:

Rougher/Scavenger Cu Recovery = 24.382 × Cu head grade + 0.8012

Rougher/Scavenger Cu Concentrate Grade = 6.3425 × Cu head grade + 0.0016

Cleaner Cu Recovery = -0.0022 × (Cleaner Concentrate Grade)/(Rougher Concentrate Grade) + 0.9754

Overall Cu Recovery = Rougher Recovery × Cleaner Recovery

Model Accuracy

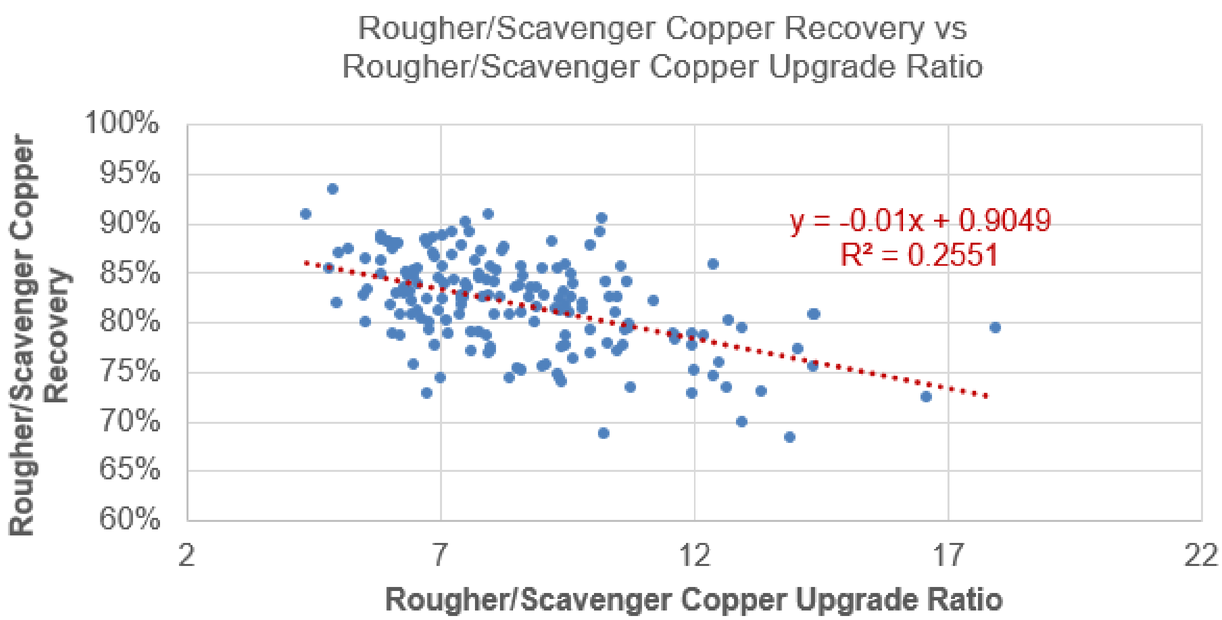
The DR model was compared to the actual plant performance to determine the model accuracies. The standard deviation of the residuals for the model was 3.79%. The actual plant copper recoveries were compared to the predicted copper recoveries using the DR model and were seen to be within 6.5% with 90% confidence.

Model Limitation

Based on the equations presented above, the rougher/scavenger copper recovery can be calculated as greater than zero when the copper head grade is equal to zero. This is mitigated by adding a further constraint to the DR model. The assumption is made based on historical operating data that the lowest acceptable rougher/scavenger concentrate grade is 1% copper. The equation generated in Figure 13-5 would yield a rougher/scavenger concentrate grade of less than 1% copper for a head grade less than 0.132% copper.

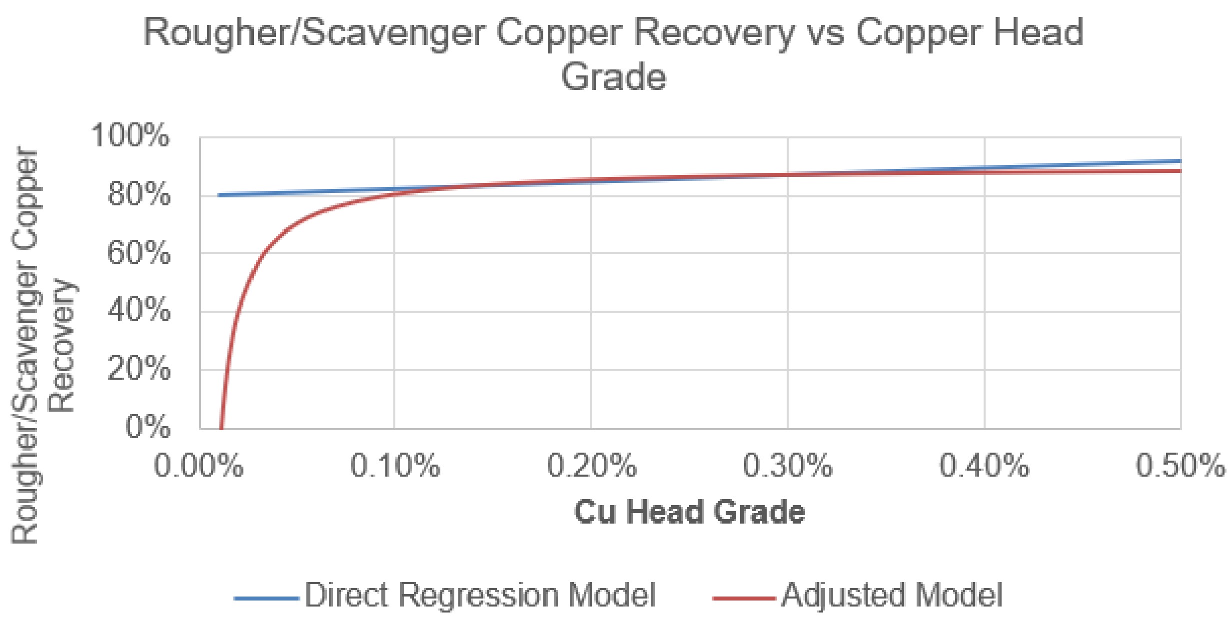
By correcting the rougher/scavenger concentrate grade to 1% copper for head grades below 0.132% copper, the rougher/scavenger copper recovery is predicted as a function of the rougher/scavenger upgrade ratio instead of copper head grade (Figure 13-7).

Figure 13-7: Rougher/Scavenger Copper Recovery vs Rougher/Scavenger Copper Upgrade Ratio



When the rougher/scavenger upgrade ratio equation is applied with a 1% copper rougher/scavenger concentrate grade, the head grade at the inflection point (Figure 13-8) becomes 0.132% Cu. In this case, the predicted rougher/scavenger copper recovery rapidly drops with head grades below 0.132%.

Figure 13-8: Corrected Rougher/Scavenger Copper Recovery Model



Despite the divergence at 0.35% Cu, the validity of both the DR and Adjusted models was accepted due to the plant's operating feed grade range between 0.10% - 0.35% copper. If the head grade is below 0.132% copper, the corrected rougher/scavenger copper recovery is calculated as:

$$Rougher/Scavenger\ Cu\ Recovery = -0.01 \times (1\%)/(Cu\ head\ grade) + 0.904$$

13.6. GOLD RECOVERY CURVES

Definitions

In the 2017 Technical Report the following gold recovery formulae were defined and are summarized below. For each formula the "x" term refers to the gold head grade in g/t and the "y" term refers to gold recovery.

- Feasibility: $y = 78.289x^{0.0434}$ (life of mine gold recovery formula used in the feasibility study)
- 43-101LOM: $y = 69.796x^{0.0152}$ (life of mine gold recovery formula used in the 2017 Technical Report)
- 43-101ST: $y = 62.871x^{0.101}$ (short-term, 2017 & 2018, gold recovery formula used in the 2017 Technical Report)

Background

The 43-101LOM and 43-101ST formulae were developed for the publication of the 2017 Technical Report. The 43-101ST formula was meant to be used for two years, specifically 2017 and 2018. During that time, various upgrades, targeting throughput and recovery, were expected to be completed through capital expenditure, after which the 43-101LOM formula was to be used. Since the publication of the 2017 Technical Report, Mount Milligan has been using the Modified 43-101 formula for calculation of the gold recovery in budgeting and forecasting exercises. The Modified 43-101 formula is the 43-101ST formula with adjustments depending on the percentage content of various types of ore.

The Modified 43-101 formula was used for budgeting and forecasting purposes. A Base Gold Recovery was used as the 43-101ST curves shown below:

MBX Zone: Gold Recovery = 62.87 x (Au g/t)$^{0.101}$

66 Zone: Gold grade ≤ 0.90 g/t: Gold Recovery = 50%

66 Zone: Gold grade>0.90 g/t: Gold Recovery = 62%

Based on historical plant performance, various other adjustments were made dependent on certain throughput and ore conditions.

If throughput was artificially controlled due to water conservation efforts:

*Gold Recovery = [Base Gold Recovery × (1 + 4.42%) - 1.5%] * (105%)*

If the throughput was not artificially controlled but the MBX content of the ore was more than 90%:

Gold Recovery = Base Gold Recovery × (1 + 4.42%)

If the throughput was not artificially controlled but the MBX content of the ore was less than 90%:

Gold Rec = Base Gold Recovery × (1 + 4.42%) – 1.5%

The formulae developed for the 2017 Technical Report were used in the scripts for the pit economic modeling. These scripts had conditional statements for calculating the gold recovery. For 2017 and 2018, the 43-101ST formula was to be used and for 2019 onwards, the 43-101LOM formula was to be used.

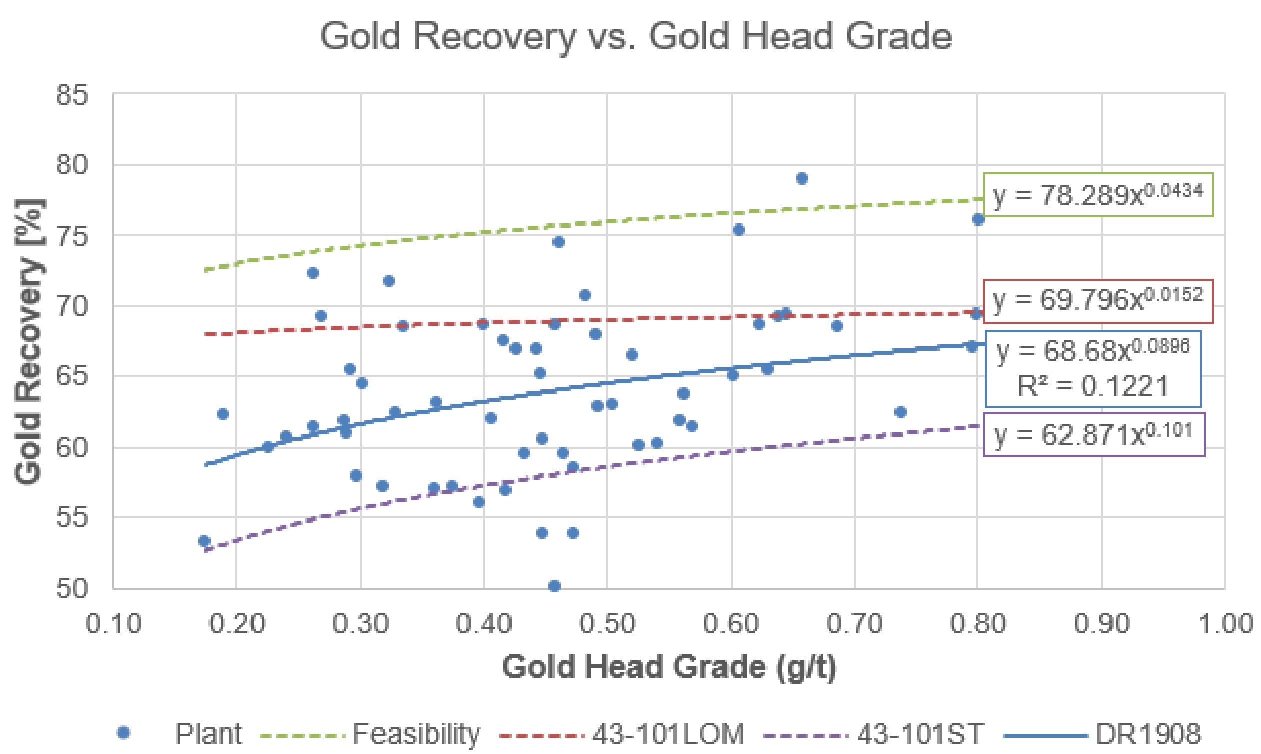
The 43-101LOM formula is quite aggressive in gold recovery improvements at lower head grades. The exponent of 0.0152 is approaching 0 which would make the recovery curve near linear and independent of gold grade; this is unrealistic. However, due to improvements in gold recovery between 2017 and 2019, a new gold recovery model was developed to more accurately reflect the process plant's performance and to be used in the budgeting, forecasting, and life-of-mine gold recovery estimation.

Model Development

The model was developed in August 2019 for which the actual plant performance from April 15 to early August 2019 was collated and analyzed to compare to the Feasibility, 43-101LOM, and 43-101ST curves. The data was filtered to only include days between 55,000 and 65,000 tonnes processed per operating day. April 15 was significant due to the removal of the throughput restriction through the winter of 2018/2019 due to water inventory management. Data from 2019 was solely used due to the implementation of continuous improvement projects in Q1 2019 further discussed in subsequent sections.

The resulting curve, named Direct Regression 2019-08 (DR1908), can be seen plotted alongside the abovementioned three curves in Figure 13-9.

Figure 13-9: Gold Recovery as a Function of Head Grade



The DR1908 curve is similar in shape to both the 43-101ST and the Feasibility curve and contains more data points than the curves generated for the 2017 Technical Report. As expected with the current operation of the plant and the available equipment, the gold recovery drops as the head grade is reduced. Efforts continue to improve the gold recovery, therefore the DR curve will be evaluated and updated over time.

13.7. LIFE-OF-MINE RECOVERY CURVE DISCUSSION

The previous sections describe the development and use of the copper and gold recovery curves for budgeting and forecasting purposes. Also described was the importance of the Py:Cpy ratio, ensuring all efforts are undertaken to blend the process plant feed ore, maintaining the ratio below 12.

During the LOM update in preparation for this Technical Report, blending of all the process plant feed ore to a Py:Cpy ratio below 12 was not possible. Production data from 2016 to 2019 was analyzed, comparing copper and gold recoveries segregated by less than and greater than a Py:Cpy ratio of 12. The results indicated that, for Py:Cpy ratio greater than 12, the copper and gold recoveries should be discounted by 3% and 7% respectively, which has been included as part of this LOM plan and related economic analysis.

13.8. HGLC DISCUSSION

Definitions

- NSR: Net Smelter Return calculation that considers metallurgical recovery, metal prices, and treatment & transportation costs.
- HGLC: Material above the NSR economic cut off value of $7.64 containing copper grade below 0.12%.
- ORE: Material above the NSR economic cut off value $7.64 containing copper grade above 0.12%.

Feasibility Study 66 Zone Testwork

HGLC ore is located in many locations in the pit with the greatest concentration in the 66 zone. As part of the feasibility study work completed in 2007, a variety of standard tests were conducted to define the metallurgical performance of composite samples from the 66 zone. The following summarizes the key findings:

- The low copper master composite recovered approximately 90% of the gold to a rougher concentrate requiring 18-20% mass recovery to the rougher concentrate.
- Similarly, the high copper master composite recovered approximately 90% of the gold to a rougher concentrate requiring 18-20% mass recovery to the rougher concentrate.
- For a final concentrate of 21.5% copper, copper recovery was 80% and 50% for the high and low copper master composites, respectively.
- Corresponding gold recoveries, for the low and high copper composites, were 63% and 80% respectively.

Duplicate locked cycle tests were carried out on the 66 zone master composites. Further locked cycle tests were carried out on various 66 zone composites samples with varying copper and gold content. The following summarizes the key findings:

- The low copper master composite recovered approximately 81% and 66% of the copper and gold respectively, yielding a 13.1% final concentrate copper grade.
- The high copper master composite recovered approximately 83% and 78% of the copper and gold respectively, yielding a 23.3% final concentrate copper grade.
- Two 66 zone composites containing both low and high gold content were generated for variability testing. They produced gold recoveries of 75% and 81%, yielding approximately 5% final concentrate copper grade.

Locked cycle tests were then completed blending MBX, WBX, SS, and high/low copper 66 zone master composites. The blends recovered 80-90% and 71-81% of the copper and gold respectively, yielding 20-30% final concentrate copper grade.

Geometallurgical Locked Cycle Testwork (Since 2017 Technical Report)

Based on the feasibility study 66 zone test results discussed in the previous section, the common practice at Mount Milligan has been to blend the MBX and 66 zone feeds to provide a reasonable head grade such that a saleable concentrate can be produced at reasonable recovery rates. A test program was initiated in 2017 to determine whether continuing to blend the MBX and 66 zone ores is financially beneficial compared to the individual, “single-feed” processing of those respective zone ores. The secondary objective was to further confirm maintaining a feed blend to the process plant of the pyrite to chalcopyrite ratio (Py:Cpy) less than twelve.

Composite samples from a 2017 geometallurgical drilling program were generated to cover a large range of head grades representing HGLC and ORE from the 66 and MBX zones. The seven single composites were distributed such that three were low grade and four were higher grade copper ranging from 0.01-0.46% copper and 0.25-0.88g/t gold. The single composites were used to generate seven 50:50 blended composites that reflected the typical process plant feed copper and gold head grades expected for 2018, ranging from 0.10-0.24% copper and 0.28-0.68g/t gold.

Locked cycle tests were performed on the unblended single composites yielding a range of 2.26-18.8% copper in the concentrate for the low-grade feed and 20.8-26.6% copper in the concentrate for higher grade copper feed. Further locked cycle tests were performed on the blended composites yielding a range of 18.6-23.3% copper in the concentrate.

This study concluded that, based on the test results, MBX and 66 zone ores should be blended in order to justify recoveries as tested without process plant modifications requiring major capital investment. Additionally, the geometallurgy team confirmed that maximizing the 66 zone HGLC value would be achieved by feeding the plant a blended feed Py:Cpy ratio less than twelve and a head grade near 0.18% copper.

13.9. CONTINUOUS IMPROVEMENT PROJECTS

Secondary Crushing

The secondary crushing plant project was completed in late 2016. As discussed in section 17, the gyratory crusher product passes through a transfer point with a diverter gate which allows for partial to total diversion of the material to the secondary crushing circuit. The material that is not diverted proceeds to the ore stockpile to feed the SAG mill. Upon operation in the winter, the secondary crushing circuit experienced blinding of the screen deck due to frozen fines build-up on the screen surfaces, which continued throughout the winter of 2016/2017. Since then, various mechanical and screen configuration modifications were implemented in combination with the use of an anti-freezing agent to maximize the availability of the secondary crushing circuit. Additional improvements are ongoing and are planned to be completed in 2021.

Comminution Circuit Optimization

The original SAG liner and discharge configuration consisted of top-hat style shell liners/lifters and radial pulp lifters, which allowed bi-directional rotation of the mill in order to maximize wear life of the liner and discharge components. Unfortunately, this combination had significant disadvantages in terms of operating performance at higher mill rotating speeds due to ball-on-liner impacts and/or mill discharge rate, which effectively capped the speed at less than 72% of critical speed. Through several iterations, a revised liner package was installed in October 2018, enabling the SAG mill to operate effectively at its maximum of 81% of critical speed.

The original design of the Mount Milligan SAG mill discharge screen called for a single screen in that application. Due to debottlenecking efforts it was replaced with two parallel screens. The SAG mill discharge box was redesigned and installed in January 2018 to resolve the issue of uneven loading on the screens.

The SAG screen bottom deck aperture was changed from 16mm to 12.5mm, allowing for more material to be sent to the pebble crushers, targeting near full time utilization of both crushers. This change eliminated the surging of crushed pebbles to the SAG mill, ball mill circuit, and flotation feed; stabilizing the whole plant operation. The change yielded reduction in primary grind size (Figure 13-10), reduction of operating work index of the grinding circuit indicating that the grinding is now more efficient (Figure 13-11), and an increase in rougher flotation copper and gold recoveries due to finer grind size sent to the rougher circuit (Figure 13-12 and Figure 13-13). The standard deviations of primary grind size and operating work index also decreased, indicating a more stable operation.

Figure 13-10: Primary Grind Size Comparison

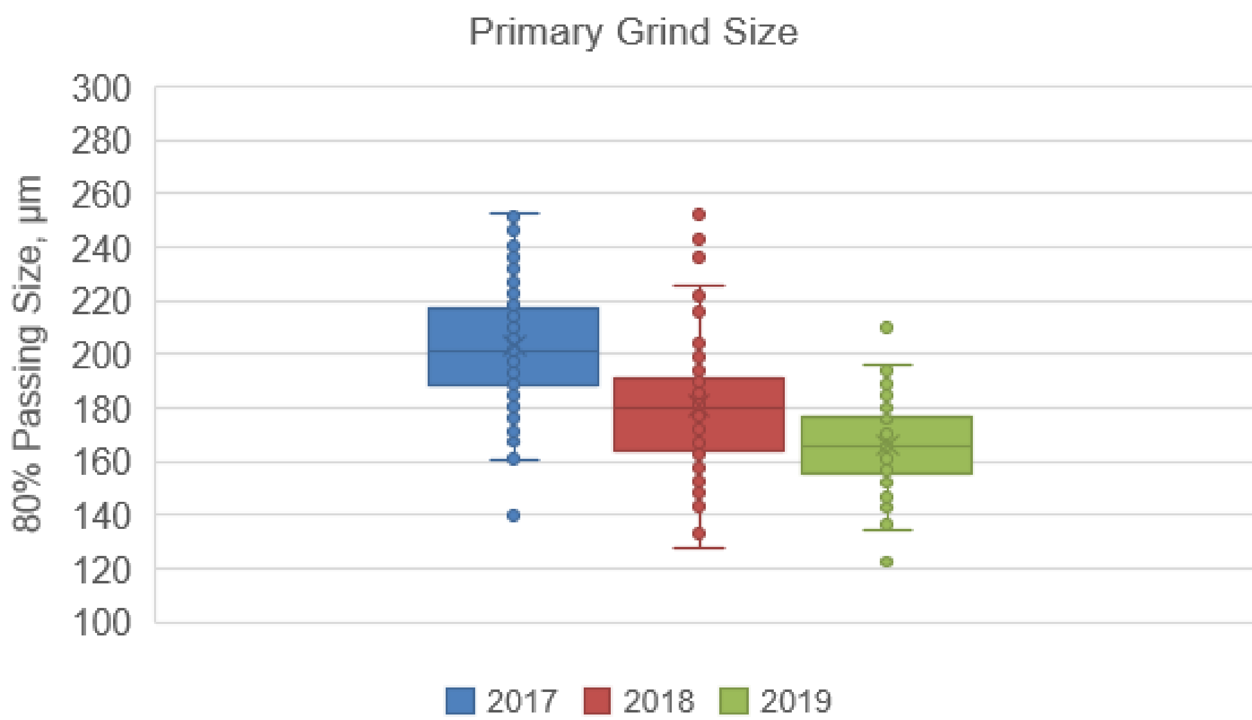


Figure 13-11: Operating Work Index Comparison

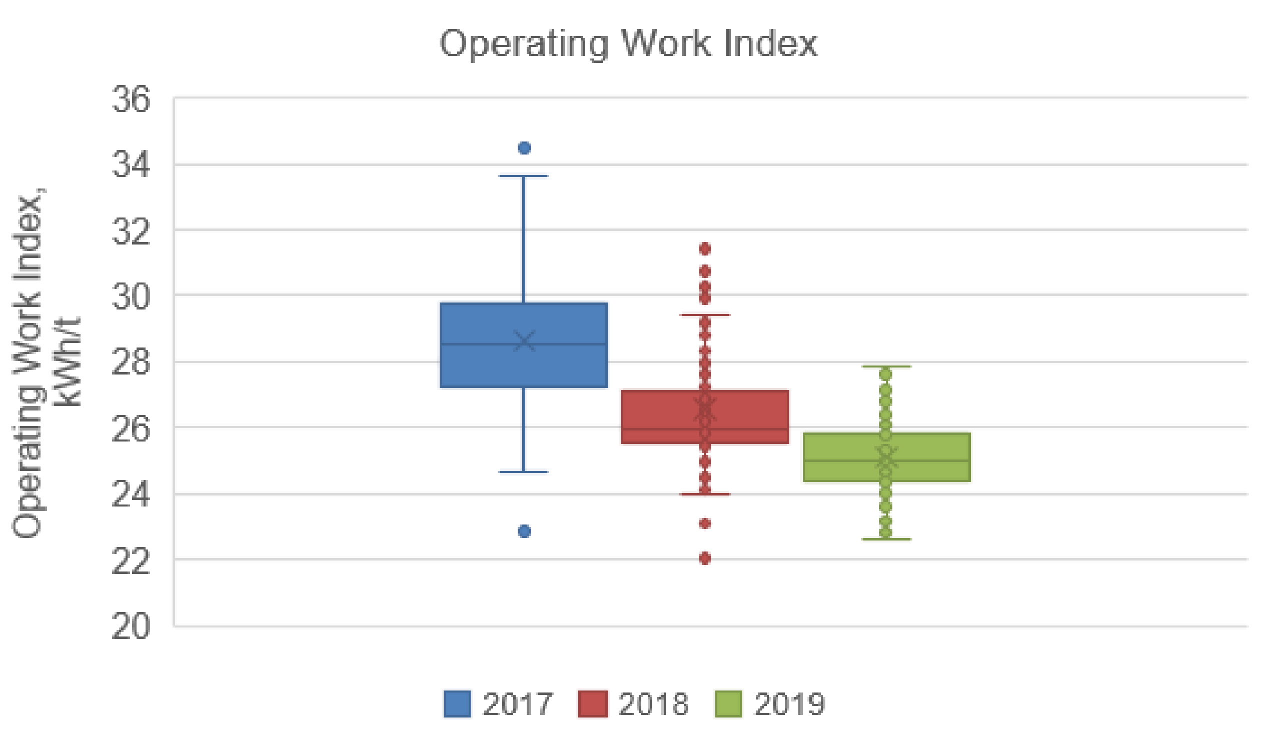


Figure 13-12: Rougher Flotation Gold Recovery Comparison

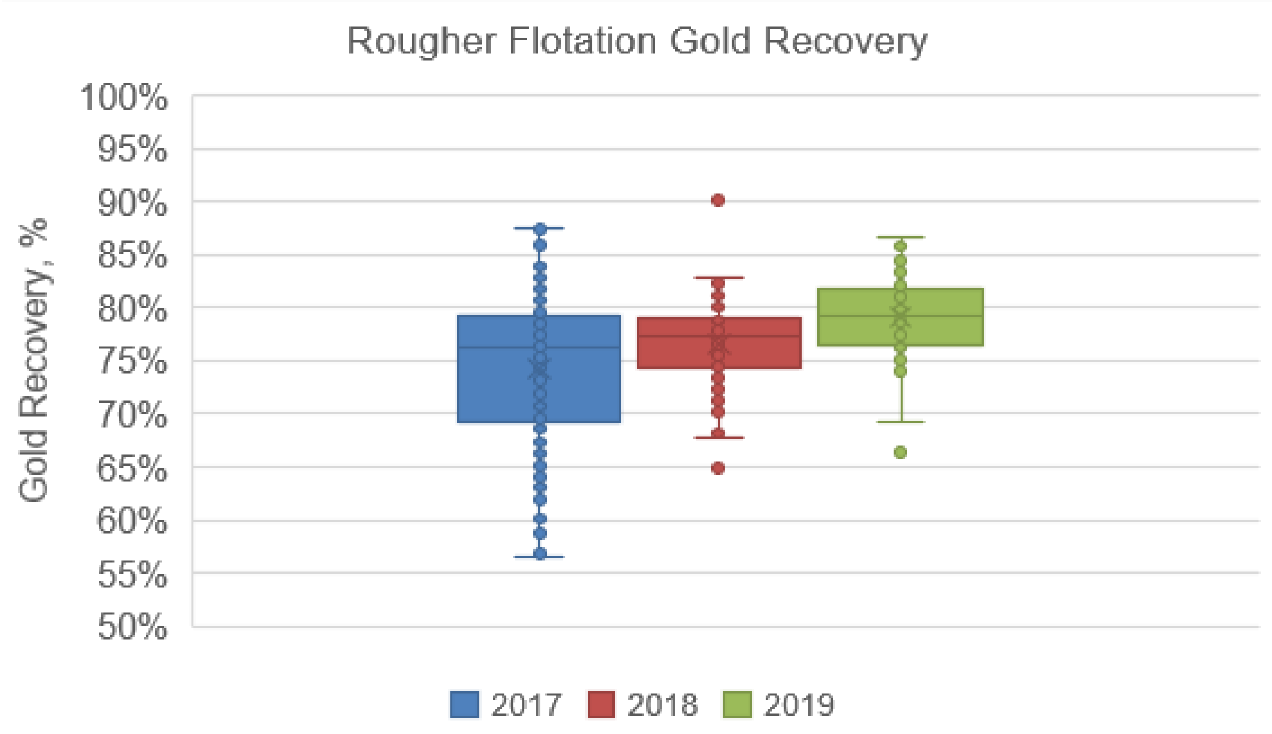
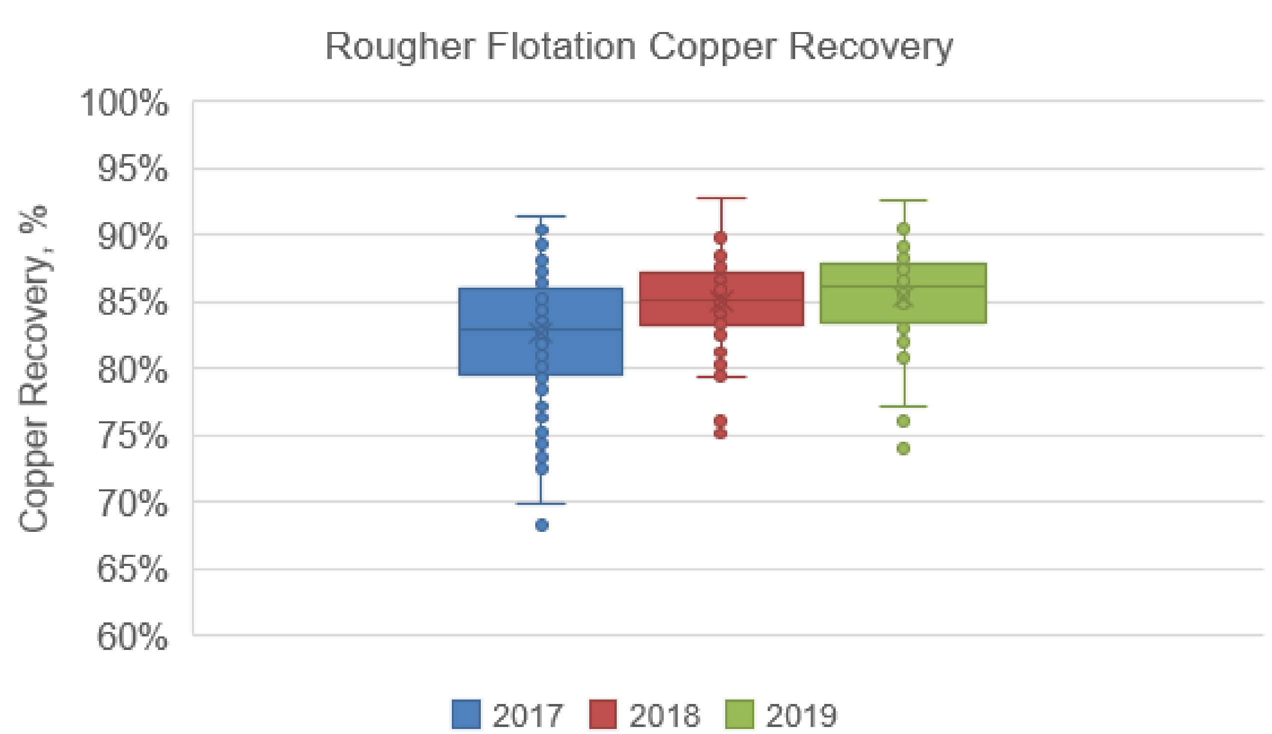


Figure 13-13: Rougher Flotation Copper Recovery Comparison



Staged Flotation Reactor (SFR) Test Programs

In 2017, a Staged Flotation Reactor (SFR) pilot test program was conducted using plant samples from the flotation circuit. Historically, Mount Milligan has received a range of low to high Py:Cpy ratio ore. As discussed previously, the optimal performance of the flotation circuit occurs at a ratio below 12. The pilot tests for each location occurred in two phases, the first being at low Py:Cpy ratio and the second being at high Py:Cpy ratio. The results of the test program indicated that the most successful location to install the SFRs were in the scalping of the 1st cleaner feed, producing a 25% Cu concentrate with a stage Cu recovery of 60%. This would reduce the mass flow to the cleaning circuit by up to 5% which would increase its capacity. The pilot test results were not immediately utilized in a plant upgrade feasibility study or implementation of plant improvements due to the focus on debottlenecking and optimization projects to reach design throughput capacity.

Across 2018 and early 2019, many debottlenecking and optimization efforts were undertaken. Through a multitude of mechanical, operational, and process control projects, the process plant was consistently and reliably exceeding 60,000 tonnes throughput per operating day.

In mid-2019, the SFR project was revived with a goal of completing a feasibility study by the end of the year. Mount Milligan conducted a study on the impact of process changes; specifically, the addition of flotation equipment. As part of the project, a test program was initiated to provide additional engineering data so the study could be completed with greater accuracy. The first part of the study was to validate the 2017 pilot program's results given the process improvements since that test program. The second part of the study was to evaluate the potential for SFRs to recover the ultrafine (<20 µm), liberated gold that exits the process plant in the cleaner scavenger tailings (CST) stream; a cleaner scavenger scalper application was investigated.

The results of the test program for each scope showed significant improvements in the upgrade ratios for each of the respective stages for those scopes. In the 1st cleaner scalper application, the copper and gold upgrade ratios increased from 3.7 and 3.2 baselines respectively to 9.8 and 6.6 in the SFR concentrates respectively. In the cleaner scavenger scalper application, the copper and gold upgrade ratios increased from 16.1 and 13.5 baselines respectively to 73.7 and 41.3 in the SFR concentrates respectively.

Since each of the above-mentioned scopes were analyzed independently, the design for the cleaner scavenger scalper SFR targeted a final concentrate grade so as not to dilute the overall concentrate grade of the plant. Due to this, it was initially overdesigned to include four rougher stages and four cleaner stages. Since the primary purpose of the SFR application was to target the ultra-fine liberated gold in the CST, combining both SFR applications was investigated, using only the rougher stages in cleaner scavenger scalper, with its concentrate reporting to the first cleaner scalper.

Recovery improvements were quantified, and a pilot plant is planned for 2020 to validate the results of the bench-scale tests performed in the 2019 study.

Metallurgical Testing of Cleaner Tailings

As discussed in Section 17, the Mount Milligan process produces low-sulfide rougher tailings and high-sulfide cleaner tailings. Due to the nature of the tailings, they are stored separately to minimize the environmental impact. In 2018 a study was conducted to investigate further recovery of copper and gold from the tailings stream using representative samples that were extracted from the high-sulfide cleaner tailings impoundment. A variety of flotation tests resulted in average recoveries of 10.8% and 28.6% for copper and gold respectively. Cyanide leach tests were carried out yielding average gold recoveries of 77.5%, 80.4%, and 80.8% after 6-hour, 24-hour, and 48-hour leaches respectively.

Cleaner Column Optimization

Operation of the cavitation tube spargers since March 2019 has shown a shift in grade-recovery curve allowing higher concentrate grade without sacrificing recovery.

Process Control

Significant efforts were undertaken in 2018 and 2019 on stabilizing the flotation circuit. Control loops were audited, tuned, and put back in service. Regular investigation of tuning occurs when feedback is received from process plant operations personnel. Recommissioning of the plant-wide expert systems will be considered a priority in 2020 to further optimize the operation of the plant.

13.10. REQUIRED IMPROVEMENTS

The LOM metallurgical recovery curves have been redefined in the earlier subsections of this section. The maintenance of these recoveries, as discussed, is dependent on the content of the ore that is fed to the process plant. The HGLC ore should be blended such that there is sufficient copper to produce a saleable grade concentrate. The Py:Cpy ratio should be blended to as low as possible, with all efforts to keep it below 12.

Further improvements in metal recoveries can be achieved by expansion of the flotation circuit. Comminution debottlenecking continues; however, operation of the plant has shown that it has the capacity for significantly higher throughput. The 2017 Technical Report stated that “data indicates that as throughput increases there will be a shift in the bottleneck from primary grinding downstream to two areas: the ball mills being able to achieve target grind and the flotation circuits to achieve recovery.”

Grinding efficiency improvements in the SAG and ball mill circuits have led to a decreased product size feeding the rougher flotation circuit, averaging 175µm for 2019 at nominal throughput rates. In those instances where the process plant was operating above the nominal throughput rates in 2019, the grind size rarely exceeded 200µm. This addresses the first of the statements from the 2017 Technical Report.

In the future, as the throughput is consistently maintained at higher rates, the retention time in the flotation circuit will become a bottleneck. The original design for the flotation circuit is 8 tph of copper feeding the circuit. With the recovery and stability improvements, 10-11 tph of copper can currently be successfully fed to the flotation circuit without major detriments to the copper and gold recoveries. However, the bottleneck can be seen in the cleaner circuit when higher throughputs are maintained. This, coupled with the fine gold losses seen in the sulfide cleaner tailings, has led to investigations discussed in this section related to SFR technology, alternative processing flowsheets to include cyanide, and the reprocessing of tailings in an effort to collect liberated gold.

As a result, the following is being progressed:

- Evaluate the use of alternative flotation equipment such as Staged Flotation Reactors or Direct Flotation Reactors. The 2017 and 2019 test work discussed in this section has shown potential to increase both copper and gold metallurgical recoveries. Confirmation work by operating a pilot plant on site in 2020 is in progress.
- Evaluate alternative flowsheets for treatment of the HGLC and high Py:Cpy ratio ore to improve metal recoveries. This includes the potential use of flowsheets using cyanide to enhance gold metallurgical recovery.
- Evaluate high-sulfide cleaner tailings re-processing, an opportunity that was previously assessed at a high level.

14. MINERAL RESOURCE ESTIMATES

14.1. INTRODUCTION

The corporate geology team of Centerra prepared a mineral resource estimate for the Project using data from both historic drilling and drilling from recent years undertaken by TCM. A database was compiled using data from 1,134 core holes, with collar and down-hole survey, geological and assay information. It is important to note that approximately 65% of the current drill hole database do not have azimuth values due to magnetite content which would cause errors for the acid test used prior to 1990 drilling programs. Consequently, those drill holes are considered to create a degree of error in mineralized domain interpretation and grade distribution interpretation. The mineral resource model discussed herein considers assay data available as of October 31, 2019. Table 14-1 summarizes the records in the database used for the updated resource estimation.

Table 14-1: Summary of Geological Database

	Number of DDH	Length (m)	Survey	Lithology	Assay
Records 2019	1,134	295,961	8,559	22,363	1,133,869

In assembling the database for the mineral resource estimate, a systematic approach was used to select assay data from the core drill samples. Intervals noted in the drill logs as being subject to any discrepancy with the assay certificates were investigated and corrected or removed from the resource estimate database.

The mineral resource estimate was derived from a computerized resource block model. The construction of that model started with drill hole data, which served as the basis for the definition of 3D mineralized envelopes with resources limited to the material inside those envelopes. The next step was the selection of drill hole data within the mineralized envelopes in the form of fixed length composites. This was used to interpolate the grade of blocks on a regular grid and populate the mineralized envelopes from the grade of composite samples within the same envelopes. All the interpolated blocks below the overburden/bedrock contact or pit bottom surface comprise the mineral resources which were then classified based on their proximity to composites and the corresponding precision/confidence level. The Centerra team, based on information provided by the exploration database, have re-interpreted the mineralized domains and completed the mineral resources estimation.

14.2. GEOLOGY AND MINERALIZATION MODELS

Geology Model

Geological Interpretation

The most recent interpretation of geology, alteration and structure was used, based on the regional and local (mine) litho-structural interpretation. This interpretation is constantly in the process of being updated by Centerra geology staff as new data becomes available. The Mount Milligan intrusive complex comprise dominantly of monzonitic rocks: MBX (Magnetite Breccia), SS (Southern Star), Goldmark, and North Slope stocks. The Mount Milligan property has been sub-divided into different zones based on prevailing geology and structural controls: Southern Star (SS), Saddle, 66, MBX, and WBX (please refer to Figure 7-2).

Overburden

An extensive layer of overburden covers the deposit and low-lying areas to the east and south of the deposit. This overburden consists of till and fluvial material, and thicknesses are highly variable, ranging from metres across parts of the deposit to 100 m thick to the north and east of the MBX stock.

Overburden was typically tricone drilled during exploration, with descriptions collated from the drillers' logs. The overburden-bedrock contact was defined as the base of casing, and a points file was generated in Vulcan to mark the top of bedrock for all drill holes. An overburden surface was generated through triangulation of the top of bedrock points file.

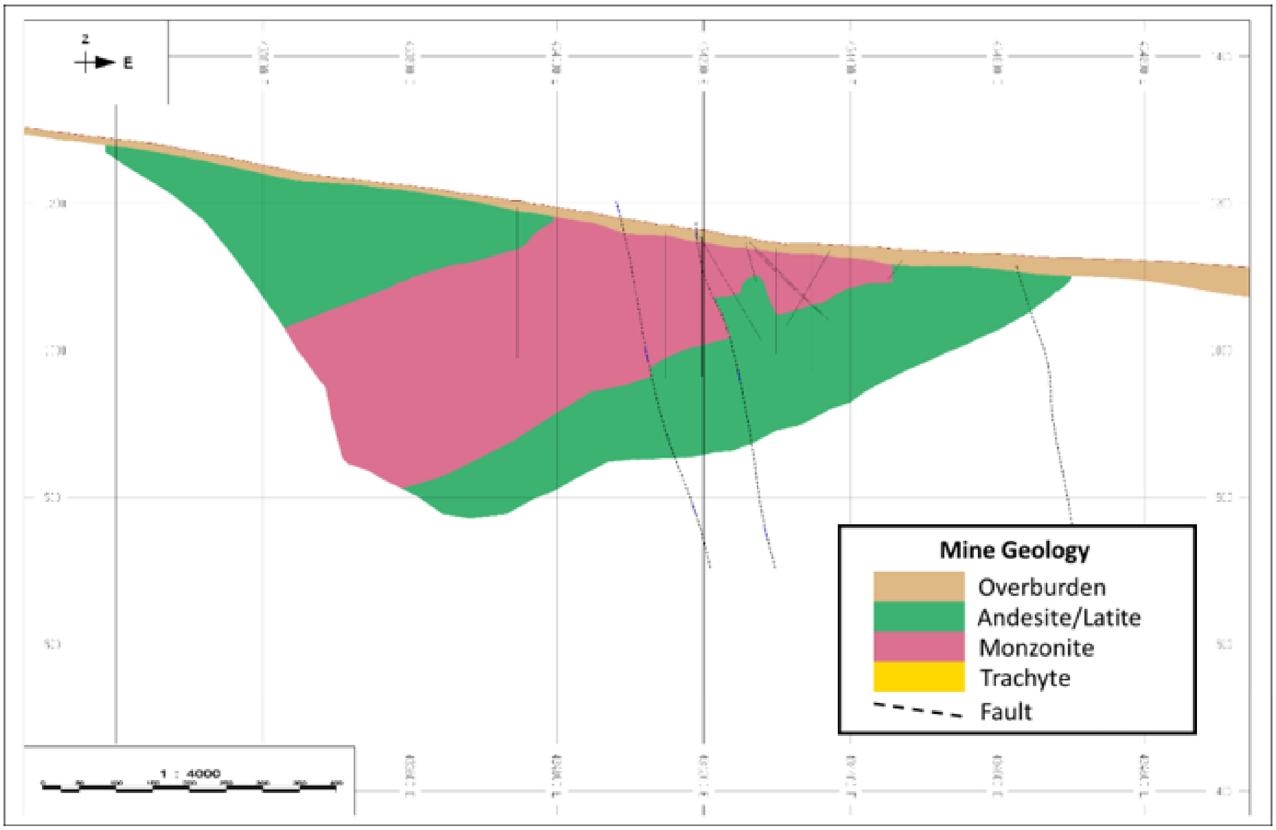
A geotechnical site investigation program consisting of 17 HQ diameter drill holes was completed by Knight Piésold (KP). Six of these holes cored overburden within the perimeter of the Main open pit, and each hole was completed in weathered bedrock. These holes are considered more accurate than exploration drill holes in defining the base of overburden.

The 2007 site investigation drill logs were obtained from KP. Drill holes KP07-10, KP07-11, KP07-12, KP07-14, KP07-15, and KP07-16 were collared within the pit limits. The depth to bedrock was compared to modelled depths (Table 14-3).

Lithology and Structure

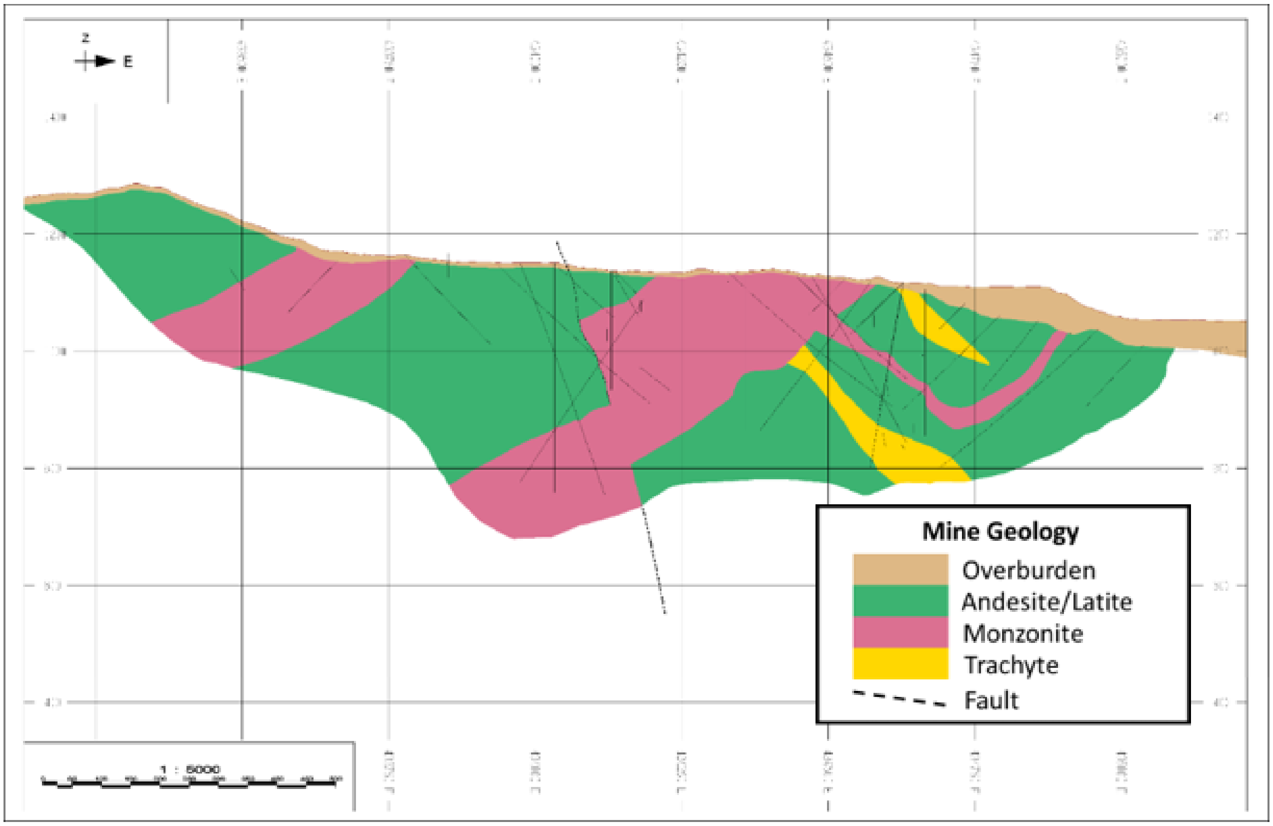
The lithology of the Project was simplified to three major units: volcanics (andesite and latite), trachyte and monzonite. These units were split into discrete shapes by major geologic structures. Major geologic structures were modelled in section (Figure 14-1 and Figure 14-2). All features were tied to individual drill holes where relevant. An estimation envelope was used to limit extrapolation beyond the base of drilling.

Figure 14-1: Southern Star Zone Cross Section – 6108220N



Source: Centerra, Geology Department

Figure 14-2: MBX Deposit Cross Section – 6109360N



Source: Centerra, Geology Department 2017

A series of post-mineral dyke swarms occur in both the Main and SS deposits. These dykes are irregular, discontinuous, and vary from sub-meter to 15m in width, making them difficult to model in three dimensions and, as such, the dykes were not modelled. The dyke intervals were included in the drill hole assay samples, thus their impact is included in the copper and gold grades dataset for resource estimation.

Mineralization Models

The Centerra team, using Datamine RM modelling software (version 1.5.62.0) and data based on the complete drilling database, has re-interpreted and created updated mineralized wireframe models for the 31st December 2019 mineral resource estimate. Updated models were interpreted using the data received from the database manager with the cut-off date of October 31, 2019. A model of the mineralization, above 0.1% CuEq cut-off, was created by digitizing polylines on east-west sections at approximately 40 metres intervals throughout the deposit. In some instances, the section intervals were greater, up to 60m, relative to the drilling density. The interpretation of the mineralization models was correlated with the litho-structural interpretation completed by Centerra's exploration and mine geology groups (Figure 14-3). Polyline interpretation was verified in north looking sections and horizontal view for consistency. These polylines were linked and triangulated to create three-dimensional wireframe solids. Figure 14-3 to Figure 14-7 display the interpreted mineralization.

Mineralized domains were characterized by:

- Delineation of waste inclusions.
- Strong characterization of the anticlinal signature with the variability in geometry (dip and plunge).
- Distinctive gold distribution generally along the eastern flank of the deposit (66, Saddle and SS zones) but relatively high variability.

The mineralization occurring outside of the modeled domains was captured within the shell and interpreted accordingly. An isometric view of lithology and several cross-sections of colour-coded domains are provided in Figure 14-3 and Figure 14-4 to Figure 14-7.

Figure 14-3: 3D Litho-Structural Interpretation- Isometric View-Looking N-NE

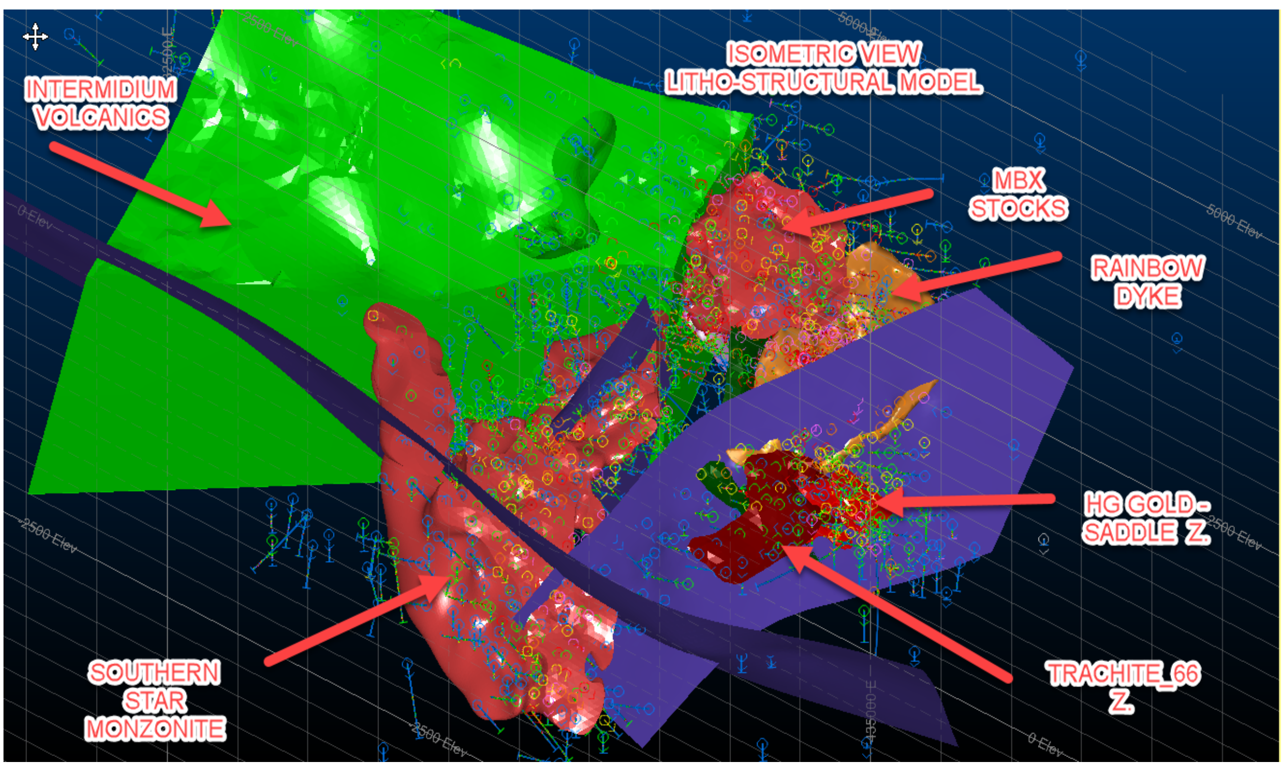


Figure 14-4: 3D Mineralized Domains- Isometric View-Looking N-NE

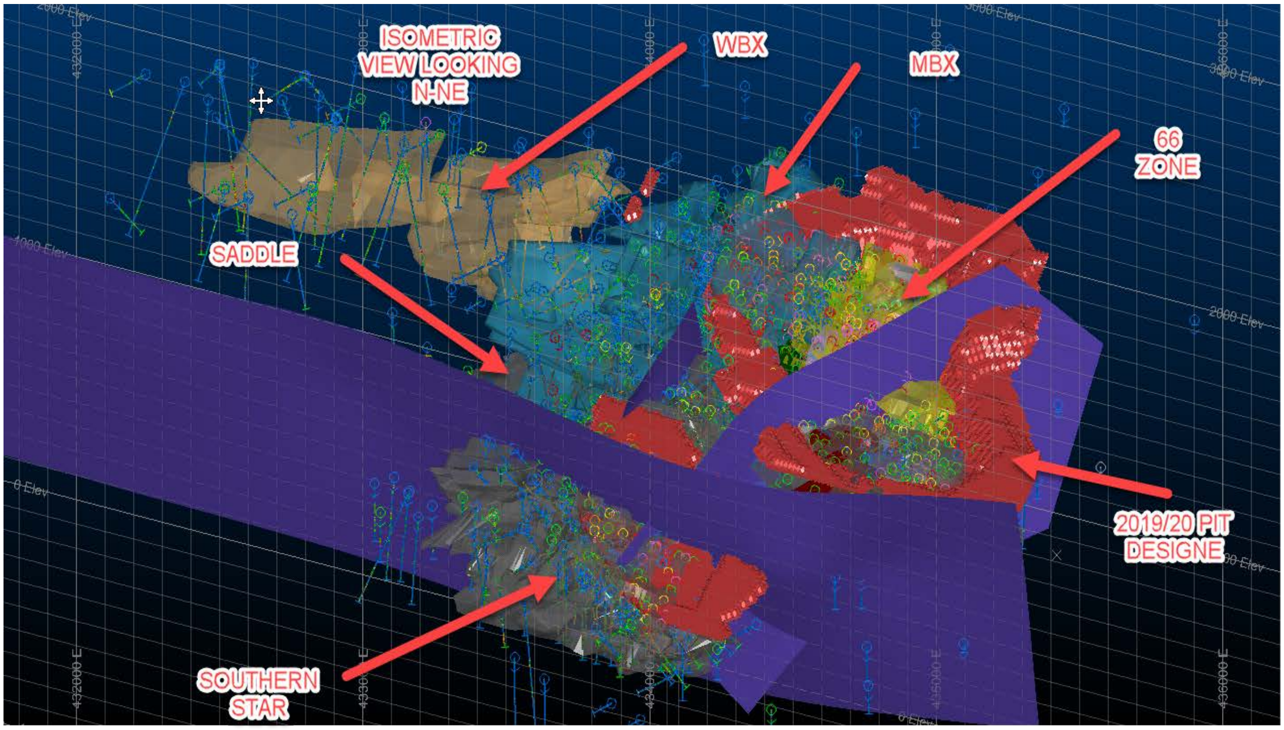


Figure 14-5: Drill Holes and Limits of Domains in Cross Section 6,108,900N, Looking North (50-m grid spacing)

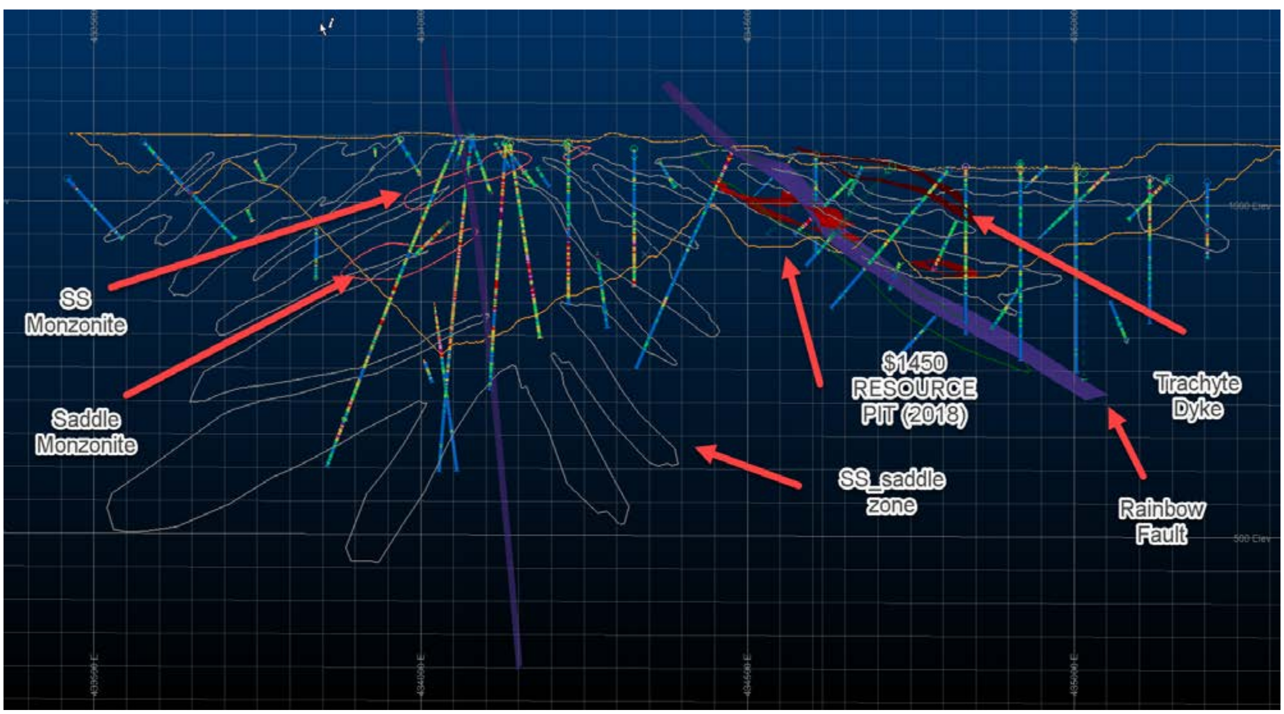


Figure 14-6: Drill Holes and Limits of Domains in Cross Section 6,108,700N, Looking North (50-m grid spacing)

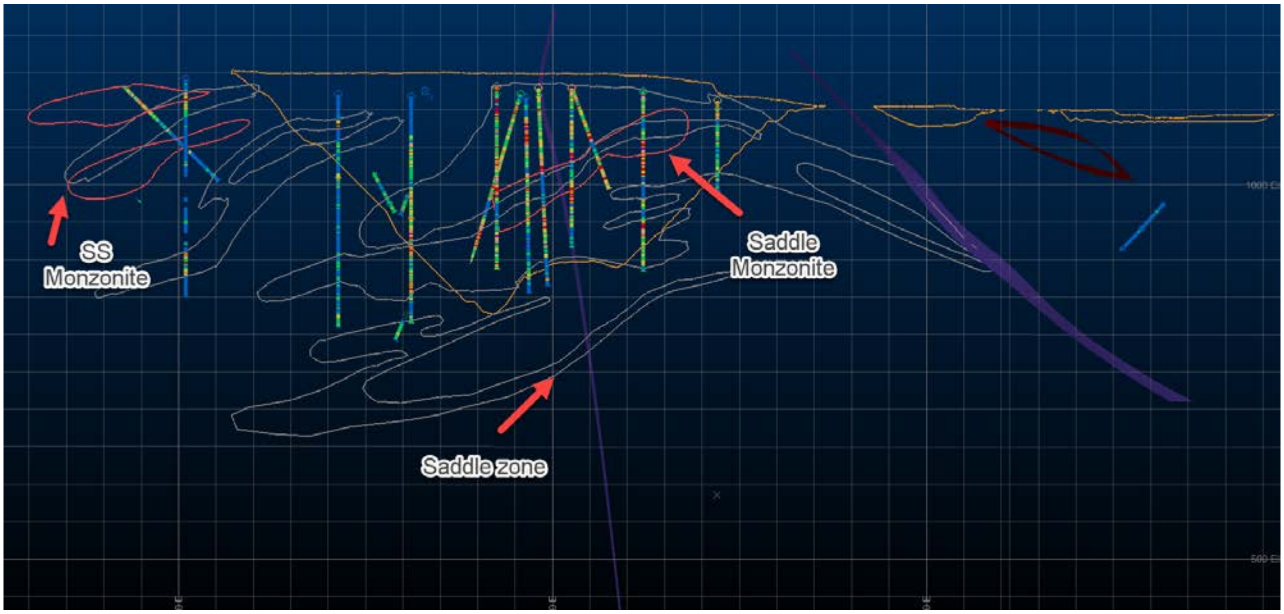
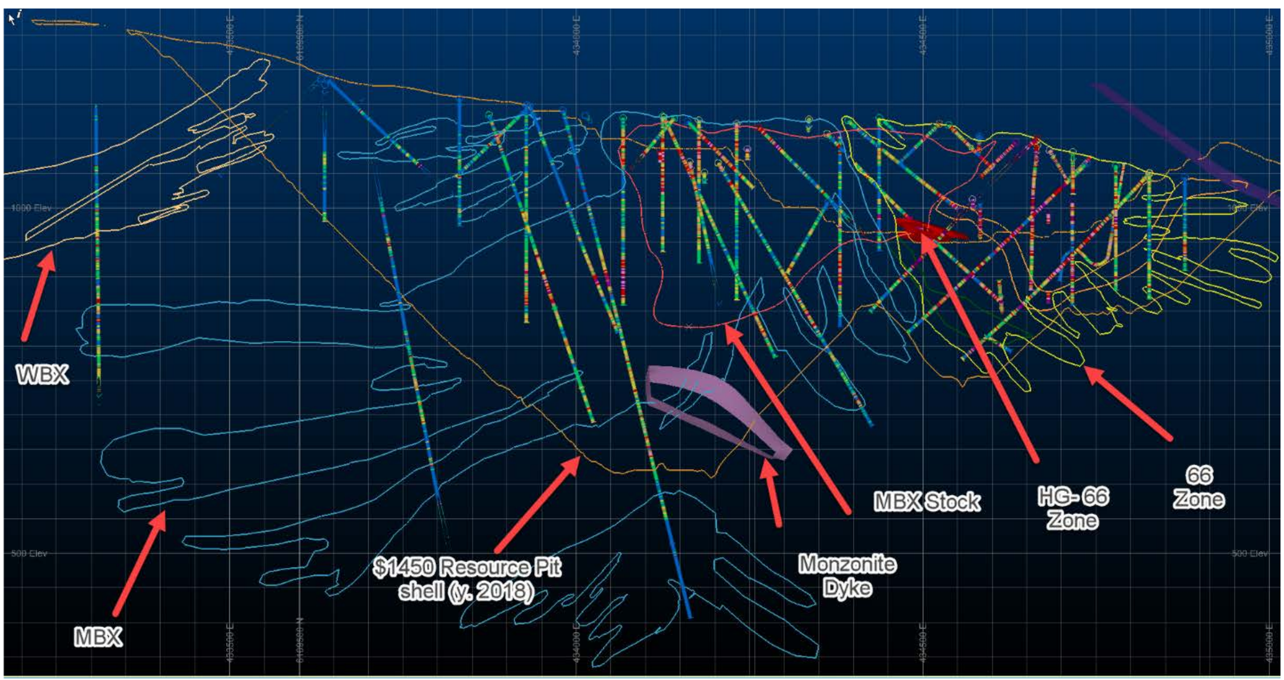


Figure 14-7: Drill Holes and Limits of Domains in Cross Section 6,109,460N Looking North (50-m grid spacing)



Mineralized Domains

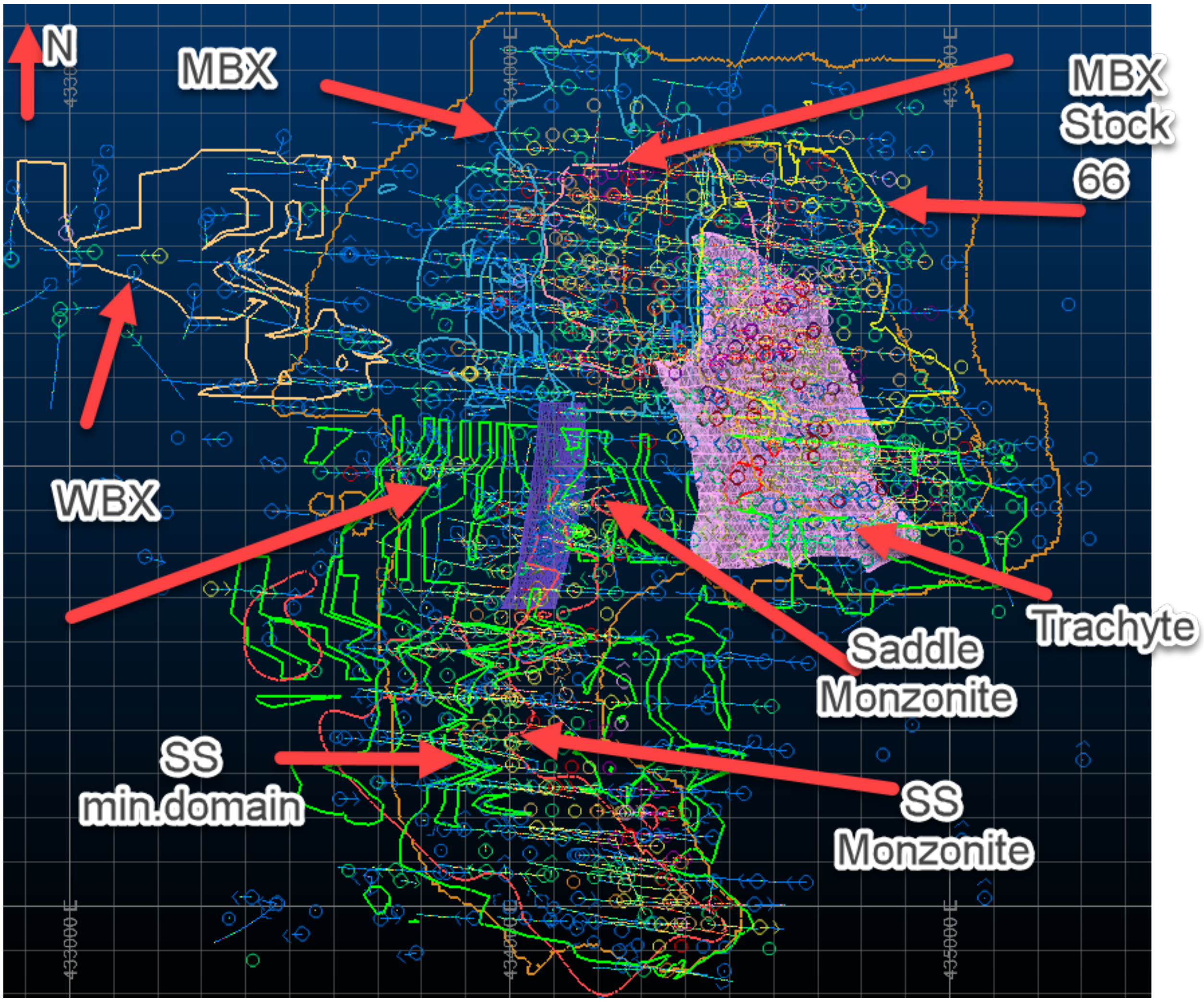
Major mineralized domains were defined and interpreted with hard boundaries for grade estimation purposes, broken primarily by the distribution of the copper and gold using copper equivalency as the main guideline (Table 14-2).

Table 14-2: Mineralized Domain Coding

Domain Code	Name	Description
1000	Southern Star (SS)	Southern most situated copper-gold zone; Intrusive controlled characterized with high grade gold breccia bodies
2000	Saddle	Central part, south from MBX, north from Southern Star Intrusive and Domain
3000	66 zone	Gold Hosted zone, east of the MBX, adjacent to the Rainbow fault
4000	MBX Domain	Central deposit, good Cu-Au correlation; distributed within the MBX stock and volcanogenic unit to the west
5000	WBX Domain	West from MBX, relatively narrow, flat lying zone

Figure 14-8 is the level plan at 1040 EL with a +/-50m window showing mineralized domains interpretation relative to the main lithological units.

Figure 14-8: 1040 Elevation Bench – Mineralized Domains within the Main Lithological Units



14.3. EXPLORATORY DATA ANALYSIS (EDA)

Exploratory data analysis was completed on raw assay and composite samples within the interpreted mineralized zones as discussed below.

Assay

There are 133,869 assay intervals in the database used for the current mineral resource estimate. A total of 1,227 intervals with no assay values, in the historical drill holes, were replaced with default values of 0.01g/t Au. This was done to diminish the possible bias in gold values of the historical holes.

Table 14-3 displays the range of gold and copper values from the utilized assay data.

Table 14-3: Range of Gold and Copper Assay Data for Resource Estimation

	Length mean (m)	Records	Au g/t and Cu %		
			Minimum	Maximum	Mean
Resource Estimate - Au	1.88	133,857	0.001	236.8	0.27
Resource Estimate - Cu	1.88	133,869	0.0001	7.2	0.12

The assay cut-off date used in the construction of this mineral resource model is October 31, 2019 (Table 14-4). The majority of the surface drill holes are inclined between -45 and -70 degrees to the east or west on E-W cross-sections at 40-60 metres spacing and 40-60 metres vertical spacing on the same section.

Table 14-4: Detailed Database Summary Table

Period	DH Series	Location	No. of DH	Length (m)	No. of Assays	Assay Length (m)	Assayed (%)
Pre - 2010	87-1 to 04-920	MBX, Saddle, SS, 66	913	214,133	90,533	177,441	83
2010	10-1003 to 10-1012		10	5,767	3,156	5,714	99
2011	11-1013 to 11-1019	MBX, Saddle, DWBX,	17	5,636	2,857	5,361	95
2017	17-1029 to 17-1051	SS; DWBX, WBX, 66, Saddle	21	7,692	5,232	7,486	97
2018	18-1052 to 18-1127		81	25,751	13,923	24,390	95
2019	19-1128 to 19-1223	Saddle, SS, MBX	92	36,982	18,168	31,517	85
Totals			**1,134**	**295,961**	**133,869**	**251,911**	**85**

Capping

Most of the copper and gold assay values in more than 1,134 drill holes are low grade, but there were some results with extremely high copper and gold values that needed to be capped for block model grade interpolation.

A standard approach to capping high-grade outlier values consists of developing probability plots and examining the distribution to search for any natural gap in those distributions.

Cumulative probability plots are shown in Figure 14-9 and Figure 14-10 for gold and copper respectively. In almost all cases, the log scale cumulative frequency plots for high grades do show obvious abrupt change of slope, which correspond to a natural gap in the grade population data. Therefore, it is the opinion of the QP that the selected capping limits are relatively objective.

Table 14-5 displays the capping limits and global statistics of gold and copper values. The overall copper and gold removed by this approach was below 1%, which remained at the same level as the December 2016 mineral resource estimation. The "Shell" domain relates to mineralized blocks that were not defined within other domains due to modelling complexity and/or grade distribution.

Table 14-5: Proposed Capping Limits Raw Data

Domain	Nb. GT	Avg. Raw Au g/t	Capping Grade g/t Au	% Capped	Avg. Capped Au g/t	% Au Removed
MBX	20,987	0.28	8.5	1.08%	0.26	0.90%
WBX	1,839	0.24	4	1.17%	0.21	0.90%
66	17,175	0.58	8.5	1.08%	0.54	0.90%
SS-Saddle	38,923	0.31	10	1.03%	0.3	1.00%
Shell	120,397	0.28	3.6	1.12%	0.25	0.90%
Domain	**Nb. GT**	**Avg. Raw Cu %**	**Capping Grade % Cu**	**% Capped**	**Avg. Capped Cu %**	**% Cu Removed**
MBX	20,987	0.18	1.5	1.01%	0.18	1.10%
WBX	1,839	0.13	1	1.01%	0.12	0.80%
66	17,175	0.23	2.5	1.01%	0.23	1.30%
SS-Saddle	38,923	0.15	1.6	1.01%	0.15	0.70%
Shell	120,397	0.13	2	1.00%	0.13	0

Figure 14-9: Cumulative Probability Plots for Gold

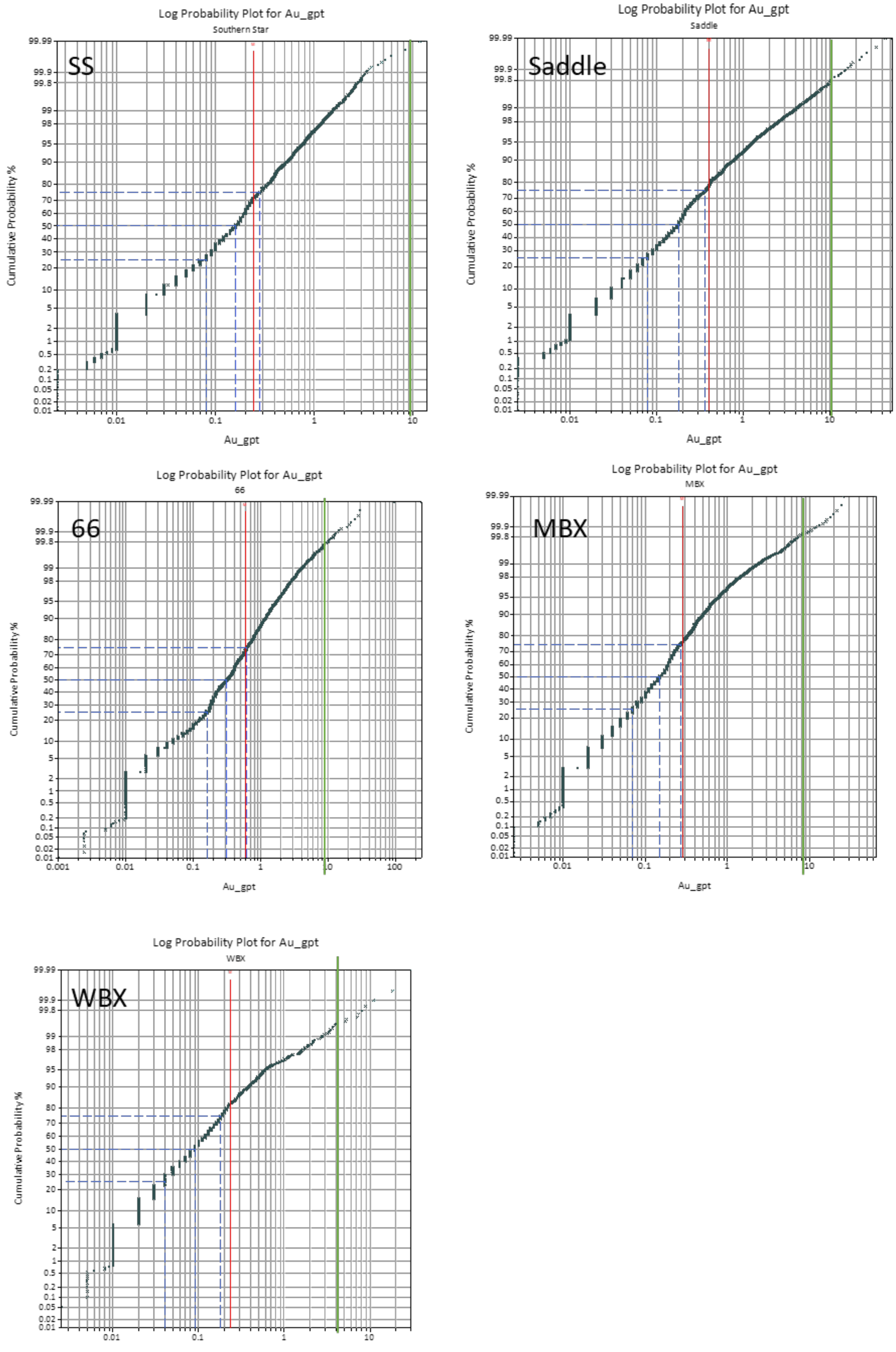


Figure 14-10: Cumulative Probability Plots for Copper

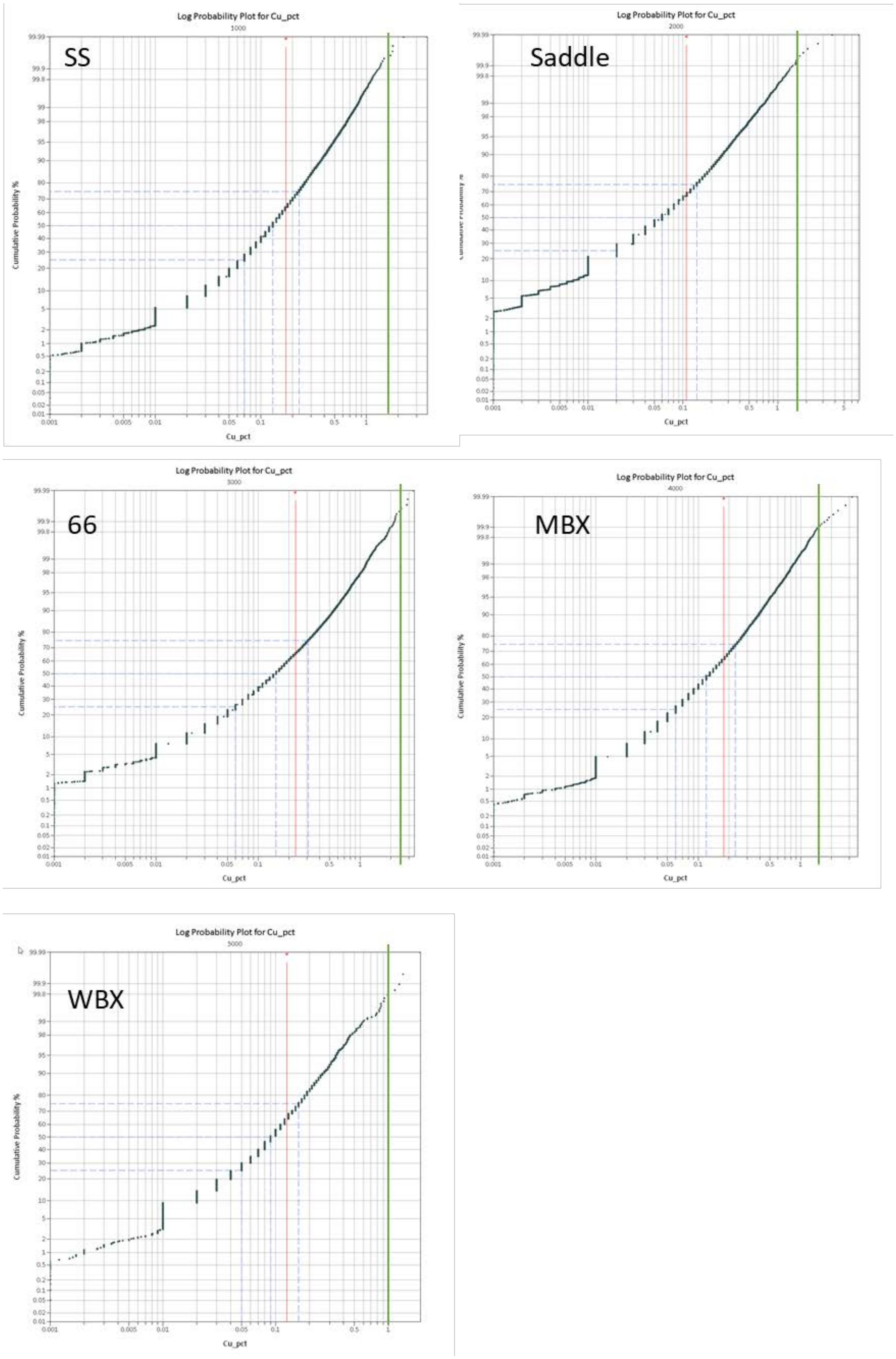


Table 14-6 shows the summary statistics for capped raw gold and copper assays with a comparison of the coefficient of variance (CV) for capped and un-capped raw assay values.

Table 14-6: Summary Statistics for Raw Copper and Copper Composites, by Domain

	SS_Sad Au	SS_Sad Cu	SS_Sad CuEQ	66 Au	66 Cu	66 CuEQ	WBX Au	WBX Cu	WBX CuEQ	MBX Au	MBX Cu	MBX CuEQ
Total Records	38,923	38,923	38,923	17,175	17,175	17,175	1,839	1,839	1,839	20,987	20,987	324
Total Samples	38,923	38,923	38,923	17,152	17,174	17,175	1,839	1,839	1,839	20,987	20,987	324
No. of Missing Values	0	0	0	23	1	0	0	0	0	0	0	0
No. of Values > Trace	38,923	38,923	38,923	17,152	17,174	17,175	1,839	1,839	1,839	20,987	20,987	324
Minimum	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0.02**
Maximum	**10.0**	**1.6**	**3.5**	**8.5**	**2.5**	**117.9**	**30.0**	**1.9**	**15.1**	**63.9**	**3.6**	**15.0**
Mean	**0.3**	**0.2**	**0.3**	**0.5**	**0.2**	**0.5**	**0.2**	**0.1**	**0.3**	**0.3**	**0.2**	**2.6**
Variance	0.36	0.03	0.12	0.65	0.07	2.00	1.00	0.02	0.29	0.81	0.03	6.14
Standard Deviation	0.60	0.16	0.34	0.81	0.26	1.42	1.00	0.13	0.54	0.90	0.19	2.48
Standard Error	0.00	0.00	0.02	0.01	0.00	0.01	0.02	0.00	0.01	0.01	0.00	0.14
Coefficient of Variation-Capped	**2.01**	**1.09**		**1.49**	**1.13**		**1.56**	**0.89**		**1.56**	**0.89**	
Coefficient of Variation-UnCapped	**2.68**	**1.14**		**4.66**	**1.14**		**4.16**	**1.06**		**3.21**	**1.04**	
Skewness	9.06	2.70	4.25	5.16	2.61	64.93	19.45	3.97	16.12	28.56	3.53	2.23
Kurtosis	115.30	11.92	28.06	37.98	10.72	5263.00	494.65	30.27	374.77	1432.00	29.64	6.23
5th Percentile	0.02	0.00	0.03	0.02	0.01	0.03	0.22	0.01	0.02	0.02	0.01	0.30
10th Percentile	0.03	0.01	0.05	0.05	0.02	0.08	0.43	0.02	0.04	0.03	0.03	0.45
25th Percentile	0.08	0.04	0.11	0.16	0.06	0.16	0.94	0.04	0.08	0.07	0.06	1.02
50th Percentile	0.17	0.10	0.20	0.31	0.15	0.33	1.75	0.09	0.15	0.15	0.12	1.91
75th Percentile	0.31	0.20	0.36	0.62	0.31	0.64	3.22	0.16	0.26	0.27	0.24	3.44
90th Percentile	0.60	0.33	0.60	1.17	0.55	1.10	5.14	0.26	0.45	0.50	0.39	5.05
95th Percentile	0.94	0.45	0.85	1.73	0.75	1.49	7.46	0.35	0.65	0.79	0.51	8.31

Composites

For the current resource estimate, the analysis of the spatial continuity of gold and copper grade in each mineralized domain was assessed based on the capped grade of the 4 metre down-hole composites. This composite length was selected as a multiple of the 2 metres sample length, which was the most common, and provided a sufficient number of composites within the 15x15x15m block size.

Table 14-7 and Table 14-8 list statistics of the computed grade of the raw/uncapped data and 4 metre capped composites. A composite was kept if its computed grade was derived from assay data over a minimum of 0.5 metres in length. Compositing started at the overburden-bedrock contact.

Note: the statistical analysis of the raw/uncapped data and composited capped 4m data presented in Table 14-7 below is only shown within modelled mineralized domains.

Table 14-7: Comparison of Raw, Capped Raw and Capped Composite of Gold Assays within Modelled Wireframes

		Count	Minimum	Maximum	Mean	Standard deviation	CV	Variance	Skewness
SS	Raw	22,535	0.00	13.58	0.24	0.36	1.49	0.13	11.32
	Capped Raw	22,535	0.00	3.00	0.24	0.30	1.27	0.09	4.03
	Capped Composite	10,768	0.00	3.00	0.24	0.26	1.08	0.07	3.22
Saddle	Raw	15,950	0.00	51.50	0.40	1.20	3.01	1.44	18.42
	Capped Raw	15,950	0.00	10.00	0.38	0.84	2.19	0.70	7.09
	Capped Composite	7,298	0.00	10.00	0.38	0.70	1.84	0.49	6.26
66	Raw	17,208	0.00	236.85	0.59	2.72	4.66	7.41	73.12
	Capped Raw	17,208	0.00	8.50	0.54	0.81	1.49	0.65	5.16
	Capped Composite	8003	0.00	8.50	0.52	0.66	1.29	0.44	4.50
MBX	Raw	21,632	0.00	63.90	0.28	0.90	3.20	0.82	27.91
	Capped Raw	21,632	0.00	8.50	0.27	0.57	2.12	0.33	9.07
	Capped Composite	9,959	0.00	8.50	0.26	0.42	1.58	0.17	7.09
WBX	Raw	1,933	0.00	30.00	0.24	0.98	4.17	0.97	19.60
	Capped Raw	1,933	0.00	4.00	0.20	0.45	2.21	0.20	5.97
	Capped Composite	883	0.01	3.83	0.20	0.36	1.76	0.13	4.93

Note: *Excludes composites outside the mineralized domains which has been listed as resource model box.

Table 14-8: Comparison of Raw, Capped Raw and Capped Composite of Copper Assays within Modelled Wireframes

		Count	Minimum	Maximum	Mean	Standard deviation	CV	Variance	Skewness
SS	Raw	22,535	0.00	3.75	0.17	0.17	0.95	0.03	2.94
	Capped Raw	22,535	0.00	1.60	0.17	0.16	0.94	0.03	2.39
	Capped Composite	10,768	0.00	1.28	0.17	0.14	0.84	0.02	1.97
Saddle	Raw	15,950	0.00	7.20	0.11	0.17	1.51	0.03	9.81
	Capped Raw	15,950	0.00	1.60	0.11	0.15	1.37	0.02	3.64
	Capped Composite	7,298	0.00	1.59	0.10	0.13	1.23	0.02	2.96
66	Raw	17,230	0.00	3.37	0.23	0.26	1.14	0.07	2.81
	Capped Raw	17,230	0.00	2.50	0.23	0.26	1.13	0.07	2.62
	Capped Composite	7,912	0.00	2.36	0.22	0.23	1.02	0.05	2.17
MBX	Raw	21,631	0.00	3.58	0.18	0.18	1.04	0.03	3.54
	Capped Raw	21,631	0.00	1.50	0.18	0.18	1.00	0.03	2.48
	Capped Composite	9,958	0.00	1.50	0.17	0.16	0.90	0.03	2.13
WBX	Raw	1,933	0.00	1.91	0.13	0.13	1.05	0.02	3.95
	Capped Raw	1,933	0.00	1.00	0.13	0.12	1.00	0.02	2.80
	Capped Composite	883	0.00	0.70	0.12	0.10	0.83	0.01	1.88

Rock Density

Dataset

Specific gravity (SG) data were systematically collected across the Project, with a total of 13,802 determinations in the raw database. These measurements were routinely collected, with one measurement approximately every 10 m of core length. The measurements relied upon the Archimedes Principle, which determines SG through a water displacement methodology. SG was calculated as follows:

- The sample was weighed to determine the dry mass in air (MA).
- The sample was suspended in water where the mass in water was measured (MW).
- The bulk density was then calculated as: SG = MA / (MA – MW).
- The determinations were recorded on a template that captured hole name, depth, MA and MW. This information was later transferred to spreadsheets, with SG calculated from primary measurements. This method is considered appropriate for competent non-porous rocks, which are typical of the Project.
- Validation of the SG data compares favourably with previous and historical SG determinations.

A small subset of core, covering a range of representative lithologies, was selected from the 2007 HQ diameter Terrane drill program. These samples were sent to ALS Chemex for external verification of SG, using method OA-GRA-08 which is effectively a water-displacement method similar to that employed at the Project. The results are summarized in Table 14-9.

Table 14-9: Specific Gravity Summary – ALS Chemex Data

Lithology	SG
Monzonite	2.70
Hybrid Volcanics	2.77
Volcanics	2.86

A separate study completed by Melis Engineering Ltd. (Melis) in August 1990 provided five SG measurements on ore composites derived from NQ drilling. The average SG of this data was 2.81, with individual results as displayed in Table 14-10.

Table 14-10: Melis SG Measurements

Composite Zone	SG
MBX	2.81
66	2.87
WBX	2.81
SS	2.85
Oxide MBX	2.73

Specific Gravity of Overburden

An internal study was completed to determine the specific gravity of overburden materials. Results were determined from proctor compaction tests and truck weigh scales. Averages for in-situ density values concluded glacial till at 2.50 tonnes per cubic metre (t/m3) and common glacial fluvial overburden material at 2.37t/m3. Values were averaged and overburden was assigned a value of 2.40t/m3 in the model. This remains unchanged in the 2017 model.

Specific Gravity Data Validation

An internal validation of SG data identified that 1,504 specific gravity results were missing from the data set. The missing data was identified and added back into the database; in addition, geologic logs and SG data forms were examined for all SG determinations greater than 3.0 or less than 2.0 in order to establish validity of measurements. A total of nine corrections were made to erroneous data, and in addition, data was removed from the database in cases where validity could not be established.

Estimation of Specific Gravity

In order to preserve density trends, blocks were assigned SG values through estimation by inverse distance squared (ID2), with a soft boundary between lithologies. The search ellipse was set to 200 m x 200 m x 100 m with a minimum of one and maximum of 20 determinations, and a maximum of eight per octant required to make an estimate.

Coordinate System and Topography

The geologic modelling and resource estimates were completed in UTM coordinates (Zone 10, NAD83).

In the fall of 2008, McElhanney completed a LiDAR and photogrammetric survey over the Project area and the powerline transmission route. From this survey, 1m contour intervals were generated in UTM NAD83.

In order to accommodate the slight shift in the surveyed elevation, all drill hole collar elevations of holes drilled prior to 2017 have been registered to the LiDAR surface. In October 2016, a new LiDAR survey of the property was completed by Eagle Mapping, however, no adjustments have been made to the collar locations or the model surface.

Drill hole collar elevations in the resource area drilled from 2017-2019 were surveyed using a Trimble DGPS equipment.

The Mineral Resource estimation was constrained and reported using the open pit surface as of November 30, 2019 for the updated resource estimation.

14.4. MINERAL RESOURCE ESTIMATION

This Mineral Resource estimate was completed by the Centerra Project and Technical Services (P&TS) geology team. Updates in this model include:

- Mineralized domains were interpreted using a 0.1% CuEq cut-off grade on 40-60m section-by-section interpretation.
- The economical domains were based on litho-structural interpretation and has been defined by several monzonite bodies relative to the volcanic rocks as well as the major structural features;
- Correlation between copper and gold, which are critical to interpretation of the mineralization and grade distribution, were considered.
- Two high-grade gold zones were statistically identified within the 66 zone and the saddle zone. They were interpreted and modelled using hard boundaries; having positive impact on the interpretation of the gold distribution and overall validation of the model.
- Statistical analysis resulted in the following parameters being updated:

- Composite sample lengths were reduced from 10m to 4m
- The search radius and the number of composites/drill holes used for grade estimation, being more restricted and closer to the composites as discussed in section 14.6.

Block Model Limits

The updated block model for the Project was completed using the Datamine Studio RM version 1.5.62.0 modeling software with the limits shown below in Table 14-11.

Table 14-11: Block Model Parameters

Parameter	X	Y	Z
Minimum (UTM83)	433080	6107500	620
Maximum (UTM83)	435690	6110245	1445
Number of Blocks	174	183	55
Block Size (m)	15	15	15

The block size selected was 15 m x 15 m x 15 m as this reflected an appropriate Selective Mining Unit (SMU) relative to the bench height for the Project. Total number of blocks in the model is 1,751,310.

14.5.SPATIAL ANALYSIS

The spatial continuity of the composite grades in each mineralized domain is assessed through pairwise relative variograms in Datamine and normal score variograms in Snowden Supervisor; that is, the calculated correlation coefficient of grades from pairs of composites separated by a given distance in a given direction. Variograms for the resource estimate were defined in Datamine and Snowden Supervisor software based on capped composite samples. Nugget values were derived from downhole variograms, and 3-D variography was determined from variogram maps and sweeps in the major and semi-major directions, based on geological understanding of mineralization controls. All variograms were modelled as double-structured spherical models, from experimental variograms.

Experimental variograms are on file at Mount Milligan Mine. Directions investigated were the N-S horizontal average strike, the vertical average dip and the horizontal E-W across average strike and dip. In each case, in addition to the principal directions, an average variogram along drill holes was calculated to better assess the magnitude of any nugget effect.

In most domains, the spatial continuity of the grade of composites is characterized by: (1) low relative nugget effects from 0.5% to 10%, and (2) generally poorly defined anisotropies with best continuity (lowest curve) along the average N-S horizontal strike or the dip to east or west.

Both copper and gold mineralization exhibit strong local trends, which are a function of the mineralizing system. In the SS deposit, mineralization is strongest along the footwall contact of the Monzonite-Volcanic contact, with decreasing intensity away from this contact. Continuity is greatest in the plane of the contact.

Copper mineralization in the Main deposit is most pronounced along the MBX intrusive and Volcanic boundary, forming a concentric pattern around the MBX stock. Copper grade diminishes away from the stock towards the 66 Zone. Gold mineralization also forms a concentric pattern around the MBX stock, but is more dispersed, exhibiting continuity in the north-south direction, parallel to the stratigraphic strike of the footwall rocks.

In most domains, the spatial continuity of the both copper and gold grade of composites were characterized by:

- Relatively low nugget effect.
- Generally poorly defined anisotropy but rather being isotropic distribution.

14.6. RESOURCE BLOCK MODELING – GOLD AND COPPER GRADE INTERPOLATION

Drill hole spacing across the deposit is generally on the order of 40-80 metres in the north-south and east-west directions. A block size of 15 x 15 x 15 metres, with no rotation, was selected to accommodate the drill hole spacing, width of the mineralization, and the mining bench height (15m).

Table 14-12 shows the details of the assignment of blocks to the mineralized domains with the corresponding volume and tonnage after excluding the mined block portions/volumes. The “Shell” domain relates to mineralized blocks that were not defined within other domains due to modelling complexity and/or grade distribution.

Table 14-12: Summary of Wireframe Volumes of the Mount Milligan Deposit

Domain	Volume below overburden ('000 m^3)	Volume Below overburden + pit ('000 m^3)	Volume in Pit, including overburden ('000 m^3)
SS	138,149	98,813	43,160
Saddle	103,052	77,450	27,084
WBX	28,256	28,256	0
MBX	177,167	117,740	60,740
66	62,870	47,642	15,400
All above	509,493	369,900	146,384
Shell			77,716

In order to preserve trends, multiple estimation passes (Ordinary Kriging and Inverse Distance Squared) were used in the estimation of copper and gold. The first pass established the search at approximately 1/3 variogram range, the second pass at 1/2 variogram range, the third pass at the 2/3 variogram range and the final pass at full variogram range.

The search parameters for copper and gold are displayed in Table 14-13.

Table 14-13: Search Parameters used in Grade Estimation

Domain	Estimation Pass	Range 1 (m)	Range 2 (m)	Range 3 (m)	Datamine Rotation			Datamine Rotation Axis			Min. Comp.	Max. Comp.	Max. Comp/Hole
Southern Star	1	45	30	35	90	140	160	3	1	2	10	18	4
	2	68	45	52							8	22	4
	3	90	60	70							6	24	4
	4	135	90	105							3	28	3
Saddle	1	45	30	35	60	120	-35	3	1	2	10	18	4
	2	68	45	52							8	22	4
	3	90	60	70							6	24	4
	4	135	90	105							3	28	3
66	1	45	30	35	80	10	170	3	1	3	10	18	4
	2	68	45	52							8	22	4
	3	90	60	70							6	24	4
	4	135	90	105							3	28	3
MBX East	1	45	30	35	40	-65	135	3	1	2	10	18	4
	2	68	45	52							8	22	4
	3	90	60	70							6	24	4
	4	135	90	105							3	28	3
MBX West	1	45	30	35	-60	-50	45	3	1	2	10	18	4
	2	68	45	52							8	22	4
	3	90	60	70							6	24	4
	4	135	90	105							3	28	3
WBX	1	45	30	35	-60	-50	45	3	1	2	10	18	4
	2	68	45	52							8	22	4
	3	90	60	70							6	24	4
	4	135	90	105							3	28	3
Outside the Wireframes (Shell)	1	45	30	35	90	140	160	3	1	2	10	18	4
	2	68	45	52							8	22	4
	3	90	60	70							6	24	4

Search directions were established as per variogram orientation. The estimation strategy is summarized in Table 14-14. For all blocks, a minimum of three or two drill holes were required per each pass to make an estimate, with generally a minimum of 8-10 and a maximum of 18-22 composites.

Table 14-14: Grade Estimation Parameters

Interpolation Parameters	**1st Pass**	**2nd Pass**	**3rd Pass**
Primary Interpolation Method	Inverse Distance	Inverse Distance	Inverse Distance
Search Volume (multiple of 1/3 semi-variogram)	1	1.5	2
Minimum number of Composites	10	8	6
Maximum number of Composites	18	22	24
Maximum number of Composite per hole	4	4	4
Discretisation	3X3X3		3X3X3

14.7.COPPER EQUIVALENT CALCULATIONS

Copper equivalent (CuEq) grade was calculated for individual blocks, using estimated in situ copper and gold grades and the following parameters:

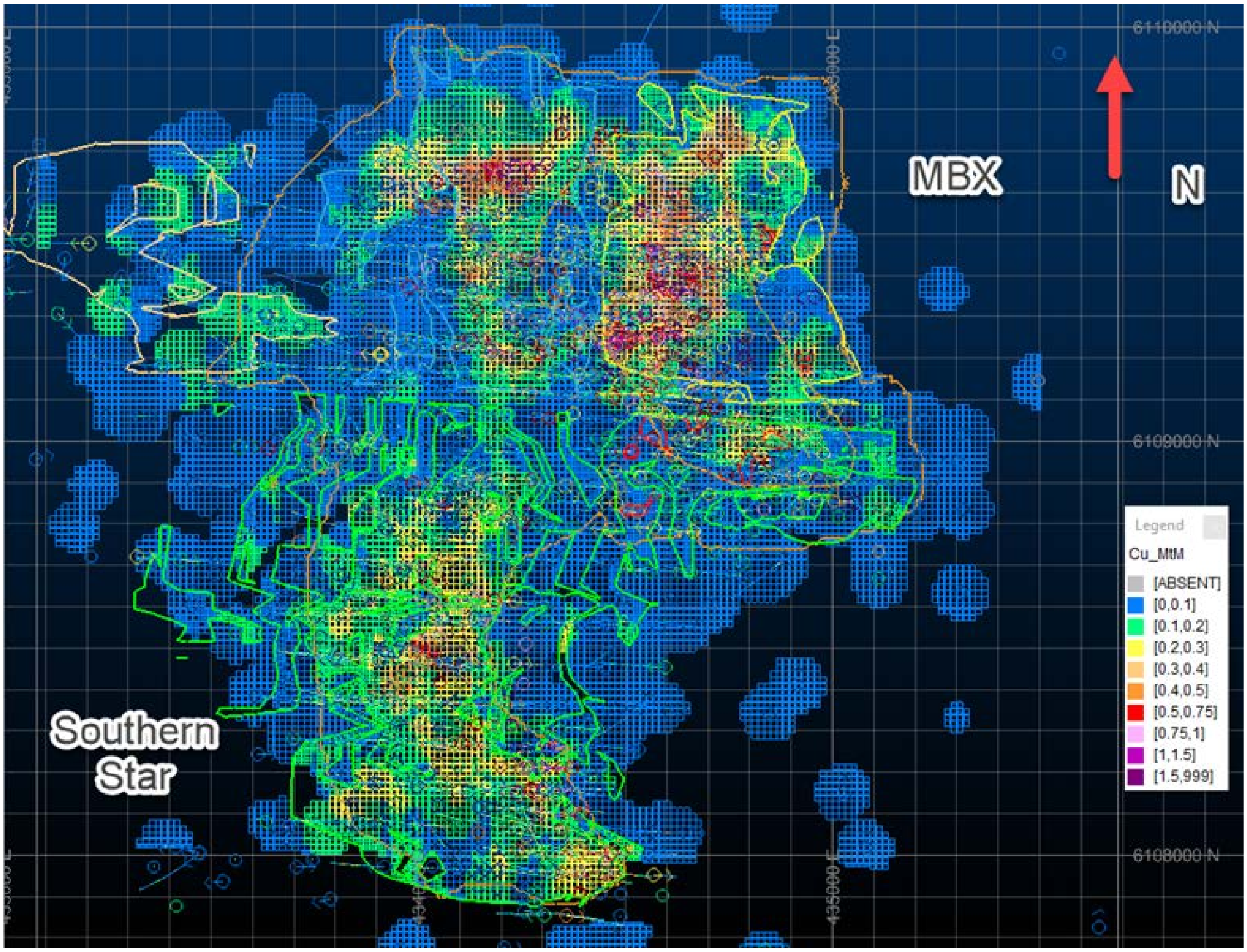
- Gold Price (AuPrice): $1,500/oz
- Copper Price (CuPrice): $3.50/lb
- Copper Recovery (CURE): 79.65%
- Gold Recovery (AURE): 65%

Formula used for CuEq calculation:

$$CuEq = CU_{\%} + \left(\frac{AURE * AuPrice * 14.5833}{CURE * CuPrice)}\right) * \frac{Au_{gpt}}{10000}$$

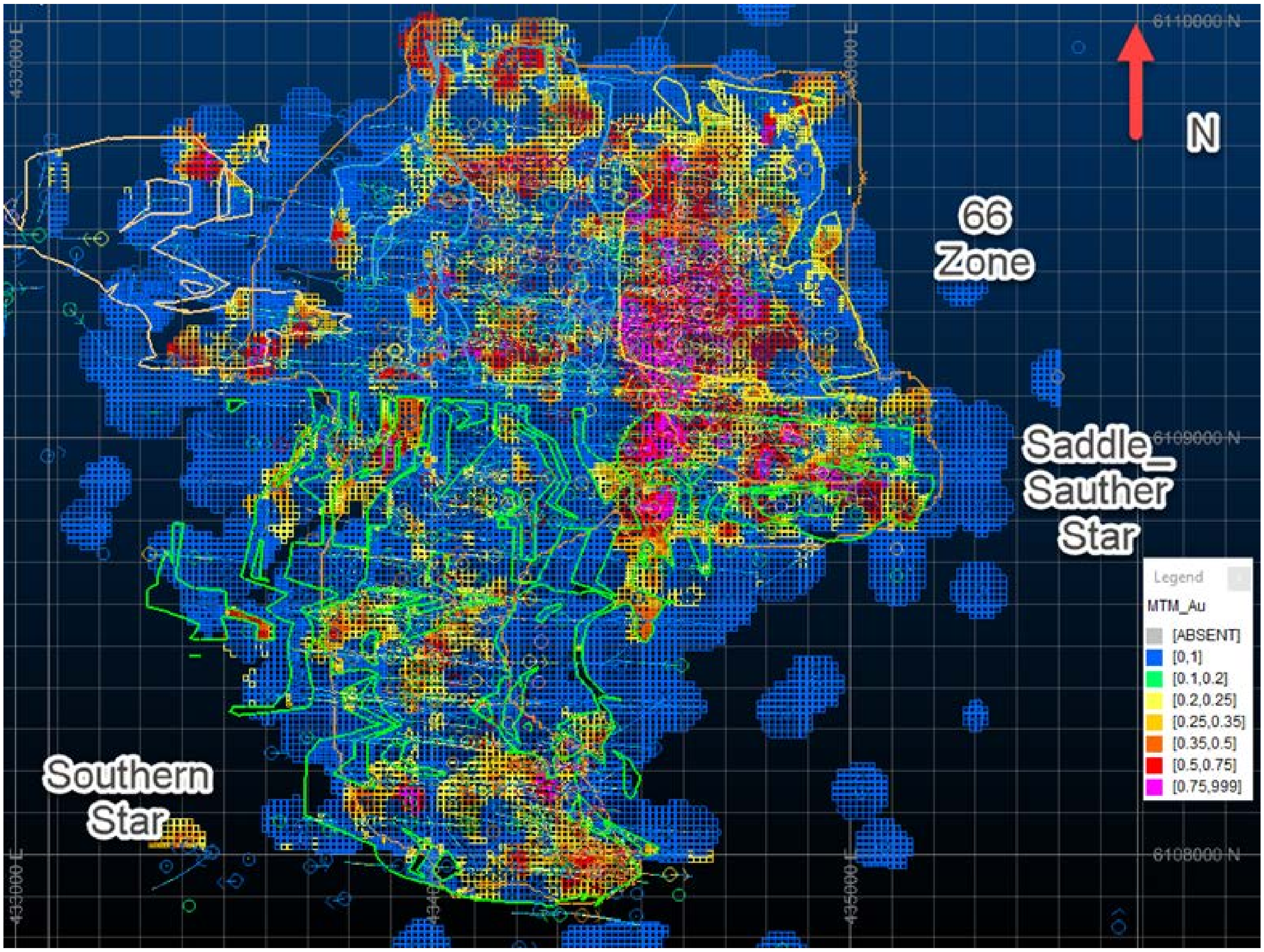
A typical bench plan and cross section of the copper and gold block estimate distribution is shown in Figure 14-11 and Figure 14-12.

Figure 14-11: 1010 M EL Bench – Copper Block Estimate Distribution



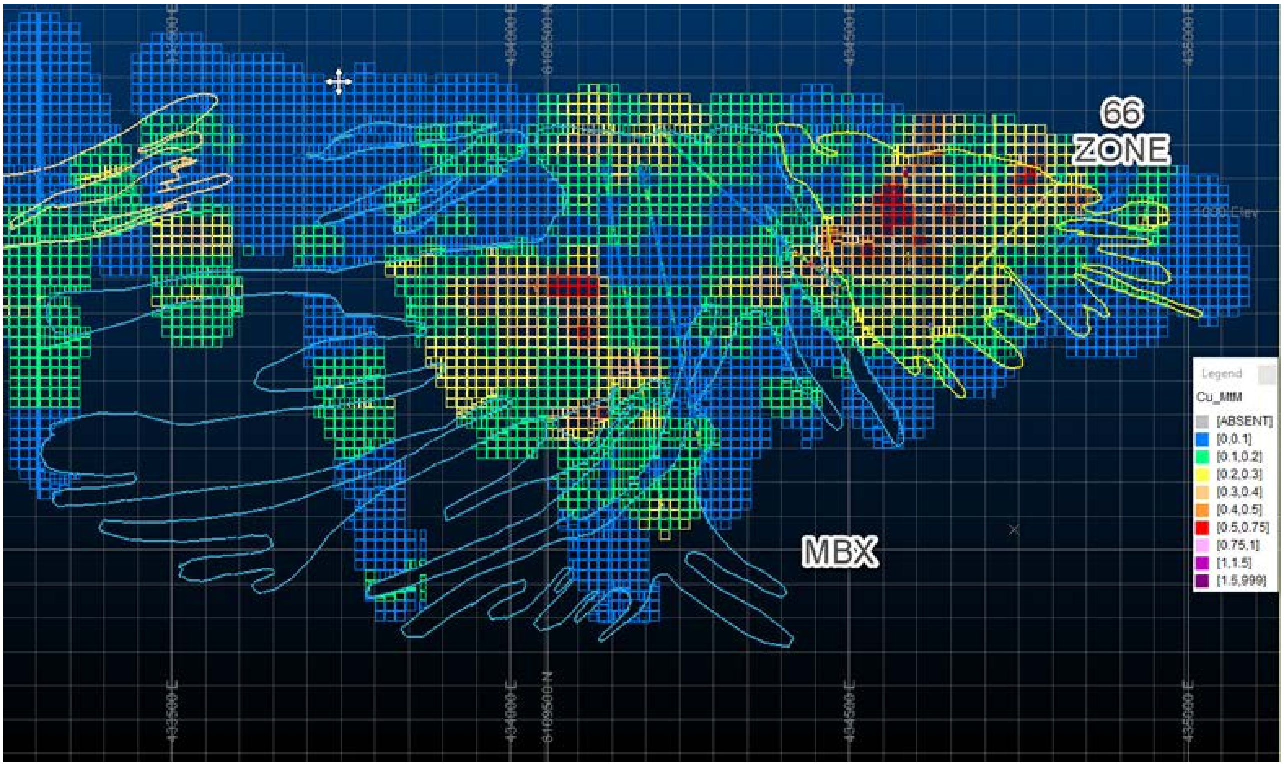
Source: Centerra, 2019. Gridlines are 100m.

Figure 14-12: 1010 M EL Bench – Gold Block Estimate Distribution



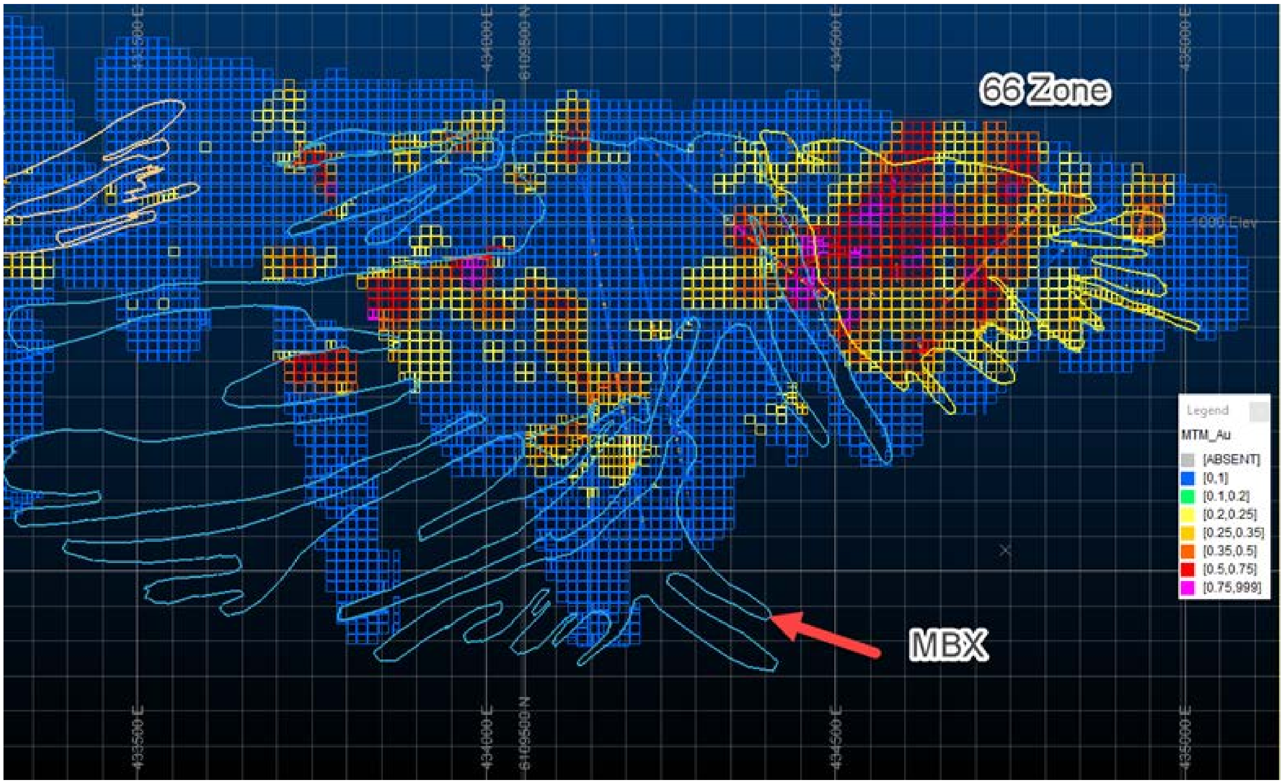
Source: Centerra, 2019. Gridlines are 100m.

Figure 14-13: Section 6109500N – Copper Block Estimate Distribution – Looking North



Source: Centerra, 2019. Gridlines are 100m

Figure 14-14: Section 6109500N – Gold Block Estimate Distribution - Looking North



Source: Centerra, 2019. Gridlines are 100m.

14.8. MODEL VALIDATION

A visual comparison of the block model grades and composite data was conducted in multiple cross-sections and plan view. The estimated block grades were found to be consistent with the composite grades, with no major discrepancies observed. An example of a cross section is shown in Figure 14-15 and Figure 14-16.

Figure 14-15: Cross Section at 6,108,480N Showing Mineralized Blocks

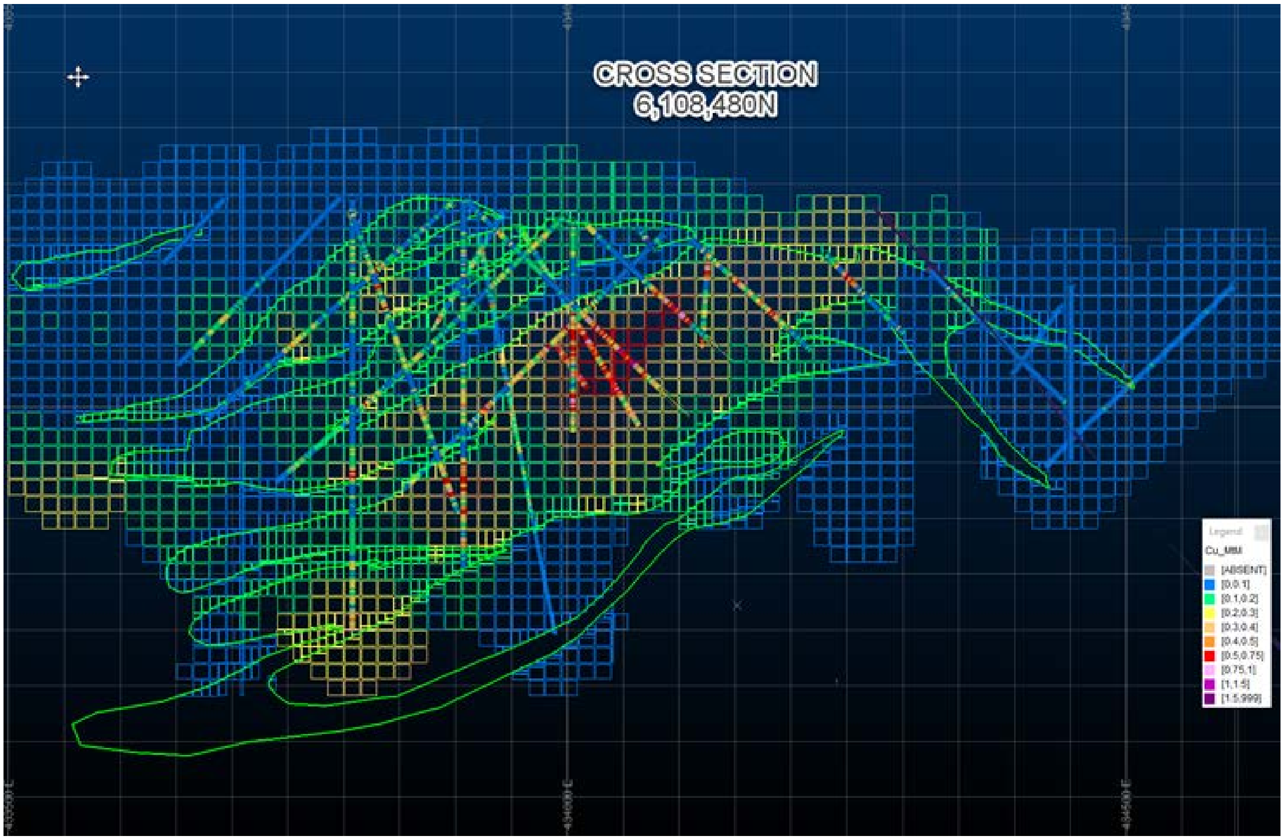
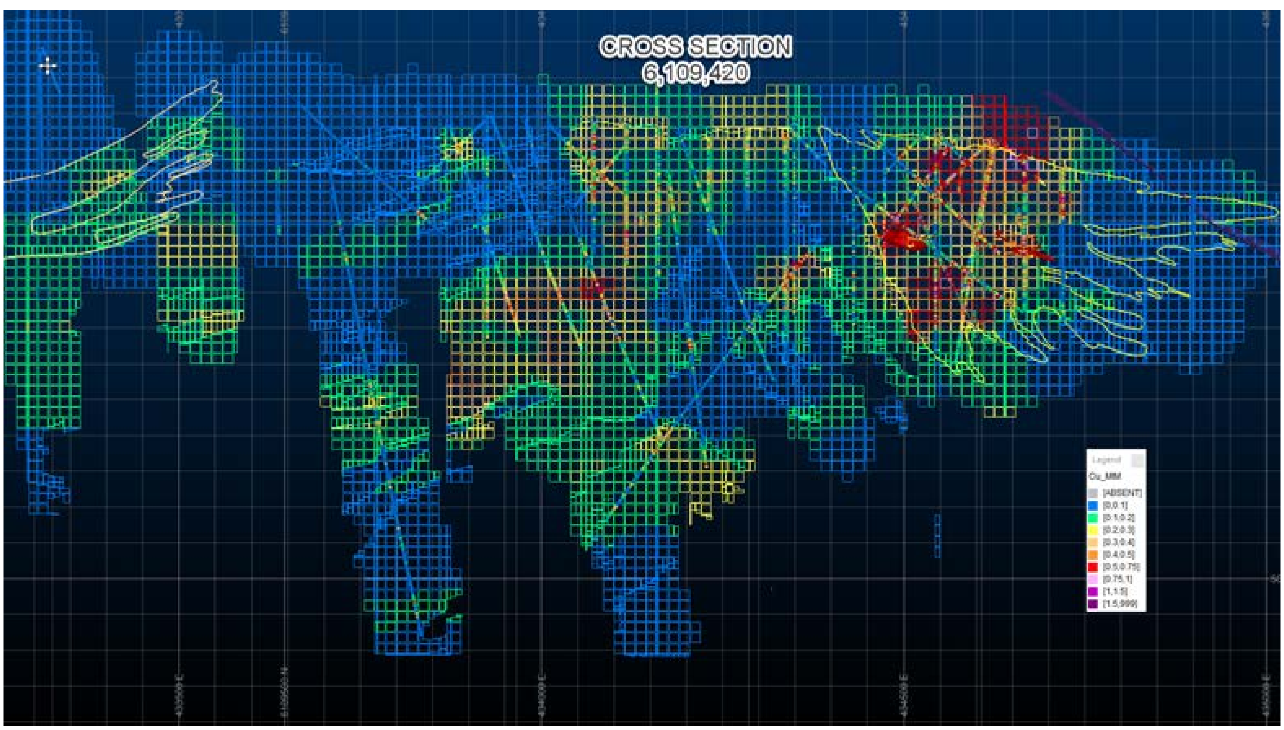


Figure 14-16: Cross Section at 6,109,420N Showing Mineralized Blocks



In addition, global statistical validation was conducted among the two modelling methodologies used during the estimation process.

Based on improved statistical parameters and SWATH plots correlation, ID2 was the final modelling methodology selected for reporting the Mineral Resource estimate.

This deposit is characterized by relatively high variability in the gold grade distribution which is reflected in the grade estimation and block model validation. It is possible that the gold distribution may have been slightly underestimated (see Table 14-15) which requires further study and modified interpretation for the next update.

Table 14-15 presents comparison of the means and variances of block estimates with that of informing capped composites.

Table 14-15: Global Statistical Validation of OK vs ID2 vs NN

Gold	Southern Star				Saddle				66				MBX				WBX			
	Decl Samp	OK Est	ID Est	NN Est	Decl Samp	OK Est	ID Est	NN Est	Decl Samp	OK Est	ID Est	NN Est	Decl Samp	OK Est	ID Est	NN Est	Decl Samp	OK Est	ID Est	NN Est
Mean (g/t)	0.24	0.20	0.21	0.20	0.37	0.29	0.30	0.28	0.49	0.41	0.42	0.40	0.26	0.24	0.25	0.24	0.20	0.19	0.19	0.18
Std Dev	0.26	0.14	0.14	0.24	0.69	0.32	0.33	0.54	0.65	0.36	0.35	0.57	0.42	0.17	0.18	0.39	0.36	0.15	0.16	0.33
Variance	0.07	0.02	0.02	0.06	0.48	0.10	0.11	0.29	0.42	0.13	0.12	0.33	0.17	0.03	0.03	0.15	0.13	0.02	0.03	0.11
CV	1.08	0.69	0.70	1.23	1.87	1.11	1.10	1.96	1.32	0.88	0.83	1.42	1.58	0.72	0.73	1.63	1.76	0.80	0.83	1.85
Maximum (g/t)	3.00	2.44	2.57	3.00	10.00	5.25	5.34	9.61	8.50	4.93	4.31	8.50	8.50	3.42	3.95	8.50	3.83	1.54	1.39	3.83
Minimum (g/t)	0.00	0.01	0.01	0.00	0.00	0.01	0.01	0.00	0.00	0.01	0.02	0.00	0.00	0.01	0.01	0.00	0.01	0.02	0.01	0.01

Copper	Southern Star				Saddle				66				MBX				WBX			
	Decl Samp	OK Est	ID Est	NN Est	Decl Samp	OK Est	ID Est	NN Est	Decl Samp	OK Est	ID Est	NN Est	Decl Samp	OK Est	ID Est	NN Est	Decl Samp	OK Est	ID Est	NN Est
Mean (%)	0.17	0.16	0.16	0.15	0.10	0.10	0.10	0.10	0.22	0.19	0.19	0.19	0.17	0.16	0.17	0.16	0.12	0.12	0.12	0.11
Std Dev	0.14	0.08	0.09	0.13	0.13	0.08	0.09	0.12	0.23	0.13	0.14	0.21	0.16	0.10	0.10	0.14	0.10	0.05	0.05	0.09
Variance	0.02	0.01	0.01	0.02	0.02	0.01	0.01	0.01	0.05	0.02	0.02	0.04	0.03	0.01	0.01	0.02	0.01	0.00	0.00	0.01
CV	0.84	0.52	0.54	0.86	1.23	0.81	0.83	1.16	1.02	0.70	0.73	1.12	0.90	0.59	0.61	0.91	0.83	0.40	0.44	0.82
Maximum (%)	1.28	0.67	0.67	1.28	1.59	0.94	0.94	1.35	2.36	0.98	1.02	2.36	1.50	0.97	1.01	1.50	0.70	0.40	0.43	0.70
Minimum (%)	0.00	0.00	0.01	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.01	0.01	0.00

Note: Additional block model validation Swath Plots are on file at Mount Milligan Mine.

Validation between the Ordinary Kriging, ID2 and NN stands as acceptable with reasonable discrepancies. ID2 shows better correlation with the composited values and has been chosen as the final estimation methodology.

14.9. RESOURCE CLASSIFICATION

Block model quantities and grade estimates were classified according to the CIM Definition Standards for Mineral Resources and Mineral Reserves (2019) and adopted by NI 43-101, by Mr. Slobodan Jankovic. Mr. Jankovic is a Qualified Person for the purpose of National Instrument 43-101. Mineral resource classification is typically a subjective concept, and industry best practices suggest that resource classification should consider both the confidence in the geological continuity of the mineralized structures, the quality and quantity of exploration data supporting the estimates, and the geostatistical confidence in the tonnage and grade estimates. Appropriate classification criteria should aim at integrating these concepts to delineate regular areas at similar resource classification.

This classification was based upon a review of data distribution, data density, and the spatial continuity of both gold and copper. Plots of kriging variance, average distance to composites, and the minimum and maximum number of composites used were generated and reviewed. The following criteria were used in the classification of the Project's Mineral Resource:

- Distance as defined by kriging pass, established from variogram models;
- Kriging variance;
- Number of drill holes used to make an estimate; and
- Number of samples used to make an estimate.

The classification of Mineral Resources for the MBX, WBX, 66 Zone, SS and Saddle Zones was based primarily on copper equivalent. All blocks were estimated using composites from at least two drill holes. All blocks were trimmed by a limiting open-pit shell (or cone) before reporting.

Measured Mineral Resource

Blocks classified as Measured Mineral Resource were estimated in the first estimation pass, thus lie within one-third of the variogram range. In addition, three or more holes were used to make an estimate and were estimated by a minimum of eight composites with kriging variance less than or equal to 0.33.

Indicated Mineral Resource

Blocks classified as Indicated Mineral Resource were estimated in the second estimation pass, thus lie within one half of the variogram range. In addition, these blocks have a kriging variance of less than 0.66, used three or more holes to make an estimate and were estimated by a minimum of six composites.

Inferred Mineral Resources

Inferred Mineral Resource blocks were classified from two-thirds of variogram search and were estimated by a minimum of six and maximum of 24 composites and used a single drill hole.

14.10. MINERAL RESOURCE STATEMENT

In order to determine the quantities of material offering “reasonable prospects for economic extraction” (as defined in CIM Definition Standards for Mineral Resources & Mineral Reserves, 2019 by an open pit, Centerra personnel used Datamine NPV Scheduler to evaluate the profitability of each resource block based on its value. Optimization parameters used are presented in Table 14-16. The reader is cautioned that the results from this pit optimization are used solely for the purpose of testing the “reasonable prospects for economic extraction” by an open pit and do not represent an attempt to estimate mineral reserves.

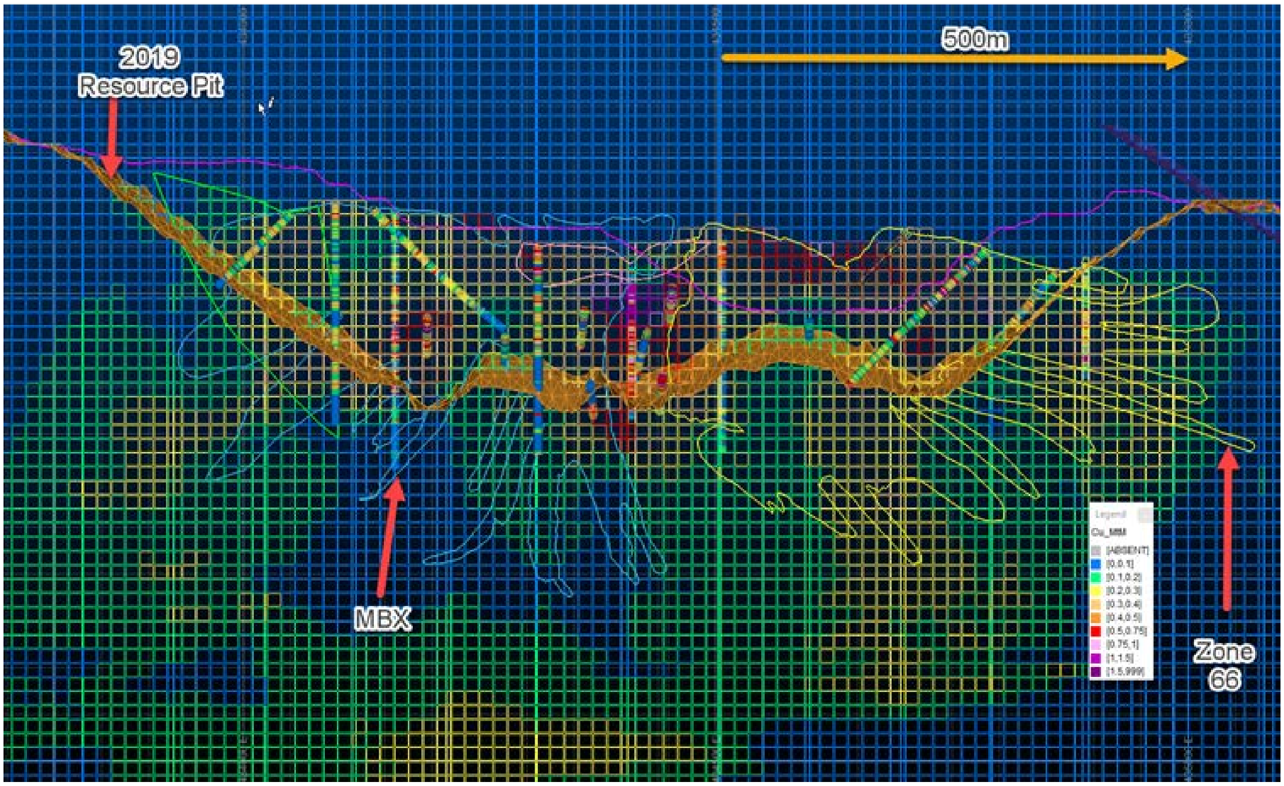
Table 14-16: Pit Optimization Parameters for Mineral Resource Estimation

Parameters	Unit	Value
Gold Price	US$ per ounce	1,500
Copper Price	US$ per pound	3.50
Exchange Rate	C$:US$	1.25:1.00
Mining Cost	US$ per tonne mined	2.23
Processing Cost	US$ per tonne of feed	5.45
G&A Cost	US$ per tonne of feed	1.82
Mining Dilution	percent	5.0
Mining Loss	percent	1.5
Pit Slope	degrees	Variable – see section 16
Process Rate	Tonnes feed per day	60kt average
Copper Recovery	percent	Variable – see section 13
Gold Recovery	percent	Variable – see section 13

Analysis of resource pit optimization results suggests that it is appropriate to report the mineral resources at an NSR cut-off of $7.64/t or C$9.55/tonne. After reviewing the optimization results and block CuEq calculations, it was found that 0.20% CuEq is an appropriate cut-off grade to report mineral resources for the Mount Milligan Mine, please refer to cross section below (Figure 14-17)

That is, the Mineral Resources at 0.20% CuEq correspond well with the C$9.55/tonne NSR value. Centerra's qualified person considers that the blocks above the 0.20% CuEq cut-off grade and located within the conceptual pit envelope show "reasonable prospects for economic extraction" and therefore can be reported as a mineral resource.

Figure 14-17: Cross Section 6,109,740N (MBX and 66 Zone)



The Mineral Resource Statement presented in Table 14-17 was prepared by Mr Slobodan Jankovic P. Geo. (APGO#1388). The Mineral Resources are exclusive of Mineral Reserves. The effective date of the Mineral Resource Statement is December 31, 2019.

Table 14-17: Mineral Resource Statement, Effective Date December 31, 2019 (exclusive of Mineral Reserves)

Category	Cut-off CuEq (%)	Tonnes (kt)	Copper Grade (%)	Gold Grade (g/t)	Contained Copper (Mlb)	Contained Gold (koz)
Measured (M)	0.2	50,582	0.16	0.44	182	713
Indicated (I)	0.2	74,788	0.20	0.29	335	695
Total M+I	**0.2**	**125,370**	**0.19**	**0.35**	**517**	**1,408**
Inferred	0.2	3,736	0.12	0.46	10	55

Notes:
(1) CIM definitions were followed for Mineral Resources.
(2) Mineral Resources are reported at a 0.2% CuEq cut-off value using metal prices of $3.50 per pound copper and $1,500 per ounce gold, and a US$/C$ exchange rate of US$1.00/C$1.25.
(3) All figures have been rounded to reflect the relative accuracy of the estimates.
(4) Mineral resources that are not mineral reserves do not have a demonstrated economic viability.
(5) Mineral Resources reported exclusive of Mineral Reserves.

The Mount Milligan Au-Cu porphyry deposit contains a combined Measured and Indicated Mineral Resource, exclusive of Mineral Reserves, of 125.4 million tonnes (Mt) at 0.19% Cu and 0.35g/t Au containing 517 million pounds of copper and 1.4 million ounces of gold and an Inferred Mineral Resource of 3.7 million tonnes at 0.12% copper and 0.46g/t gold.

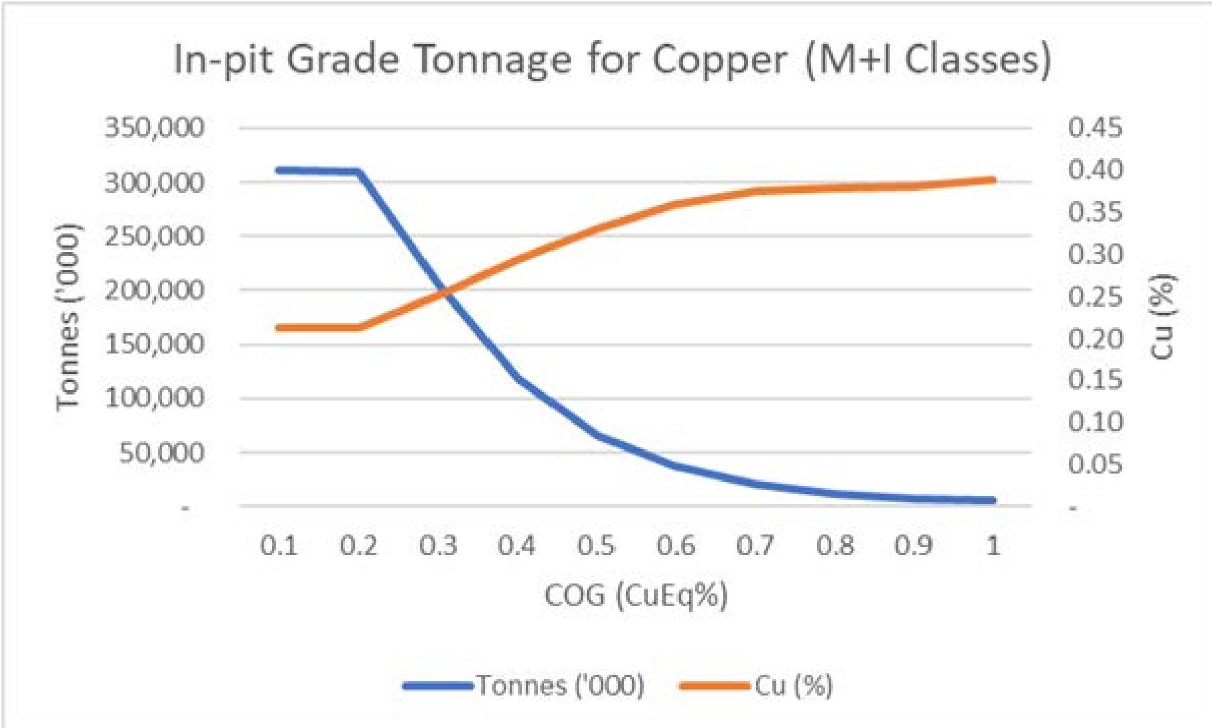
14.11. GRADE SENSITIVITY ANALYSIS

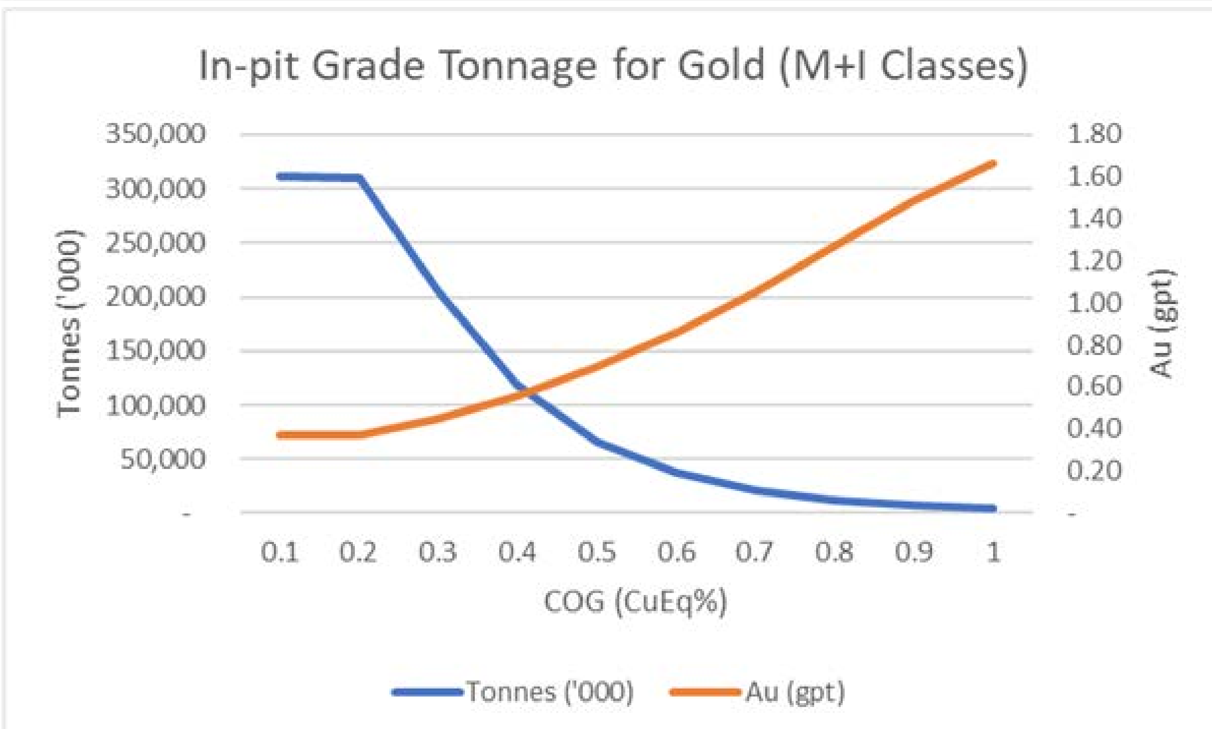
Estimated mineral resources are sensitive to the selection of the reporting cut-off grade. To illustrate this sensitivity, the block model quantities and grade estimates within the pit used to constrain the mineral resources are presented in Table 14-18 for selected CuEq values with these resource figures being inclusive of reserves. The reader is cautioned that the figures presented in these tables should not be misconstrued as a Mineral Resource Statement. The figures are presented to show the sensitivity of the block model estimates to the selection of cut-off grade. A comparison of Table 14-18 and Table 15-2 suggests that grade and tonnages at 0.20% CuEq and C$9.55/tonne NSR cut-off are reasonably matched. Figure 14-18 presents this sensitivity as grade and tonnage curves for CuEq for both copper and gold.

Table 14-18: Measured and Indicated In-Pit Model Quantity and Grade Estimates at Various CuEq%

Category	COG CuEq (%)	Tonnes (kt)	Cu grade (%)	Au grade (g/t)	Copper (Mlb)	Gold (koz)
M+I	0.1	310,933	0.21	0.37	1,458	3,737
M+I	**0.2**	**310,201**	**0.21**	**0.37**	**1,457**	**3,730**
M+I	0.3	206,055	0.25	0.45	1,140	2,973
M+I	0.4	118,115	0.29	0.56	766	2,122
M+I	0.5	65,840	0.33	0.70	480	1,475
M+I	0.6	36,724	0.36	0.86	292	1,013
M+I	0.7	21,141	0.38	1.05	175	714
M+I	0.8	12,510	0.38	1.27	105	510
M+I	0.9	7,716	0.38	1.48	65	368
M+I	1.0	5,107	0.39	1.66	44	273

Figure 14-18: In-Pit Grade Tonnage Curves (Measured and Indicated Classes) for CuEq% Cut-Off





14.12. PREVIOUS MINERAL RESOURCE ESTIMATES

- The previous Mineral Resources were estimated as of December 31, 2018 within a conceptual open pit shell using spot metal prices of $3.50/lb copper and $1,450/oz gold and are reported exclusive of the Mineral Reserve.
- The Mineral Resources were reported using a C$8.12 NSR cut-off grade and are not comparable with the current resource estimation.
- Combined Measured and Indicated Mineral Resources, exclusive of reserves, totalled 342.2 Mt at 0.14% Cu and 0.25 g/t Au containing 1,028 million lb of copper and 2.7 million oz of gold.
- Inferred Mineral Resources totalled 41.0 Mt at 0.13% Cu and 0.31 g/t Au containing 115 million lb of copper and 0.41 million oz of gold.

14.13. OTHER MATERIAL FACTORS

The Qualified Person as defined by National Instrument (NI) 43-101 and the person who completed the Resource Estimation from this report, is not aware of any other known material factors that may affect the Mineral Resource.

15. MINERAL RESERVE ESTIMATES

The Mineral Reserve estimate was prepared by Mount Milligan Mine and Centerra Corporate Technical Services personnel under the supervision of John Fitzgerald P.Eng., Vice President Projects & Technical Services for Centerra. This reserve estimate is effective 31st December 2019.

15.1. OPEN PIT OPTIMIZATION PARAMETERS

The pit optimization parameters shown on Table 15-1 were developed to define the optimized pit shell based on Measured and Indicated Resources using the resource model discussed in Section 14. In addition to the parameters listed, the following factors were also used to derive the optimized pit shell:

- Mining dilution = 5.0%
- Mining loss = 1.5%
- Inter-ramp pit slope angles:
  - 30° in overburden
  - 42° for east wall
  - 47° for all other pit walls
- Metallurgical recoveries as discussed in Section 13

Operating and sustaining capital unit cost estimates were derived using first principles cost models that assume improved operating practices compared to historic costs. These improvements have been identified and quantified by site management and will be a key focus moving forwards.

NPV Scheduler software allows the user to develop multiple pit shells by adjusting the revenue factor (RF) from base case metal prices. By modelling changes in the metal prices, the mine design engineer can assess the ultimate pit shell, on a profitability basis. Using Table 15-1 parameters, the base case parameters (RF=1.00) result in a pit shell having approximately 150Mt ore and a strip ratio of 0.65. This strip ratio is considered relatively low compared to historic mining at Mount Milligan where the strip ratio has averaged approximately 1.35 so the RF=1.06 optimized pit shell was used as the basis for ultimate pit design. The RF=1.06 pit shell contains approximately 197Mt ore and a 0.92 strip ratio. This higher RF shell generated a material increase in strip ratio, which was deemed necessary to ensure sufficient waste material for tailings storage facility construction.

Table 15-1: Pit Optimization Parameters (RF=1.00)

Item	Units	Value	Comments
Au price	US$/oz	1,250	Assumed long-term Au price for reserve estimation
Cu price	US$/lb	3.00	Assumed long-term Cu price for reserve estimation
Royalty	%	2.0	Haslinger royalty, calculated using spot prices
Exchange rate	C$:US$	1.25	Assumed long-term exchange rate assumption
Mining opex - waste	US$/t mined	2.37	Estimated LOM average, including mining sustaining capex
Mining opex - ore	US$/t mined	2.09	Estimated LOM average, including mining sustaining capex
Mining incremental opex	US$/t ore	0.02	Incremental haulage cost per bench below 995 bench
Processing opex	US$/t ore	5.45	Estimated LOM average, excluding processing sustaining capex
G&A opex	US$/t ore	1.82	Estimated LOM average, excluding G&A sustaining capex
Mining capex	US$/t ore	0.00	Included in mining opex figures
Processing capex	US$/t ore	0.21	Estimated sustaining capex LOM average
G&A capex	US$/t ore	0.16	Estimated sustaining capex LOM average
Production rate	kt/d ore	60	Nominal (permitted) process plant production rate
Discount rate	%	5.0	Applied to discount future cash flows
Concentrate grade	%	21.5	Assumed average concentrate grade
Cu deduction	%	1.07	Assumed average smelter term
Au deduction	%	2.50	Assumed average smelter term
Smelting charge	US$/dmt	80.0	Assumed average smelter term
Cu refining charge	US$/lb payable	0.08	Assumed average smelter term
Au refining charge	US$/oz payable	5.0	Assumed average smelter term
Truck transport	US$/wmt	21.45	Assumed truck transport cost, site to Mackenzie
Rail transport	US$/wmt	42.44	Assumed rail transport cost, Mackenzie to Vancouver
Port charges	US$/wmt	23.46	Assumed port charges for initial 120,000wmt/year
Port charges	US$/wmt	20.58	Assumed port charges for amount >120,000wmt/year
Ocean transport	US$/wmt	45.34	Assumed ocean transport cost to Asia
Representation	US$/wmt	1.28	Assumed representation charges
Other fees	US$/wmt	1.48	Assumed assay, analysis & umpire fees
Loss	%	0.5	Assumed concentrate quantity loss
Moisture content	%	8.50	Estimated moisture content site to port
Moisture content	%	7.96	Estimated moisture content from port

15.2. ULTIMATE PIT DESIGN

For Mineral Reserve classification, the resource pit shell output from NPV Scheduler is brought into computer-aided drafting (CAD) design software Datamine Studio to convert the optimised pit shell (or cone) into a practical pit design. Locations of ore and waste within the cone, metal content, haul roads, and minimum mining widths based on loading fleet size are considered to develop the final (or ultimate) pit design. The location of ore blocks is important, as this helps determine the amount of waste rock that will be extracted in order to reach the ore. Datamine Studio 3D design software provides computer aided drafting tools to map out how to access ore based on the NPV Scheduler optimised pit shell. Datamine Studio 3D software was utilized to quantify and tabulate according to cut-off value (or grade) the quantity and quality of economic mineral reserves by tonnage and average grade(s) by bench and mining pushback (or phase). The tabulated reserves were then scheduled with recoveries and costs applied to determine saleable product and the resultant economics.

15.3. NET SMELTER RETURN CUT-OFF CALCULATION

Following definition of the ultimate pit shell, a net smelter return (NSR) based cut-off was used to define ore reserves. This NSR cut-off comprises processing and G&A operating and sustaining capital unit costs shown on Table 15-1 for a total $7.64/t or C$9.55/t. Mining opex is excluded from this calculation as the definition of ore (and waste) is made at the pit rim; with mining opex having been considered in definition of the optimized pit shell. One-time processing or G&A sustaining capex items were also excluded from the NSR cut-off.

Note that the 2017 Technical Report for Mount Milligan Mine calculated a NSR cut-off of $6.25/t or C$8.12/t.

15.4. MINERAL RESERVE STATEMENT

Table 15-2 summarizes the proven and probable reserves estimated as of 31st December 2019, with CIM definitions used for the estimate.

Table 15-2: Mineral Reserve Statement, Effective Date December 31, 2019

Mineral Reserve Category	Tonnes (kt)	Copper Grade (%)	Gold Grade (g/t)	Contained Copper (Mlb)	Contained Gold (koz)
Proven	114,753	0.23	0.41	571	1,525
Probable	76,275	0.23	0.36	389	882
Proven + Probable	191,028	0.23	0.39	959	2,407

Notes:
(1) CIM definitions were followed for Mineral Reserves estimation.
(2) Mineral Reserves are estimated at $7.64/t (C$9.55/t) NSR cut-off value using metal prices of $3.00 per pound copper and $1,250 per ounce gold, and a US$/C$ exchange rate of US$1.00/C$1.25.
(3) Figures may not total exactly due to rounding.

The proven reserve estimate includes stockpiled material totalling 6,197kt with grades of 0.14% copper and 0.43g/t gold and contained metal of 19Mlb copper and 86koz gold. As such, the proven and probable reserve within the ultimate pit is 184.8Mt.

End-2018 reserves at Mount Milligan Mine were estimated to be 447.6Mt comprising 211.6Mt proven and 235.9Mt probable reserves. Adjusting this estimate for end-2019 open pit topography results in a total 426.1Mt proven and probable reserves. The following factors are estimated to account for the reduction from this figure to the end-2019 reserve estimate of approximately 185Mt within the ultimate pit:

- 87Mt decrease due to applying the December 31, 2019 resource model against the end-2018 ultimate pit design.
- 107Mt decrease due to applying costs per Table 15-1 to the December 31, 2019 resource model.
- 47Mt decrease due to applying metallurgical recoveries per Section 13 to the December 31, 2019 resource model.

For clarity, the aforementioned decreases are cumulative to give the total 241Mt proven and probable reserve decrease from the end-2018 reserve estimate (adjusted to end-2019 open pit topography) to the end-2019 reserve estimate, with the latter based on the December 31, 2019 resource model. It should be noted that the order in which these factors are applied could affect the decrease estimated for each factor.

16. MINING METHODS

16.1. INTRODUCTION

Only Measured and Indicated Mineral Resources were considered as ore in the pit optimization process, Mineral Reserve estimate and life of mine (LOM) plan. All blocks classified as Inferred Resource or below the incremental NSR cut-off, regardless of metal grade values, were excluded.

The mining operation is a conventional shovel and truck open pit mine feeding a 60,000 t/d (permitted throughput) processing plant. The planned mine life is 9 years with a Proven and Probable Reserve of 191.0 million tonnes at a grade of 0.23% copper and 0.39 g/t gold. The pit has been planned as a series of discrete pushbacks and is scheduled to maximize the production of high-value ore, especially during the next three years. This strategy may necessitate mining at a higher cut-off value and the stockpiling of lower value material for processing later in the mine life. The average stripping ratio for the LOM is estimated to be 1.24:1.

The mine currently uses two electric rotary blasthole drills which drill 311 mm diameter holes, two diesel blast hole drills for 203 mm holes, two 41 m^3 electric rope shovels and two 19 m^3 front end loaders loading haul trucks of 220 and 190 tonnes capacity from 15m high production benches. The operation is supported by standard ancillary equipment including track and rubber tire dozers, graders and a fleet of service vehicles.

The mine operates continuously, seven days per week, using a roster schedule of predominantly 14 days on, 14 days off, with 12-hour shifts.

The overall mining sequence was developed through a series of mining pushbacks (phases) specific to the MBX, 66 and WBX subzones (collectively, the Main Zone), and the Southern Star (SS) phases.

Mineralized material with a value greater than $7.64 (C$9.55) per tonne and with a copper grade greater than or equal to 0.12% is designated as standard ore. Material with a value greater than $7.64 (C$9.55) per tonne with copper grades less than 0.12% are designated as HGLC (high-gold low-copper) ore.

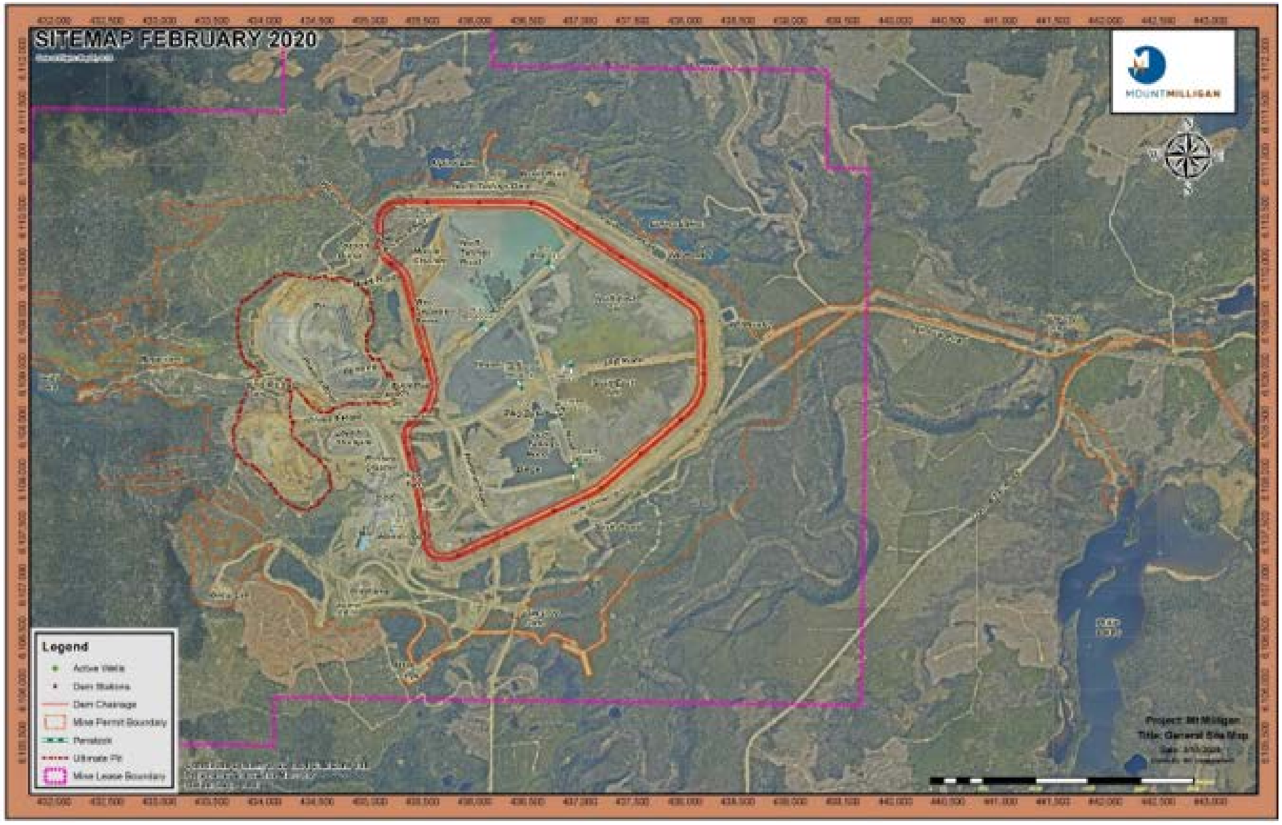
The following objectives were considered whilst finalizing pit designs and sequencing pushbacks:

- maintain higher value production in the early years of the mine life.
- maintain a smooth waste/ore ratio and also providing the required annual Tailings Storage Facility (TSF) construction materials.
- develop an ore blending strategy to honour metallurgical constraints such as ORE/HGLC ratio, Hg content and Pyrite to Chalcopyrite (Py:Cpy) ratio
- provide potential waste rock and cleaner tailings storage in the mined-out areas of the Main Zone pit for waste generated from the WBX and SS mining phases

In 2020, mining will continue in Phases 3, 4 (Main pit) and Phase 8 (Southern Star pit). The pits will expand as phased pushbacks until the ultimate pit limits are reached. In total, nine phases will be mined encompassing the Main and Southern Star pits. Each pit phase will expose a portion of each ore body facilitating control of process plant head grade by blending ore types. Separate accesses have been designed for each phase, permitting sequencing, flexibility and equipment movement between phases.

Mining and placement of overburden, non-acid generating (NAG) sulphide rock, weathered and oxide rock from the mine is integrated at the TSF with the construction of the south, southeast, northeast, and north embankments, Pipeline Corridor Causeway (PCC), Eskers Road and West Separator Berm (WSB). Delivery of potentially acid generating (PAG) rock for sub-aqueous storage at the TSF will be integrated in the cleaner/scavenger tailing Separator Dyke and bulk storage area within the TSF. Residual overburden will be stockpiled in the WSB laydown area and a temporary stockpile located west of the primary crusher (South Stockpile). Overburden will be reclaimed from the South Stockpile during the later years of the mine life for shortfalls of TSF material requirements, and for the completion of the last construction stages of the TSF embankment. The current general arrangement drawing for the mine, process plant, other infrastructure and overall TSF is illustrated in Figure 16-1.

Figure 16-1: Site Layout



16.2.MINE DESIGN AND SCHEDULING

Geotechnical Parameters

Pit slope angles vary by sector within the open pit and are based on recommendations by Knight Piésold (2007 Feasibility Open Pit Slope Design report and KP memo dated May 14, 2009, entitled "Mount Milligan Project –Updated Feasibility Pit Slope Design). Pit slope designs are based on double-benching of 15m height and consider haulage ramp positioning, safety berms and other geotechnical features required to maintain safe inter-ramp and overall slope angles. Slope recommendations from the report can be summarized into three categories, namely overburden, MBX East, and all remaining. The respective pit slope design parameters that have been used are illustrated in Table 16-1.

Table 16-1: Pit Slope Design Parameters

Material/Lithology-Area	Inter-Ramp Angle (degrees)	Face Angle (degrees)	Bench Height (m)	Bench Width (m)
Overburden	30	60	30	17
Main (East, Southeast) Intrusives, Volcanics	42	60	30	16
Remaining Main, SS Intrusives, Volcanics	47	65	30	14

As mining operations progress, further geotechnical drilling and stability analysis will be completed by Mount Milligan to further optimize the geotechnical parameters in the LOM designs. Mount Milligan staff and external consultants developed the pit water management program that consists of surface water diversion ditches and drilling of horizontal drains into pit walls. Management of water inflows to-date have been appropriate, and no hydrological or hydrogeological issues that could significantly impact pit wall stability have been encountered.

Ultimate Pit Design and Mining Phase Layouts

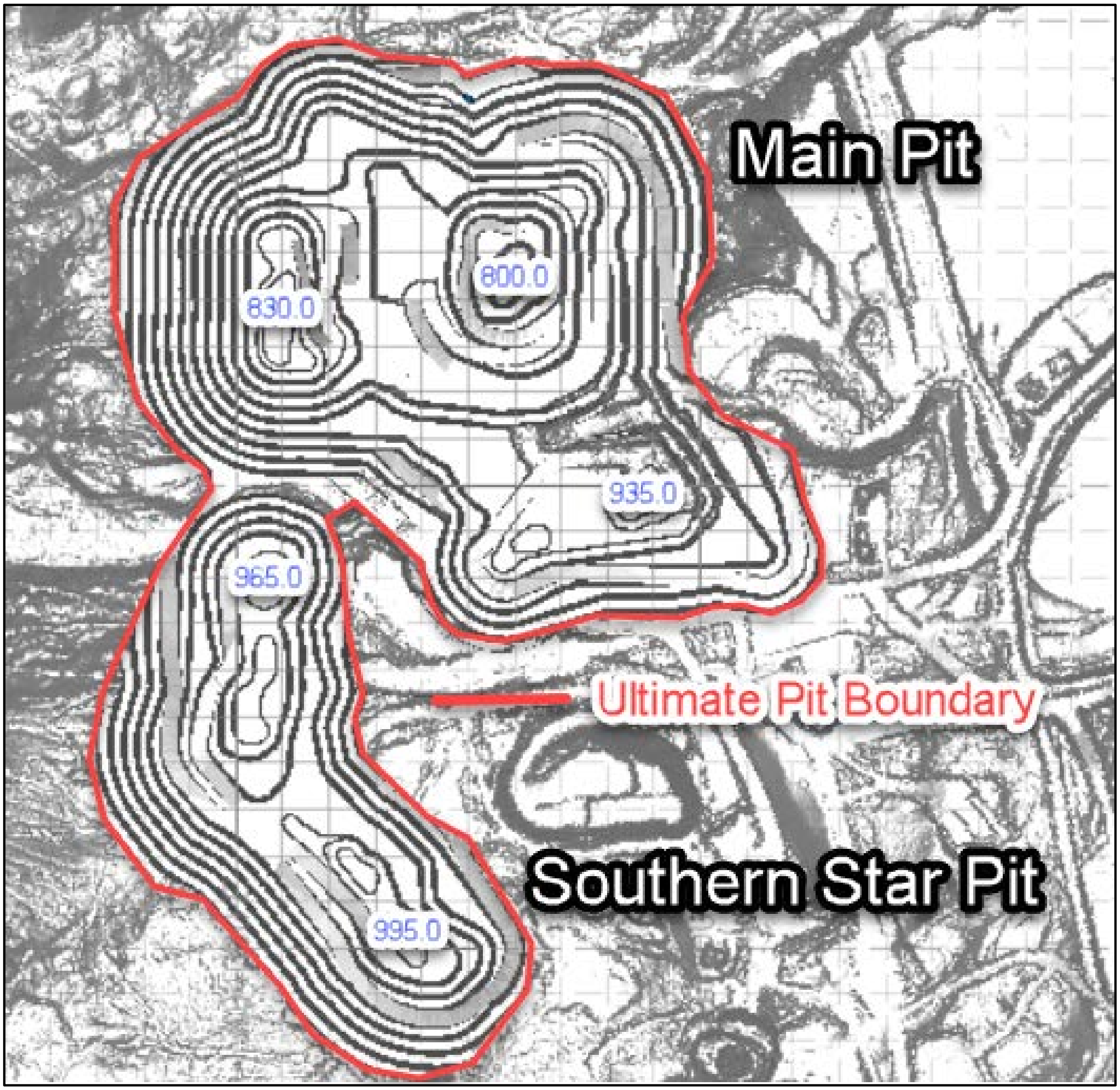
The ultimate pit dimensions and design, including the final haul ramps extending to an ultimate depth at the 800m bench in the Main Zone and 965m bench in the SS Zone, are shown in Table 16-2 and Figure 16-2.

Table 16-2: Ultimate Pit Dimensions

Designs	Main Pit	SS Pit
Dimensions, m	N-S 1,200	N-S 1,300
	E-W 1,290	E-W 540
Pit Bottom Elevation, m	800	965
Maximum Depth, m	330	255

The ultimate pit haul roads have been designed with an overall 34 m width that will include an outside berm (4.84 m wide, 1.82 m high) and ditches (1.00 m) for dual lane passing of 227-t haul trucks (8.28 m) with a one-half truck width (4.12 m) for truck separation. On each side of the truck, an allowance of 4.23 m for rubble buildup and safety berm creep has been included. Roads have been designed with a maximum 10% down grade.

Figure 16-2: Ultimate Pit Design



The current LOM Plan consists of nine Phases, building off previous Phases, 1 and 2, which are now complete. The mine is currently operating in Phases 3 and 4 with initial stripping started in Phase 8 in the Southern Star Pit. Phases 3, 4, 5, 6a, 6 and 7 are focused on the Main Zone while Phases 8, 9a and 9 will focus on the Southern Star Pit. The phases are not mined sequentially. The LOM schedule is focused on returning the highest value ore as early as possible while honouring metallurgical constraints. The spatial arrangement of the current phases is illustrated in Figure 16-3 and Figure 16-4.

Figure 16-3: Spatial Arrangement of Phases

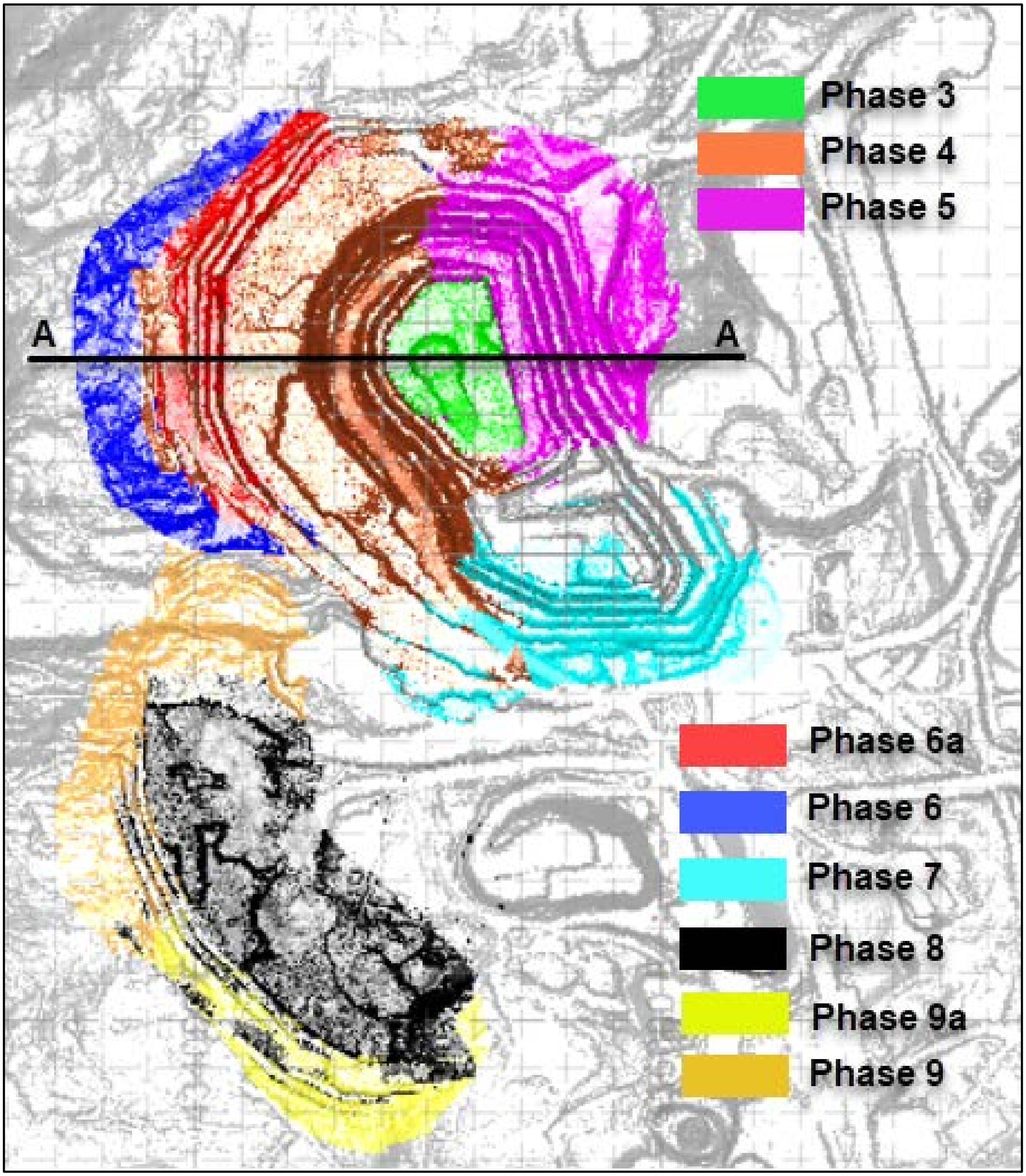
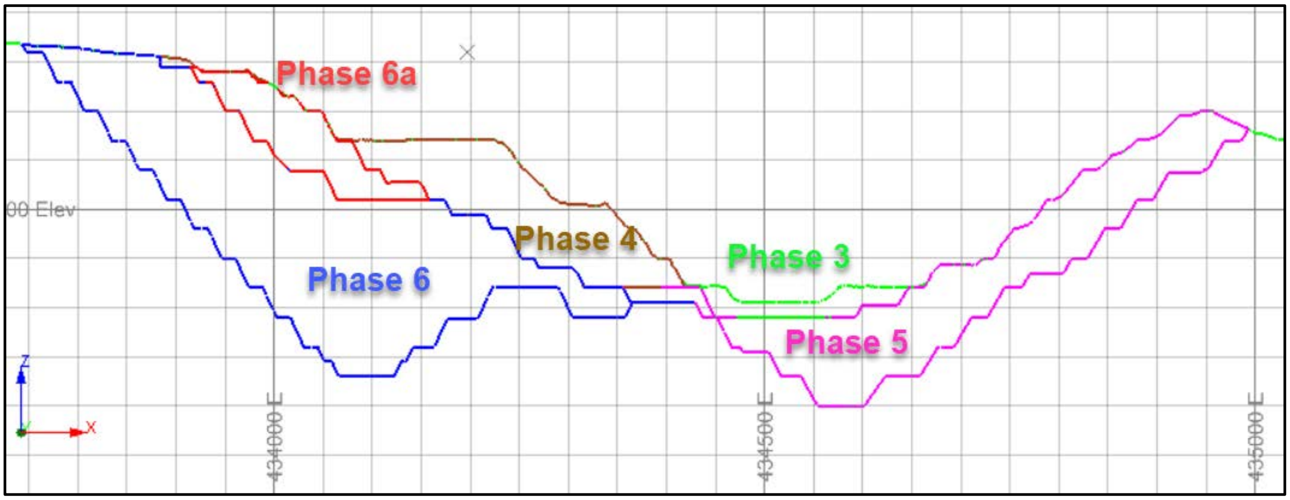


Figure 16-4: A-A Section Through the Main Pit

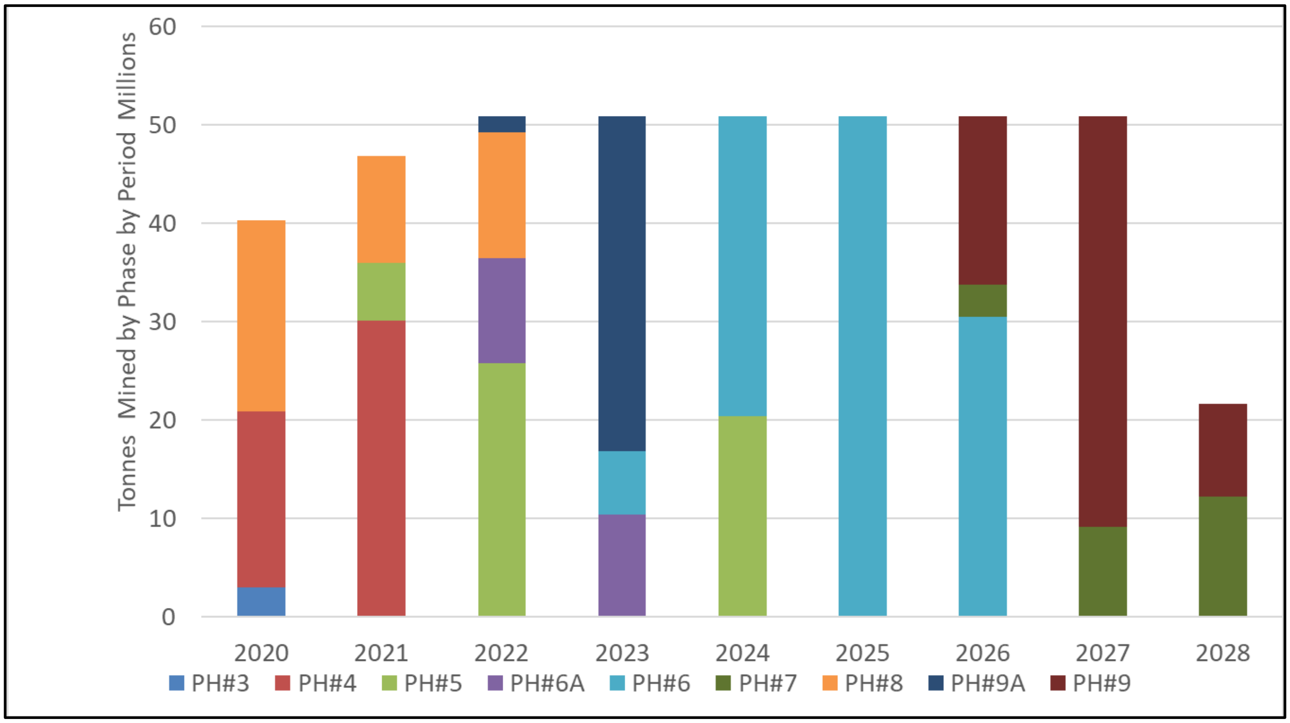


Mining Schedules

Mining schedules were developed with considerations for ore blending, waste scheduling for the TFS dam construction, metal delivery and economics. Open pit phase developments are overlapped to smooth the stripping, equipment and manpower requirements. Figure 16-5 illustrates the LOM profile by annual total tonnes mined (ore and waste) segmented by phases.

In the course of scheduling, the vertical advance rate (or sink rate) was restricted to a maximum 10 benches per year per phase. Where possible, development was restricted to two concurrent phases in each pit, with some years requiring development of three concurrent phases to ensure consistent ore supply.

Figure 16-5: Annual Material Movement by Phase



Material requirements for Tailings Storage Facility (TSF) dams, road building and waste disposal were followed to ensure tailings disposal requirements of the process plant would be achieved. Final scheduling assumptions incorporated the following process plant throughput:

- 2020 at 57,500 t/d
- 2021 onward at 60,000 t/d

Total ore and waste will be mined at an average rate of 40.2 Mt/a in 2020, 46.8 Mt/a in 2021 and 50.8 Mt/a in 2023 through 2027, and decreasing to 21.6 Mt/a in 2028 when waste mining decreases; yielding an overall LOM waste:ore ratio of 1.24:1.0. The mining schedule by year is illustrated in Figure 16-6 and in Table 16-3.

Figure 16-6: Annual Material Mined

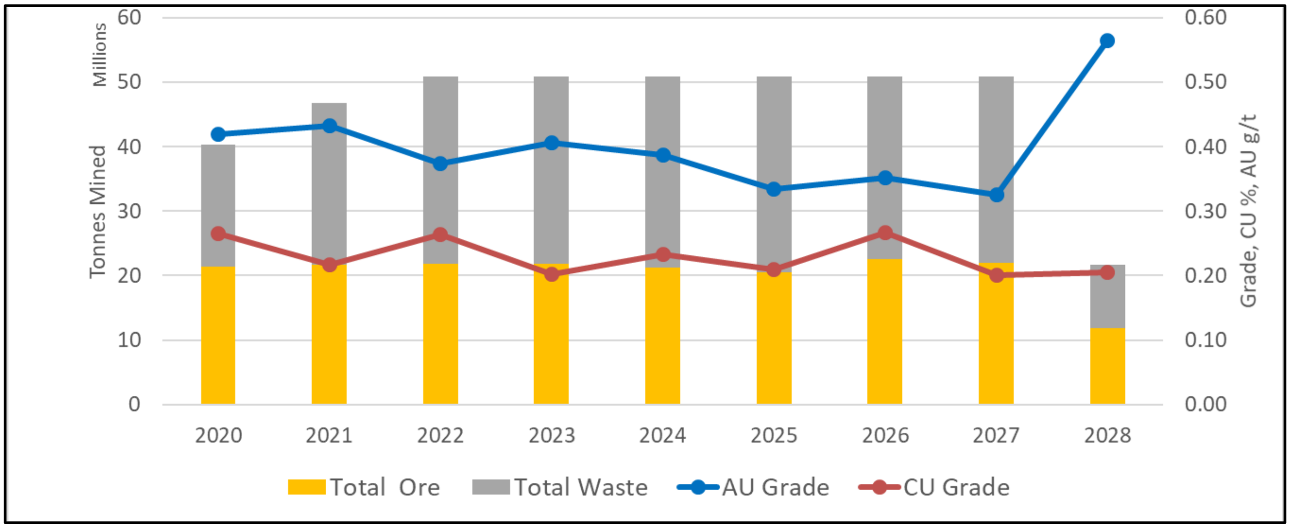


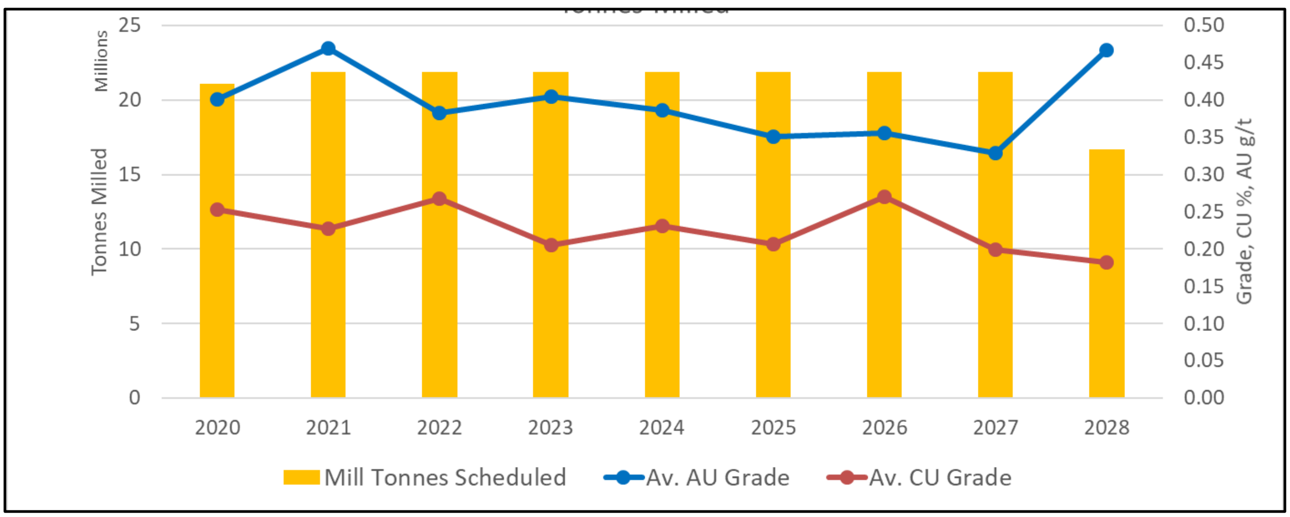
Table 16-3: Annual Material Mined (excludes existing stockpile)

Indicators	Units	Total LOM	Year								
			2020	2021	2022	2023	2024	2025	2026	2027	2028
Total Mined	(kt)	413,797	40,277	46,802	50,848	50,854	50,848	50,847	50,853	50,846	21,621
Total Waste	(kt)	228,966	18,929	25,047	28,988	29,061	29,580	30,321	28,328	28,876	9,836
Total Ore	(kt)	184,831	21,348	21,756	21,860	21,793	21,268	20,526	22,525	21,970	11,785
Strip Ratio	(t)/(t)	1.24	0.89	1.15	1.33	1.33	1.39	1.48	1.26	1.31	0.83
Total Ore: Ore + HGLC											
Tonnes	(kt)	184,831	21,348	21,756	21,860	21,793	21,268	20,526	22,525	21,970	11,785
AG Grade	(g/t)	1.59	1.60	1.59	1.82	1.56	1.69	1.54	1.50	1.44	1.50
AU Grade	(g/t)	0.39	0.42	0.43	0.37	0.41	0.39	0.33	0.35	0.33	0.56
CU Grade	(%)	0.23	0.27	0.22	0.26	0.20	0.23	0.21	0.27	0.20	0.20
VLPTO	(CAD/t)	18.5	20.8	19.0	19.5	17.6	18.4	16.1	19.0	15.5	22.0
PYCPY Ratio	n/a	10.0	5.3	7.3	9.1	7.7	11.9	10.7	5.8	11.9	28.7
Hg content	(ppm)	0.28	0.23	0.20	0.30	0.17	0.21	0.24	0.32	0.58	0.28
AG Cont.	(kOz)	9,431	1,096	1,115	1,279	1,096	1,158	1,016	1,088	1,016	569
AU Cont.	(kOz)	2,321	288	302	263	285	265	221	254	230	214
CU Cont.	(Mlb)	940	125	104	127	97	109	95	132	97	53

The ore stockpile at end-2019 is estimated to contain 6.2Mt ore which, when added to the scheduled 184.8Mt to be mined over the remaining mine life sums to a total of 191.0Mt, per the Mineral Reserve estimate.

The blending strategy to deliver an overall average mill feed of 0.23% copper to yield a marketable 21.5% copper concentrate was achieved through sequencing by bench and phase. The annual material processing schedule is graphically presented in Figure 16-7.

Figure 16-7: Annual Material Processed (includes existing stockpile)



Annual construction material for TSF embankments will be sourced from overburden, NAG waste rock and oxide-weathered rock from the pit. The overburden and NAG rock can be used for all downstream (outside) embankment mass fills. Weathered and oxide material with some addition of overburden and NAG rock will be used for constructing the upstream (inside) portion of the TSF. PAG material will be placed inside the facility, either in the PAG Separator Dykes, the PAG dump (interior to the TSF), the PCC and Eskers access roads or other areas inside the facility where it will ultimately be submerged by tailings or water following the operations phase. Table 16-4 represents the waste mined from the pit in relation to material requirements for TSF dam construction.

Table 16-4: Waste material Requirements for TSF Dam Construction

Destination	Material type	Units	Tonnage			Comments
			Material required	Material Mined	Difference	
Core	S	(t)	7,622,400	11,056,164	3,433,764	Sufficient amount
Upstream	Oxide	(t)	14,949,600	9,400,655	(5,548,945)	Deficit of oxide material will be replenished from excess of NAG material
Downstream	NAG, OVB, C	(t)	37,128,000	63,947,602	26,819,602	Sufficient amount

The excess of waste material identified in Table 16-4 will be managed either by storing excess material in the open pit or by re-designing the ultimate pit so that it contains less waste relative to ore.

Ore Quality Requirements and Blending

Although mining scheduling was primarily concerned with the sequenced removal of waste and ore within mining constraints, metallurgical blending requirements were also considered. The objective is to mine in such a way that the resulting ore mix meets the quality and quantity specifications of the process plant as per Table 16-5. Maintaining process plant ore feed quality requirements at Mount Milligan requires honouring metallurgical constraints such as the ORE:HGLC ratio, Py:Cpy ratio and Hg content.

Table 16-5: Ore Blending Requirements

Blending constraint	Units	Blending requirements			Mill's KPI affected by constraint	Importance 1 - most important for the processing; 2 - less important
		Min	Target	Max		
Ore to HGLC ratio	(t/t)	Cu grade > 0.10% Cu	Cu grade 0.2-0.24% Cu	Not restricted	Recovery, concentrate grade	1
Pyrite-Chalcopyrite ratio	n/a	as low as possible	as low as possible	12	Recovery	1
Mercury (Hg) content	(ppm)	as low as possible	as low as possible	0.35	Concentrate purity	2

The final mining and stockpile handling schedule ensures that the average qualities of the material being mined in each year meet the stipulated processing requirements in terms of copper grade, Ore to HGLC ratio and Hg content. However, in 2027 and 2028, the Py:Cpy ratio remains higher than the stipulated threshold of 12. The reason is that HGLC material with average Py:Cpy ratio of 67 is mined in Zone 66 in Phase 7 (ore material with high Py:Cpy ratio in this zone is also characterized by relatively very high gold content of 0.97 g/t). Metallurgical recovery for 2027-2028 has been adjusted to account for this material. Due to uncertainties related to processing of material with high Py:Cpy ratio, the mining of Phase 7 (and hence processing of this material) has been delayed to later stages of the LOM (2026-2028) to ensure metallurgical tests on this material will be completed to further optimize related metallurgical recoveries.

Stockpiling Strategy

An ore stockpiling strategy is included for the LOM plan update. All material with a NSR value greater than 7.64/t (C$9.55/t) is considered to be ore (and included in the Reserve estimate). However, there are periods when lower NSR value ore above this NSR cut-off value is stockpiled to increase overall Project value. Also, in some instances, ore is stockpiled to ensure the required ore quality is provided for processing (as per metallurgical constraints). As a result, while the start-2020 stockpile balance is 6.2Mt, additional ore is stockpiled over the mine life such that a total 19.4Mt is re-handled from the stockpile for process plant feed; with the permitted stockpile capacity not exceeded at any time.

Annual Mining Plans

Annual mining plans have been developed showing the development of the Main and SS pits for years 2020 through Year 2028 as shown on Figure 16-8 to Figure 16-16.

Figure 16-8: Pit Advances at End-2020

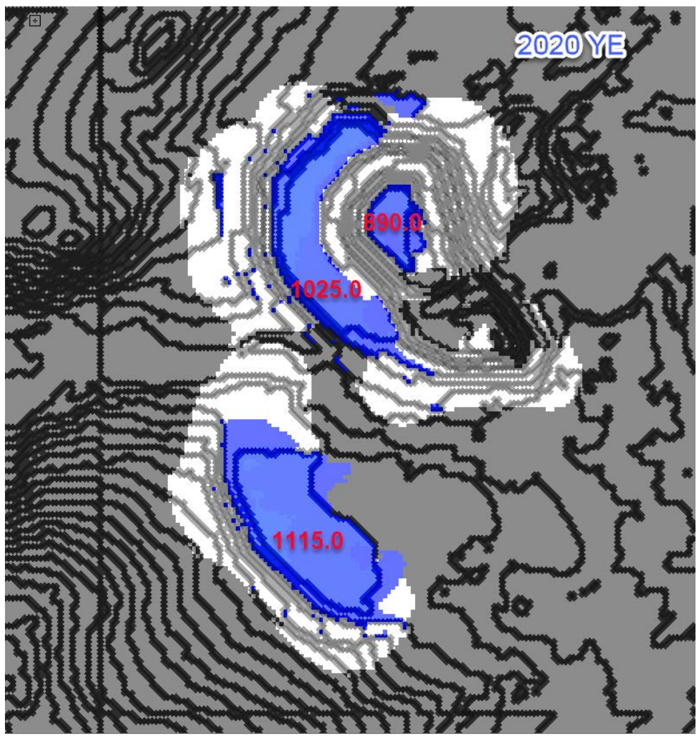


Figure 16-9: Pit Advances at End-2021

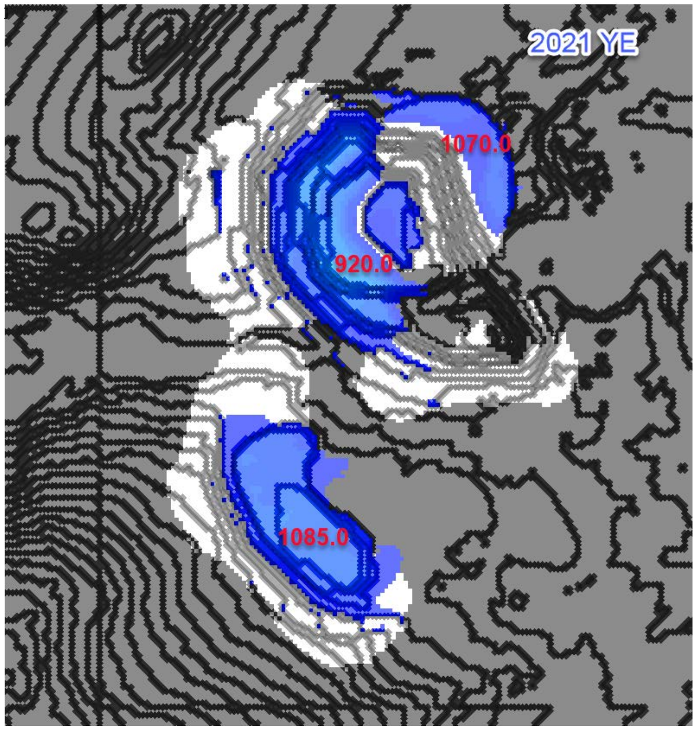


Figure 16-10: Pit Advances at End-2022

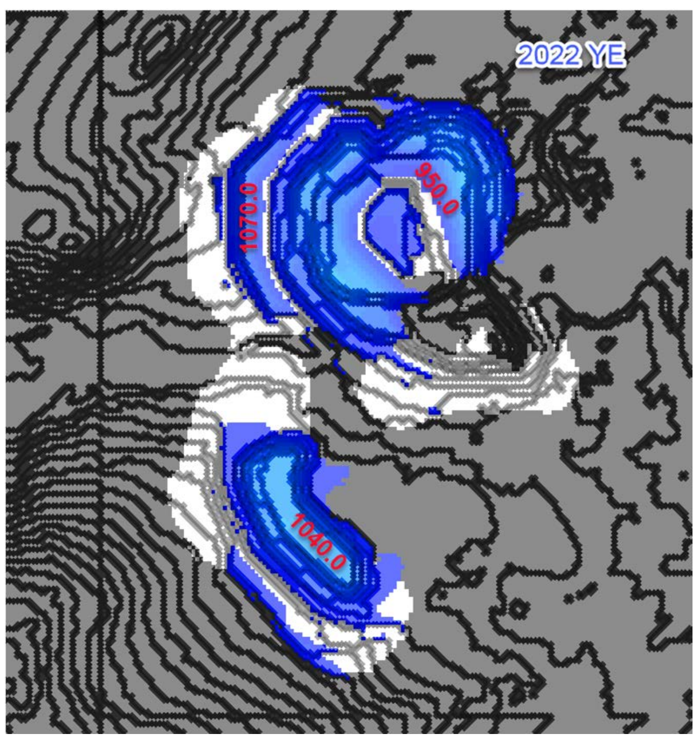


Figure 16-11: Pit Advances at End-2023

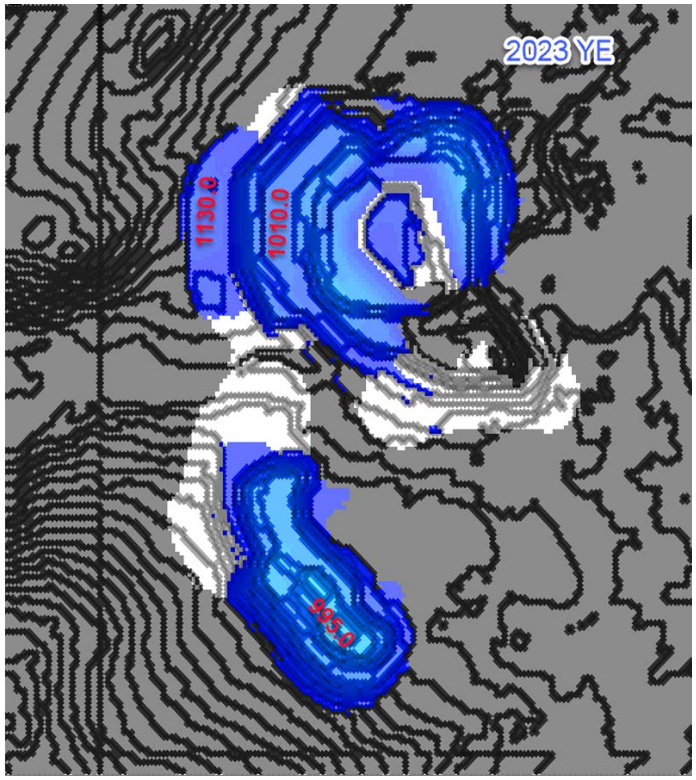


Figure 16-12: Pit Advances at End-2024

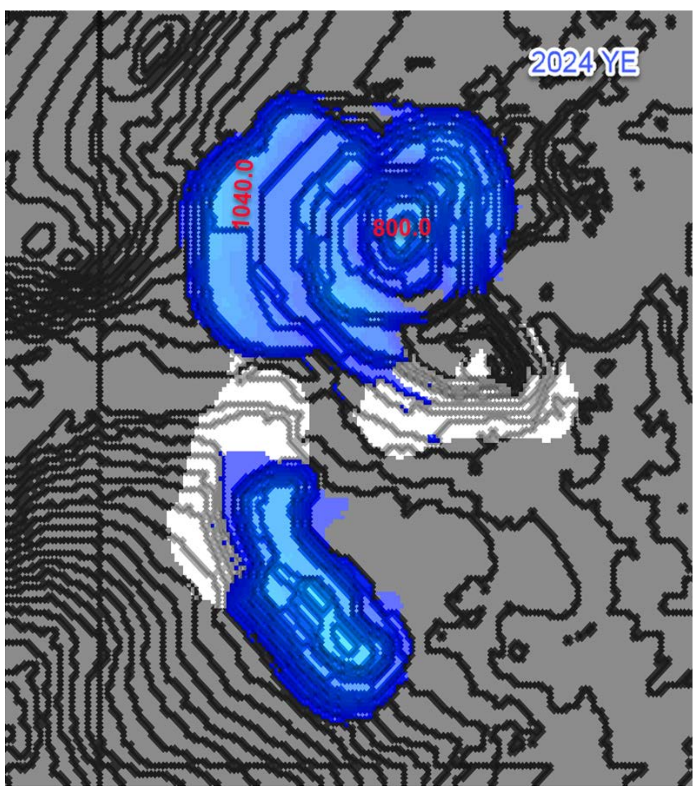


Figure 16-13: Pit Advances at End-2025

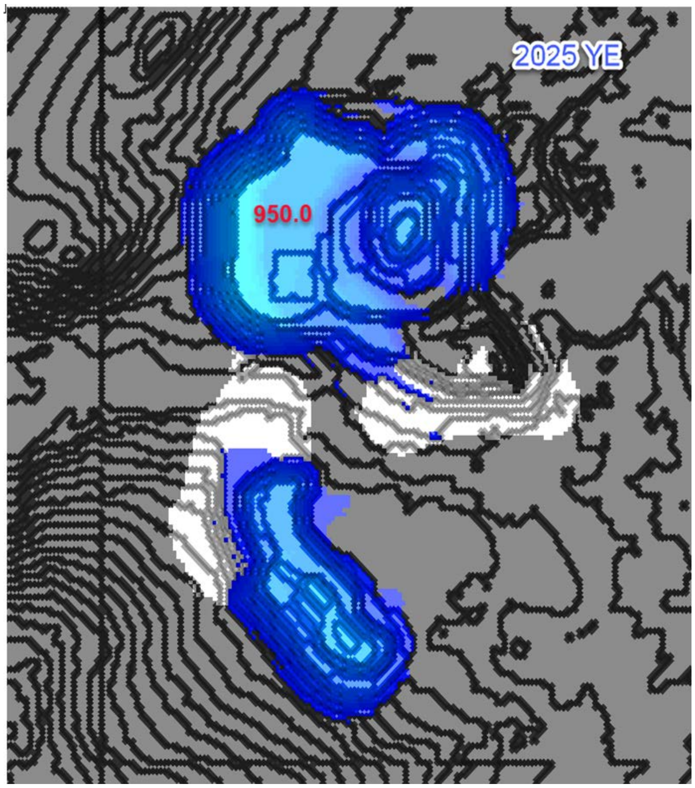


Figure 16-14: Pit Advances at End-2026

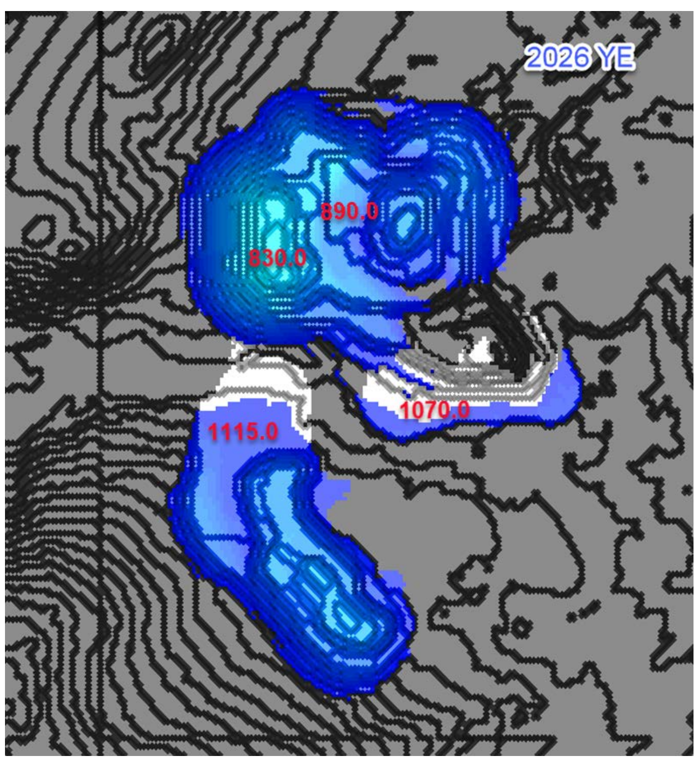


Figure 16-15: Pit Advances at End-2027

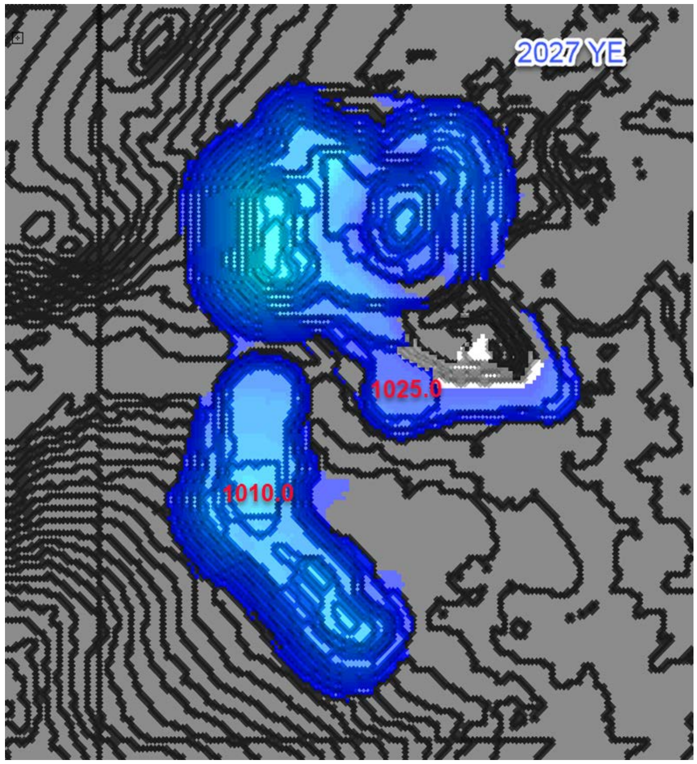
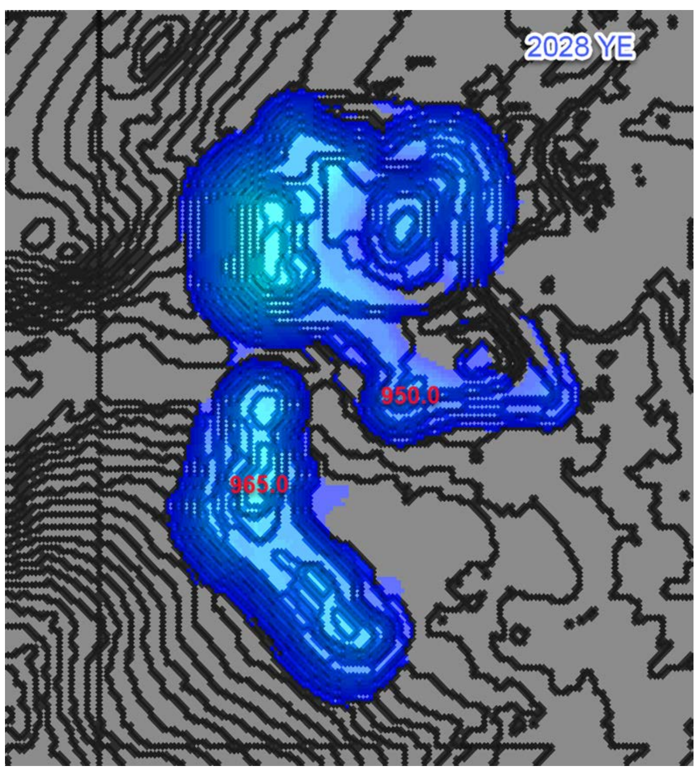


Figure 16-16: Pit Advances at End-2028



Mining Equipment

Open pit mining is undertaken using a conventional truck-and-shovel fleet. A list of the existing major open pit mobile equipment is shown in Table 16-6. Maintenance of mine mobile equipment is undertaken by Mount Milligan Mine personnel with the assistance of external specialists as required. The loading capacity of the mining fleet is sufficient for the next nine years. However, five additional haul trucks will need to be added to the existing fleet over the next several years as haulage distances and cycle times increase with greater pit depths and distance to the Tailings Dam.

Table 16-6: List of Major Mining Mobile Equipment

Type	Quantity	Unit
Major Loading Tools	CAT 7495HR2 Electric Rope Shovels	2
	CAT 994F Wheel Loader	2
Drills	CAT 6640 Electric	2
	CAT 6290 Diesel	1
	Epiroc D65 Smartroc Diesel	1
Haul Trucks	CAT 793F	13
	CAT 789C	2
Dozers	CAT D6	1
	CAT D8	1
	CAT D10	5
	CAT 834 Wheel Dozer	2
Graders	CAT 14M	1
	CAT 16M	2
	CAT 24M	1
Support Loaders	CAT 988	2
	CAT 930	1
	CAT 980	1
	CAT 966	1
Support Excavators	CAT 390	1
	CAT 349	2
	CAT 329	1

17. RECOVERY METHODS

17.1. SUMMARY

The Mount Milligan process plant was originally designed to process ore at a nominal rate of 60,000 t/d, producing a marketable concentrate containing copper, gold, and silver. A secondary crushing circuit, installed in 2016, together with process plant optimization projects, increased the throughput to a nominal rate of 62,500 t/d. Key process equipment consists of:

- Primary crushing plant with a 1.525 m x 2.794 m gyratory crusher
- Secondary crushing plant with two cone crushers prior to the grinding circuit, each powered by one 1,000 kW motor
- SAG/ball mill/pebble crusher grinding circuit:
  - one SAG mill with one 23.5 megawatt (MW) gearless motor drive
  - two ball mills each driven by two 6.5 MW variable speed synchronous motors (26 MW total installed ball mill power), operated in closed circuit with cyclones
  - two cone crushers, each powered by one 750 kW motor
- Flotation circuits:
  - rougher flotation: two parallel trains of two 200 m^3 tank cells
  - rougher-scavenger flotation: two parallel trains of three 200 m^3 tank cells
  - first cleaner flotation: two 100 m^3 tank cells
  - second cleaner flotation: four 30 m^3 tank cells
  - third cleaner flotation: two 30 m^3 tank cells
  - one 4 m x 12 m flotation column
  - cleaner scavenger flotation: five 100 m^3 tank cells
- Regrinding and gravity concentration circuits:
  - one 1,125 kW tower mill operated in closed circuit with six 500 mm and two 375 mm cyclones
  - two 3,000 kW IsaMills™ operated in closed circuit with twenty-six 250 mm cyclones
  - one centrifugal gold concentrator

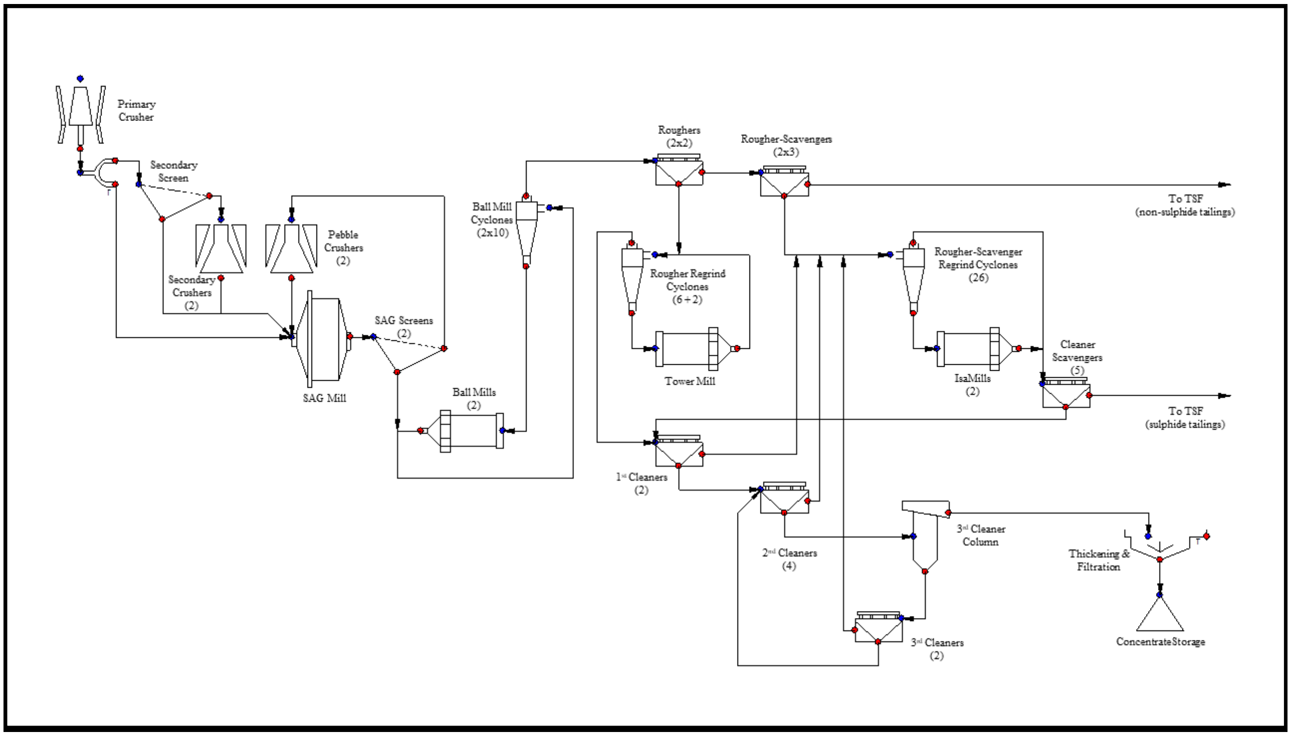
Run-of-mine ore is crushed to a top size of 150 mm. Up to 100% of the gyratory crusher product is diverted to the secondary crushing circuit by a moveable gate. Prior to being introduced into the SAG and ball mill circuits, a variable percentage of the ore, depending on hardness, is sent to the secondary crushing circuit targeting a 60 mm, or finer, product. The SAG discharge screens' oversize is recycled to the pebble crushing circuit targeting a 12 mm, or finer, product. The design for the final feed to flotation from the ball mill circuit is a product size of 80% passing 212 µm. Grinding efficiency improvements in the SAG and ball mill circuits have led to a decreased product size feeding the rougher flotation circuit, averaging 175 µm for 2019 at nominal throughput rates.

The rougher/rougher-scavenger flotation circuit produces a higher-grade rougher concentrate and a lower-grade rougher-scavenger concentrate. These concentrates are separately reground and then upgraded in three cleaner flotation stages to produce a final flotation concentrate targeting approximately 21.5% Cu, containing 30-40 gpt Au.

A gravity circuit is used to scalp coarse gold from the rougher concentrate prior to the cleaning circuit with this material added at the concentrate storage stock tank to the final concentrate produced from flotation. The final flotation concentrate is thickened and stored in the stock tank prior to filtering. The combined concentrate is pressure-filtered to a targeted moisture content of 8.5%, stockpiled, and then trucked to the rail loadout facility in Mackenzie. The concentrate is then railed to North Vancouver where it is loaded onto ships and sent to purchasers located around the Pacific Rim.

The rougher-scavenger tailings, containing mostly non-sulphide gangue minerals, is stored in the tailings storage facility (TSF), while the cleaner scavenger tailings, containing most of the sulphide gangue minerals, is stored in a separate area of the TSF. The latter is kept in a lined pond and underwater to prevent acid generation from the oxidation of the sulphide minerals. The simplified process plant flowsheet is shown in Figure 17-1.

Figure 17-1: Simplified Process Plant Flowsheet



Source: Centerra, 2020

17.2. PROCESS PLANT DESCRIPTION

Crushing

A conventional gyratory crusher facility is used to crush ROM ore at an average rate of 3,500 to 4,000 tpoh. The facility includes a 1.525 m x 2.794 m (60 inch x 110 inch) gyratory crusher which crushes the ore to a top size of 150 mm.

The originally installed coarse ore stockpile feed conveyor was split into two conveyors with a transfer station that allows the material to either feed the original coarse ore stockpile feed conveyor or be diverted to a new conveyor that feeds the secondary crushing circuit.

The secondary crushing circuit consists of a vibrating screen with oversize material conveyed to a surge stockpile and the undersize reporting to the ore stockpile feed conveyor via the secondary crushing circuit product conveyor. The screen oversize material is reclaimed from the surge stockpile and transferred to the secondary crushers. The secondary crusher product is combined with the screen undersize on the secondary crushing circuit product belt and transported to the coarse ore stockpile feed conveyor.

The ore stockpile (COS) has a live capacity of 60,000 t.

Grinding

The original grinding process was a SAG mill, ball mill, and pebble crushing (SABC) circuit designed to process ore at a nominal rate of 2,717 t/h. The installation of the secondary crushing circuit, described above, increased the nominal rate to 2,830 t/h. The grinding circuit includes:

- one 12.20 m diameter x 6.71 m EGL (40' x 22') SAG mill with one 23.5 MW gearless motor drive;
- two 7.32 m diameter x 12.50 m EGL (24' by 41') ball mills each driven by two 6.5 MW variable speed synchronous motors (26 MW total installed ball mill power);
- two cone crushers, each powered by one 750 kW motors;
- two 3.66 m wide x 7.3 m long double-deck (DD) vibrating screen; and
- twenty 840 mm (33") cyclones.

The SAG mill is currently equipped with 70 mm grates to maximize the removal of critical size material. The mill discharge is screened by two vibrating screens with screen oversize transferred to two pebble crushers producing 12 mm top size material. The crushed product is conveyed back to the SAG mill feed conveyor. The vibrating screen undersize, screened on a bottom deck aperture of 12.5 mm, is pumped by two pumps, each with a standby pump, one for each ball mill cyclone cluster. The cyclone underflows flow by gravity to each of the ball mills, while the overflows flow by gravity to the rougher flotation circuit.

Reagents, including lime, PAX (potassium amyl xanthate), and A3409, can be added to the SAG feed chute if necessary. Lime is used for pH control while KAX and A3409 are bulk and selective collectors, respectively.

Flotation

The rougher flotation circuit consists of two trains of five 200 m^3 flotation tank cells. Each train has two rougher and three rougher-scavenger flotation tank cells.

The concentrates from the first two cells of each train (rougher concentrate) and the concentrates from the last three cells of each train (rougher-scavenger concentrate) are reground separately to produce an 80% passing size of 40-50 µm and 18-25 µm, respectively. The rougher and rougher-scavenger concentrates target an overall mass pull of 12%.

Rougher and rougher-scavenger flotation is carried out at natural pH and a slurry density of 35%-40% solids. PAX and A3409 are added to the flotation cells and Polyfroth H28C is used as the frother.

Two separate regrinding circuits are used for the rougher and the rougher-scavenger concentrates, respectively. The major equipment includes:

- One 1,125 kW tower mill operated in closed circuit with six 500 mm and two 375 mm cyclones; and
- Rwo 3,000 kW IsaMills™ operated in closed circuit with twenty-six 250 mm cyclones.

The rougher concentrate is reground to 80% passing 40-50 µm in the tower mill while the rougher-scavenger concentrate together with the first cleaner, second cleaner, and third cleaner flotation tailings are reground to 80% passing 18-25 µm in the IsaMills™.

To recover coarse metallic gold particles, approximately 20% of the rougher concentrate regrind cyclone underflow is diverted to a centrifugal gravity concentrator. The gravity concentrate is currently pumped directly to the copper concentrate stock tank. Gravity concentrator tailings reports to the tower mill for further regrinding.

The reground concentrates undergo three stages of cleaning flotation to produce a final copper concentrate containing approximately 21.5% Cu and 30 to 40 gpt Au. The major equipment currently in use in the cleaner flotation circuit includes:

- Two 100 m^3 first cleaner flotation tank cells;
- Five 100 m^3 first cleaner/scavenger flotation tank cells;
- Four 30 m^3 second cleaner flotation tank cells;
- Two 30m^3 third cleaner flotation tank cells;
- One 4 m x 12 m flotation column.

The reground rougher concentrate is cleaned in the first cleaner tank cells with the concentrate pumped to the second cleaner circuit. The second cleaner flotation concentrate is cleaned in a third cleaner column. The concentrate from the third cleaner column, which is the final concentrate, is pumped to the concentrate thickener. The third cleaner column tailings are recycled to the third cleaner tank cells as a scavenging circuit.

The tailings from the first, second, and third cleaner circuits are combined and pumped to the rougher-scavenger concentrate regrind circuit. The reground rougher-scavenger concentrate together with the combined first, second, and third cleaner tailings are cleaned in the cleaner-scavenger tank cells. The concentrate from these cells is pumped to the head of the first cleaner bank while the tailings are pumped to the sulphide cleaner tailings impoundment area.

For optimum pyrite rejection, the pH is maintained at approximately 11.3 in the cleaner circuit.

For environmental reasons, the first cleaner-scavenger tailings, containing most of the sulphide gangue minerals, is discharged and stored in a separate tailings area located within the TSF and is always kept submerged under water to prevent oxidation of the sulphide minerals.

The reagents used in the rougher flotation circuits are also added to the cleaner flotation stages, at significantly lower dosages. Lime is used to maintain the optimum pH in the cleaning circuit.

Thickening and Dewatering

The final flotation concentrate is thickened to 60%-65% solids in a 12 m diameter high rate thickener. The thickener underflow is pumped to the concentrate stock tank, and then fed to a 96 m^2 pressure filter. The filtered concentrate, containing approximately 8.5% moisture, is conveyed to the concentrate storage shed. Concentrate is transported by truck to a rail loadout facility located in Mackenzie. The concentrate is then railed to North Vancouver where it is loaded onto ships and sent to purchasers located around the Pacific Rim.

Tailings

Two tailings streams, the rougher-scavenger tailings and the cleaner-scavenger tailings, are deposited and stored in separate tailings storage areas within the TSF. The tailings pond supernatant is recycled to the process plant for re-use.

A tailings pump station will be installed in 2021 or 2022 to aid the transportation of the rougher-scavenger tailings. Starting in 2013 and to date, the tailings have been gravity-flowing to the tailings pond.

Tailings embankment construction and management is further detailed in Section 18.

Reagents

The reagent preparation and storage facility is located within a spill containment area within the process plant building, designed to accommodate 110% of the content of the largest tank. The storage tanks and reagent systems are equipped with instrumentation and systems to enhance safety and control.

The collectors include A3409, a specialty gold collector containing 30-40% sodium diisobutyldithiophosphinate supplied in bulk tanker deliveries, and a solid type collector, PAX, shipped to the mine site in supersacks. Polyfroth H28C is used as the frother and is shipped as liquid in bulk tankers. Flocculant, added to the thickener, is used for dewatering and is prepared in a wetting and mixing system. Pebble lime, used for pH control, is delivered by bulk tanker trucks and prepared in a slaking system to produce a 15% solids slaked lime slurry distributed throughout the process plant.

New reagents are occasionally tested to enhance metal recovery and concentrate grading. These reagents are handled in accordance with regulatory requirements.

Assay and Metallurgical Laboratory

The on-site assay laboratory is equipped with necessary analytical instruments to provide all routine assays for the mine, the process plant, and the environment department. The most important of these instruments includes:

- an atomic absorption spectrophotometer (AA);
- a mineral liberation analyzer (MLA); and
- a Leco furnace.

The metallurgical laboratory has equipment to conduct all necessary test work to monitor metallurgical performance and to improve the process flowsheet and efficiency.

Site Water and Air

Two separate water supply systems for fresh water and process water are provided to support the operation. Fresh and potable water is supplied to storage tanks from three ground water wells, located approximately 1 km south of the plant site.

Fresh water is used primarily for the following:

- Firewater for emergency use;
- Reagent preparation;
- Dust suppression; and
- Potable water supply (treated by chlorination and ultraviolet lamps).

The fresh-water tank, by design, is full at all times and provides at least two hours of firewater pumping volume in case of an emergency.

Process water consists primarily of reclaim water from the TSF and the copper concentrate thickener overflow, fresh make-up water that is supplied from the Philip Lake and Meadows Creek pump stations, and water from the mine dewatering wells.

Separate air service systems supply air to flotation, filtration, crushing and general plant services.

Power Consumption

The 2019 total site power consumption was 535,708 MWh, or 32.76 kWh/t of ore processed. Just under 75% of the electrical power for the operation is consumed by the plant operations of rock crushing and grinding. Table 17-1 breaks down power consumption by area for 2019.

Table 17-1: 2019 Power Consumption (by area)

Unit Operation	**Power Consumption**	
	% of Total	**MWh/a**
Crushing/Conveying	2.0%	10,960
Pebble Crushing	3.4%	18,106
SAG mill grinding	33.4%	178,859
Ball mill grinding	29.1%	155,729
Grinding pumps	6.1%	32,687
Flotation/Thickening	7.4%	39,652
Regrinding/Classification/Gravity	7.3%	39,109
Fresh Water/Reclaim Water	4.5%	23,903
Assay/Met Lab	1.5%	8,245
Mining	3.9%	20,845
Admin/Workshop	1.4%	7,612
Total	**100%**	**535,708**

17.3. PROCESS CONTROL AND INSTRUMENTATION

The plant control system consists of a Distributed Control System (DCS) with PC-based Operator Interface Stations (OIS) located in two separate control rooms: one in the primary crusher station and one in the main administration building, alongside the mine dispatch centre. The plant control rooms are staffed by trained personnel 24 hours per day.

18. PROJECT INFRASTRUCTURE

18.1. PROJECT INFRASTRUCTURE

Mount Milligan Mine is accessible by commercial air carrier to Prince George, British Columbia, then by vehicle from the east via Mackenzie on the Finlay Philip Forest Service Road and the North Philip Forest Service Road, and from the west via Fort St. James on the North Road and Rainbow Forest Service Road. Road travel to Mount Milligan Mine is 254 km from Prince George. The forestry-based communities of Mackenzie and Fort St. James are within daily commuting distance of the mine, and both of these communities are serviced by rail.

The infrastructure at Mount Milligan Mine includes a concentrator, a TSF and reclaim water ponds, as described elsewhere in this report, an administrative building and change house, a workshop/warehouse, a permanent operations residence, a first aid station, an emergency vehicle storage, a laboratory, and sewage and water treatment facilities. The power supply is provided by B.C. Hydro via a 91 km hydroelectric power line. Concentrate is transported by truck from the Project site to Mackenzie, transferred onto railcars of the Canadian National Railway to existing port storage facilities of Vancouver Wharves in North Vancouver and loaded as lots into bulk ore carriers. Concentrate is then shipped to customers via ocean transport.

18.2. TAILINGS STORAGE FACILITY

The TSF at the Mount Milligan Mine is designed to store tailings solids and potentially acid generating (PAG) and oxide/weathered waste rock materials in designated areas. The TSF embankment is constructed as a centreline dam using open pit overburden and non-acid generating (NAG) waste rock materials. The TSF is comprised of two dams: the Main Embankment and the West Separator Berm (WSB). The dams will eventually join as additional raises are completed. The highest portion of the TSF embankment is located in the King Richard Creek valley and is approximately 54 meters in height as measured from crest to downstream toe. The embankments are zoned earthfill structures comprised of a low permeability glacial till core zone, and appropriate filter and transition zones to mitigate piping of the core zone material. The TSF embankment core zone is keyed into the low permeability glacial till in the foundation.

The Main embankment is subdivided into segments designated: South, Southeast, Northeast, and North Dam. The South Dam is situated across the King Richard Creek valley; the Southeast/Northeast Dams are along the eastern plateau towards the Esker Lakes; the North Dam is constructed through the esker deposit. The WSB is constructed along the western edge of the impoundment providing containment between the TSF and the Open Pit. The WSB will be extended towards the south and north until it connects into the Main embankment creating a continuous impoundment.

The maximum embankment height will eventually be 85 meters at the South Dam across King Richard Creek valley. The current constructed height at the South Dam is approximately 54 m. Construction of each of the embankment stages is scheduled to correspond with material availability from the Open Pit and the projected rate of rise. Construction of the Stage 1 TSF and ancillary facilities commenced in December 2010 and was substantially completed in July 2013 to allow for operations to discharge tailings into the impoundment. Stage 6 is planned for 2020 with Stages 7-14 planned as annual raises based on the current mine schedule and planning. General arrangements illustrating the staged development together with a typical embankment cross-section are shown on Figure 18-1 to Figure 18-4.

Figure 18-1: Stage 6 TSF General Arrangement

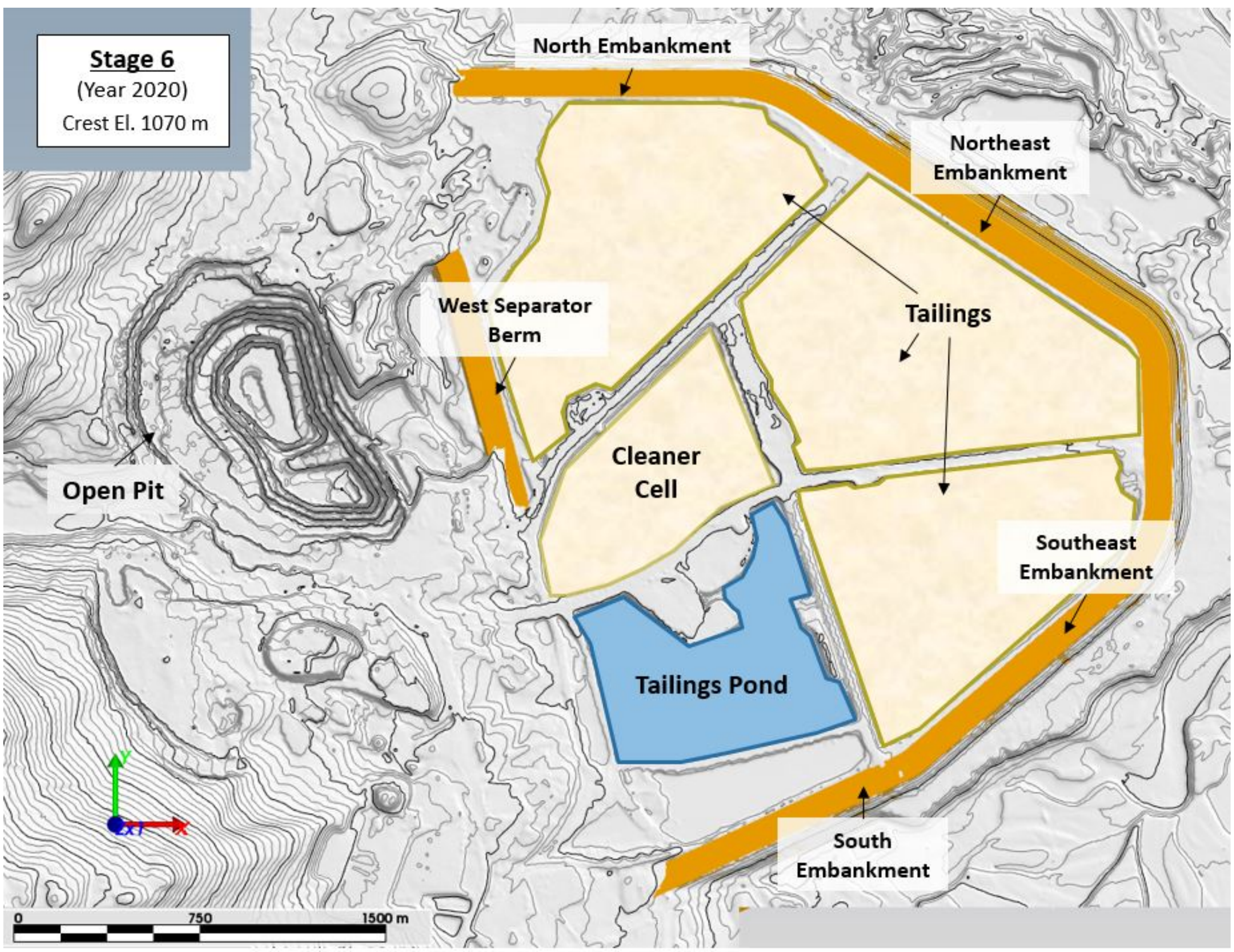


Figure 18-2: Stage 10 TSF General Arrangement

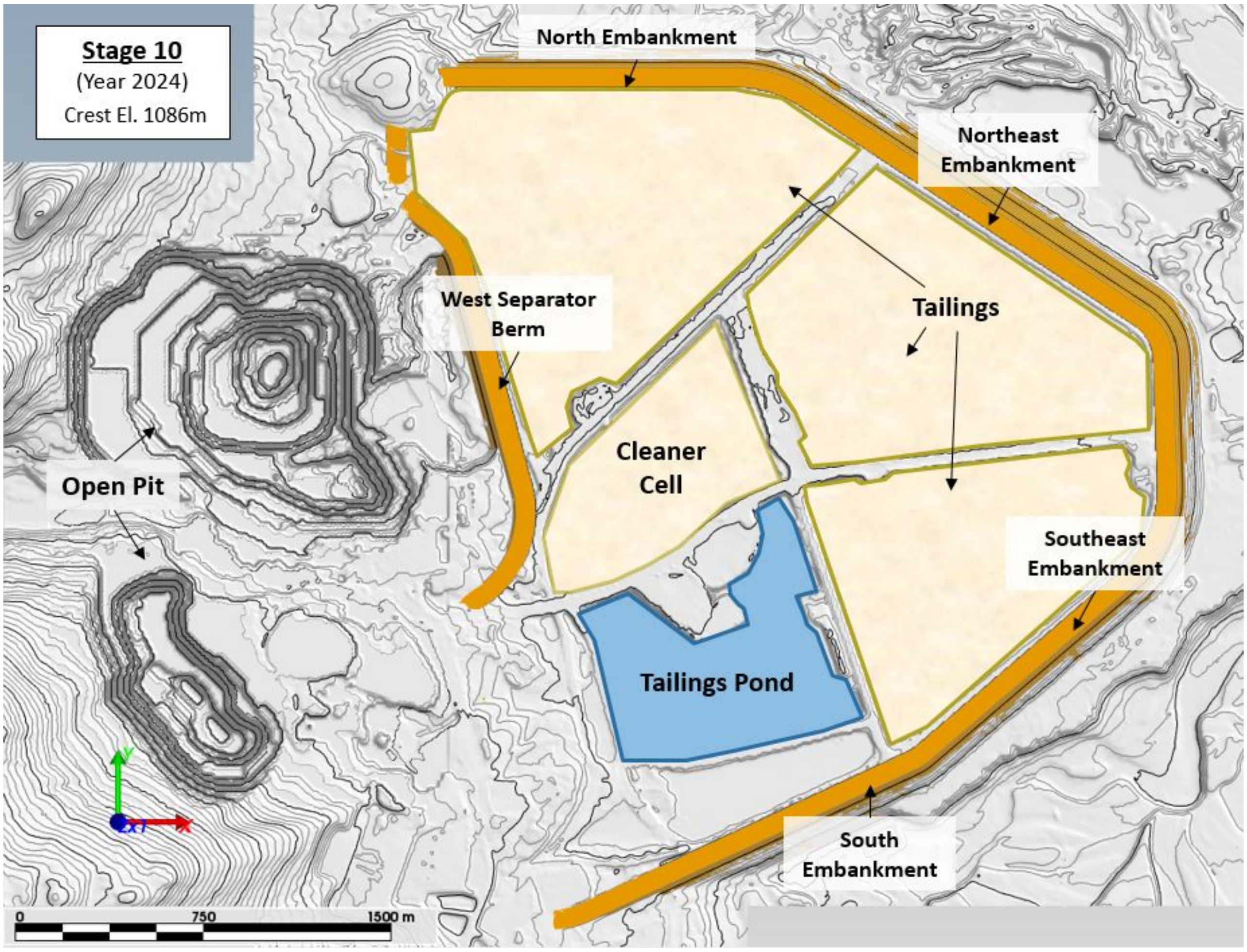


Figure 18-3: Stage 14 TSF General Arrangement

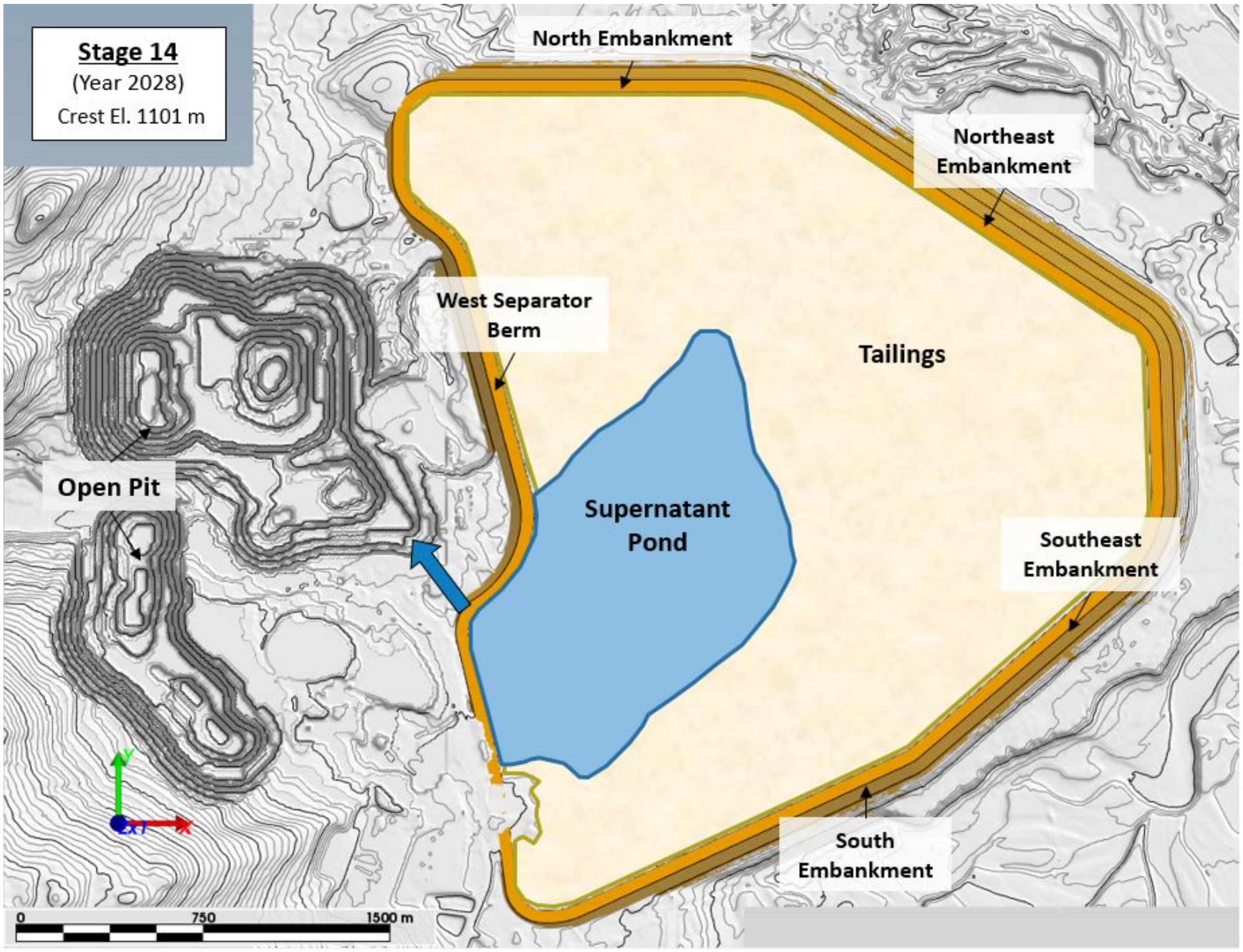
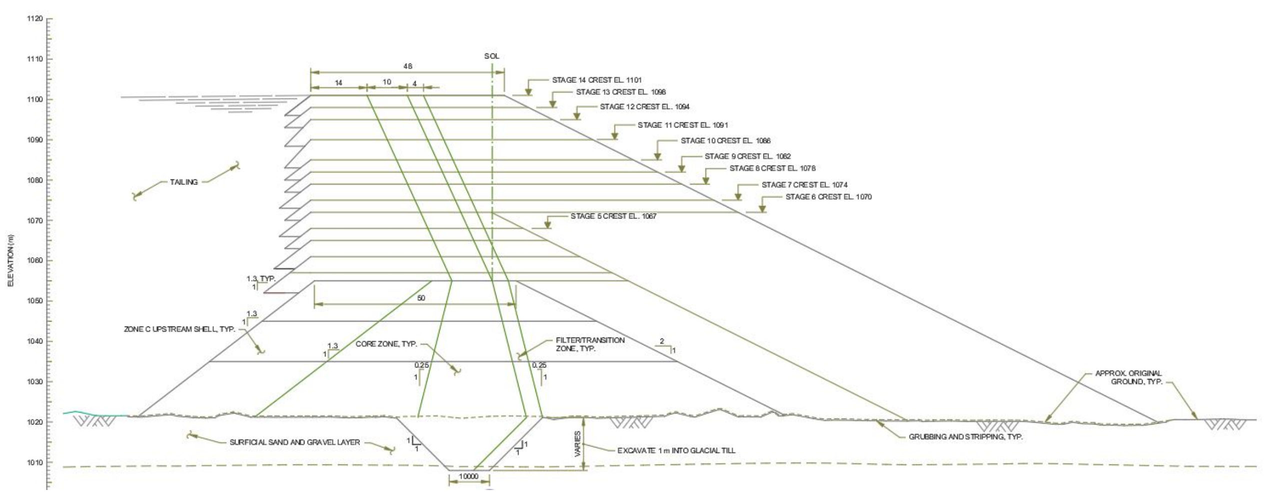


Figure 18-4: Typical TSF Embankment Section



A summary of the remaining stages and required construction materials are provided in Table 18-1.

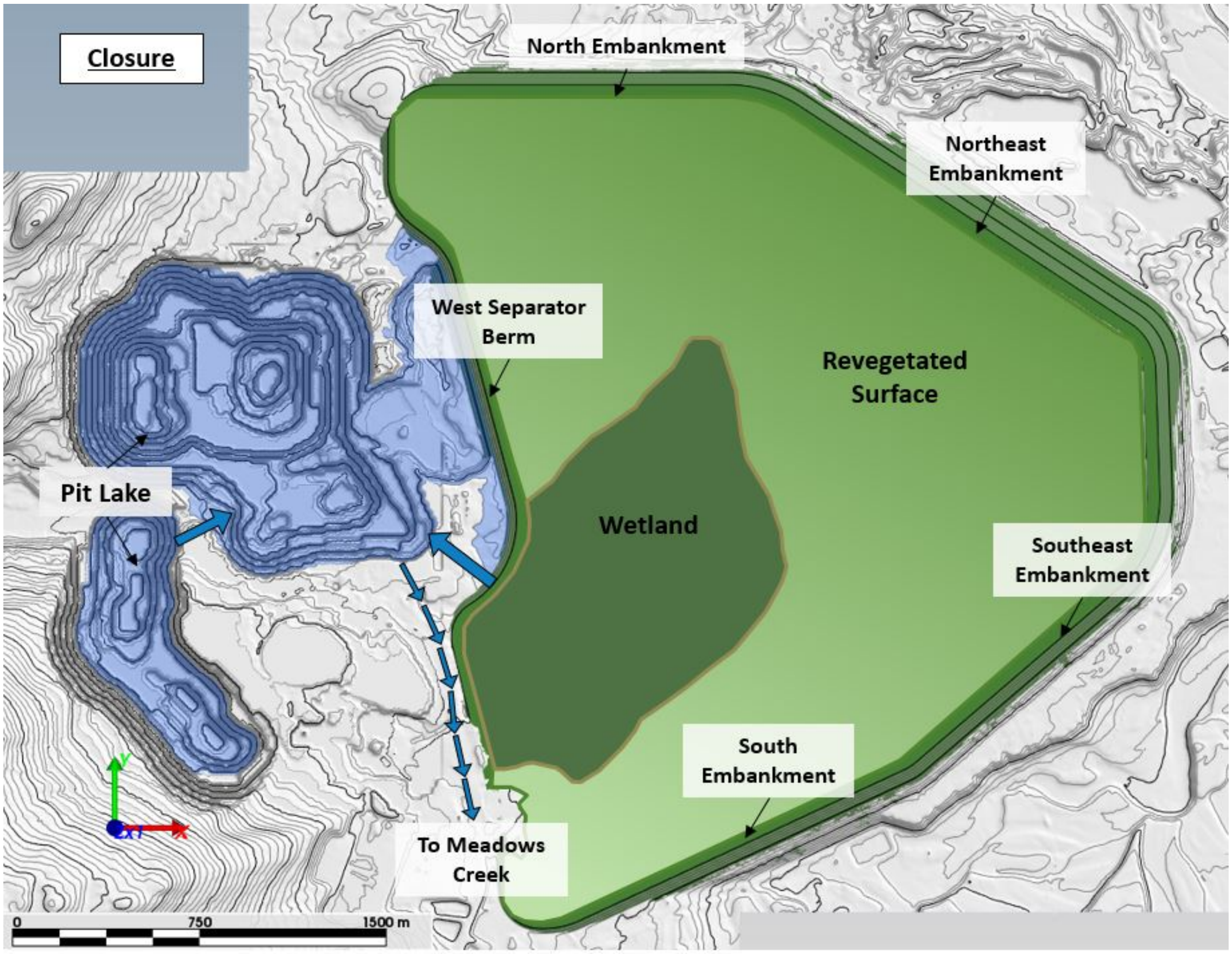
Table 18-1: TSF Embankment Staging and Construction Material Summary

Tailings Embankment and West Separator Berm Combined									
Stage	Year	Elevation (m)	Cut-off Trench		Embankment				Total (m³)
			Core (m³)	Zone F (m³)	U/S Zone C (m³)	Core (m³)	Zone F/T (m³)	D/S Zone C (m³)	
6	2020	1,070	0	0	608,000	217,000	87,000	3,730,000	**4,642,000**
7	2021	1,074	21,000	1,900	806,000	304,000	122,000	1,960,000	**3,214,900**
8	2022	1,078	25,000	2,200	796,000	322,000	129,000	1,880,000	**3,154,200**
9	2023	1,082	20,000	1,800	774,000	337,000	135,000	1,750,000	**3,017,800**
10	2024	1,086	19,000	1,700	743,000	350,000	140,000	1,610,000	**2,863,700**
11	2025	1,091	34,000	3,100	901,000	469,000	188,000	1,820,000	**3,415,100**
12	2026	1,094	19,000	1,700	510,000	292,000	117,000	950,000	**1,889,700**
13	2027	1,098	24,000	2,100	647,000	406,000	163,000	1,100,000	**2,342,100**
14	2028	1,101	8,000	700	444,000	309,000	124,000	670,000	**1,555,700**
Total (m³)			**170,000**	**15,200**	**6,229,000**	**3,006,000**	**1,205,000**	**15,470,000**	

Scavenger tailings are discharged from pipelines into the TSF along the South, North East and Southeast embankments and WSB. Cleaner tailings are discharge from a separate pipeline into the cleaner tailings cell. The coarse fraction of the tailings is expected to settle rapidly and will accumulate closer to the discharge points, forming a gentle beach. Finer tailings particles will travel further and settle at a flatter slope adjacent to and beneath the supernatant pond. Beaches are developed to maximize storage volume and to control the location of the supernatant pond. Selective tailings deposition is used to maintain the supernatant pond away from the embankments to reduce seepage and ensure that reclaimed water is clear and accessible for reuse in the process plant. The tailings deposition strategy is expected to be modified towards the end of mine life. The intention is to shift the cleaner tailings discharge to the open pit for the last eight months of processing operations so that a scavenger cap of tailings can be placed over the cleaner tailings cell and the PAG cell so that these deposits will remain saturated.

At closure the tailing pond will be drained to the open pit via a constructed overflow channel. A wetland will be established above the western end of the cleaner cell and, where accessible, will be planted with native wetland plants. This wetland will result in reducing conditions in its bottom sediments over part of the cleaner tailing preventing oxygen transfer to the cleaner tailing. The wetland will polish surface runoff over the scavenger tailing surface, and after the pit lake fills, will polish the pit lake overflow. A 0.3m layer of topsoil will be placed onto the accessible portion of the scavenger tailing surface (i.e., beach adjacent to the tailings dam) and wetland area, if feasible, to facilitate revegetation. The TSF closure general arrangement is shown in Figure 18-5.

Figure 18-5: TSF Closure General Arrangement



18.3. WATER MANAGEMENT

Mount Milligan Mine was designed and permitted as a zero-discharge facility during operation. The objective of this design is to prevent discharge of surface contact water from the Mine to the receiving environment. Water required for ore processing is supplied via the reclaim barge from the TSF. The objective for water volume in the TSF is to maintain a minimum volume of approximately 5Mm^3 as well as to store the inflow design flood without discharge during operations. Based on the LOM mine plan, the annual water demand for operation is between 8Mm^3 and 10Mm^3, which is a function of the ore processing rate and climatic conditions.

The operation originally relied on four main sources of water to operate the process plant:

- Surface runoff into the TSF catchment;
- Direct precipitation onto the TSF;
- Water pumped to the TSF from Meadows Creek, and
- Recycling of process water.

To address recent shortages however, the company obtained authorization from the BC Environmental Assessment Office (EAO), Ministry of Energy, Mines and Petroleum Resources (EMPR) and Forest, Lands and Natural Resources Office (FLNRO) for water extraction permits and associated infrastructure that allowed surface water withdrawals from Meadows Creek during freshet 2018 and from Rainbow Creek and Philip Lake 1 until November 2021. The mine has also obtained additional water licenses to extract groundwater from the Meadows Creek Well Field and the Lower Rainbow Wellfield until December 2023.

The groundwater resources are not yet proven over the long-term (i.e. life-of-mine), and the mine is in the process of securing permits for one or more long-term surface water sources from local water courses as further supply to meet its operational requirements; which will include additional infrastructure such as pumps and piping, electrical power supply (transmission line) and mine boundary expansion.

To ensure adequate water supply for the life-of-mine, the mine is actively investigating additional long-term water sources. On-going investigation activities include hydrological and hydrogeological desk-top analysis and field work, environmental impact assessment, community consultation and engineering evaluation. At the time this report was prepared, the mine is considering multiple sources, including surface water from the Nation River and Philip Creek. However, to remain flexible as additional analysis is ongoing, the potential for long-term water to be provided from a combination of one or more of the following is being evaluated: Philip Lake 1, Rainbow Creek and Meadows Creek and groundwater from various wellfields. Due to the variability of, among other things, environmental factors, including the amount of water available from such potential long-term water sources, climatic conditions as well as the relative costs of such long-term solutions, there can be no assurance that the long-term water solution ultimately selected by the mine will be able to supply all its water needs in all future circumstances.

The following Table 18-2 provide a summary of the currently permitted water sources and the potential long-term water source options under investigation.

Table 18-2: Surface and Groundwater Sources

	Surface and Groundwater Sources	Estimated annual volume	Expire Date
Currently Permitted Sources	EAC #M09-01, Amendment # 6- additional surface water sources from Philip Lake 1, Rainbow Creek and Meadows Creek	4.5 -12.6 Mm^3	30-Nov-21
	Groundwater license #501625 from the Lower Rainbow Well Field	Up to 3.8 Mm^3	31-Dec-23
Potential Long-Term Sources	Nation River	>10 Mm^3	LOM
	Philip Creek	10 Mm^3	LOM
	Combined surface water sources including, Philip Lake 1, Rainbow Creek and Meadows Creek.	4.5-10 Mm^3	LOM
	Extension of the groundwater license #501625 from the Lower Rainbow Wellfield	Up to 3.8 Mm^3	LOM
	Other groundwater targets	TBD	LOM

19. MARKET STUDIES AND CONTRACTS

19.1. MARKETABILITY

Mount Milligan Mine is strategically located for delivery to Asian custom smelters and the concentrate analysis is low in deleterious impurities such as arsenic, antimony, bismuth, chlorine, and fluorine. The presence of significant gold and payable silver values has been welcomed by custom smelters and has proven to be a positive factor when negotiating sales contracts, particularly with South Korea, Philippines and Japan. Historically, smelters located in China have offered less favourable terms for the recovery of gold and thus is a less attractive market for the Mount Milligan concentrates, however Chinese gold payables have improved in recent years. Delivery to North American smelters is also a possibility; however, the logistic costs are not as favourable as bulk transport to Asia so this would be an unlikely outcome.

19.2. CONTRACTS

Concentrate Sales

Copper-gold concentrate produced by the Mount Milligan Mine is sold to various smelters and off-take purchasers. TCM is currently party to four multi-year concentrate sales agreements for the sale of copper-gold concentrate produced at Mount Milligan Mine. Pursuant to these agreements, TCM has agreed to sell an aggregate of approximately 130,000 t in 2020, 100,000 t in 2021 and 40,000 t in 2022.

Pricing under these concentrate sales agreements will be determined by reference to specified published reference prices during the applicable quotation periods. Payment for the concentrate will be based on the price for the agreed copper and gold content of the parcels delivered, less smelting and refining charges and certain other deductions, if applicable. The copper smelting and refining charges are negotiated in good faith and agreed by the parties for each contract year based on terms generally acknowledged as industry benchmark terms. The gold refining charges are as specified in the agreements.

Remaining concentrate produced at the Mount Milligan Mine will be sold under short-term contracts or on a spot basis. TCM may also choose to enter into another multi-year concentrate sales agreement, if appropriate, given the mine’s expected concentrate production.

The commitments under the current multi-year concentrate sales agreements cover approximately 65% of the expected concentrate production in 2020 and are also less than the total expected production in 2021 and beyond. TCM intends to either extend the current multi-year agreements as the terms expire or may enter into additional multi-year sales agreements. To the extent that production is expected to exceed the volume committed under these agreements, TCM will sell the additional volume under short-term contracts or on a spot basis.

Stream Agreement with Royal Gold

Pursuant to the Stream Agreement with Royal Gold, TCM has agreed to sell to Royal Gold 35% of the gold produced and 18.75% of the copper production at the Mount Milligan Mine. Royal Gold pays $435 per ounce of gold delivered and pays 15% of the spot price per metric tonne of copper delivered. When the Stream Agreement was originally entered into by TCM (prior to the Acquisition by Centerra), TCM received an upfront payment of $781.5 million for the rights to receive future gold production. The Stream Agreement was amended by TCM as part of the Acquisition to reduce the percentage of gold production allocated to Royal Gold (from 52.25% to 35%) and to include 18.75% of the copper production from the Mount Milligan mine.

The Stream Agreement has an initial 50-year term, with automatic successive 10-year renewal periods.

TCM sells copper and gold concentrate from Mount Milligan Mine to customers and in connection with such sales, TCM purchases gold ounces and copper warrants in the market for delivery to Royal Gold in an amount based on a portion of the gold and copper content contained in the copper and gold concentrate sold to customers.

The Stream Agreement covers substantially the entire Property.

The Stream Agreement includes certain restrictions on assignment or transfer of the respective rights of both parties to the Stream Agreement.

20. ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

Environmental programs and initiatives are essential to mine success. Mount Milligan Mine's environmental permit requirements are implemented in accordance with conditions of the permits and other regulatory approvals. The Mount Milligan Mine was specifically designed with a limited spatial footprint and environmental impacts generally occur within that footprint. TCM continues to interact with local communities, Indigenous groups and stakeholders to educate on and improve its environmental performance.

All design, engineering, construction, operation and management of mine facilities and components incorporates criteria for responsible management of process flows, effluent and waste products to meet established capture and containment guidelines and permit requirements. Environmental protection, as well as operational safety and maintenance requirements, are incorporated into the design of the plant processes, as well as in the transportation, storage, and disposal of materials within and outside the boundaries of the plant.

As part of the Environmental, Health and Safety Management Systems, all employees and contractors working on the site are provided environmental orientation training to develop awareness of, and facilitate compliance with, site environmental requirements.

20.1. ENVIRONMENTAL ISSUES

Environmental impacts and their associated mitigations were considered at the beginning of the original Project plan and further assessment as mine plans were updated and changed. These assessed impacts and mitigations have been approved in various certificates, permits and management plans with implementation either complete or significantly underway.

This updated plan contains no activities that change the existing environmental considerations or materially alter the success of any mitigations that are currently underway or in place. A long-term water plan for life-of-mine operations is being developed and has been applied for.

20.2.WASTE DISPOSAL, MONITORING AND WATER MANAGEMENT

The pit waste material is being used for the construction of the TSF and WSB embankments, the PCC and the PAG Separator Dyke (which separates the cleaner and scavenger tailing areas). Material remaining after construction requirements are met in any time period will be placed in storage areas that meet long-term containment requirements.

The main TSF embankment is constructed in stages using annual raises throughout the LOM, from low permeability glacial till, overburden and waste rock materials from stripping operations at the open pit and borrow areas within and near the TSF. With the use of overburden and NAG waste rock for downstream TSF embankment construction this eliminates the need for conventional waste rock dumps. Delivery of PAG and oxide/weathered waste rock to the interior of the TSF and Main Zone pit, once depleted, ensures secure underwater disposal.

Tailings from the mill are being delivered by gravity to the TSF for as long as possible. Each delivery pipeline has been sized to carry up to 100% of the design scavenger tailing production from the circuit. One of the three delivery pipelines is required for use at all times while allowing for maintenance work to be completed on the other two pipelines. Discharge into the TSF will be from valve controlled off-takes along the pipeline.

A series of pumps transfer water from active mining bench sumps for discharge into the TSF. The Project was implemented, prior to construction and continuing in operations, surface water and groundwater monitoring to ensure that there is no off-site impact of the mining operation. Additional hydrological monitoring has been implemented to ascertain if pumping programs (both surface and groundwater) from operations have an impact on local water supplies and the environment. The associated adaptive management plan includes a Trigger Action Response Plan (TARP) to manage pumping volumes if an attributed change is noted in the environment. The Project generates an annual water quality report that describes the monitoring program and results for a variety of parameters.

As per a requirement of the Mines Act permit (M-236), Centerra-TCM submitted a revision to its 5-Year Reclamation Plan in 2019. Pursuant to the recent revision to the Reclamation Plan, the most recent estimate of the undiscounted and uninflated cost of reclamation is $31.5 million (C$40.9 million). The Mount Milligan Mine currently has a posted reclamation bond in the amount of $40 million (C$52 million).

Environmental and other management plans are reviewed and updated as necessary and are submitted to applicable regulatory ministries for review and or approval as part of an adaptive management process.

20.3. PROJECT PERMITTING

All necessary permitting requirements to operate the mine have been applied for and have been approved by the applicable regulatory agencies. Additional requirements such as long-term water supply is in the application stages with an expectation of operational implementation of the infrastructure by 2022; see section 18.3 for more details.

20.4. SOCIAL OR COMMUNITY REQUIREMENTS

Consultation with local communities and impacted Indigenous groups is required for all new or amended mine permits. Permit-specific consultation strategies are developed and implemented whenever operational changes requiring either new or amendment permits are contemplated for the mine. These strategies outline the manner in which the Company will engage with local communities and impacted Indigenous groups on current or proposed permitting activities.

In addition, the mine's Community Sustainability Committee formed in 2008 and made up of local community stakeholders and Indigenous groups, acts as TCM's primary mechanism for ongoing engagement concerning the mine's activities and investments into the region. A Sustainability Management Plan for the mine, which was reviewed and approved by the Committee, remains in place.

TCM is party to a Socio-Economic Agreement with the McLeod Lake Indian Band and an Impact Benefit Agreement (IBA) with Nak'azdli Whut'en. Both agreements commit TCM to the provision of financial payments and these amounts have been incorporated into the economic analysis in this Technical Report.

21. CAPITAL AND OPERATING COSTS

The estimated future capital and operating costs for Mount Milligan are discussed and tabulated below in US dollars using an exchange rate of 1.25 US$/CA$, however all cost estimates were developed in Canadian dollars. Please see section 22 of this document for sensitivity analyses.

Total operating and capital costs over the LOM are estimated at $2,839 million, including $828 million for mining costs, $1,029 million for processing costs, $333 million for administrative (G&A) costs and $140 million for transportation costs, total selling and marketing costs are estimated at $88 million, total treatment and refining charges are estimated at $199 million and total capital expenditures are estimated at $222 million, as shown in Table 21-1.

The total LOM capital expenditures required to exploit the Mineral Reserves in the LOM plan is estimated at $222 million, which includes capital equipment and component replacements, planned improvements to crushing equipment, the tailings pumping system, and site facilities, as well as water management, but excludes $125 million TSF construction costs (included with mining opex). Waste mined at Mount Milligan is used for routine TSF raises, the cost of which is capitalized to the TSF rather than as capitalized stripping. The current mine plan does not contemplate any growth capital.

The all-in sustaining cost per ounce sold, on a by-product basis, which includes sustaining capital and copper revenue credits, averages $704/oz of gold for the period from 2020 to the end of the LOM. All-in sustaining cost per ounce sold, on a by-product basis, is a non-GAAP financial performance measure.

Table 21-1: Operating and Capital Cost Summary

Costs Summary (Total LOM)	**Total $M**
Mining	828
Processing (including water supply)	1,029
Admin	333
Transportation	140
Selling and Marketing	88
Treatment and Refining	199
Capital	222
Total	**2,839**

21.1.MATERIAL ASSUMPTIONS

The following material assumptions have been used in the LOM plans, estimates of operating and capital costs and Mineral Reserve estimates:

- A gold price of $1,250/oz,
- A copper price of $3.00/lb,
- Exchange rate: US$1.00:C$1.25;
- Diesel fuel price assumption: US$0.85/litre delivered to site.
- Electrical Pricing of US$0.052/kWh.

The diesel price includes British Columbia taxes and freight to the onsite tank farms at Mount Milligan. Diesel fuel is sourced from the Prince George distribution terminal and correlates well with world oil prices.

21.2.CAPITAL COSTS

The LOM capital costs for the Mount Milligan mine are summarised in Table 21-2.

Table 21-2: LOM Capital Costs

Capital Category	Total ($'000)
Mining Capital	
Mine Equipment Replacement	32,820
Equipment Component Replacement	69,280
Mine Sustaining Capital	1,326
Sub Total Mining	**103,425**
Processing Capital	
Crushing Equipment Improvement	4,000
Process Plant Pumping System Facility Improvement	9,586
Water Management Capital	8,045
Process Plant Sustaining Capital	23,309
Sub Total Processing	**44,940**
Admin Capital	
Site Facility Improvement and Other Administration	3,946
Long-Term Agreements requiring capitalization (IFRS16)	69,190
Sub Total Mining	**73,136**
Total	**221,501**

The LOM capital spending has been included in the cash flow model.

The capital cost estimate assumes the replacement of four and the addition of five 227t haul trucks and some auxiliary equipment. Major component rebuilds of the mobile fleet has been estimated based on expected operating hours per component.

Also included in the sustaining capital estimate is an upgrade to the secondary crusher to winterize it, the installation of a tailings pumping system to replace the current gravity system and upgrading of the water supply infrastructure.

21.3. OPERATING COSTS SUMMARY

Operating costs were developed from first principles, using site historic costs as a basis for calibrating the operating cost models, for a nine-year mine life. This includes detailed estimates of personnel for all required roles/functions. Total LOM operating costs are summarized in Table 21-3.

Table 21-3: Operating Cost Summary

LOM Operating Costs Summary	LOM Cost ($'000)	$/tonne
Mining	828,477	4.34
Milling	1,028,532	5.38
Administration	333,434	1.75
Total	**2,190,442**	**11.47**

All unit costs have been calculated per tonne processed. Unit mining costs are $2.00/tonne mined.

21.4. MINE OPERATING COSTS

Key inputs for the mining costs are labour, maintenance, fuel, drill and blast consumables and mine general, and the costs are summarised by cost category in Table 21-4 and by activity in Table 21-5.

Table 21-4: Mining Costs by Category

Cost Category	LOM Cost ($'000)	$/tonne
Labour	309,559	0.75
Parts and Spares	204,512	0.49
Fuel	184,234	0.45
Drill and Blast consumables	99,666	0.24
Mine General	30,507	0.07
Total	**828,477**	**2.00**

Note: Unit mining rate is based on material mined.

Table 21-5: Mining Costs by Activity

By Activity	LOM Cost ($'000)	$/tonne
Hauling	239,920	0.58
Road and Dump	163,620	0.40
Drill and Blast	155,320	0.38
Loading	77,930	0.19
Mine Admin	75,063	0.18
Mine Shops	95,214	0.23
Mine General	21,410	0.05
Total	**828,477**	**2.00**

Note: Unit mining rate is based on material mined.

Open pit mining for the Mount Milligan Mine will provide process plant feed at a nominal rate of 60,000 t/d.

The total LOM operating cost per tonne of material mined is $2.00/tonne.

21.5. PROCESSING OPERATING COSTS

The processing costs are derived using a process plant feed rate of 60,000 tonnes per day. Main inputs for the processing costs are electricity, grinding and reagents, labour, maintenance materials, water sourcing and contract labour, and the costs are summarised by cost category in Table 21-6 and by activity in Table 21-7.

Table 21-6: Processing Costs by Category

Cost Category	LOM Cost ($'000)	$/tonne
Electricity	276,689	1.45
Grinding and Reagents	237,799	1.24
Labour	195,130	1.02
Parts and Spares	126,540	0.66
Contract Labour Services	80,016	0.42
Liners	97,134	0.51
Professional Services	11,155	0.06
Processing General	4,068	0.02
Total	**1,028,532**	**5.38**

Table 21-7: Processing Costs by Activity

By Activity	LOM Cost ($'000)	$/tonne
Grinding	584,617	3.06
Crushing	125,927	0.66
Water Sourcing	51,132	0.27
Flotation and Thickening	50,760	0.27
Processing Admin	29,174	0.15
Maintenance Shop and General	186,922	0.98
Total	**1,028,532**	**5.38**

Unit cost is calculated based on material processed.

Consumable costs were based on current contract pricing and historical consumption rates. Water management costs are included to extract process water from nearby sources based on historical experience and near-term expectations.

Plant maintenance has been factored in for mechanical, electrical and instrumentation together with an allowance for outside contractors to perform major shutdowns. Overall plant availability is estimated to be 94%.

The total LOM unitized material processing cost is $5.38/tonne processed.

21.6. ADMINISTRATION COSTS

The administration costs for the operation include all site support, camp, bussing, insurances and general administration as well as an administrative office in Prince George. Administration costs are depicted by cost category in Table 21-8 and by activity in Table 21-9.

Table 21-8: Administration Costs by Category

Cost Category	LOM Cost ($'000)	$/tonne
Labour and Contractors	134,646	0.70
Site Services and Freight	20,644	0.11
Bus Services	22,307	0.12
Property Tax	15,587	0.08
Rentals	2,242	0.01
Camp Charges	41,429	0.22
Professional Services	14,195	0.07
IT Supplies	17,816	0.09
Insurance	20,771	0.11
Admin General	43,797	0.23
Total	**333,434**	**1.75**

Unit cost is calculated based on material processed.

Table 21-9: Administration Costs by Activity

By Activity	LOM Cost ($'000)	$/tonne
Property Taxes and Insurances	21,725	0.11
Site Admin, HR and IT	96,357	0.50
Camp Services	61,530	0.32
Purchasing/Warehouse	28,412	0.15
Crew Transportation	26,081	0.14
Safety	20,752	0.11
Environment	23,464	0.12
Site Facilities Maintenance	30,368	0.16
Sustainability	23,241	0.12
General	1,504	0.01
Total	**333,434**	**1.75**

Unit cost is calculated based on material processed.

The total LOM administration cost per tonne of material processed is $1.75/tonne.

22. ECONOMIC ANALYSIS

The material assumptions used for the calculations presented in this section have been stated in Section 21.

22.1.LOM PROCESSING

Table 22-1 provides an annual summary for the ore processing operation and resultant concentrate and metal production. Total ore tonnes processed is aligned with the Mineral Reserve estimate, so including no portion of Mineral Resources exclusive of Mineral Reserves. Metallurgical recoveries are based on recovery formulae discussed in section 13. Payable metals are based on typical smelter terms.

Table 22-1: Processing Schedule

Item	Units	Total	2020	2021	2022	2023	2024	2025	2026	2027	2028
Ore processed - total	Mt	**191.0**	21.1	21.9	21.9	21.9	21.9	21.9	21.9	21.9	16.7
Ore processed - direct from pit	Mt	**171.6**	18.1	19.5	19.8	19.8	20.1	19.5	21.9	21.2	11.8
Ore processed - from stockpile	Mt	**19.4**	3.0	2.4	2.1	2.1	1.8	2.4	0.0	0.7	4.9
Cu feed grade	%	**0.23**	0.25	0.23	0.27	0.21	0.23	0.21	0.27	0.20	0.18
Au feed grade	g/t	**0.39**	0.40	0.47	0.38	0.40	0.39	0.35	0.36	0.33	0.47
Py:Cpy feed ratio	ratio	**9.6**	3.9	7.8	8.3	7.5	11.2	9.4	5.6	12.2	24.1
Concentrate produced DMT	ktonnes	**1,632**	203	187	224	168	191	169	226	156	108
Cu recovery	Mlb	**80.6%**	81.9%	81.0%	82.3%	80.3%	81.1%	80.3%	82.4%	77.0%	76.4%
Cu recovered	Mlb	**773.3**	96.2	88.8	106.1	79.6	90.4	79.9	107.3	73.9	51.0
Cu Payable Produced	Mlb	**735.6**	91.5	84.5	100.9	75.7	86.0	76.0	102.1	70.3	48.5
Au recovery	koz	**61.8%**	63.3%	64.2%	63.0%	63.3%	63.1%	62.5%	62.6%	55.2%	57.1%
Au recovered	koz	**1,487.5**	171.7	212.0	169.9	180.2	171.7	154.4	156.6	127.7	143.1
Au Payable Produced	koz	**1,450.3**	167.5	206.7	165.7	175.7	167.4	150.6	152.7	124.5	139.5

22.2.LOM CASH FLOW FORECAST

Using a gold price of $1,250 per ounce, copper price of $3.00/lb and exchange rate of C$:US$ of 1.25:1.00, as assumed for the Mineral Reserve estimation process, the LOM physicals and all the operating, transport and capital cost forecasts have been used to estimate the net cash flow for the Mount Milligan Mine from 2020 to the end of 2028 to be $398M as shown in Table 22-2. The after-tax NPV at a discount rate of 5% is $342M. RG on Table 22-2 refers to amounts per the Royal Gold stream.

Table 22-2: Cash Flow Summary

	Units	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	Total
Gold Sales	Ounces('000's)	172	203	171	171	170	151	156	123	153	0	1,470
Gold Price	$	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250		
Gold Sales	$ Millions	215	253	214	214	212	188	195	154	191	0	1,837
Gold RG Share	$ Millions	(49)	(58)	(49)	(49)	(48)	(43)	(45)	(35)	(44)	0	(419)
Smelter Charges	$ Millions	(1)	(1)	(1)	(1)	(1)	(1)	(1)	(1)	(1)	0	(7)
Net Gold Revenue	$ Millions	**165**	**195**	**164**	**164**	**163**	**144**	**150**	**118**	**147**	**0**	**1,410**
Copper Sales	Lbs (Millions)	91	84	101	76	86	76	102	70	56	0	743
Copper Price	$	3.00	3.00	3.00	3.00	3.00	3.00	3.00	3.00	3.00		
Copper sales	$ Millions	273	253	303	227	258	228	306	211	169	0	2,228
Copper RG Share	$ Millions	(44)	(40)	(48)	(36)	(41)	(36)	(49)	(34)	(27)	0	(355)
Smelter Charges	$ Millions	(23)	(22)	(26)	(19)	(22)	(19)	(26)	(18)	(14)	0	(190)
Net Copper Revenue	$ Millions	**206**	**191**	**229**	**172**	**195**	**172**	**231**	**159**	**127**	**0**	**1,683**
Total Revenue	$ Millions	**372**	**386**	**393**	**336**	**358**	**317**	**381**	**277**	**274**	**0**	**3,094**
Operating Costs	$ Millions	255	273	266	266	271	259	275	257	171	0	2,294
Silver Credit	$ Millions	(6)	(6)	(7)	(6)	(7)	(6)	(6)	(6)	(5)	0	(55)
Royalties	$ Millions	7	7	7	6	7	6	7	5	5	0	58
Capital Expenditures	$ Millions	56	44	41	37	48	43	31	32	15	0	346
Lease Interest	$ Millions	1	1	0	0	0	0	0	0	0	0	4
Working Capital	$ Millions	4	(1)	1	(3)	(3)	(10)	6	(3)	(2)	0	(11)
Reclamation	$ Millions	0	0	0	0	0	0	0	0	0	35	35
Taxes	$ Millions	4	5	5	2	2	1	2	1	1	0	24
Total Cash flow	**$ Millions**	**51**	**64**	**80**	**33**	**39**	**23**	**65**	**(10)**	**89**	**(35)**	**398**

Offsite costs contemplate delivery of copper concentrate to Pacific Rim Asian smelters. Concentrate from the mine site will be trucked to a storage and loadout facility at Mackenzie and transferred onto railcars for transport to port storage facilities at Kinder Morgan's loading facility in North Vancouver. It should be noted that TCM has leased 95 rail cars to ensure more reliable transport of concentrate to Vancouver.

22.3. SENSITIVITY ANALYSIS

Table 22-3 to Table 22-6 summarise sensitivities to gold price, copper price, change in cost estimates and currency fluctuations to the USD respectively.

Table 22-3: Sensitivity of NPV to Gold Price Changes

NPV $ millions	***Sensitivity to Gold Price at 0%, 5%, and 8% Discount Rates***		
Discount Rate Gold Price ($/ounce)	**0%**	**5%**	**8%**
-20%	78	72	68
-10%	284	245	227
$1,250	398	342	315
+10%	513	438	382
+20%	627	534	465

Table 22-4: Sensitivity of NPV to Copper Price Changes

NPV $ millions	***Sensitivity to Copper Price at 0%, 5% and 8% Discount Rates***		
Discount Rate Copper Price ($/lb)	**0%**	**5%**	**8%**
-20%	39	41	41
-10%	219	191	178
$3.00	398	342	315
+10%	578	492	451
+20%	758	642	588

Table 22-5: Sensitivity of NPV to the Impact of Change in Cost Estimates

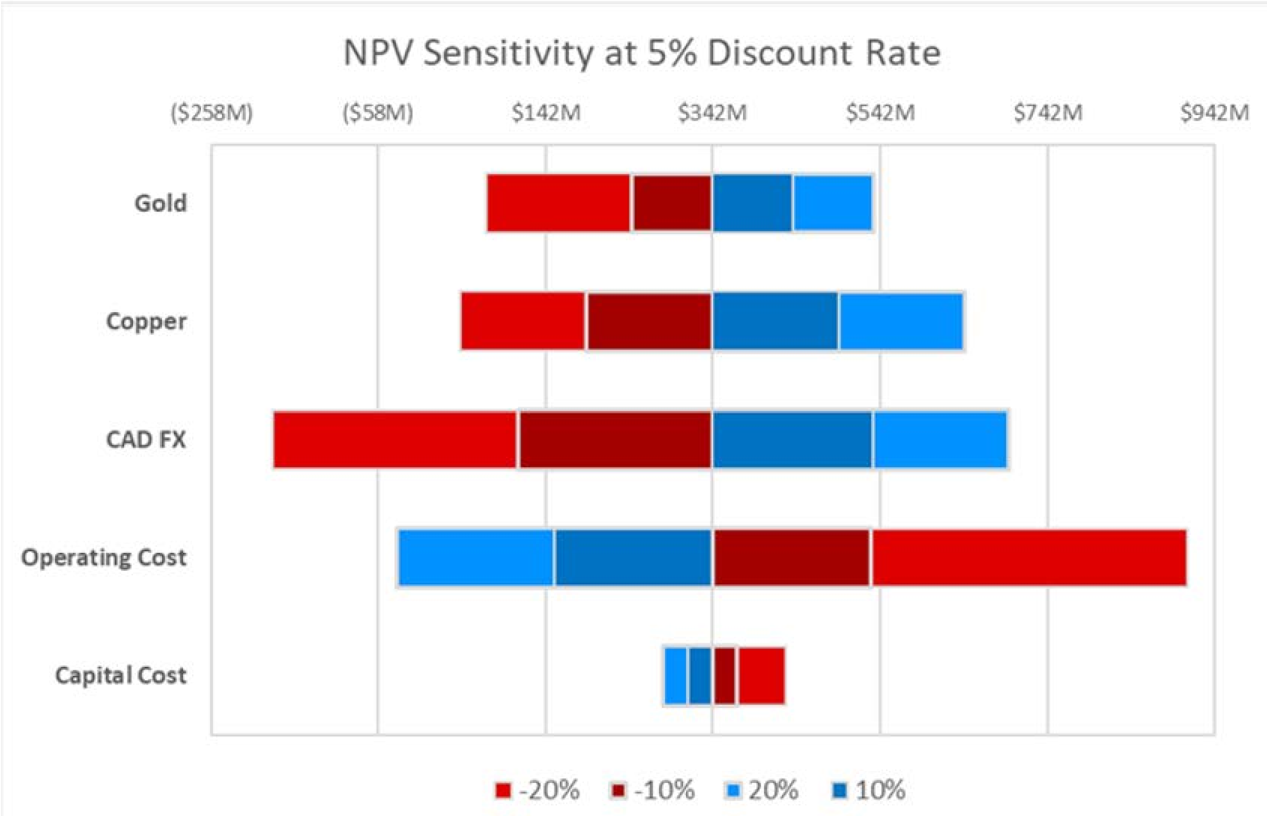
NPV $ millions	***Sensitivities to costs at $1,250/oz gold, $3.00/lb copper and 5% Discount Rate***	
Variable	**Operating Costs**	**Capital Costs**
-20%	908	430
-10%	531	371
Base Case	342	342
+10%	153	312
+20%	(36)	282

Table 22-6: Sensitivity of NPV to the Impact of Change in Exchange Rate

NPV $ millions	***Sensitivity to FX at 0%, 5% and 8% Discount Rates***		
Discount Rate FX (CAD/USD)	**0%**	**5%**	**8%**
-20%	(230)	(184)	(164)
-10%	121	109	103
1.25:1.00	398	342	315
+10%	630	534	488
+20%	821	696	636

Figure 22-1 provides a comparison of the sensitivity of NPV (at 5% discount rate) for the various parameters for +/-10% and +/-20% changes from the base case valuation for metal prices of $1,250/oz gold and $3.00/lb copper. As can be seen, the highest sensitivity is to OPEX and exchange rate changes, both for increases and decreases, with capex being the least sensitive parameter.

Figure 22-1: Comparison of Sensitivities



22.4. TAXATION

The determination of taxes involves significant estimation and judgment requiring a number of assumptions. The actual taxes payable will be subject to assessments by taxation authorities who may interpret tax legislation differently. The cash flow is based on management's best estimate of the probable outcome of these matters.

Corporation Taxes – Federal

Based on the pricing assumptions noted above, the mine is not expected to pay any federal income taxes at the statutory rate of 15% for approximately 9 years from 2020 to 2028 as there are sufficient deductions during that period to offset taxable income. These deductions include:

- Exploration and pre-production development expenditures allowed to be claimed discretionarily at 100% acceleration, limited to mine operating profit;
- Initial and sustaining capital expenditures generally allowed to be claimed discretionarily at 25%, limited to mine operating profit;
- Debt financing costs;
- Net operating loss carry forward allowed for up to 20 years; and
- Provincial mining taxes

Corporation Taxes – Provincial

The provincial corporate taxable income base is the same as the federal tax base. Based on the pricing assumptions noted above, the mine is not expected to pay any provincial income taxes at the statutory rate of 11%.

Mining Taxes – Provincial

The mine will be subject to the greater of two different taxes; either 2% tax on net current proceeds (net revenue less operating costs) or 13% tax on net revenue (net revenue less operating costs and capital expenditures). Additionally, to the extent the mine has previously paid the 2% tax on net current proceeds, this amount can be applied as a credit against the 13% tax on net revenue. Based on the pricing assumptions noted above, the mine is expected to pay the 2% net current proceeds tax for approximately 9 years from 2020 to 2028 as there are sufficient deductions and credits during that period to offset the 13% tax on net revenue. In lieu of allowing a deduction of debt financing costs, the net revenue can be reduced by an investment allowance which is earned on expenditures incurred to the extent they have not yet been deducted.

22.5. ENVIRONMENTAL LIABILITIES

The Project is operated in compliance with all applicable environmental obligations arising from federal and provincial statutes and regulations that are necessary to the current operation of its business. The Mount Milligan Mine submitted the five-year revision to its Reclamation Plan in 2019 and will submit another revision as per permit and regulatory requirements in 2024. The updated mine plan may require an updated version of the 5-year Closure Plan to be developed and submitted. Pursuant to the 2019 revision to the Reclamation Plan, the most recent estimate of the undiscounted and uninflated cost of reclamation is $35 million.

23. ADJACENT PROPERTIES

There are no adjacent properties relevant to the assessment of the Project.

24. OTHER RELEVANT DATA AND INFORMATION

There is no additional relevant data or information that should be included in this Technical Report.

25. INTERPRETATION AND CONCLUSIONS

Based on the information contained herein, the Qualified Persons, as authors of this Technical Report, offer the following interpretations and conclusions.

25.1. GEOLOGY AND MINERAL RESOURCES

Mount Milligan is a roughly tabular, near-surface, alkalic copper-gold porphyry deposit currently being mined.

The procedures for drilling, sampling, sample preparation and analyses are appropriate for the type of mineralization and estimation of Mineral Resources.

Combined Measured and Indicated Mineral Resources total 125.4Mt at 0.19% Cu and 0.35g/t Au containing 517Mlb copper and 1.4Moz gold. Inferred Mineral Resources total 3.7Mt at 0.12% Cu and 0.46g/t Au containing 10Mlb copper and 0.05Moz gold.

The classification of Mineral Resources conforms to CIM Definition Standards.

Mineral Resources were estimated as of December 31, 2019 within a conceptual open pit shell using metal prices of $3.50/lb copper and $1,500/oz gold, and are reported exclusive of the Mineral Reserve. The estimate takes into consideration factors such as metallurgical recoveries, concentrate grades, transportation costs, smelter treatment charges, and royalty and streaming arrangements in determining economic viability.

25.2. MINING AND MINERAL RESERVES

Proven and Probable Mineral Reserves total 191.0Mt at 0.23% Cu and 0.39g/t Au containing 959Mlb copper and 2.4Moz gold, effective 31st December 2019. This estimate was derived through the design of an ultimate pit, conforming to practical mining requirements, based on an optimized pit shell using metal prices of US$3.00/lb Cu and US$1,250/oz Au.

The Mineral Reserve estimate has been prepared using industry standard best practise methodologies with the classification of Proven and Probable Mineral Reserves conforming to CIM definitions and NI 43-101 requirements. The Mineral Reserves estimate is based on metal prices of $1,250/oz Au and $3.00/lb Cu, and a CAD:USD exchange rate of 1.25.

Mining is carried out using a conventional drill-blast, load and haul approach. All waste rock is either used for TSF dam construction or stored within the TSF to comply with environmental approvals requirements.

The aforementioned Mineral Reserve estimate is significantly lower than that reported at end-2018 which was 447.6Mt at 0.19% Cu and 0.33g/t Au containing 1,836Mlb copper and 4.7Moz Au. The causes of this change are summarised in section 1.11.

25.3. MINERAL PROCESSING

Metallurgical recoveries are derived from operating results for the current process plant flowsheet configuration.

This LOM update assumes that an average 60kt/d process plant throughput will be achieved from 2021 onwards. While this has not yet been achieved over a calendar year, various operational and maintenance improvement initiatives are in progress with this throughput goal as the ultimate objective; including improvements to the secondary crushing circuit and SAG mill liner performance.

In 2017, low water inventories resulted in a temporary shutdown of the process plant. During winter 2018-2019, a shutdown was avoided, but throughput was decreased to align with available water supply. The risk to throughput due to low water inventories in the TSF is being mitigated by additional wells in the TSF and the newly commissioned Lower Rainbow Valley Well Field. There is no anticipated tonnage reduction in 2020 due to a lack of process water.

25.4. ECONOMIC ANALYSIS

Capital costs have been estimated for a 9-year mine life and operating costs have been estimated for an average 60,000t/d process plant feed rate. Operating cost estimates were developed from first principles using site historic costs as a basis for calibrating the estimates.

The base case economic assumptions align with parameters used for Mineral Reserve estimation including metal prices of $1,250/oz Au and $3.00/lb Cu, and a CAD:USD exchange rate of 1.25.

The life-of-mine all-in sustaining cost per ounce sold, on a by-product basis, before taxes is estimated at $704/oz gold sold. This is a non-GAAP financial performance measure.

The NPV for the base case model is $346M after tax at a 5% discount rate.

25.5. RISKS AND MITIGATIONS

The most significant technical risk relates to ensuring sufficient water supply to avoid disruption to the process plant operation.

Two other more significant risks relate to indigenous relations and process plant reliability affecting throughput.

Other less significant risks relate to increased opex and sustaining capex, metallurgical recoveries, resource modeling, environmental impact, geotechnical stability, human resources and climate change impacts.

The reader is referred to section 26, Risk Assessment, for details on identified risks and risk mitigation plans and actions.

26. RISKS AND MITIGATION

26.1. KEY RISKS

Throughout the various sections of this report, risks have been discussed. Table 26-1, on the following page, presents a summary of the key risks that could have a material effect on the Mt. Milligan Mine if not mitigated effectively.

In addition to these key risks, the Mt. Milligan site is impacted by a number of additional risks, both specific to the operation as well as to the industry in general. The management of these and the identified key risks are undertaken as part of a broader risk management program detailed below.

26.2. ONGOING MANAGEMENT OF RISK

The Board of Directors for Centerra has created a Risk Committee whose mandate is Enterprise Risk Management (ERM) governance and oversight. The VP Risk & Insurance at Centerra prepares and presents a quarterly report for the Risk Committee on the key strategic, operational, project and exploration risks, as well as emerging risks.

The Mt. Milligan Mine participates in the Centerra ERM program which has been implemented to ensure risk-informed decision making throughout the organization. The program is based on leading international risk management standards such as ISO 31000 as well as industry best practice. The program employs both a bottom-up and top-down approach to identify and address risks from all sources that threaten the achievement of company objectives. Each operating site and project is responsible for identifying, assessing, treating and monitoring risk. These risks and their mitigation action plans are recorded in a site risk register. Efforts are coordinated by appointed “Risk Champions” who facilitate the process to ensure consistency and continuity.

The Centerra Risk & Insurance Team, stationed at the Centerra corporate head office, is responsible for providing the requisite tools, guidance, oversight and strategic direction for the ERM program. The risk management program at Centerra considers the full life of mine cycle from exploration through to closure. All aspects of the operation, the surrounding environment and our stakeholders are considered when identifying risks. As such, the risk program encompasses a broad range of risks including technical, financial, commercial, social, reputational, environmental, health and safety, political and human resources related risks.

Table 26-1: Key Risks

RISK RATING:	Priority Attention	Active Management Required	Diligent Monitoring	Monitor / Manage as Appropriate

Risk Category	Risk Description	Risk Rating	Mitigation Strategy
Water Management	Lower water levels resulting in possible reduction in mill throughput.	Priority Attention (red)	• Dedicated resources focused on water related activities from construction to operations and maintenance. • Water balance model updated using Goldsim modeling software and is being validated through measurements of actual flows and movements. • Build upon initial success with groundwater sourcing. Continue exploration for additional groundwater sources for Life of Mine purposes. • Assessing options for long term water sources.
Production	Process plant reliability and throughput lower than planned.	Active Management Required (orange)	• Process plant throughput and recovery optimization projects are being prioritized and planned out by the Capital Projects team. • Secondary crusher bottleneck review being completed and improvements planned over 2020 and 2021. • Completed comprehensive identification of critical/capital spares. Majority of critical spares have arrived and spares inventory almost complete. • Improve reliability and condition monitoring programs. • Subject Matter Expert review of grinding mill liner designs with a goal of having SAG Mill grate life equal to that of the shell liners thus reducing liner related down time. • Original Equipment Manufacturers engaged to provide oversight on operating and maintenance programs. • Third party expertise to assist with continuous improvement Initiatives/metallurgical recovery technologies.
Infrastructure	Aging machinery and equipment could possibly result in process plant downtime and/or delay in pit development/lower tonnes mined.	Active Management Required (orange)	• Continually invest in preventative maintenance programs. • Identification and Implementation of continuous improvement Initiatives. • Identification and investment in critical and capital spares for process plant/mine equipment.

Risk Category	Risk Description	Risk Rating	Mitigation Strategy
Community Relations/ Indigenous Peoples	Relationships with Indigenous Groups resulting in possible operational disruption and/or delayed or denied permits.		• Implementing consultation plan to support permitting activities, and monitor/evaluate effectiveness. • Support new and existing training and community development programs that engage and provide benefits for Indigenous groups.
Capital & Operating Costs	Inability to transport concentrate due to railroad operational issues, rail/port labour strikes and/or blockades.		• Monitoring of emerging issues. • Enhanced rail car lease arrangements, including maintenance. • Contractual arrangements with railroad company to ensure access and compensation for delays. • Contingency plan including alternative storage and transportation. • Increase inventory stocking levels to compensate for longer delivery times and use truck transport when required.
Capital & Operating Costs	Costs are higher than estimated in the plan.		• Ongoing monitoring for continuous improvement opportunities.
Metallurgy	The impact of Py:Cpy on recoveries is greater than estimated.		• Refine recovery model for when ratio is above ~ 12 (current model ratio). • Perform metallurgical drilling for more testwork and investigate recovery options for pyrite associated gold.
Geology & Modelling	Reconciliation between new resource model and actual production results.		• New block model created for 2020 currently being evaluated against 2019 production. • Monthly reconciliation process in place.
Site Environment	Ineffective tailings dam management resulting in loss of life, production reductions or interruptions, reputational damage, and severe long-term environmental degradation.		• Third party independent annual reviews of dam structure and integrity. • Inundation studies completed. • Dam construction maintains the designed maximum inflow of water. • Conduct daily dam inspections. • Segmented construction of tailings.
Mining	Pit wall instability/failure could result in		• Complete structural mapping and modelling, pit slope stability analysis.

Risk Category	Risk Description	Risk Rating	Mitigation Strategy
	lower production by blocking access to the pit.		• Continuous movement monitoring. • Wall controlled blasting. • Perform third party independent annual inspections.
Human Resources	Increasing challenges attracting and retaining employees caused by tight job market resulting in employee turnover and loss of knowledge.		• Implementation of robust performance management program and succession plan for critical positions. • Annual review of compensation strategy focusing on total rewards mix.
Surrounding Environment	Forest fires in area resulting in inability to evacuate employees and business interruption.		• Mine Rescue team in place with firefighting training. • Evacuation strategy in place in case of emergency. • Fire breaks around camp. • Annual inspections of power line and managing tree undergrowth. • Weather station on site that monitors fire danger.
Health & Safety	Inability to operate as a result of injuries and/or death of employees.		• Supervisor and employee safety training including Centerra's "Work Safe Home Safe" and "Visible Felt Leadership" programs. • Job Hazard Assessments conducted on all activities prior to commencement. • Implementation of Critical Controls program for key safety risks. • Ongoing site inspections.

27. RECOMMENDATIONS

The following recommendations are provided by the various Qualified Persons.

27.1. EXPLORATION RECOMMENDATIONS

It is recommended that mineral exploration continue targeting both of the mineralization styles that have been identified in the Mount Milligan deposit; early stage porphyry Au-Cu and late stage structurally controlled high-gold low-copper (HGLC). This should be done through continued development of a comprehensive 3D exploration model developed and updated from information in the drilling database which compiles lithology, alteration, mineralization, structural and geophysical data. The 3D model will improve understanding of the geometry of the deposit from fault block to fault block and better determine controls on both styles of mineralization and where to find extensions of known mineralization that could potentially be added to resources and reserves.

Ongoing exploration programs should continue to identify, delineate and test target zones within the three strategic mineral exploration domains: Near-pit infill/expansion, Brownfield (outside the ultimate pit boundary but inside the mine lease) and Greenfield (outside the mine lease). Budgets for programs over the next three years should be on par with Centerra-TCM programs in recent years (2018-2020). Annually, these have been USD $5.5-6.5 million comprised of core drilling (30,000 to 43,000 m), geophysics surveys (15 to 60 line-km IP surveys and other surveys) and in-house team building.

27.2. RESOURCE ESTIMATION AND GRADE CONTROL

The following are recommendations related to the resource estimate and grade control:

- Reconciliation of the head grade obtained from the recommended test bench to the 31st December 2019 resource model.
- Additional work to improve local grade estimation (Cu:Au correlation) within the resource model, including remodeling the current domain interpretation.
- Grade control testing and short-term block model development.
- Additional drilling to determine the extension of mineralization along the western ultimate pit wall.
- Progress the geometallurgical program with compilation of a block model.

27.3.WATER MANAGEMENT

Continue to identify water sources for LOM operations and progress related approvals in consultation with Indigenous partners.

Continue to monitor water supply, storage and use very closely to minimize potential process plant downtime due to lack of water.

27.4.MINE PLANNING AND OPERATIONS

Evaluate using an optimized pit with a lower revenue factor as the basis for designing the ultimate pit, which is expected to reduce the strip ratio and possibly provide a more manageable balance between ore and waste tonnes. However, this is also likely to decrease the mined ore tonnes and may reduce the mine life.

Re-work the mining schedule, including considerations for additional waste rock stockpiles, to provide adequate TSF dam building material for each construction season.

Evaluate opportunities to further improve mining efficiencies, such as through the application of automation.

Evaluate opportunities for in-pit waste and tailings storage.

27.5.PROCESSING AND METALLURGY

Evaluate use of alternative flotation equipment such as Staged Flotation Reactors or Direct Flotation Reactors. An initial assessment for the Mt Milligan flowsheet and ore has shown potential to increase both gold and copper metallurgical recoveries.

Evaluate alternative flowsheets for the treatment of high-gold low-copper (HGLC) and high Py:Cpy ratio ore to improve metal recoveries. This includes the use of flowsheets using cyanide to enhance the metallurgical recovery of gold.

Evaluate tailings re-processing, an opportunity that was previously assessed at a high level.

Evaluate increasing the mining rate and process plant throughput to reduce unit costs. This might increase the portion of the mineral resource which can be extracted profitably.

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28.1. GLOSSARY OF UNITS, ABBREVIATIONS, AND SYMBOLS

Glossary of Units

Symbol	Definition
”	seconds (geographic)
'	foot/feet
‘	minutes (geographic)
"	inches
#	number
%	percent
/	per
<	less than
>	greater than
µm	micrometer (micron)
a	annum/ year
Å	angstroms
asl	above sea level
BQ	36.5 mm diameter core
c.	circa
d	day
d/wk	days per week
dmt	dry metric tonne
fineness	parts per thousand of gold in an alloy
ft	feet
g	gram
g/cm^3	grams per cubic centimeter
g/dmt	grams per dry metric tonne
g/m^3	grams per cubic meter
Ga	billion years ago
ha	hectares
HP	horsepower
HQ	63.5 mm diameter core
kg/m^3	kilograms per cubic meter
kL	kiloliters
km	kilometer
km^2	square kilometers
koz	thousand ounces
kton	thousand tonnes
kV	kilovolt
kVA	kilovolt–ampere
kW	kilowatt
kWh	kilowatt hour
kWh/t	kilowatt hours per tonne
lb	pound
M	million
m	meter
m^3	cubic meter
m^3/hr	cubic meters per hour
Ma	million years ago

Symbol	Definition
Mesh	size based on the number of openings in one inch of screen
mg/L	milligrams per liter
mi	mile/miles
Mlbs	million pounds
Mm	million meters
mm	millimeter/millimeters
Moz	million ounces
mRL	meters relative level
Mton	million tonnes
Mtpa	million tonnes per annum
MW	megawatts
NQ/NQ2	47.6 mm size core
º	degrees
ºC	degrees Celsius
oz	ounce/ounces (Troy ounce)
P	Passing, i.e. % passing through a screen
pH	measure of the acidity or alkalinity of a solution
pop	population
ppb	parts per billion
ppm	parts per million
PQ	85 mm diameter core
t	metric tonne
t/yr	tonnes per annum (tonnes per year)
tpd	tonnes per day
tph	tonnes per hour
tpod	tonnes per operating day
t/m^3	tonnes per cubic meter
TDS	total dissolved solids
TSS	total suspended solids
µm	micrometers
wt%	weight percent

Glossary of Abbreviations

Abbreviation	Definition
®	registered name
AAS	atomic absorption spectroscopy
AAL	Australian Assay Laboratories
AC	Aircore
Alcoa	Alcoa of Australia Ltd
Amdel	Amdel Laboratory
ANC	acid-neutralizing capacity
ANP	acid-neutralizing potential
ARD	acid-rock drainage
AuAA	cyanide-soluble gold
AuEq	gold equivalent
AuFA	fire assay
AuPR	preg-rob gold
AuSF	screen fire assay
AusIMM	Australasian Institute of Mining and Metallurgy
BFA	bench face angle
BLEG	bulk leach extractable gold
BLM	US Bureau of Land Management
BMCO	breakeven mill cut-off
BSCO	breakeven stockpile cut-off
C	Canadian
C.P.G.	Certified Professional Geologist
Capex	capital expenditure
CAF	cost adjustment factor
CDN	Canadian
CER	Consultative Environmental Review
CIL	carbon-in-leach
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
CN_{wad}	Weak acid-dissociable cyanide
CRF	capital recovery factor
CRM	certified reference material
CST	cleaner scavenger tailings
CTOT	carbon total
Cu Eq	copper equivalent
CuCN	cyanide-soluble copper
E	east
EDA	exploratory data analysis
EIA	Environmental Impact Assessment
EIS	Environmental Impact Statement
EOM	end of month
EOY	end of year
EPA	Environmental Protection Authority
ERMP	Environmental Review and Management Program
FAusIMM	Fellow of the Australasian Institute of Mining and Metallurgy
GAAP	Generally Accepted Accounting Principles
Golder	Golder Associates Ltd.
Golder	Golder Associates Inc.
GN	mine grid north

Abbreviation	Definition
GPS	global positioning system
GRG	gravity recovery gold
H	horizontal
HC	high capacity
HPGR	high pressure grinding rolls
ICP	inductively-coupled plasma
ICP-AES	inductively-coupled plasma atomic emission spectroscopy
ICP-MS	inductively-coupled plasma mass spectrometry
ICP-OES	inductively-coupled plasma optical emission spectrometry
IRSA	Inter-ramp slope angle
IW	Impacted Water
JCR	joint condition rating
JORC	The Joint Ore Reserves Committee of The Australasian Institute of Mining and Metallurgy, Australian Institute of Geoscientists and Minerals Council of Australia
JV	joint venture
KV	kriging variance
L–G	Lerchs–Grossman
LC	low capacity
LOA	length overall
LOM	life-of-mine
LSK	large-scale kinetic
MAIG	Member of Australian Institute of Geoscientists
MAusIMM	Member of the Australasian Institute of Mining and Metallurgy
MIK	multiple-indicator kriging
MN	magnetic north
MPA	maximum potential acidity
MRF	Mine Rehabilitation Fund
MWMS	mine water management system
MWMT	meteoric water mobility testing
N	north
NAG	net acid generation/net acid generating
NAPP	net acid-producing potential
Newmont	Newmont Mining Corporation
NI 43-101	Canadian National Instrument 43-101 “Standards of Disclosure for Mineral Projects”
NOI	Notice of Intent
NN	nearest-neighbor
NNP	net neutralizing potential
NSR	net smelter return
NW	northwest
OK	ordinary kriging
Opex	operating expenditure
P.Eng.	Professional Engineer (CAN)
P.E.	Professional Engineer (US)
P.Geol	Professional Geologist (CAN)
P.G.	Professional Geologist (US)
PAG	potentially acid-generating
PLI	point load index
PoO	Plan of Operations
PSI	pounds per square inch
QA/QC	quality assurance and quality control

Abbreviation	Definition
QLT	quick leach test
QP	Qualified Person
RAB	rotary air blast
RC	reverse circulation
RDA	Residue Disposal Area
RM SME	Registered Member, Society for Mining, Metallurgy and Exploration
RMR	rock mass rating
ROM	run-of-mine
RPL	Environmental Monitoring Plan
RQD	rock quality designation
S	south
SAG	semi-autogenous grind
S&ER	Sustainability and External Relations
SE	southeast
SEIS	Supplemental Environmental Impact Statement
SG	specific gravity
SME	The Society for Mining, Metallurgy & Exploration
SME-RM	Registered Member of The Society for Mining, Metallurgy & Exploration
SMU	selective mining unit
SPET	State Plane East Truncated, Local Mine Grid
SRM	standard reference material
SS	sulfide sulfur
ST	scavenger tailings
STOT	sulfur total
SX-EW	solvent extraction–electrowin
TF	tonnage factor
TN	true north
Topo	topography
UC	uniform conditioning
UG	underground
UHF	ultra-high frequency
USGS	United States Geologic Survey
US	United States
V	vertical
US$ or USD	United States Dollar
VHF	very high frequency
VWP	vibrating wire piezometer
W	west
WD	waste dump
XRD	X-ray diffraction
WDX	waste dump expansion
WRF	waste rock formation
XRF	X-ray fluorescence

Glossary of Symbols

Symbol	Element
Ag	silver
Al	aluminum
As	arsenic
Au	gold
B	boron
Ba	barium
Be	beryllium
Bi	bismuth
C	carbon
Ca	calcium
$CaCO_3$	calcium carbonate
CaO	calcium oxide
$CaSO_4 \cdot 2H_2O$	calcium sulfide dehydrate
Cd	cadmium
Ce	cerium
Cl	chlorine
CN	cyanide
CO	carbon monoxide
Co	cobalt
Cr	chromium
Cs	cesium
Cu	copper
Fe	iron
FeOx	iron oxides
Ga	gallium
Ge	germanium
H	hydrogen
Hf	hafnium
Hg	mercury
In	indium
K	potassium
La	lanthium
Li	lithium
Mg	magnesium
Mn	manganese
$Mn(OH)_2$	manganous hydroxide
MnO_2	manganese dioxide
Mo	molybdenum
N	nitrogen
Na	sodium
Nb	niobium
NH_3	ammonia
Ni	nickel
NOx	nitrogen oxide compounds
O_2	oxygen
P	phosphorus
Pb	lead
Pd	palladium

Symbol	Element
Pt	platinum
Rb	rubidium
Re	rhenium
S	sulfur
Sb	antimony
Sc	scandium
Se	selenium