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Casino Project

Form 43-101F1 Technical Report

Preliminary Economic Assessment

Yukon, Canada

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DATE AND SIGNATURES PAGE

The effective date of this report is 22 June 2021. The issue date of this report is 2 August 2021. See Appendix A, Preliminary Economic Assessment (PEA) Study Contributors and Professional Qualifications, for certificates of qualified persons. These certificates are considered the date and signature of this report in accordance with Form 43-101F1.

CASINO PROJECT
FORM 43-101F1 TECHNICAL REPORT
PRELIMINARY ECONOMIC ASSESSMENT

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1 SUMMARY

This Report was prepared for Western Copper and Gold Corporation ("Western") by M3 Engineering & Technology Corporation (M3) in association with Independent Mining Consultants (IMC), Aurora Geosciences, GeoSpark Consulting Inc., and Knight Piésold Ltd.

The purpose of this report is to provide a preliminary economic assessment on the Casino Property. This report conforms to the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) National Instrument (NI) 43-101, Standards of Disclosure for Mineral Projects.

1.1 KEY DATA

The key details about this project are as follows:

1. Casino is primarily a copper and gold project that is expected to process 120,000 dry tonnes of material per day (t/d) or 43.8 million dry tonnes per year (t/y). Metals to be recovered are copper (Cu), gold (Au), molybdenum (Mo) and silver (Ag).

2. Based on the economic analysis, the Property will produce the following over the life of the mine from the concentrator and heap leach facility:

    a. Gold – 6.52 million ounces
    b. Silver – 34.38 million ounces
    c. Copper – 4.08 billion pounds
    d. Molybdenum – 330 million pounds

3. The process will include a conventional single-line SAG mill circuit (Semi-Autogenous Ball Mill Crusher, or SABC) followed by conventional flotation to produce concentrate for sale. In addition to the concentrator, there will be a separate carbon-in-column facility to recover precious metals from heap leached oxide material. Gold and silver bullion (doré) produced will be shipped by truck to metal refineries.

4. The Property will require the construction of a power plant and will generate its own electrical power using LNG to fuel the generator drivers.

5. The Property has several routes of access, including by the Yukon River, by aircraft, winter roads, and existing trails. A network of paved highways provides access to the region from the Port of Skagway, Whitehorse and northern British Columbia. Paved roads to the Property currently exist up to Carmacks. A new, all weather, gravel road will be constructed by the project to connect Casino to Carmacks via the existing Freegold Road. The new access road will, in general, follow the existing Casino Trail that will be upgraded to support trucking from Carmacks to Casino.

6. Fresh water will be sourced from the Yukon River.

1.2 PROPERTY DESCRIPTION AND OWNERSHIP

The Casino porphyry copper-gold-molybdenum deposit is located at latitude 62° 44'N and longitude 138° 50'W (NTS map sheet 115J/10), in west central Yukon, in the northwest trending Dawson Range mountains, 300 km northwest of the territorial capital of Whitehorse.

To the west, Newmont is developing the Coffee Project. To the north and to the west, White Gold Corp. has a large number of claims and is actively exploring them. Approximately 100 km to the east, Minto Explorations Ltd.operates the Minto Mine, which produces copper concentrate.

The project is located on Crown land administered by the Yukon Government and is primarily within the Selkirk First Nation traditional territory. The Tr'ondek Hwechin traditional territory lies to the north and the proposed access road crosses into Little Salmon Carmacks First Nation traditional territory to the south. The White River First Nation and Kluane First Nation are also potentially impacted by the project. The Casino Property lies within the Whitehorse Mining District and consists of 1,136 full and partial Quartz Claims and 55 Placer Claims acquired in accordance with the Yukon Quartz Mining Act. The total area covered by Casino Quartz Claims is 21,276.61 ha. The total area covered by Casino Placer Claims is 490.32 ha. CMC is the registered owner of all claims, although certain portions of the Casino property remain subject to royalty agreements. The claims covering the Casino property are discussed further in Section 4 of this document.

Figure 1-3 at the end of this section shows the site's location in Yukon Territory as well as other points of interest relevant to this Report.

1.3 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

The Casino Mine is located in Central Yukon, roughly 150 km due northwest of Carmacks, at approximately N62° 44' 25", W138° 49' 32". Current site access is by small aircraft using the existing 760 m airstrip, by winter road and from the Yukon River.

Either road or barge service will provide early access for construction equipment, camp construction and initial equipment. A barge landing area at Britannia Creek and the Yukon River is currently in service.

The project plan includes a new airstrip. The project also plans a new 132 km year-round access road from the end of the Freegold Road, presently extending 70 km northwest of the village of Carmacks.

The climate at the Casino Project area can generally be described as continental and cold. Winters are long, cold and dry, with snow generally on the ground from late September through mid-May. Summers are short, mild and wet, with the greatest monthly precipitation falling in July. Average daytime temperature in winter reaches a maximum of -13 degrees Celsius in January, dropping to -22 degrees Celsius overnight. On average, the daytime temperatures in July reach a maximum of 20 degrees Celsius, with overnight lows of 7.7 Celsius. The mean annual precipitation for the Casino Project area is estimated to be 500 mm, with 65% falling as rain and 35% falling as snow.

1.4 HISTORY

The first documented work on the Casino Property was the working of placer claims in the area of the Casino Deposit recorded in April 1911, following a placer gold discovery on Canadian Creek by J. Britton and C. Brown. A study by D.D. Cairnes, of the Geological Survey of Canada in 1917, recognized huebnerite ($MnWO_4$) in the heavy-mineral concentrates of the placer workings that the gold and tungsten mineralization was derived from an intrusive complex on Patton Hill. The total placer gold production from the area of the property is unknown, but during the period of 1980-1985 placer mining yielded about 50 kg (1,615 troy ounces) of gold. During the Second World War, a small amount of tungsten was recovered from placer workings.

The first recorded bed rock mineral discovery was in 1936 when J. Meloy and A. Brown located silver-lead-zinc veins approximately 3 km south of the Canadian Creek placer workings. Over the next several years the Bomber and Helicopter vein systems were explored by hand trenches and pits. In 1943, the Helicopter claims were staked and in 1947 the Bomber and Airport groups were staked.

Lead-silver mineralization was the focus of exploration on the property until 1968. Noranda optioned the property in in 1948 and Rio Tinto in 1963. During this time trenching, mapping and sampling were conducted.

L. Proctor purchased the claims in 1963 and formed Casino Silver Mines Limited to develop the silver-rich veins. The silver-bearing veins were explored and developed intermittently by underground and surface workings from 1965 to 1980. In total, 372.5 tonnes of hand-cobbled argentiferous galena, assaying 3,689 g/t Ag, 17.1 g/t Au, 48.3% Pb, 5% Zn, 1.5% Cu and 0.02% Bi, were shipped to the smelter at Trail, British Columbia.

Based on the recognition of porphyry copper potential, the Brynelsen Group acquired Casino Silver Mines Limited and from 1968 to 1973 exploration was directed jointly by Brameda, Quintana, and Teck Corporation towards a porphyry target. Exploration included extensive soil sampling surveys, geophysical and trenching programs, eventually leading to the discovery of the Casino deposit in 1969. From 1969 to 1973, various parties including Brameda Resources, Quintana Minerals and Teck Corporation drilled the property.

Archer, Cathro & Associates (1981) Ltd. (Archer Cathro) optioned the property in 1991and assigned the option to Big Creek Resources Ltd. In 1992 a program consisting of 21 HQ (63.5 mm diameter) holes totalling 4,729 m systematically assessed the gold potential in the core of the deposit for the first time. In 1992, Pacific Sentinel Gold Corp. (PSG) acquired the property from Archer Cathro and commenced a major exploration program. The 1993 program included surface mapping and 50,316m of HQ and NQ (47.6 mm diameter) drilling in 127 holes. All but one of the 1992 drill holes were deepened in 1993. PSG drilled an additional 108 drill holes totalling 18,085 metres in 1994. This program completed the delineation drilling commenced in 1993. PSG also performed metallurgical, geotechnical and environmental work which was used in a scoping study in 1995. The scoping study envisioned a large-scale open pit mine, conventional flotation concentrator that would produce a copper-gold concentrate for sale to Pacific Rim smelters.

First Trimark Resources and CRS Copper Resources obtained the property and using the Pacific Sentinel Gold data published a Qualifying Report on the property in 2003 to bring the resource estimate into compliance with National Instrument 43-101 requirements. The two firms combined to form Lumina Copper Corporation in 2004. An update of the Qualifying Report was issued in 2004.

Western Copper Corporation acquired Lumina Copper Corporation, and the Casino Deposit, in November 2006. In the fall of 2011, Western Copper Corporation spun out all other assets except the Casino Deposit and changed its name to Western Copper and Gold Corporation.

In 2007, Western conducted an evaluation of the Bomber Vein System and the southern slope of the Patton Hill by VLF-EM and Horizontal Loop EM survey and soil geochemistry. Environmental baseline studies were also initiated in 2007. In 2008, Western reclaimed the old camp site, constructed a new exploration camp next to the Casino airstrip and drilled three drill holes (camp water well and two exploration holes) totalling 1,163 m. The main purpose of the drilling was to obtain fresh core samples for the metallurgical and waste characterization tests. Both exploration holes twinned PSG's holes to confirm historical copper, gold and molybdenum grades. Later that year, M3 Engineering produced a pre-feasibility study for Western. In 2009, Western completed 22.5 km of DC/IP surveying and MT surveying using the Quantec Geosciences Ltd. Titan system. Additionally, the company drilled 10,943 m in 37 diamond drill holes. Approximately 27 holes were infill holes drilled to convert inferred resource and non-defined material to the measured and indicated resource categories. Infill drilling covered the north slope of the Patton Hill. Drilling has identified supergene Cu mineralization and Mo mineralization in this area. The remaining 10 holes, totalling 4,327 m, were drilled to test geophysical targets. In 2010, infill and delineation drilling continued with most of the drilling done to the north and west of the deposit as outlined by PSG. The drilling program also defined hypogene mineralization at the southern end of the deposit. In addition, the company drilled a series of geotechnical holes at the proposed tailings embankment area and within the pit, and several holes for hydrogeological studies. The geotechnical drilling continued in 2011 (41 holes, 3,163 m) and 2012 (6 holes, 228 m). This work culminated in the publishing of a pre-feasibility study in 2011 and a feasibility study in 2013.

The design envisioned for Casino by the 2013 Feasibility Study was a four-year construction project resulting in a facility expected to process 120,000 dry tonnes of material per day or 43.8 million dry tonnes per year with a 22-year mill production schedule. Metals to be recovered would be copper, gold, molybdenum and silver. Gold and silver bullion (doré) produced would be shipped by truck to metal refineries.

In 2019, Western carried out a program of infill drilling designed to convert mineralization in the inferred resource category located along the margin of the deposit to the indicated category.

In mid 2019, Western acquired the adjacent property to the west referred to as the Canadian Creek property from Cariboo Rose Resources Ltd., which led to the issuance of a new Mineral Resource Statement in late 2020. Exploration on the Canadian Creek property dates from 1992 when Archer Cathro & Associates staked the Ana Claims. In 1993 Eastfield Resources Ltd. acquired the Ana Claims, expanded the Ana Claim block and explored the expanded property with soil geochemical sampling grids, trenching and drilling, (Johnston, 2018). This work was directed at the discovery of additional porphyry deposits. The 1993 program was followed by extensive field programs in 1996, 1997 and 1999 consisting of induced polarization (IP) surveying, road construction, and trenching on the Ana, Koffee, Maya and Ice claims. In 2000, another drill campaign was undertaken by Eastfield on the Ana, Koffee Bowl, and the newly acquired Casino "B" claims located immediately to the east of the Casino deposit. The Casino "B" holes confirmed the existence of gold mineralization which had first been discovered here in 1994 by Pacific Sentinel, who encountered 55.17 m averaging 0.71 g/t gold in hole 94 - 319. Modest exploration programs were conducted, mostly over the Casino "B" area, in 2003, 2004 and 2005. In 2007 a five-hole core drill program at Casino "B" targeted gold and copper soil anomalies and ground magnetic high features.

In 2009, the discovery of gold on Underworld Resources' White Gold property sparked new interest in gold exploration on the Canadian Creek property. This led to the implementation of a major exploration program at Canadian Creek directed at the gold potential of the property, away from the previous porphyry copper focused work areas. A soil survey revealed large areas of greater than 15 ppb gold in soils, associated with anomalous values in arsenic, bismuth and antimony, which extend for over four kilometres in an east-northeast direction from the Casino deposit. The induced polarization surveys showed numerous strong chargeability highs, many of which coincided with the gold- in-soil anomalies, which were subsequently tested with 10 core holes. The holes intersected clay altered structures with sheeted pyrite veins and narrow, structurally-controlled clay-altered structures with pyrite and quartz-caronate veins. With few exceptions gold grades of the mineralization were less than 1 gpt, and widths were less than 3 m.

In 2011 additional soil sampling, ground geophysics, and trenching were completed. The soil sampling completed coverage of the entire Canadian Creek property, while a limited-extent induced polarization identified two zones of greater than 20 mv/V chargeability. The trenching program identified a number of areas with anomalous gold values, including high values of 2,890 and 4,400 ppb.

As a follow up of the 2011 program a modest 2016 program of trenching, prospecting, and in-fill soil sampling was carried out by Cariboo Rose, who had acquired the property from Eastfield. Trenching work conducted in three areas in the Ana portion of the Canadian Creek property returned locally anomalous gold, widely spread anomalous arsenic, bismuth and antimony, and local high silver values generally confined to narrow structures.

Cariboo Rose's 2017 exploration program consisted of surface work directed at the Kana and Malt West gold targets and a reverse circulation (RC) drill program that tested a variety of gold targets across the property. A total of 2,151.27 m of reverse circulation (RC) drilling was conducted in 24 holes. This work confirmed gold and silver mineralization to be limited to narrow, less than 3-metre-wide structures rarely traceable over more than 100 m.

1.5 GEOLOGY

The geology of the Casino deposit is typical of many porphyry copper deposits. The deposit is centered on an Upper Cretaceous-age (72-74 Ma), east-west elongated porphyry stock, the Patton Porphyry, which intrudes Mesozoic granitoids of the Dawson Range Batholith and Paleozoic schists and gneisses of the Wolverine creek Suite of the Yukon Tanana Terrane (YTT). Intrusion of the Patton Porphyry into the older rocks caused brecciation of both the intrusive and the surrounding country rocks along the northern, southern and eastern contact of the stock. Brecciation is best developed in the eastern end of the stock where the breccia can be up to 400 m wide in plan view. To the west, along the north and south contact, the breccias narrow gradually to less than 100 m. The overall dimensions of the intrusive complex are approximately 1.8 by 1.0 km.

The main body of the Patton Porphyry is a relatively small, mineralized, stock measuring approximately 300 by 800 m and is surrounded by a potassically-altered Intrusion Breccia in contact with rocks of the Dawson Range Batholith. Elsewhere, the Patton Porphyry forms discontinuous dikes ranging from less than one to tens of metres wide, cutting both the Patton Porphyry Plug and the Dawson Range Batholith. The overall composition of the Patton Porphyry is rhyodacite, with phenocrysts of a dacitic composition and the matrix being of quartz latite composition. It is more commonly made up of abundant distinct phenocrysts of plagioclase and lesser biotite, hornblende, quartz and opaque minerals.

The Intrusion Breccia surrounding the main Patton Porphyry body consists of granodiorite, diorite, and metamorphic fragments in a fine-grained Patton Porphyry matrix. It may have formed along the margins, in part, by the stoping of blocks of wall rock. An abundance of Dawson Range inclusions is prominent at the southern contact of the main plug, whereas Wolverine Creek metamorphic rocks increase along the northern contact, and bleached diorite increases along the eastern contact of the main plug. Strong potassic and phyllic alteration locally destroys primary textures.

Primary copper, gold and molybdenum mineralization was deposited from hydrothermal fluids that exploited the contact breccias and fractured wall rocks. Higher grades occur in the breccias and gradually decrease outwards away from the contact zone, both towards the centre of the stock and outward into the granitoids and schists. The main mineralization types are:

- Leached Cap Mineralization (CAP) – This oxide gold zone is copper-depleted due to weathering processes and has a lower specific gravity of this zone relative to the other supergene zones. Weathering has replaced most minerals with clay. The weathering is most intense at the surface and decreases with depth.

- Supergene Oxide Mineralization (SOX) – This zone is copper-enriched, with trace molybdenite. It generally occurs as a thin layer above the Supergene Sulphide zone. Where present, the supergene oxide zone averages 10 m thick, and may contain chalcanthite, malachite and brocanthite, with minor azurite, tenorite, cuprite, and neotocite.

- Supergene Sulphide Mineralization (SUS) – Supergene copper mineralization occurs in a zone of sulphide mineral enrichment up to 200 m deep, located below the leached cap and above the hypogene zone. It has an average thickness of 60 m. Grades of the Supergene sulphide zone vary widely, but are highest in fractured and highly pyritic zones, due to their ability to promote chalcocite precipitation. The copper grades in the Supergene Sulphide zone are almost double the copper grades in the Hypogene (0.43% Cu versus 0.23% Cu).

- Hypogene Mineralization (HYP) – Hypogene mineralization occurs throughout the various alteration zones of the Casino Porphyry deposit below the Supergene zone, as mineralized stock-work veins and breccias. Significant Cu-Mo mineralization is related to the potassically-altered breccia surrounding the core Patton

Porphyry, as well as in the adjacent phyllically-altered host rocks of the Dawson Range Batholith. The breccias surrounding the core Patton Porphyry unit are host to the highest Cu values on the property.

1.6 DEPOSIT TYPE

The Casino deposit is best classified as a Calc-Alkalic porphyry type deposit associated with a tonalite intrusive stock. Primary copper, gold and molybdenum mineralization was deposited from hydrothermal fluids that exploited the contact breccias and fractured wall rocks. Higher Cu-Au grades occur in the breccias and gradually decrease outwards away from the contact zone both towards the centre of the stock and outward into the granitoids and schists. A general zoning of the primary sulphides occurs, with chalcopyrite and molybdenite occurring in the tonalite and breccias grading outward into pyrite-dominated mineralization in the surrounding granitoids and schists. Alteration accompanying the sulphide mineralization consists of an earlier phase of potassic alteration and a later overprinting of phyllic alteration. The potassic alteration typically comprises secondary biotite and K-feldspar as pervasive replacement and veins. Quartz stockwork zones and anhydrite veinlets also occur. Phyllic alteration consists of sericite and vein and replacement-style silicification.

The Casino Copper deposit is unusual amongst Canadian porphyry copper deposits in having a well-developed secondary enriched blanket of copper mineralization similar to those found in deposits in Chile and the southwestern United States, such as the Escondida and Morenci deposits. Unlike other porphyry deposits in Canada, the Casino deposit's enriched copper blanket was not eroded by the glacial action during the last ice age. At Casino, weathering during the Tertiary Period leached the copper from the upper 70 m of the deposit, forming the Leached Cap, and re-deposited it lower in the deposit, forming the Supergene enrichment zones. This created a layer-like sequence consisting of an upper leached zone up to 70 m thick where all sulphide minerals have been oxidized and copper removed, leaving behind a bleached, limonitic leached cap containing residual gold. Beneath the leached cap is a zone up to 100 m thick of secondary copper mineralization consisting primarily of chalcocite and minor covellite with a thin, discontinuous layer of copper oxide minerals at the upper contact with the overlying leached cap. The copper grades of the enriched, blanket-like zone can be up to twice that of the underlying unweathered primary copper mineralization. Beneath the secondary enriched mineralization, the primary mineralization consists of pyrite, chalcopyrite and lesser molybdenite. The primary copper mineralization is persistent at depth, extending more than 600 metres below surface, and beyond the ends of the deepest drill holes.

1.7 EXPLORATION STATUS

In 2009, Quantec Geoscience Limited of Toronto, Ontario performed Titan-24 Galvanic Direct Current Resistivity and Induced Polarization (DC/IP) surveys, as well as a Magnetotelluric Tensor Resistivity (MT) survey over the entire grid. Magnetotelluric Resistivity results in high resolution and deep penetration (to 1 km) and the Titan DC Resistivity & Induced polarization provides reasonable depth coverage to 750 m.

In 2010, all Pacific Sentinel's historic drill core stored at the Casino Property was re-logged. The purpose of the re-logging was to provide data for the new lithological and alteration models.

In 2011 and 2012, Western focused on geotechnical, metallurgical and baseline environmental studies, however however, some exploration holes were also drilled. In 2011, the program involved 41 drill holes for a total of 3,163.26 m. In 2012, six holes (228.07 m) were drilled for geotechnical purposes and 5 holes (1,507.63 m) were drilled for metallurgical sampling.

During the 2019 field season, Western focused on exploration drilling for the primary purpose of updating the resource base of the Casino Project. A total of 72 holes were drilled, logged and sampled in 2019 for a final tally of 13,594.63 m.

During the 2020 field season, Western completed a diamond drilling program of 12,008 m in 49 core holes. The program focused on identification of high-grade gold intercepts in the "Gold Zone", as well as expansion of the main deposit to the north and west. Results are not included in this resource estimate within this preliminary economic assessment (PEA).

1.8 EXPLORATION PROCEDURES

Exploration on the property over its history has included prospecting, geological mapping, multi-element soil geochemistry, magnetic and induced polarization surveys, trenching and drilling. Targeting of early drilling on the Casino Deposit was based mainly on coincident copper and molybdenum-in-soil anomalies. Since 1993, with the exception of a Titan TM Survey, exploration in the vicinity of the Casino deposit has comprised drilling on a grid pattern using a core drill with a core diameter primarily of NQ and NTW widths, with a smaller number of holes drilled with HQ diameter core.

To the west of the Casino deposit on the recently acquired Canadian Creek Property, exploration utilized grid soil sampling surveys, ground magnetic and induced polarisation surveys to generate targets for trenching and drilling. Initially the focus of the geochemical and geophysical surveys was to locate porphyry copper mineralization. After 2016, the focus of this work switched to the identification of gold mineralization similar to that discovered at nearby Coffee Creek.

Soil sampling surveys to the west of the Casino Deposit were done over the time period from the mid 1990s through to 2011. The soil results show a co-incident copper and gold-in soil-anomaly at the 50 ppm Cu and 15 ppb Au threshold levels respectively, extending west from the western limits of the Casino Deposit for approximately 3 km. The coincident anomaly has been tested by 16 core holes. The holes closest to the Casino Deposit have moderate potassic alteration to strong propylitic alteration. The four closest holes intersected zones of leached capping or incipient leaching, underlain by weak enrichment and hypogene copper-gold-molybdenum mineralization, and are typical of the outer edges of a porphyry copper – gold – molybdenum deposit. Copper grades are in the 0.03 to 0.07% range, gold grades are in the 0.1 to 0.3 g/t range, and moly grades range from 0.002 to .004%. Copper, gold and molybdenum grades in the Casino B drill holes increase eastward towards the Casino deposit. These holes define the western limits of the Casino deposit system.

Ground magnetic surveys were undertaken over the Canadian Creek portion of the Casino Property in 2011 and in 2017. Line spacing was 100 m. The survey detected a number of lineaments, oriented mostly to the northwest, though none obviously align with the soil geochemical anomalies. The ground magnetic data shows a trend of high magnetic values stretching from the Casino Deposit through the Ana to the Koffee Bowl areas. This west-southwest trend follows the trend of Patton Porphyry dykes extending from the main intrusive complex.

Induced polarization surveys were carried out in 1993, 1996, 2009 and 2011. The surveys in the 1990s used a pole-dipole array with an a spacing of 75 m and an n 1 to 4 depth profile. The 2009 survey was a pole-dipole survey using an a spacing of 25 m and an n 1 to 6 depth profile, and the 2011 pole dipole survey used an a spacing of 25 m and an n 1 to 8 depth profile. In general, the surveys used small “n” spacings and have a limited depth profile. The surveys identified a number of high chargeability anomalies which remain to be tested.

1.9 MINERAL RESOURCE ESTIMATE

The Mineral Resource for the Casino Project includes Mineral Resources amenable to milling and flotation concentration methods (mill material) and Mineral Resource amenable to heap leach recovery methods (leach material). Table 1-1 presents the Mineral Resource for mill material. Mill material includes the supergene oxide (SOX), supergene sulphide (SUS) and hypogene sulphide (HYP) mineral zones. Measured and Indicated Mineral Resources amount to 2.17 billion tonnes at 0.16% total copper, 0.18 g/t gold, 0.017% moly and 1.4 g/t silver and contained metal amounts to 7.43 billion pounds of copper, 12.7 million ounces gold, 811.6 million pounds of moly and 100.2 million ounces of silver. Inferred Mineral Resource is an additional 1.43 billion tonnes at 0.10% total copper, 0.14 g/t gold, 0.010% moly and 1.2 g/t silver and contained metal amounts to 3.24 billion pounds of copper, 6.4 million ounces of gold, 322.8 million pounds moly and 53.5 million ounces of silver for the Inferred Mineral Resource in mill material.

Table 1-2 presents the Mineral Resource for leach material. Leach material is oxide dominant leach cap (CAP or LC) mineralization. The emphasis of leaching is the recovery of gold in the leach cap. Copper grades in the leach cap are low, but it is expected some metal will be recovered. Measured and Indicated Mineral Resources amount to 217.4 million tonnes at 0.03% total copper, 0.25 g/t gold and 1.9 g/t silver and contained metal amounts to 166.5 million pounds of copper, 1.8 million ounces gold and 13.3 million ounces of silver. Inferred Mineral Resource is an additional 31.1 million tonnes at 0.03% total copper, 0.17 g/t gold and 1.7 g/t silver and contained metal amounts to 17.2 million pounds of copper, 200,000 ounces of gold and 1.7 million ounces of silver for the Inferred Mineral Resource in leach material.

Table 1-3 presents the Mineral Resource for combined mill and leach material for copper, gold, and silver. Measured and Indicated Mineral Resources amount to 2.39 billion tonnes at 0.14% total copper, 0.19 g/t gold and 1.5 g/t silver. Contained metal amounts to 7.60 billion pounds copper, 14.5 million ounces gold and 113.5 million ounces of silver for Measured and Indicated Mineral Resources. Inferred Mineral Resource is an additional 1.46 billion tonnes at 0.10% total copper, 0.14 g/t gold and 1.2 g/t silver. Contained metal amounts to 3.26 billion pounds of copper, 6.6 million ounces of gold and 55.2 million ounces of silver for the Inferred Mineral Resource. The Mineral Resource for moly is as shown with mill material since it will not be recovered for leach material.

The Mineral Resources are based on a block model developed by IMC during June 2020. This updated model incorporated the 2019 Western Copper drilling and updated geologic models. It also includes some 2010 through 2012 Western Copper drilling that was not available in 2010 when the model used for the January 2013 Feasibility Study was developed.

The Measured, Indicated, and Inferred Mineral Resources reported herein are contained within a floating cone pit shell to demonstrate "reasonable prospects for eventual economic extraction" to meet the definition of Mineral Resources in NI 43-101.

Table 1-1: Mineral Resource for Mill Material at C$ 5.70 NSR Cutoff

Resource Class	Tonnes (Mt)	NSR ($/t)	Copper (%)	Gold (g/t)	Moly (%)	Silver (g/t)	CuEq %	Copper (Mlbs)	Gold (Moz)	Moly (Mlbs)	Silver (Moz)
Measured	145.3	38.08	0.31	0.40	0.025	2.1	0.74	985.8	1.9	80.6	9.8
Indicated	2,028.0	19.10	0.14	0.17	0.016	1.4	0.33	6,448.5	10.9	731.0	90.4
M+I	2,173.3	20.37	0.16	0.18	0.017	1.4	0.36	7,434.3	12.7	811.6	100.2
Inferred	1,430.2	14.50	0.10	0.14	0.010	1.2	0.24	3,240.4	6.4	322.8	53.5

Table 1-2: Mineral Resource for Leach Material at C$ 5.46 NSR Cutoff

Resource Class	Tonnes (Mt)	NSR ($/t)	Copper (%)	Gold (g/t)	Silver (g/t)	AuEq (g/t)	Copper (Mlbs)	Gold (Moz)	Silver (Moz)
Measured	37.2	19.72	0.05	0.45	2.8	0.48	39.3	0.5	3.3
Indicated	180.2	9.54	0.03	0.21	1.7	0.23	127.2	1.2	10.0
M+I	217.4	11.28	0.03	0.25	1.9	0.27	166.5	1.8	13.3
Inferred	31.1	7.60	0.03	0.17	1.7	0.18	17.2	0.2	1.7

Table 1-3: Mineral Resource for Copper, Gold, and Silver (Mill and Leach)

Resource Class	Tonnes (Mt)	NSR ($/t)	Copper (%)	Gold (g/t)	Silver (g/t)	Copper (Mlbs)	Gold (Moz)	Silver (Moz)
Measured	182.4	34.34	0.25	0.41	2.2	1,025.1	2.4	13.1
Indicated	2,208.3	18.32	0.14	0.17	1.4	6,575.6	12.1	100.5
M+I	2,390.7	19.54	0.14	0.19	1.5	7,600.7	14.5	113.5
Inferred	1,461.3	14.35	0.10	0.14	1.2	3,257.6	6.6	55.2

Notes:

1. The Mineral Resources have an effective date of 3 July 2020 and the estimate was prepared using the definitions in CIM Definition Standards (10 May 2014).
2. All figures are rounded to reflect the relative accuracy of the estimate and therefore numbers may not appear to add precisely.
3. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
4. Mineral Resources for leach material are based on prices of US$2.75/lb copper, US$1500/oz gold and US$18/oz silver.
5. Mineral Resources for mill material are based on prices of US$2.75/lb copper, US$1500/oz gold, US$18/oz silver, and US$11.00/lb moly.
6. Mineral Resources are based on NSR Cutoff of C$5.46/t for leach material and C$5.70/t for mill material.
7. NSR value for leach material is as follows: NSR (C$/t) = $12.65 x copper (%) + $41.55 x gold (g/t) + $0.191 x silver (g/t), based on copper recovery of 18%, gold recovery of 66% and silver recovery of 26%.
8. NSR value for hypogene sulphide mill material is: NSR (C$/t) = $60.18 x copper (%) + $41.01 x gold (g/t) + $214.94 x moly (%) + $0.355 x silver (g/t), based on recoveries of 92.2% copper, 66% gold, 50% silver and 78.6% moly.
9. NSR value for supergene (SOX and SUS) mill material is: NSR (C$/t) = $65.27 x recoverable copper (%) + $42.87 x gold (g/t) + $142.89 x moly (%) + $0.425 x silver (g/t), based on recoveries of 69% gold, 60% silver and 52.3% moly. Recoverable copper = 0.94 x (total copper – soluble copper).
10. Table 14 6 accompanies this Mineral Resource statement and shows all relevant parameters.
11. Mineral Resources are reported in relation to a conceptual constraining pit shell in order to demonstrate reasonable prospects for eventual economic extraction, as required by the definition of Mineral Resource in NI 43-101; mineralization lying outside of the pit shell is excluded from the Mineral Resource.
12. AuEq and CuEq values are based on prices of US$2.75/lb copper, US$1500/oz gold, US$18/oz silver, and US$11.00/lb moly, and account for all metal recoveries and smelting/refining charges.

1.10 MINING METHODS

This PEA is based on a conventional open pit mine plan. Mine operations will consist of drilling large diameter blast holes (31 cm), blasting with a bulk emulsion, and loading into large off-road trucks with cable shovels and a hydraulic shovel. Resource amenable to processing will be delivered to the primary crusher or various resource stockpiles. Waste rock will be placed inside the limits of the tailings management facility (TMF). There will be a fleet of track dozers, rubber-tired dozers, motor graders and water trucks to maintain the working areas of the pit, stockpiles, and haul roads.

The following general parameters guided the development of the mining plan:

- Mill material is limited to about 1.1 billion tonnes,
- Total mine waste to be co-disposed with tailings is limited to about 500 million tonnes,
- Mill capacity is a nominal 120,000 tonnes per day (t/d), but actual plant throughput for the schedule is based on hardness of the various material types, and usually exceeds 120,000 t/d.

Based on the mining plan developed for this study, the commercial life of the project is 25 years after an approximate 3-year pre-production period. Total mill material is 1.13 billion tonnes at 0.197% copper, 0.226 g/t gold, 0.0219% moly, and 1.70 g/t silver. Only measured and indicated mineral resources are considered as potential plant feed. Inferred mineral resources are considered as waste for this study.

In addition to the potential mill material, there is material mined from the leach cap zone that is amenable to processing by crushing, and heap leaching. This amounts to 203.8 million tonnes at 0.259 g/t gold, 1.95 g/t silver, and 0.034% total copper.

Total waste in the mine plan amounts to 500.1 million tonnes. This material is disposed in the tailing management facility. Figure 1-1 shows three facilities for mine waste: 1) North Waste which contains 200.6 million tonnes, 2) South

1 Waste which contains 154.8 million tonnes, and 3) South 2 Waste which contains 144.7 million tonnes. The material will be placed by trucks and dozers, the rising water level in the TMF facility will saturate the material relatively quickly, usually one to two years.

Additional rock storage facilities during the life of the project include:

- The heap leach pad which at the end of the project will contain 203.8 million tonnes of spent, non-reactive material, assuming all the potential leach material is processed.
- A low-grade stockpile southeast of the pit that has the capacity for 178.3 million tonnes, and a low-grade stockpile east of the pit that contains 84.7 million tonnes, both which will be processed at the end of the mine life.
- There will also be supergene oxide (SOX) stockpile south of the pit to store mining phase 1 SOX. It will be reclaimed during mining Years 4 through 13. The maximum size of this facility is estimated at 35.8 million tonnes. The SOX stockpile and the leach pad overlap by a small amount, but the SOX stockpile will be reclaimed before the leach pad gets to its final limits.
- There will be a stockpile for leach resource east of the pit. This stockpile is necessary because there will be many years when the mine production of leach resource will exceed crushing and stacking capacity. This is expected to reach a maximum size of 74.1 million tonnes and will be reclaimed by the end of Year 20.

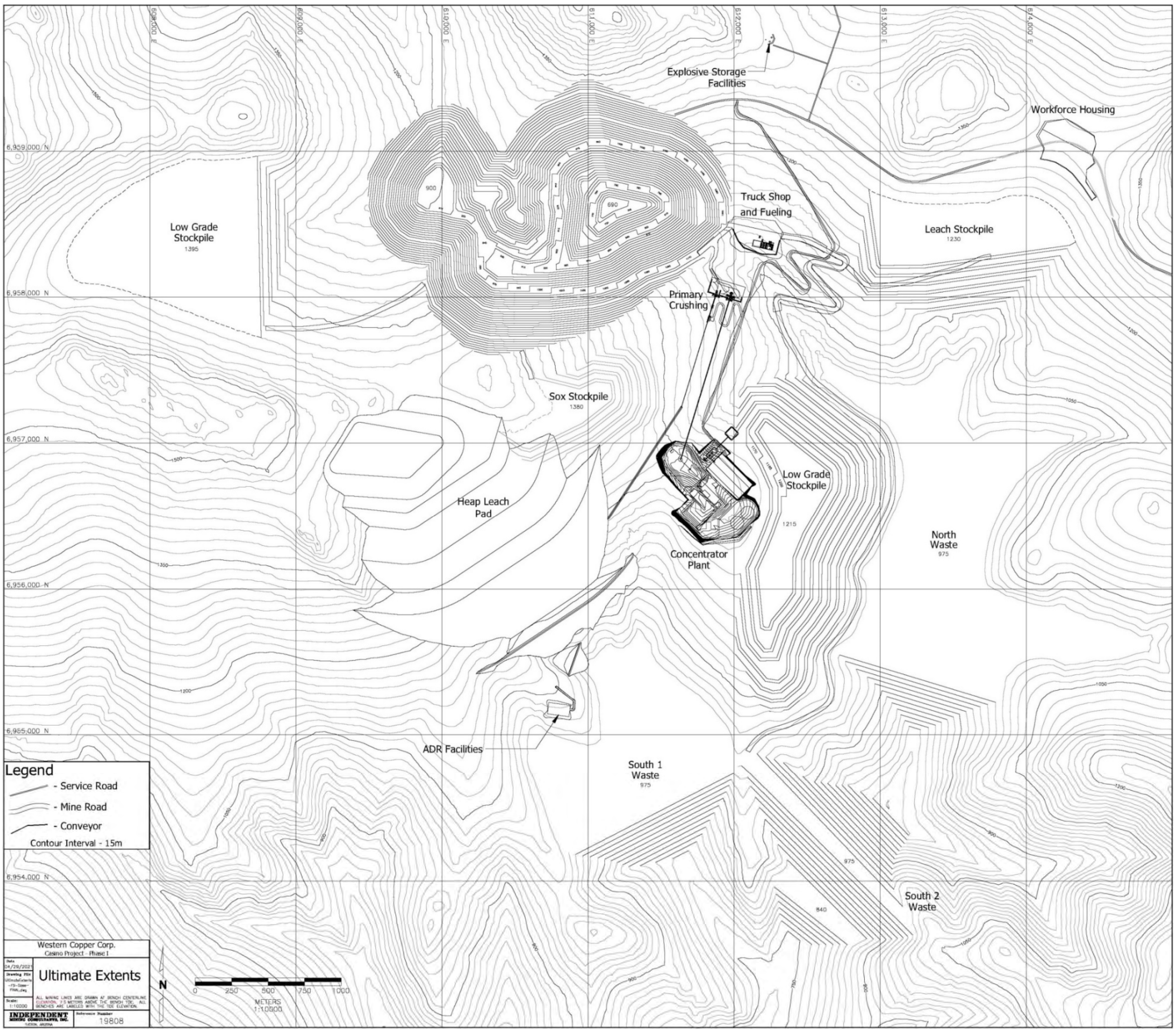


Figure 1-1: Maximum Extent of Waste Storage Areas and Stockpiles (IMC, 2021)

1.11 METALLURGICAL TESTING

Flotation testing by ALS Metallurgy from 2008 to 2012 indicated that copper concentrate grades of 28% copper could be routinely achieved at good copper recoveries with a primary grind size of 80% passing 200 µm and a regrind of 80% passing 25 µm. Gold and silver will be recovered with the copper concentrate. Molybdenum will be recovered to a molybdenum concentrate in a separate flotation circuit.

The average metal recoveries expected from mill processing following the planned mill feed schedule are noted below:

- Copper recovery to copper concentrate, percent 86
- Gold recovery to copper concentrate, percent 67
- Silver recovery to copper concentrate, percent 53
- Molybdenum recovery to molybdenum concentrate, percent 71

Column leach test work by SGS E&S Engineering Solutions Inc.on the oxide cap material crushed to minus 3.8 cm (1.5 inch) showed that good recoveries of gold and acceptable cyanide consumptions could be obtained by integrating the cyanide heap leach with the SART process. This process has been adopted for this feasibility study.

The metal recoveries expected from oxide cap heap leach processing are based on:

- Gold recovery, percent 70
- Silver recovery, percent 26
- Copper recovery to SART precipitate, percent 18

1.12 RECOVERY METHODS

A mine plan was developed to supply mill material to a conventional copper sulphide flotation plant with the capacity to process mill material at a nominal rate of 120,000 t/d, or 43.8 million tonnes per year (Mt/y). Actual annual throughput will vary depending on the mill material hardness encountered during the period. The mine is scheduled to operate two 12 hour shifts per day, 365 days per year.

Both sulphide copper-molybdenum mill material and oxide gold leach material will be processed. Copper-molybdenum mill material will be transported from the mine to the concentrator facility and oxide gold leach material will be transported from the mine to a crushing facility ahead of a heap leaching facility and a gold recovery facility.

Copper-molybdenum mill material will be processed by crushing, grinding, and flotation to produce copper and molybdenum sulphide mineral concentrates. Copper concentrate will be loaded into highway haul trucks and transported to the Port of Skagway for ocean shipment to market. Molybdenum concentrate will be bagged and loaded onto highway haul trucks for shipment to market.

Oxide gold mill material will be leached with an aqueous leach solution. Gold in the enriched (or pregnant) leach solution will be recovered using carbon absorption technology to produce gold doré bars. The enriched leach solution will also be treated to recover copper and cyanide and produce a copper sulphide precipitate. The copper sulphide precipitate will be bagged and loaded onto highway haul trucks for shipment to market. Recovery methods are discussed more in depth in Section 17.

1.13 INFRASTRUCTURE

The region is serviced by paved all-weather roads connecting the towns of Carmacks and Whitehorse in the Yukon with the Port of Skagway Alaska. With the completion of the 132 km Casino access road, the project will have an all-weather access route through Carmacks to Whitehorse (approx. 380 km) and to the Port of Skagway (550 km). The

Port of Skagway has existing facilities to store and load-out concentrates as well as facilities to receive bulk commodity shipments, fuels and connection to the Alaska Marine Highway. The Port of Skagway is developing plans to expand these facilities to better serve the expanding mining activity in the Yukon and Alaska.

The City of Whitehorse is the government, financial and commercial hub of the Yukon with numerous business and service entities to support the project and represents a major resource to staff the project. Whitehorse has an international airport and provides commercial passenger and freight services for the region. The proposed new access road alignment is shown in Section 18.2 of this report.

A new airstrip will be constructed at the mine to accommodate appropriately sized aircraft. The existing airstrip will be razed in preparation for grading for process facilities.

1.13.1 Power

Electrical power generation for the Project will be developed in two phases. An initial power plant designated the Supplementary Power Plant will be constructed in the vicinity of the main workforce housing complex to provide power to the camp, for construction activities, and to oxide crushing, conveying and heap leach facilities that go into operation before the main power plant is operational.

The Supplementary Power Plant will consist of three 2,250-kilowatt (kW) diesel internal combustion engines (ICE). Two of the generators will remain at the Workforce Housing complex and the third will be relocated to the Sand Cyclone (Area 640) facility to provide standby/emergency power to this area after the concentrator start-up.

A Main Power Plant will be constructed at the Casino main mill and concentrator complex to supply the electrical energy required for operations throughout the mine site. The primary electrical power generation will be provided by three Gas Turbine driven generators (two Single Fuel Gas Turbines, one Dual Fuel Gas Turbine) and a steam generator, operating in combined cycle mode (CCGT) with a total installed capacity of approximately 200 megawatts (MW). The nominal running load to the mine and concentrator complex is about 130 MW. Three diesel ICE driven generators will provide another 6.75 MW of power for black start capability, emergency power, and to complement the gas turbine generation when required. The gas turbines will be fuelled by natural gas (supplied as liquefied natural gas, or LNG). One of the three will have Dual Fuel capabilities - LNG and Diesel.

1.13.2 LNG Receiving, Storage and Distribution Facilities

LNG will be transported to the site from Fort Nelson, British Columbia via tanker trucks and stored on-site in a large 10,000 m^3 site-fabricated storage tank to provide fuel for the power plant. An LNG receiving station is provided to unload the LNG tankers and transfer the LNG into the storage facility. An LNG vaporization facility is provided to convert the LNG into gas at a suitable supply pressure to operate the power generation equipment.

1.13.3 Power Distribution

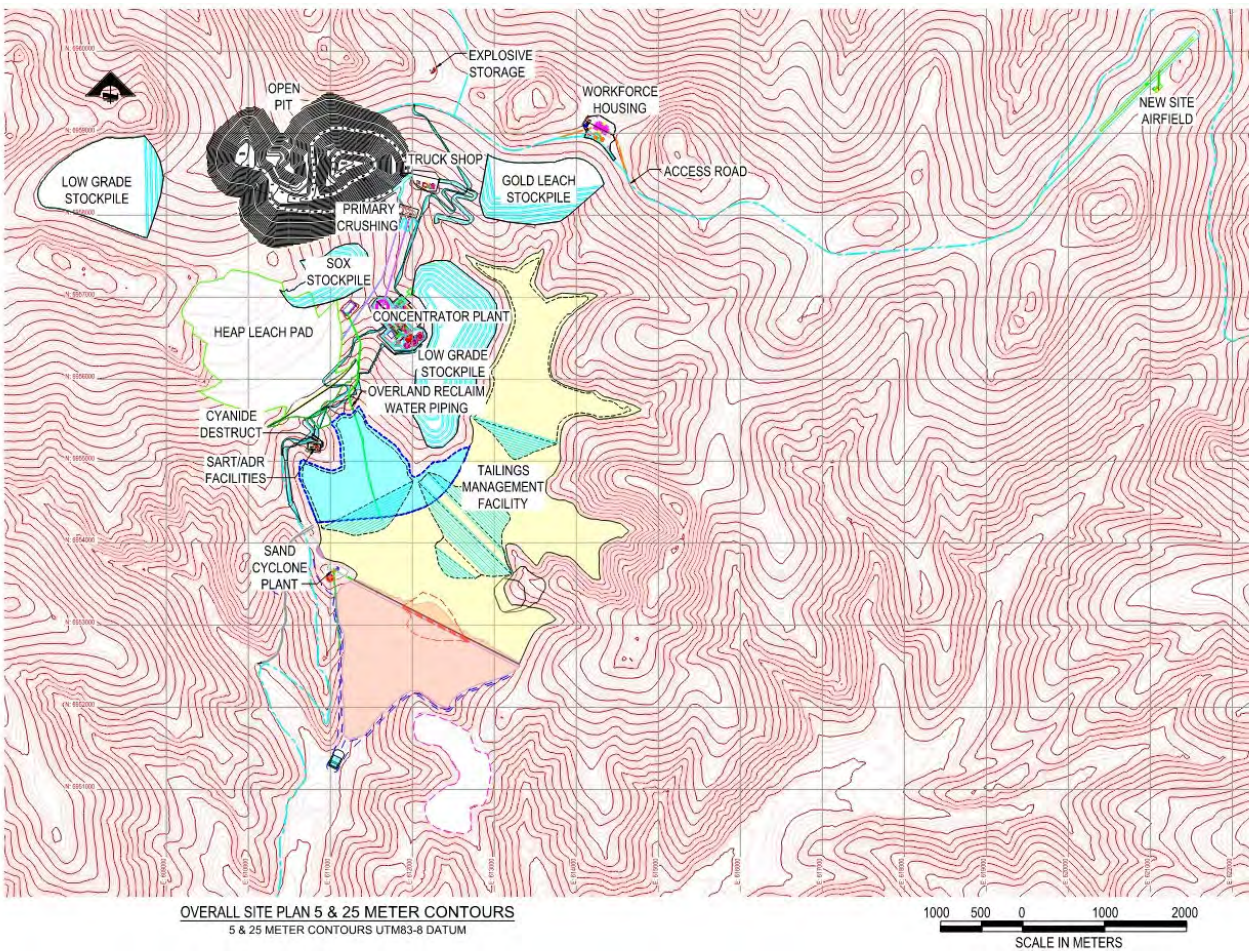
The power system for the Casino Project consists of two generating stations and the distribution system.

The main generating station will consist of a combined cycle plant with three 50.5 MW gas turbines (GT) and approximately 40 MW steam turbine (ST). There will also be three 2.25 MW diesel powered reciprocating engine generators at the main power plant. The GT and the ST units will all generate power at 13.8 kilovolts (kV) which will be stepped up to 34.5 kV through four (4) 13.8 kV to 34.5 kV transformers for the distribution system. The three ICE generators will be stepped up from 600V to 34.5 kV through three transformers.

The second generating station will be located at the main/construction camp site and will consist of three 2.25 MW diesel powered reciprocating engine generators. These units will generate power at 600V and will be stepped up through a 1,500 kVA, 33 kV Delta to 400/231 V Wye transformer. This station will be the first installed and will provide power for the Project construction.

The 34.5 kV distribution systems will radiate from a 34.5 kV switchgear line-up with feeders to the SAG mill, Ball Mill No. #1, Ball Mill No. #2, and feeders to the mill and flotation areas in cable tray using insulated copper conductors. Overhead line feeder circuits with aluminum conductor steel reinforced (ACSR) will be provided for the tailings reclaim water, fresh water from the Yukon River, crushing/conveying and SART/ADR, camp site and two feeders to the pit loop.

Electric power utilization voltages will be 4,160 volts for motors 300 horsepower (hp) and above, 575 volts for three-phase motors 250 hp and below. For lighting, small loads and building services 600/347 or 208/120 volts will be the utilization voltage.



(Source: M3, 2021)

Figure 1-2: Overall Site Plan

1.13.4 Water

The main fresh water supply will be supplied from the Yukon River. The water will be collected in a riverbank caisson and radial well system (Ranney Well) and pumped through an above-ground insulated 36" diameter by 17.4 km long pipeline with four booster stations to the 22,000 m^3 capacity freshwater pond near the concentrator. The design capacity of the freshwater collection and transfer system will be approximately 3,400 m^3/h.

1.13.5 Tailings Management Facility

A single Tailings Management Facility (TMF) will be constructed south of the open pit for storage of tailings and potentially reactive waste rock generated from mining. The TMF will store approximately 712 Mt of tailings and 500 Mt of potentially reactive waste rock and overburden materials. The TMF will be constructed using a combination of local borrow and cyclone underflow sand produced from Non-Acid Generating (NAG) tailings. A total of approximately 415 Mt of NAG tailings will be used for dam construction. The TMF will be constructed with centreline raises of the dam, to a final crest elevation of El. 981 m with a maximum height of approximately 280 m (crest to toe).

1.13.6 Heap Leach Facility

A Heap Leach Facility (HLF) will be constructed on a southeast facing hill-slope, approximately one kilometre south of the Open Pit. The HLF operations will commence during pre-production stripping of the Open Pit. The HLF has a design capacity of 204 million tonnes (Mt) of leach cap material. The heap leach pad will be stacked with mineralized material and leached from Year -3 through Year 20 of mine operations. The mineralized material will be stacked at a nominal rate of approximately 9.1 Mt per year.

The mineralized material will be stacked on a prepared pad, with a composite liner system to maximize leachate collection and minimize seepage losses. A double composite liner system will be constructed within the lower portion of the HLF and this area will function as an in-heap water management pond. The double liner system will include a leak detection and recovery system (LDRS) to intercept and collect potential leakage through the upper liner. The in-heap water management pond area will be impounded by a confining embankment, constructed from mine waste rock material.

The HLF will be developed in stages by loading in successive lifts, upslope from the base platform developed within the in-heap water management pond area, behind the confining embankment. The HLF will be developed by stacking mineralized material in eight-metre lifts to establish a final overall slope of 2.5H:1V. Intermittent wider benches will be constructed to limit the vertical height of the HLF to a maximum of 120 m.

1.14 CAPITAL COSTS

Total initial capital investment in the Project is estimated to be $3.25 billion, which represents the total direct and indirect cost for the complete development of the Project, including associated infrastructure and power plant. Table 1-6 shows how the initial capital is distributed between the various components, including $719 million for sustaining costs.

Table 1-4: Capital Cost Summary

Cost Item	Total ($M)
Process Plant and Infrastructure	
Project Directs including freight	1,777
Project Indirects	390
Contingency	412
Subtotal	2,579
Mining	
Mine Equipment	409
Mine Preproduction	211
Subtotal	620
Owner's Costs	52
Total Initial Capital Costs	3,251
Sustaining Capital	719
Total Life of Mine Capital Costs	3,970

1.15 OPERATING COSTS

Operating costs for the milling operation were calculated per tonne of material processed through the mill over the life of mine as shown in Table 1-5.

Table 1-5: Mill Operating Costs Per Tonne

Category	LOM ($/t)
Milling	$5.72
General & Administrative	$0.45
Total	$6.17

Heap leach operating costs were calculated per tonne of material processed through the heap leach over the life of the heap leach as shown in Table 1-6.

Table 1-6: Heap Leach Operating Costs

Category	LOM ($/t)
Heap Leach Operation	$1.30
ADR/SART	$4.67
Total	$5.98

Mining costs were calculated to average $1.93 per tonne of material moved and $3.10 per tonne of mineralized material.

Table 1-7: Mining Operating Costs

Category	($/t)
Cost per tonne material (material moved)	$1.93
Cost per tonne mill feed (mill + heap material)	$3.10
Cost per tonne mill feed	$3.66

The combined mining and milling costs are $9.84 per tonne material milled for the life of mine, which compares favorably to the life-of-mine net smelter return of $28.14 per tonne at Base Case metal prices.

1.16 ECONOMICS

This economic analysis is based on only measured and indicated mineral resources. Inferred mineral resources are considered as waste for this analysis. The Study indicates that the potential economic returns from the Project justify its further development and securing the required permits and licenses for operation. The financial results of the Study were developed under commodity prices that were based on analyst projections of long-term metal prices and C$:US$ exchange rate ("Base Case" prices). Note that an exchange rate of C$:US$ of 0.80 was used for the capital cost estimation for all metal price scenarios. Table 1-8 summarizes the financial results:

Table 1-8: Financial Results Summary

Category and Units	Base Case
Copper (US$/lb)	3.35
Molybdenum (US$/lb)	12.00
Gold (US$/oz)	1,600
Silver (US$/oz)	24.00
Exchange Rate (C$:US$)	0.80
NPV pre-tax (5% discount, $M)	5,790
NPV pre-tax (8% discount, $M)	3,620
IRR pre-tax (100% equity)	23.3%
NPV after-tax (5% discount, $M)	3,900
NPV after-tax (8% discount, $M)	2,330
IRR after-tax (100% equity)	19.5%
LOM pre-tax free cash flow ($M)	13,000
LOM after-tax free cash flow ($M)	9,070
Payback period (years)	3.0
Net Smelter Return ($/t milled)	28.14
Copper Cash Cost* (US$/lb)	(1.13)

*C1 cash costs, net of by-product credits.

The financial results of the Study are significantly influenced by copper and gold prices, as shown in Table 1-9.

Table 1-9: Copper and Gold Price Sensitivity

Copper Price (US$/lb)*	$2.50	$3.00	$3.35	$4.00	$4.50	$5.00
NPV pre-tax (8%) ($M)	2,290	3,070	3,620	4,630	5,410	6,190
NPV after-tax (8%) ($M)	1,400	1,950	2,330	3,040	3,590	4,140
IRR pre-tax	18.5%	21.4%	23.3%	26.6%	29.0%	31.3%
IRR after-tax	15.4%	17.9%	19.5%	22.3%	24.3%	26.2%
Payback (years)	3.7	3.2	3.0	2.7	2.5	2.3
Gold Price (US$/oz)*	$1200	$1400	$1600	$1800	$2000	$2200
NPV pre-tax (8%) ($M)	2,580	3,100	3,620	4,130	4,650	5,170
NPV after-tax (8%) ($M)	1,600	1,960	2,330	2,700	3,060	3,430
IRR pre-tax	19.3%	21.3%	23.3%	25.2%	27.1%	29.0%
IRR after-tax	16.1%	17.8%	19.5%	21.1%	22.7%	24.3%
Payback (years)	3.5	3.2	3.0	2.8	2.6	2.5

*All other metal prices except those noted are the same as the Base Case.

1.17 ADJACENT PROPERTIES

Several quartz mineral claim blocks and placer claims registered to other owners are staked adjacent to and in the general vicinity of CMC's claim block. Some of the placer claims on Canadian and Britannia Creeks overlap the Casino claims in the area of the pit. These placer claims along the upper part of Canadian creek are located within the projected pit shell, and are worked by their owners on a seasonal basis with small heavy equipment. The northwestern boundary of the Casino property adjoins the Coffee Creek project of Newmont Mining. The property hosts a structurally controlled gold deposit in metamorphic rocks of the Yukon Tanana terrane and granitoids of mid Cretaceous age. The mineralization is associated with quartz- carbonate and illite alteration and is best described as an orogenic deposit. The project is at a pre-feasibility stage of development.

The northeastern boundary of the Casino property abuts the "Betty and Hayes" property held by White Gold Corp. This property abuts the northern boundary of the narrow eastern extension of the Casino property. At this time, the property has undergone fairly early stages of exploration for similar orogenic-style gold mineralization to that within the Coffee Creek property.

Part of the eastern extension is also directly surrounded by the Idaho claim block held by Atac Resources Ltd.

1.18 PROPOSED PHASE II EXPANSION

The information presented in Sections 16 through 22 of this study is based on measured and indicated mineral resources with a mine plan constrained by the capacity of the selected site and design of the Tailings Management Facility (TMF), i.e., Phase I. This section presents the results for a larger pit design which includes inferred mineral resources and an expanded tailings capacity based on building an additional embankment south of the Phase I embankment, (i.e., the Phase II plan).

The economic assessment of the proposed Phase II expansion is preliminary in nature and includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the preliminary economic assessment will be realized.

The plant production rate is a nominal 120,000 tonnes per day and the peak mining rate is 100 Mt/y or about 275,000 t/d for this plan, the same as the Phase I plan. As with the Phase I plan, all of the waste rock will be co-disposed in the TMF facility.

The concentrator and associated facilities are as per Phase I and continue to process mill feed at a nominal 120,000 t/d. Table 1-10 shows a summary of the capital costs for both cases.

Table 1-10: Phase I vs. Phase II Capital Costs

Cost Item	Phase I Total ($M)	Phase I + II Total ($M)
Process Plant and Infrastructure		
Project Directs including freight	1,777	1,777
Project Indirects	390	390
Contingency	412	412
Subtotal	2,579	2,579
Mine		
Mine Equipment	409	419
Mine Preproduction	211	206
Subtotal	620	625
Owner's Costs	52	52
Total Initial Capital	3,251	3,256
Sustaining Capital	719	1,808
Total Life of Mine Capital Costs	3,970	5,064

Table 1-11 shows the economics of Phase I and Phase II.

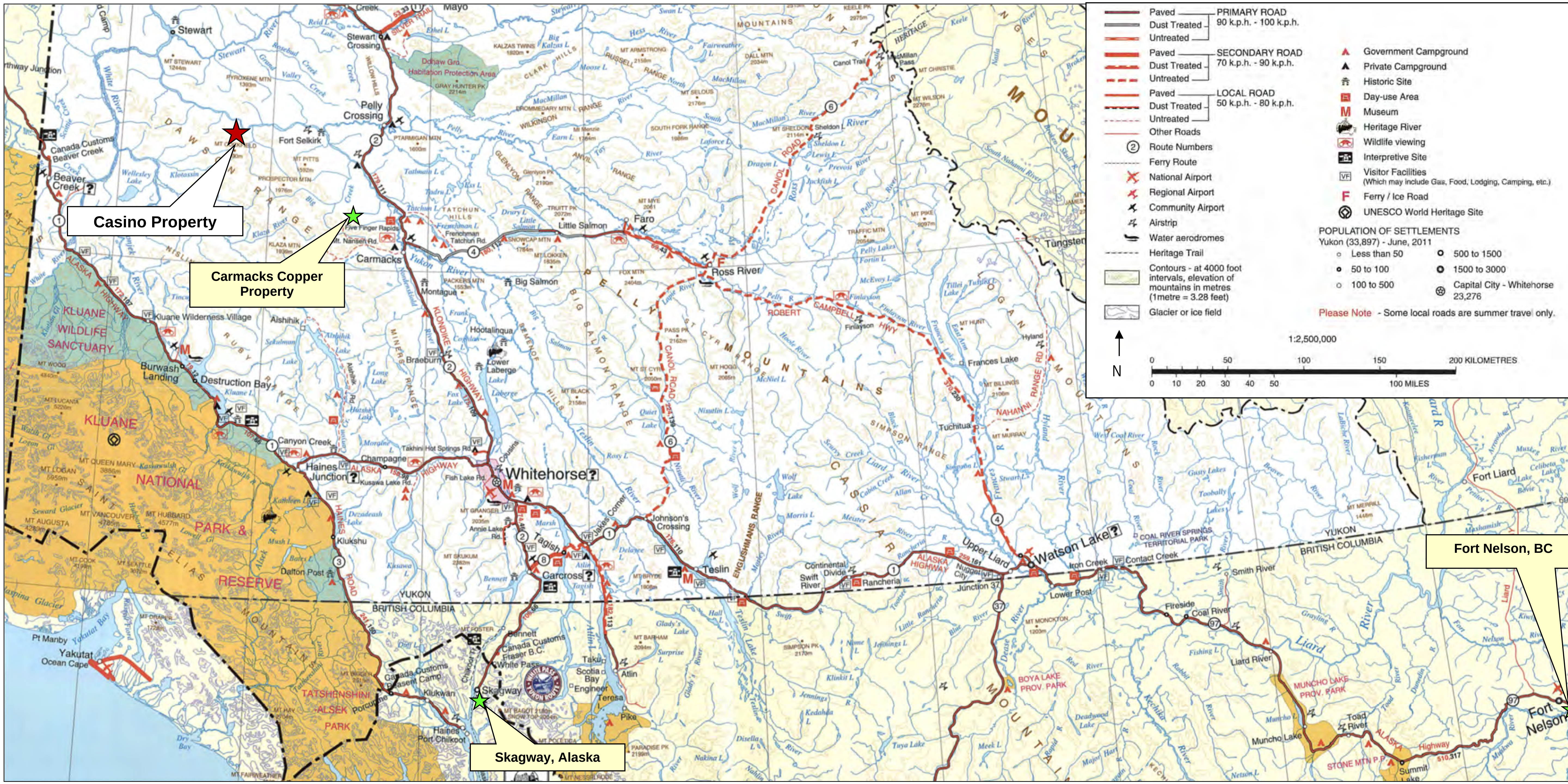
Table 1-11: Phase I vs. Phase II Economic Indicators

Economic Indicators before Taxes	Phase I	Phase II
NPV @ 0% ($M)	13,012	17,175
NPV @ 5% ($M)	5,790	6,237
NPV @ 8% ($M)	3,617	3,700
NPV @ 10% ($M)	2,632	2,640
IRR	23.3%	23.1%
Payback (years)	2.9	2.8
Economic Indicators after Taxes		
NPV @ 0% ($M)	9,073	11,968
NPV @ 5% ($M)	3,896	4,198
NPV @ 8% ($M)	2,332	2,384
NPV @ 10% ($M)	1,623	1,624
IRR (%)	19.5%	19.3%
Payback (years)	3.0	3.0

1.19 CONCLUSIONS AND RECOMMENDATIONS

The economic results of the Study demonstrate that the project has positive economics and warrants development. Standard industry practices, equipment and processes were used in this study. The project is based on conventional open pit mining and typical, well understood, processing methods. The authors of this report are not aware of any unusual or significant risks, or uncertainties that could affect the reliability or confidence in the project based on the data and information made available.

Based on the results of this study, it is recommended that the project be advanced to a Feasibility Study to establish a mineral reserve for the project. Concurrent with the later stages of the Feasibility Study, an application for environmental assessment under the Yukon Environmental and Socioeconomic Assessment Act should be prepared to continue the permitting process.



(Source: Yukon Highway Map, Yukoninfo.com)

Figure 1-3: Casino Property Location

2 INTRODUCTION

2.1 ISSUER AND PURPOSE OF ISSUE

This Report was prepared for Casino Mining Corporation (CMC), a wholly-owned subsidiary of Western Copper and Gold Corporation (Western) as well as for Western itself, by M3 Engineering & Technology Corporation (M3) in association with Independent Mining Consultants (IMC), GeoSpark Consulting Inc., Knight Piésold Ltd. (KP), and Aurora Geosciences.

The purpose of this report is to provide a Preliminary Economic Assessment on the Casino property. The estimate of mineral resources contained in this report conforms to the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Mineral Resource and Mineral Reserve definitions (May 2014) referred to in National Instrument (NI) 43-101, Standards of Disclosure for Mineral Projects.

2.2 SOURCES OF INFORMATION

The main sources of information for this Preliminary Economic Assessment estimate include the drillhole database provided to IMC in digital form. Also, various geologic solids that were reviewed by IMC and incorporated into the resource model. A geotechnical report by Knight-Piésold with slope angle recommendations was also used for the resource cone shell.

A summary of the Qualified Persons (QPs) responsible for the content of this report is shown in Table 2-1.

A site visit was conducted by Michael G. Hester on July 22, 2008 for one day to inspect the mine and waste storage areas as well as the core storage area. Mr. Hester could not conduct a site visit for this current study due to travel restrictions due to COVID-19. John M. Marek similarly did not conduct a site visit due to travel restrictions related to COVID-19.

An additional site visit was conducted by Carl Schulze as the project lead for Aurora Geosciences from September 9, 2020 through September 26, 2020.

Table 2-1: Dates of Site Visits and Areas of Responsibility

QP Name	Company	Qualification	Site Visit Date	Area of Responsibility
Daniel Roth	M3 Engineering & Technology Corporation	PE, P.Eng.	N/A	Sections 2, 3, 4, 5, 15*, 18 (except 18.5 and 18.6), 19, 20, 21 (except 21.1.5, 21.3.1 and 21.3.3), 22, 24 (except 24.2) and corresponding sections of 1, 25, 26 and 27
Michael G. Hester	Independent Mining Consultants, Inc.	F Aus IMM	22-Jul-2008	Section 14 and corresponding sections of 1, 25, 26 and 27
John M. Marek	Independent Mining Consultants, Inc.	PE	N/A	Sections 16, 21.1.5, 21.3.3, 24.2 and corresponding sections of 1, 25, 26 and 27
Laurie Tahija	M3 Engineering & Technology Corporation	MMSA-QP	N/A	Sections 13, 17, 21.3.1 and corresponding sections of 1, 25, 26, and 27
Carl Schulze	Aurora Geosciences	P. Geo.	Sept-9-2020 to 26-Sep-2020	Sections 6, 7, 8, 9, 10, 11, 12, 23 and corresponding sections of 1, 25, 26, and 27
Daniel Friedman	Knight Piésold Ltd.	P.Eng.	N/A	Sections 18.5, 18.6, and corresponding sections of 1, 25, 26, and 27

*Note that section 15 of Form 43-101F1 is not applicable to this stage of study and is listed in Table 2-1 for the sake of completeness to ensure that all sections are assigned to a QP.

2.3 UNITS AND ABBREVIATIONS

This report generally uses the SI (metric) system of units, including metric tonnes. The term “tonne” rather than “ton” is commonly used to denote a metric ton and is used throughout the report. Unless otherwise specified, currency is in Canadian dollars ($ or C$). Units and abbreviations used are listed in Table 2-2.

Table 2-2: Abbreviations Used in this Document

Units	Abbreviation
Above mean sea level	ASL
Alaska Industrial Development Authority	AIDA
ALS Global	ALS
Aluminum	Al
Aluminum conductor steel reinforced	ACSR
Amperes	A
Antimony	Sb
Argillic	ARG
Arsenic	As
Associated Engineering	AE
Barium	Ba
Beryllium	Be
Bismuth	Bi
British Colombia	BC or B.C.
B-Train Double	BTD
Cadmium	Cd
Calcium	Ca
Canadian dollars	$ or C$
Canadian Institute of Mining, Metallurgy and Petroleum	CIM
Carbon-in-column	CIC
Cariboo Rose Resources Ltd.	Cariboo Rose
Casino Mining Corporation	CMC
Central Nervous System (i.e. chemicals that affect it)	CNS
Chromium	Cr
Cobalt	Co
Combined cycle mode in gas turbines	CCGT
Copper	Cu
Copper equivalent	CuEq
CRS Copper Resources Corp.	CRS
Cubic metres	m^3
Cubic metres per hour	m^3/h
Current density	A/m^2
Dawson Range Batholith / Granodiorite	WR, WRGD
Degrees Celsius	ºC
Density	t/m^3
Direct Current Resistivity and Induced Polarization	DC/IP
Dollars per ounce	$/oz
Dollars per pound	$/lb
Dollars per tonne	$/t
Effective Grinding Length	EGL
Eighty percent passing	K80, P80

Units	Abbreviation
Electrowinning	EW
Engineering, Procurement and Construction Management	EPCM
Foot (feet)	ft
G&T Metallurgical Services	G&T
Gallium	Ga
Gas turbine	GT
General & Administrative	G&A
Gold	Au
Grams per litre	g/L or g/l
Grams per tonne	g/t
Greater than	>
Heap leach facility	HLF
Hectare(s)	ha
Horsepower	hp
Hour	h
Hour(s) per kilotonne	h/kt
Huebnerite	$MnWO_4$
Hypogene sulfide	HYP
Inch	"
Independent Mining Consultants	IMC
Induced Polarization	IP
Induced polarization	IP
Inductively Coupled Plasma-Atomic Absorption Spectroscopy	ICP-AAS
Inductively Coupled Plasma-Atomic Absorption Spectroscopy	ICP-AAS
Inductively Coupled Plasma-Atomic Emission Spectroscopy	ICP-AES
Inductively Coupled Plasma-Atomic Emission Spectroscopy	ICP-AES
Inductively Coupled Plasma-Emission Spectroscopy	ICP-ES
Internal Combustion Engine	ICE
Internal rate of return	IRR
Intrusive Breccia	IX
Inverse distance with a power weight of 2	ID2
Inverse distance with a power weight of 3	ID3
Iron	Fe
Kilo (1,000)	k
Kilogram(s)	kg
Kilogram(s) per tonne	kg/t
Kilometer	km
Kilopounds	klbs

Units	Abbreviation
Kilotonnes	ktonnes, kt
Kilotonnes per year	kt/y
Kilovolt(s)	kV
Kilowatt	kW
Kilowatt-hour	kWh
Kilowatt-hour per tonne	kWh/t
Knight Piésold Ltd.	KP
Lanthanum	La
Last-in-first-out	LIFO
Lead	Pb
Leak detection and recovery system	LDRS
Less than	<
Linear Low-Density Polyethylene	LLDPE
Liquefied natural gas	LNG
Litres	L, l
Litres per hour per square meter	$L/h/m^2$
Litres per second	L/s, l/s
Long term price	LTP
M3 Engineering & Technology Corporation	M3
Magnesium	Mg
Magnetotelluric Tensor Resistivity	MT
Manganese	Mn
Manganese	Mn
Mass Emission-Inductively Coupled Plasma Spectroscopy (ICP-MS)	ICP-MS
Material Takeoff	MTO
Mean annual precipitation	MAP
Mega (1,000,000)	M
Megawatt	MW
Mercury	Hg
Methyl Isobutyl Carbinol	MIBC
Metre(s)	m
Metric Tonne (1000 kg)	Tonne or t
Metric tonne per day	t/d
Metric tonne per year	t/y
Micrometer or micron	µm
Milligrams per litre	mg/L
Millimeter(s)	mm
Million	M
Million Canadian dollars	C$M
Million cubic metres	Mm^3
Million dollars	$M
Million ounces	Moz
Million pounds	Mlbs
Million tonnes	Mt
Million tonnes per year	Mt/y
Million years ago	Ma
Molybdenite or Molybdenum	Mo, Moly
National Instrument 43-101	NI 43-101
Nearest neighbor	NN

Units	Abbreviation
Net present value	NPV
Net profits interest	NPI
Net smelter return royalty	NSR
Nickel	Ni
Non-Acid Generating	NAG
Non-Government Organizations	NGOs
Notice to Proceed	NTP
Ordinary kriging	OK
Ounce(s)	oz
Overburden	OVB
Oxide Dominant Leach Cap, or Leached Cap Mineralization	CAP or LC
Paleozoic schists and gneisses	YM
Parts per billion	ppb
Parts per million	ppm
Patton Porphyry	PP
Percent	%
Phosphorus	P
Post-mineralization explosive breccia	MX
Potassium	K
Potassium amyl xanthate	PAX
Potentially-Acid Generating	PAG
Pound	lb
Pounds	lbs
Power of hydrogen (measure of acidity)	pH
Preliminary Economic Assessment	PEA
Qualified Person	QP
Quality Assurance and Quality Control	QA/QC
Reclamation and closure plan	RCP
Reverse Circulation	RC
Rock Quality Designation	RQD
Run of Mine	ROM
Scandium	Sc
Semi-autogenous grinding	SAG
SGS Canada Inc.	SGS
Silver	Ag
SMC	SAG Mill Comminution
Snow water equivalent	SWE
Sodium	Na
Sodium Cyanide	NaCN
Sodium Hydrosulfide	NaSH
Specific gravity	S.G.
Square metres	m^2
Steam Turbine	ST
Strontium	Sr
Sulphidization, Acidification, Recycling and Thickening	SART
Supergene oxide	SOX
Supergene sulphide	SUS
Tailing Management Facility	TMF

Units	Abbreviation
Temperature Celsius	°C
Temperature Fahrenheit	°F
Thallium	Tl
Thousand troy ounces	koz
Titanium	Ti
Tonnage factor or specific volume	m^3/tonne or m^3/t
Tonnes per day	t/d
Tonnes per year	t/y
Tungsten	W
Uranium	U
US dollars	US$
Vanadium	V

Units	Abbreviation
Volt	V
Weak Acid Soluble	WAS
Western Copper and Gold Corporation	Western
Year	y, yr
Yukon Environmental and Socioeconomic Assessment Act	YESAA
Yukon Environmental and Socio-economic Assessment Board	YESAB
Yukon First Nations	UFA
Yukon Geological Survey	YGS
Yukon-Tanana terrane	YTT
Zinc	Zn

3 RELIANCE ON OTHER EXPERTS

In cases where the study authors have relied on contributions of other qualified persons, the conclusions and recommendations are exclusively the qualified persons' own. The results and opinions outlined in this report that are dependent on information provided by qualified persons outside the employ of M3 are assumed to be current, accurate and complete as of the date of this report.

Information received from other experts has been reviewed for factual errors by CMC and M3. Any changes made as a result of these reviews did not involve any alteration to the conclusions made. Hence, the statement and opinions expressed in these documents are given in good faith and in the belief that such statements and opinions are not false and misleading at the date of these reports.

M3 relied upon Western Copper and Gold Corporation for project ownership data. M3 did not verify ownership or any underlying agreements. Mining is a risky business. The risk must be borne by the Owner. M3 does not assume any liability other than performing this technical study to normal professional standards.

The following sections describe additional information that this report relies upon beyond that which was provided by the QPs listed in Section 2.2.

3.1 METALLURGY AND PROCESS ENGINEERING

Outside reports that were relied upon included the following:

- ALS Metallurgy (formally G&T Metallurgical Services) of Kamloops, BC, performed numerous metallurgical testing to advance the flotation process design. Tom Shouldice was the official contact. International Metallurgical and Environmental and CMC managed and oversaw this work with input from FLSmidth.
- Starkey and Associates of Oakville, Ontario, performed a grinding circuit study. John Starkey is the official contact for Starkey and Associates. CMC managed and oversaw this work with input from FLSmidth.
- SGS Lakefield Research Limited of Lakefield, Ontario, performed a grinding circuit study. Carlos Lozano was the official contact for SGS Lakefield. CMC managed and oversaw SGS's work.
- SGS E&S Engineering Solutions (formerly METCON Research) of Tucson, AZ, USA performed metallurgical testing to advance design of the gold heap leach. Rodrigo Carneiro was the official contact. CMC managed and oversaw SGS's work.

M3 staff and consultants reviewed and evaluated metallurgical testing results from the tests listed above. In addition to supervising the effort, M3's Laurie Tahija also reviewed and approved design criteria, flow sheets and equipment lists for the metallurgical processes.

3.2 TRANSPORTATION

Associated Engineering (B.C.) Ltd. assisted by Lauga & Associates Consulting, Ltd. performed updates of the studies of transportation options including selection and design of the access road route. Associated Engineers and Lauga also prepared a report on port facility options. Ray Korpela is the official contact for Associated Engineers, and Tom Lauga, P. Eng. is the official contact for Lauga and Associates.

The transportation costs for concentrates and bulk commodities used in the estimate are based on information from Trimac. The concentrate storage and load out was based on criteria developed by Alaska Industrial Development and Export Authority (AIDEA). M3 performed the research for and developed costs for handling and freight.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 LOCATION

The Casino porphyry copper-gold-molybdenum deposit is located at latitude 62° 44'N and longitude 138° 50'W (NTS map sheet 115J/09, 10 and 15), in west central Yukon, in the north-westerly trending Dawson Range mountains, 300 km northwest of the territorial capital of Whitehorse. Figure 1-3 in Section 1 is a map showing the location of the Casino property in relation to the Yukon, British Columbia and the Northwest Territories (Source: Yukon Highway Map, Yukoninfo.com). The property covers a total area of 13,124 ha.

The Yukon has a population of approximately 40,800 people. Whitehorse is the nearest commercial and population centre to the project property, with a population of approximately 30,000 people. Projected land access to Whitehorse would be 380 km via the Village of Carmacks. No human settlements can be described as "local." The Village of Carmacks is located about 150 km ESE, and the settlement of Pelly Crossing is about 115 km ENE. Beaver Creek, a village on the Alaskan Highway, is located about 112 km WSW. Fairbanks, Alaska is 500 km WNW.

The Arctic Circle is 430 km to the north. The Yukon River flows about 16 km north of the site. Yukon Highway 1, the Alaskan Highway, is about 110 km west at the nearest point. Yukon Highway 2, the Klondike Highway, is about 100 km to the east at the nearest point. No year-round roads reach the property.

The international border and Alaska are about 111 km to the west at the nearest point. British Columbia is south approximately 300 km. The closest port is Skagway, Alaska.

Exploration and mining projects in the area include the following:

- To the west, Newmont is developing the Coffee project. The project is currently at the pre-feasibility stage and is undergoing environmental assessment under the Yukon Environmental and Socioeconomic Assessment Act (YESAA). They are also active with exploration on their project.
- To the north and to the west, White Gold Corp. has a large number of claims and is actively exploring them.
- Approximately 100 km to the east, Minto Explorations Ltd. operates the Minto Mine, which produces copper-silver-gold concentrate that is shipped through the port of Skagway.

The project is located on Crown land administered by the Yukon Government and is primarily within the Selkirk First Nation traditional territory. The Tr'ondek Hwechin traditional territory lies to the north and the proposed access road crosses into Little Salmon Carmacks First Nation traditional territory to the south. The White River First Nation and Kluane First Nation are also potentially impacted by the project.

4.2 LAND POSITION AND STATUS

4.2.1 Property Description

The Dawson Range forms a series of well-rounded ridges and hills that reach a maximum elevation of 1,675 m above mean sea level (ASL). The ridges rise above the Yukon Plateau, a peneplain at approximately 1,200 m ASL, which is deeply incised by the mature drainage of the Yukon River watershed.

The characteristic terrain consists of rounded, rolling topography with moderate to deeply incised valleys. Major drainage channels extend below 1,000 m elevation. Most of the project lies between the 650 m elevation at Dip Creek and an elevation of 1,400 m at Patton Hill. The most notable local physical feature is the Yukon River which flows to the west about 16 km north of the project site.

The mean annual temperature for the Casino Project area is estimated to be -2.7°C, with minimum and maximum monthly temperatures of -18.1°C and 11.1°C occurring in January and July, respectively. The mean monthly temperature values are presented in Table 5-1 in Section 5. The Mean Annual Precipitation (MAP) for the Casino Project area is estimated to be 500 mm, with 65% falling as rain and 35% falling as snow.

Characteristic wildlife in the region includes caribou, grizzly and black bear, Dall sheep, moose, beaver, fox, wolf, hare, raven, rock and willow ptarmigan, and golden eagle.

The tops of hills and ridges are sparsely covered by tundra and buckbrush, with boreal forest covering valley floors and slopes below 1,200 m of elevation. Vegetation consists of black and white spruce forests with aspen and occasional lodgepole pine. Black spruce and paper birch prevail on permafrost slopes. Balsam poplar is common along floodplains. Scrub birch and willow “buckbrush” form extensive stands in subalpine sections from valley bottoms to well above the tree line.

4.2.2 Environmental

See Section 20 for a list of permits either obtained or in progress. No environmental liabilities are expected to impact the Project.

4.2.3 Mineral Tenure

The Casino Property lies within the Whitehorse Mining District and consists of a total of 1,136 full and partial Quartz Claims, and 55 Placer Claims acquired in accordance with the Yukon Quartz Mining Act. The total area covered by Casino Quartz Claims is 21,288 ha. The total area covered by Casino Placer Leases is 490.34 ha. The 825 quartz claims (of a total of 1,136 claims) comprise the initial Casino Property and 311 claims comprise the Canadian Creek Property acquired in 2019. The claims are registered in the name of, and are 100%-owned by, Casino Mining Corp. (CMC), a wholly owned subsidiary of Western Copper and Gold Corporation (Western). A list of claims is provided in Appendix B.

The historical claims held by prior owners of the project and transferred as part of 2006 Western Copper’s plan of arrangement with Lumina Resources Corp. (“Lumina”) consist of 83 Casino “A” claims covering an area of 1,154 ha, 23 claims in the “JOE” block covering an area of 323.63 ha and 55 Casino “B” claims covering an area of 929.93 ha, 9 claims of which were repurchased from Cariboo Rose Resources Ltd. (“Cariboo Rose”) in November 2016 pursuant to an early exercise of 2020 Casino B option agreement. 46 of the Casino “B” claims were reacquired in July 2019 pursuant to the Canadian Creek Property Purchase Agreement, described in Section 4.2.4 in more detail. The Casino Deposit lies entirely on the Casino “A” claims.

CMC has significantly expanded the area of its mineral property by the staking and acquisition of mineral claims. The 188 VIK mineral claims, covering an area of 3,440 ha, were staked in June 2007 by CRS Copper Resources Corp (“CRS”), a predecessor of CMC. In June 2008, an additional 94 “CC” claims covering an area of 1,930 ha, 8 BL claims covering area of 157.24 ha, and 63 “BRIT” claims covering an area of 1,218 ha, were staked by CRS. In October 2009, CRS staked 136 AXS mineral claims, covering an area of 2,763 ha. In May of 2010, CRS staked an additional 63 AXS claims, covering an area of 1,254 ha. In 2011, CRS staked 18 FLY claims covering 327 ha. In May 2016, 87 PAL claims were staked by CMC, covering 1,818.18 ha. In July 2019, CMC acquired additional 311 mineral claims from Cariboo Rose that comprise the Canadian Creek Property and covering area of 6,001.47 ha. In September 2019, CMC staked 53 CAS19 claims covering an area of 759.88 ha.

4.2.4 Ownership and Agreements

CMC is a successor in title to the Casino Property pursuant to the Plan of Arrangement completed on October 17, 2011.

CRS, a predecessor of CMC, acquired the Casino A, B and JOE claims, comprising the historical Casino property, on August 9, 2007 by exercising its option pursuant to a Letter Agreement dated July 15, 2002 ("2002 Option") with Great Basin Gold Ltd. ("Great Basin"). The Casino deposit lies entirely on the Casino A claims.

On December 21, 2012, CMC entered into the Net Smelter Returns Royalty Agreement (the "NSR Royalty Agreement") with 8248567 Canada Ltd. ("8248567 Canada"), whereby the 2.75% Net Smelter Return Royalty ("NSR") was established on all Casino claims excluding fifty-five (55) Casino B Claims. As consideration for purchasing the 2.75% NSR, 8248567 Canada cancelled the existing 5% NPR (except on Casino B Claims).

On November 2, 2016, pursuant to the Early Exercise and Purchase Agreement (the "Early Exercise and Purchase Agreement"), Cariboo Rose exercised its right to acquire fifty-five (55) Casino B Claims, as described in the option agreement dated May 2, 2000 (the "Casino B Option Agreement") between Cariboo Rose and CMC (a successor to title by virtue of 2002 Option). As part of the Early Exercise and Purchase, CMC reacquired nine (9) Casino B Claims (the "Nine Casino B Claims"). Forty-six (46) Casino B Claims (the "Forty-Six Casino B Claims") were transferred to Cariboo Rose and became part of the Canadian Creek Property owned by Cariboo Rose.

On August 28, 2019, CMC and Cariboo Rose completed the Canadian Creek Property Purchase Agreement (the "Canadian Creek Property Purchase Agreement"), whereby Forty-Six Casino B Claims were reacquired as part of the Canadian Creek Property consisting of a total of 311 mineral claims.

4.2.5 Agreements and Royalties

Certain portions of the Casino property remain subject to certain royalty obligations. The surviving royalties and agreements are as follows:

- 2.75% NSR on the claims comprising the Casino project in favour of Osisko Gold Royalties Ltd. ("Osisko Gold") pursuant to the Royalty Assignment and Assumption Agreement dated July 31, 2017 when 8248567 Canada assigned to Osisko Gold all of its rights, title and interest in the 2.75% NSR.
- 5% Net Profits Interest (the "NPI"), as defined in the Casino B Option Agreement, remains in effect on the Casino B Claims and $1 million payment is required to be made to the original optionor within 30 days of achieving a commercial production decision.
- 5% Net Profit Interest Royalty (the "NPI Royalty") presently held by Archer-Cathro and Associates on the ANA claims pursuant to the NPI Royalty Agreement dated December 4, 1990 (the "NPI Royalty Agreement") among Big Creek Resources Ltd., Rinsey Mines Ltd., and Renoble Holdings Inc.

4.2.6 Placer Claims

In the summer of 2010, Western staked a 5-mile Placer Lease along Casino Creek and a 3-mile Placer Lease along Britannia Creek. In 2011, these leases were converted to claims. In 2014, 30 placer claims on Britannia Creek were dropped and presently, Western, through CMC, owns 55 placer claims on Casino Creek.

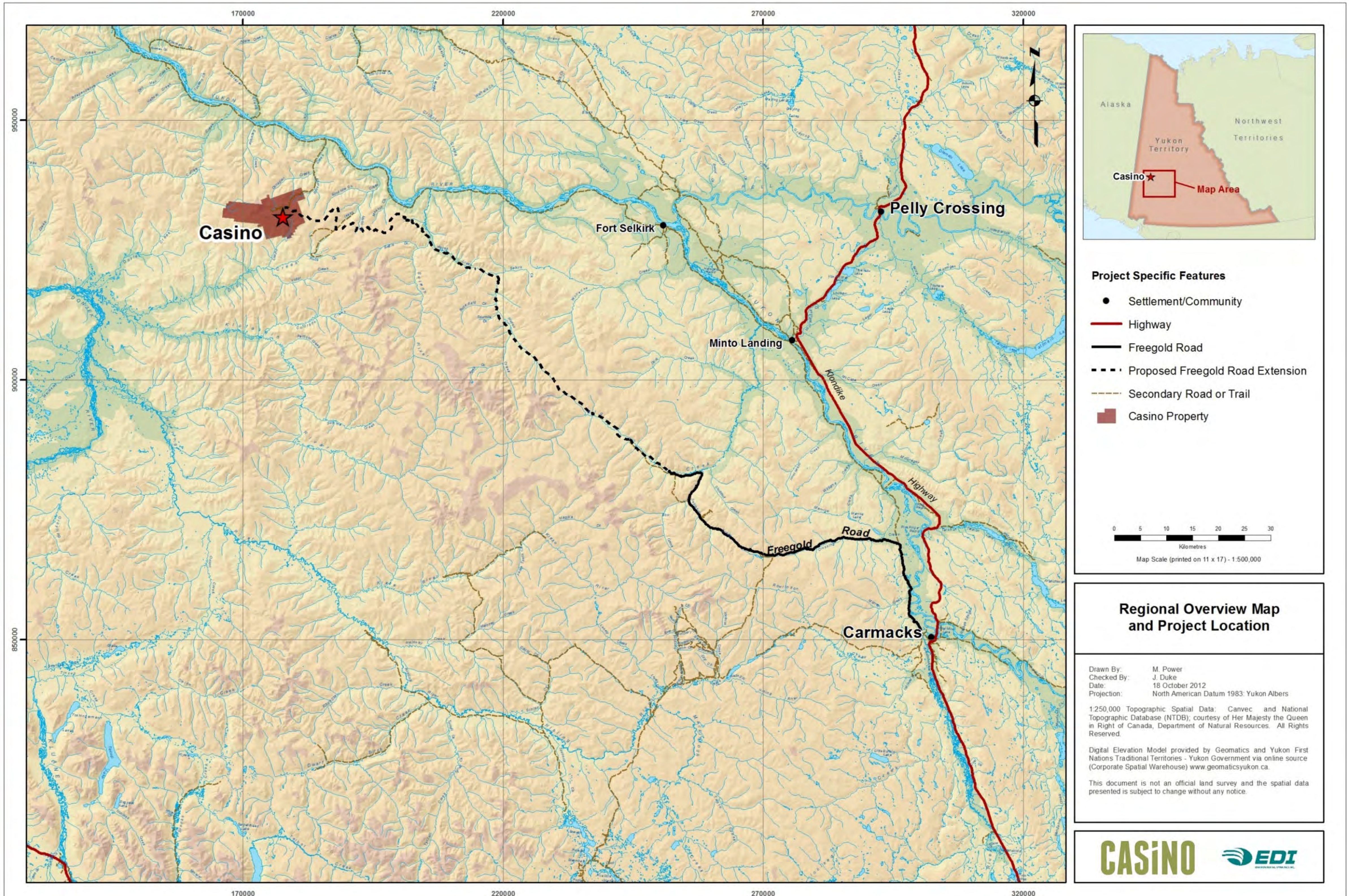


Figure 4-1: Project Road Access Map

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 ACCESSIBILITY

The Casino Mine is located in Central Yukon, at approximately N62° 44' 25", W138° 49' 32" roughly 150 km due northwest of Carmacks. Current site access is by small aircraft using the existing 760 m airstrip, by winter road from the west, and from a seasonally accessible road extending from a barge landing at the Yukon River.

The barge landing area at Britannia Creek and the Yukon River was prepared in 2010 and the lower 10 km of the 23 km access road from the landing to the site was realigned.

5.2 PHYSIOGRAPHY

The Casino property is located in the Dawson Range, a north-westerly trending belt of well-rounded ridges and hills that reach a maximum elevation of about 1,675 m. The hills rise above the Yukon Plateau, at about 1,250 m and deeply incised by mature dendritic drainages. Although the Dawson Range escaped Pleistocene continental glaciation, minor alpine glaciation has produced a few small cirques and terminal moraines.

The deposit area is situated on a small divide. The northern part of the property drains to Canadian Creek and Britannia Creek into the Yukon River. The southern part of the property flows southward via Casino Creek to Dip Creek to the Donjek River and northward to the Yukon River.

Outcrop is rare on the property. Soil development is variable ranging from coarse talus and immature soil horizons at higher elevations to a more mature soil profile and thick organic accumulations on the valley floors.

5.3 CLIMATE

The climate in the Dawson Range is subarctic. Permafrost is widespread on north-facing slopes, and discontinuous on south-facing slopes. CMC installed an automated weather station at the site in 2009 and collected a certain amount of data.

The climate at the Casino Project area can generally be described as continental and cold. Winters are long, cold and dry, with snow generally on the ground from late September through mid-May. Summers are short, mild and wet, with the greatest monthly precipitation falling in July. The climate and hydrology at the Project site have been assessed based on both short-term site data and longer-term regional data. Site data are available from a program operated from 1993 to 1995 and from the current program that was initiated in 2008.

The mean annual temperature for the Casino Project area is estimated to be -2.7°C, with mean minimum and maximum monthly temperatures of -18.1°C and 11.1°C occurring in January and July, respectively. The mean annual precipitation (MAP) for the Casino Project area is estimated to be 500 mm, with 65% falling as rain and 35% falling as snow. The mean monthly temperatures and precipitation are presented in Table 5-1.

Table 5-1: Mean Monthly Temperature and Precipitation Values

	Parameter	
Month	Precipitation (mm)	Temperature (°C)
Jan	25	-18.1
Feb	19	-14.2
Mar	16	-8.2
Apr	15	-0.1
May	42	5.7
Jun	74	9.8
July	103	11.1
Aug	65	9.1
Sept	49	4.4
Oct	35	-3.3
Nov	31	-12.7
Dec	26	-16.5
Annual	500	-2.7

The estimated average annual lake evaporation is 308 mm, based on climate data collected at site and used in conjunction with long-term regional climate data.

Based on the estimated MAP of 500 mm and a rain/snow ratio of 0.65/0.35, the annual snowfall value for Casino was estimated to be 175 mm. This is generally consistent with the 140 mm mean annual maximum snowpack value (snow water equivalent, SWE) recorded in the Project area at the Casino Creek snow course station (09CD-SC01) operated by the Yukon Department of Environment (1977-2009), Water Resources Branch.

Based on the complete years of snowpack data, the average monthly snowmelt distribution for the Casino Project area was estimated to be 40% in April and 60% in May, although there is considerable variation from year to year.

5.4 WATER RIGHTS

It is assumed that water rights can be obtained for withdrawal of water from the Yukon River.

5.5 POWER AVAILABILITY

There is no utility power available to serve the site. The Project will need to generate its own power.

5.6 SURFACE RIGHTS

CMC has sufficient rights and available land at the Project site for a mine, tailing storage areas, waste disposal areas, heap leach pad areas and process plant areas.

6 HISTORY

The first documented work on the Casino Property was the working of placer claims in the area of the Casino Deposit in April 1911, following a placer gold discovery on Canadian Creek by J. Britton and C. Brown. A study by D.D. Cairnes, of the Geological Survey of Canada in 1917, recognized huebnerite ($MnWO_4$) in the heavy-mineral concentrates of the placer workings and also that the gold and tungsten mineralization was derived from an intrusive complex on Patton Hill. During the Second World War, a small amount of tungsten was recovered from placer workings. The total placer gold production from the area of the property is unknown, but during the period 1980-1985 placer mining yielded about 50 kg (1,615 troy ounces) of gold.

The first mineral claims at Casino were staked by N. Hansen in 1917; however, the first recorded bedrock mineral discovery occurred in 1936 when J. Meloy and A. Brown located silver-lead-zinc veins approximately 3 km south of the Canadian Creek placer workings. Over the next several years the Bomber and Helicopter vein systems were explored by hand trenches and pits. In 1943, the Helicopter claims were staked and in 1947 the Bomber and Airport groups were staked.

Lead-silver mineralization remained the focus of exploration on the property until 1968. Noranda Exploration Co Ltd. optioned the property in 1948 and Rio Tinto optioned it again in 1963. During this time trenching, mapping and sampling were conducted.

L. Proctor purchased the claims in 1963 and formed Casino Silver Mines Limited to develop the silver-rich veins. The silver-bearing veins were explored and developed intermittently by underground and surface workings from 1965 to 1980. In total, 372.5 tonnes of hand-cobbled argentiferous galena, assaying 3,689 g/t silver (Ag), 17.1 g/t gold (Au), 48.3% lead (Pb), 5% zinc (Zn), 1.5% copper (Cu) and 0.02% bismuth (Bi), were shipped to the smelter at Trail, British Columbia.

In 1963, B. Hestor first recognized that the area had potential for a porphyry copper deposit, but his observations did not become generally known. In 1967, the porphyry potential was recognized again, this time by A. Archer and separately by G. Harper. Based on the recognition of porphyry copper potential, the Brynelsen Group acquired Casino Silver Mines Limited, and from 1968 to 1973 exploration was directed jointly by Brameda Resources (Brameda), Quintana Minerals (Quintana), and Teck Corporation towards a porphyry target. Exploration, including extensive soil sampling surveys, geophysical surveys and trenching programs, eventually led to the discovery of the Casino deposit in 1969.

From 1969 to 1973, various parties, including Brameda Resources, Quintana Minerals and Teck Corporation, conducted drilling on the property. During this period 5,328 m of reverse circulation drilling in 35 holes, and 12,547 m of diamond drilling in 56 holes, were completed.

Archer, Cathro & Associates (1981) Ltd. (Archer Cathro) optioned the property in 1991 and assigned the option to Big Creek Resources Ltd. In 1992, a program consisting of 21 HQ (63.5 mm diameter) holes totalling 4,729 m systematically assessed the gold potential in the core area of the deposit for the first time.

In 1992, Pacific Sentinel Gold Corp. (PSG) acquired the property from Archer Cathro and commenced a major exploration program. The 1993 program included surface mapping and 50,316 m of HQ (63.5 mm diameter) and NQ (47.6 mm diameter) drilling in 127 holes. All but one of the twenty-one 1992 drill holes were deepened in 1993.

PSG drilled an additional 108 drill holes totalling 18,085 m in 1994. This program completed the delineation drilling commenced in 1993. PSG also performed metallurgical, geotechnical and environmental work which was used in a scoping study in 1995. The scoping study envisioned a large-scale open pit mine and a conventional flotation concentrator that would produce a copper-gold concentrate for sale to Pacific Rim smelters.

First Trimark Resources and CRS Copper Resources obtained the property and, using the PSG data, published a Qualifying Report on the property in 2003 to bring the resource estimate into compliance with National Instrument 43-101 requirements. The two firms combined to form Lumina Copper Corporation in 2004. An update of the Qualifying Report was issued in 2004.

Western Copper Corporation acquired Lumina Copper Corporation, and therefore the Casino Deposit, in November 2006. In the fall of 2011, Western Copper Corporation spun out all other assets except the Casino Deposit and changed its name to Western Copper and Gold Corporation (Western).

In 2007, Western conducted an evaluation of the Bomber Vein System and the southern slope of Patton Hill by VLF-EM, Horizontal Loop EM and soil geochemical surveying. Environmental baseline studies were also initiated in 2007.

In 2008, Western reclaimed the old camp site, constructed a new exploration camp next to the Casino airstrip and drilled three holes (the camp water well and two exploration diamond drill holes) totalling 1,163 m. The main purpose of the drilling was to obtain fresh core samples for the metallurgical and waste characterization tests. Both exploration holes twinned PSG's holes to confirm historical copper, gold and molybdenum grades. Later that year, M3 Engineering & Technology Corporation produced a pre-feasibility study for Western Copper and Gold Corporation.

In 2009, Western completed 22.5 km of DC/IP surveying and MT surveying using the Titan system developed by Quantec Geosciences Ltd. As well, the company drilled 10,943 m in 37 diamond drill holes, of which 27 holes were infill holes drilled to upgrade the previously designated Inferred Resource and non-defined material to the Measured and Indicated resource categories. Infill drilling covered the north slope of Patton Hill that was mapped as a "Latite Plug" on PSG maps. The drilling also identified supergene Cu and Mo mineralization in this area. The remaining 10 holes, totalling 4,327 m, were drilled to test geophysical targets.

In 2010, Western, under the direction of the Casino Mining Corporation (CMC), a wholly-owned subsidiary of Western, completed infill and delineation drilling mostly to the north and west of the deposit, as outlined by PSG. The drilling program also defined hypogene mineralization at the southern end of the deposit. In addition, the company drilled a series of geotechnical holes at the proposed tailings embankment area and within the pit, and several other holes for hydrogeological studies. The geotechnical drilling continued in 2011 (41 holes, 3,163 m) and 2012 (6 holes, 228 m). This work culminated in the publishing of a pre-feasibility study in 2011 and a feasibility study in 2013.

In 2019, CMC carried out a program of infill drilling designed to convert mineralization from the Inferred Resource category, located along the margin of the deposit, to the Indicated Resource category.

A breakdown of drilling by Western and CMC from 2010 to the end of 2019 is as follows:

- 124 exploration holes for 27,365.37 m.
- 11 combined hydrogeological and geological holes for 1,689.58 m.
- 53 geotechnical holes in the proposed tailings embankment, heap leach pad, plant site, waste rock storage site, airstrip, access road and water well areas, for 3,786.54 m.
- 5 holes for 1,570.63 m for the metallurgical sample.

The total meterage drilled by Western and CMC from 2008 to the end of 2019 is 46,639.37 m.

In mid 2019, CMC acquired the adjacent property to the west, referred to as the Canadian Creek property, from Cariboo Rose Resources Ltd.

Exploration on the Canadian Creek property dates from 1992 when Archer Cathro & Associates staked the Ana Claims. In 1993, Eastfield Resources Ltd. (Eastfield) acquired the Ana Claims, expanded the Ana Claim block and explored the expanded property with soil grids, trenching and drilling (Johnston, 2018). This work was directed at the discovery of additional porphyry deposits. The 1993 program was followed by extensive field programs in 1996, 1997 and 1999 consisting of induced polarization (IP) surveying, road construction and trenching on the Ana, Koffee, Maya and Ice claims. In 2000, another drill campaign was undertaken by Eastfield on the Ana, Koffee Bowl, and the newly acquired Casino "B" claims located immediately to the west of the Casino deposit. The Casino "B" holes confirmed the existence of gold mineralization first discovered here in 1994 by PSG, which encountered 55.17 m averaging 0.71 g/t gold in hole 94-319. Modest exploration programs were conducted, mostly over the Casino "B" area, in 2003, 2004 and 2005. In 2007, a five-hole core drilling program at Casino "B" targeted gold and copper-in-soil anomalies and ground magnetic high features.

The discovery in 2009 of gold mineralization on Underworld Resources' White Gold property sparked new interest in gold exploration in the Yukon. This led to the implementation of a major exploration program at Canadian Creek directed at the gold potential of the property, some distance from the area of previous work, and focusing on porphyry copper mineralization.

A soil survey revealed extensive areas returning greater than 15 ppb gold in soils with associated anomalous values in arsenic (As), bismuth (Bi) and antimony (Sb). The anomalous area extends for over 4 km in an east-northeast direction. The induced polarization (IP) surveys revealed numerous strong chargeability highs, many of which coincide with the gold-in-soil anomalies.

Ten diamond drill holes were targeted within the new grid. Results include numerous intervals of anomalous gold values, commonly associated with elevated arsenic, antimony and bismuth. The mineralization is hosted in both gneiss and granodiorite, commonly in clay-altered structures, sheeted pyrite veins or quartz-carbonate veins. With few exceptions, gold grades are less than 1 gram per tonne (g/t) and widths are less than 3 m.

Resampling of old trenches in other parts of the property was undertaken to verify significant historical gold results. In trench Tr-2, excavated in 1993 and located in the Ana Pass area, a grab sample of a tourmaline-pyrite-quartz altered intrusive rock returned 2,516 ppb gold. In the Casino "B" area, trench 9076-C averaged 376 ppb gold over 50 m, including a 10 m interval of 927 ppb.

In 2011, additional soil sampling, ground geophysical surveying and trenching were completed. The soil sampling completed the coverage of the entire Canadian Creek property and increased the known extent of the arsenic anomalies. A limited-extent induced polarization survey identified two zones of chargeability with values greater than 20 mv/V. The trenching program identified a number of areas with anomalous gold values, ranging from sub-detection level up to 2,890 and 4,400 ppb Au.

As a follow up on the 2011 program, a modest 2016 program of trenching, prospecting and in-fill soil sampling was carried out by Cariboo Rose Resources Ltd. (Cariboo Rose), which had acquired the property from Eastfield. Trenching conducted in three areas of the Ana portion of the Canadian Creek property returned locally anomalous gold, and widely spread anomalous arsenic, bismuth, antimony and locally high silver values, generally confined to narrow structures.

Cariboo Rose's 2017 exploration program consisted of surface work directed at the Kana and Malt West gold targets and a reverse circulation (RC) drill program that tested a variety of gold targets across the property. A total of 2,151.27 m of reverse circulation (RC) drilling was conducted in 24 holes. This work confirmed gold and silver mineralization to be limited to narrow (less than 3 m wide) structures rarely traceable over more than 100 m.

Table 6-1: Summary of Work on the Canadian Creek Property by Previous Owners Since 1993

Category	Summary
Induced Polarization Survey	87 line km
Ground Magnetic Surveys	586.8 line km
Mechanical Trenching	170 trenches and pits (many did not reach bedrock)
Trench Samples	453 samples
Soil Samples	10,129 samples
Rock Samples	835 samples
Road Construction	16 km
Diamond Drilling	6,069.24 m in 40 holes
	(includes 1970 Bremada and 1993-94 Pacific Sentinel holes on the current Casino "B" area)
Reverse Circulation (RC) Drilling	2,151.27 m in 24 holes

Source: Johnston, 2018

7 GEOLOGICAL SETTING AND MINERALIZATION

7.1 REGIONAL GEOLOGY

The Casino deposit occurs within the Yukon-Tanana terrane (YTT), a northwest-southeast trending accreted terrane comprising Neoproterozoic to Upper Cretaceous metaigneous and metasedimentary rocks abutting the southwest side of the Tintina Fault Zone northeast of the property. This was previously described as an overlapping zone of the Yukon Cataclastic Terrane to the north and the Yukon Crystalline Terrane to the south (Templeman-Kluit, 1976). An elongate band of ultramafic rocks, 1 km north of the Casino deposit, may occur along a major tectonic suture. The YTT in this area has undergone emplacement of the 104 Ma Dawson Range Batholith, part of the Whitehorse Intrusive Suite. The Dawson Range Batholith extends WNW for about 300 km, roughly parallel to the regional orientation of strata comprising the YTT also known as the Yukon Metamorphic Complex.

The YTT is dominated by Paleozoic rocks of the Yukon Metamorphic Complex with scattered intrusions of the Coffee Creek Suite which are petrographically distinct from the Dawson Range Batholith. The YTT in the Dawson Range area is comprised of metasedimentary rocks of the Proterozoic to Devonian Snowcap assemblage, rocks of the Devono-Mississippian Wolverine Creek Metamorphic Suite, (Johnston, 1995) and rocks of the Permian Sulphur Creek assemblage (website, Yukon Geological Survey, 2020). Snowcap assemblage rocks comprise quartzites, pelites, psammites and marble (YGS, 2020). Stratigraphy of the Wolverine Creek Suite comprises sedimentary and igneous protoliths (Tempelman-Kluit, 1974; Payne et al., 1987). These meta-sedimentary rocks consist mainly of quartz-feldspar-mica schist and gneiss, quartzite, and micaceous quartzite, while the meta-igneous unit includes biotite-hornblende-feldspar gneiss and other orthogneisses, as well as hornblende amphibolite (Selby & Nesbit, 1997).

During the mid-Cretaceous period, Wolverine Creek suite rocks in this area were intruded by the Dawson Range Batholith, subsequently intruded by the Casino Intrusive Suite (Selby et al., 1999). The Dawson Range Batholith has incorporated scattered roof–pendants and blocks of the YTT, particularly Snowcap Assemblage and Wolverine Creek Suite rocks. The Dawson Range Batholith is the main country rock of the Casino Property and is represented by a relatively homogeneous, medium- to coarse-grained, hornblende-bearing, potassic quartz diorite to granodiorite, and lesser fine- to medium-grained diorite and quartz monzonitic veins, dykes, and plugs (Tempelman-Kluit, 1974).

The Casino Intrusions, also called the Casino Plutonic Suite, have been described as a suite of quartz monzonite stocks up to 18 km across (Hart and Selby, 1998) trending west-northwest parallel to the Big Creek Lineament and its northwestern extension. Mapping by Tempelman-Kluit (1974), and successively by Payne et al. (1987), associates this Casino Plutonic Suite with the mid-Cretaceous Dawson Range Batholith. Subsequently, Johnston (1995) grouped the intrusions with the late-Cretaceous Prospector Mountain Plutonic Suite, based largely on field relationships that show stocks of the Casino Plutonic Suite cutting the Dawson Range Batholith. Subsequent age determination by Mortensen and Hart in 1998, as well as geochemistry provided by Selby et al. (1999), re-evaluated the Casino Intrusions as mid-Cretaceous fractionated magmas of the Dawson Range Batholith. Recent field relationships have proven that the ‘quartz monzonites’ of the Casino property, once thought to be separate intrusions, are actually intensely altered and recrystallized diorites of the Dawson Range Batholith.

The regional geology is illustrated in Figure 7-1, which summarizes the major units with isotopic ages. All isotopic dates are based on U-Pb ratios in zircons analysed by J.R. Mortensen.

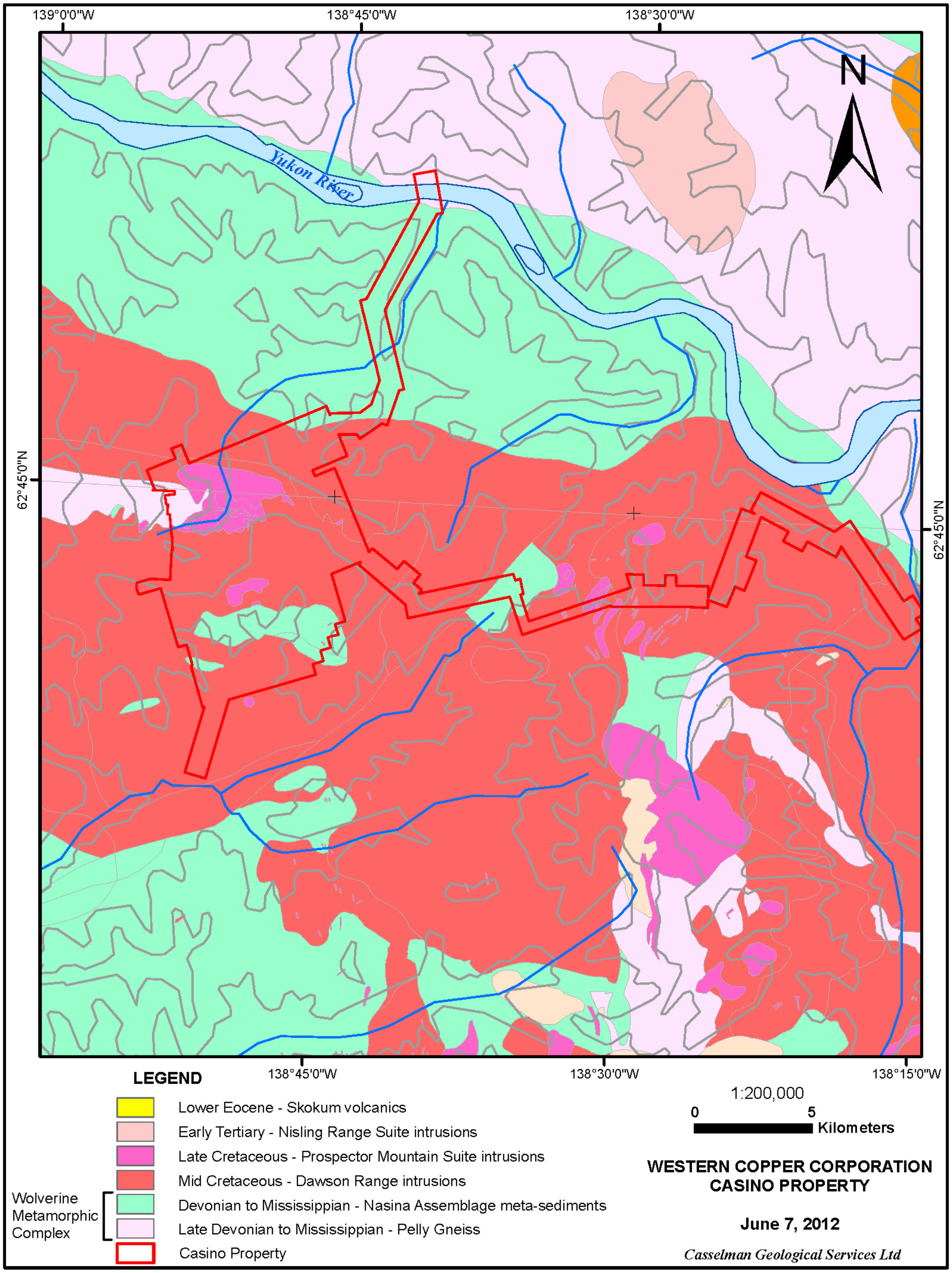


Figure 7-1: Regional Geology

During late Cretaceous time, stocks and apophyses of the Prospector Mountain Plutonic Suite were emplaced into the Dawson Range Batholith (Johnston, 1995; Selby et al, 1999). In the Casino area, this suite is represented by the 72.4 Ma Patton Porphyry intrusions, occurring as small, biotite-bearing, feldspar-porphyritic, hypabyssal rhyodacite to dacite intrusions near the centre of the deposit, and as discontinuous centimeter- to metre-wide dikes northwest of the property. Here, early phases the Patton Porphyry appear to grade into a mineralized intrusive breccia. Later, unaltered dykes of similar lithology cut surrounding hydrothermally altered and mineralized rocks (Payne et al., 1987) suggesting there are multiple phases of this unit (Bower, 1995; Selby and Creaser, 2001). Hydrothermal alteration and mineralization occur in, and adjacent to, some of these late Cretaceous intrusions.

Table 7-1: Stratigraphic Column

	Geological Unit	Isotopic Age
Late Cretaceous	PROSPECTOR MOUNTAIN PLUTONIC SUITE:	
	Intrusive Breccia (Diatreme) *Heterolithic; fine-grained matrix; angular clastic*	
	Heterolithic Intrusion Breccia *Heterolithic; Patton porphyry/potassic matrix; autobrecciated fragments*	
	Patton Porphyry *Plag-Bi Porphyry; Kf +/- Qz megacrystic porphyry*	72.4 +/-0.5 Ma
Mid-Cretaceous	DAWSON RANGE BATHOLITH:	
	Granodiorite *bi-hbld granodiorite*	104.0 +/-0.5 Ma
	Diorite *Hbld-Bi-Qtz diorite; hbld-bi diorite*	104.0 +/-0.5 Ma
Devono-Mississippian	WOLVERINE CREEK METAMORPHIC SUITE:	
	Meta-sedimentary *Micaceous Quartzite*	
	Meta-igneous *Qtz-Bi-Plag-Microcline Gneiss; KF-Qtz-Bi Gneiss; Amphibolite*	
Proterooic-Devonian	SNOWCAP ASSEMBLAGE	
	Metasedimentary: Quartzite, psammites, pelites, marble	

The Casino Property is sandwiched between parallel west-northwest-trending faults that form contacts between rocks of the Wolverine Creek Metamorphic Suite and the Dawson Range Batholith. In Figure 7-2, the fault farthest to the northeast is an extension of the Big Creek Fault interpreted as dextrally offset by 20 to 45 km. A parallel fault, 8 km to the southwest, forms the southwest boundary of a sliver of Wolverine Creek Metamorphic Suite rocks and contains outcroppings of ultramafic rocks similar to those occurring along the Big Creek Fault.

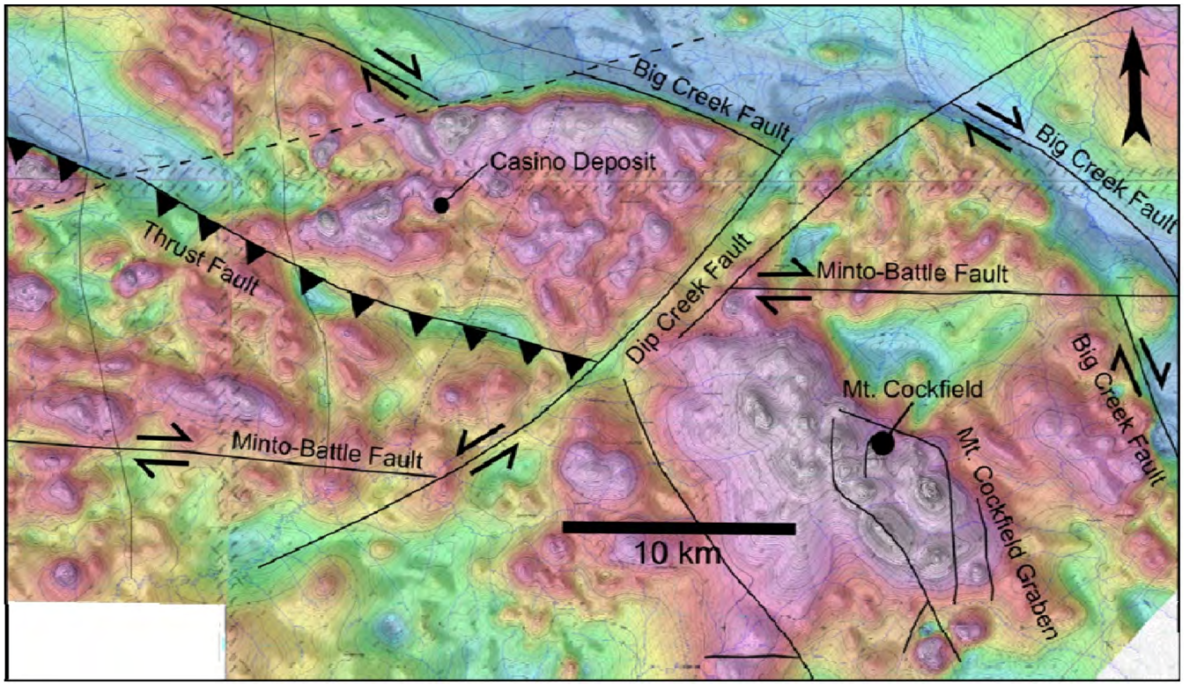


Figure 7-2: Regional Structures Overlain on Recent Aeromagnetic Survey

The Casino Property is bounded to the southeast by a northeast-trending regional structure known as the Dip Creek Fault, which has a left lateral (sinistral) displacement. The left-lateral displacement of stratigraphy along the Yukon River east of the Casino Property is a reflection of sinistral movement along this fault. The east-trending Minto-Battle Fault is also sinistrally offset by the Dip Creek Fault (Johnston, 1999). The dextrally offset Minto-Battle fault lies east of the Casino Property on the opposite side of Dip Creek, with its offset extension lying south and southwest of the Casino Property.

7.2 PROPERTY GEOLOGY

The geological setting of the Casino deposit is typical of many porphyry copper deposits. The deposit is centered on an Upper Cretaceous-age (72.4 Ma), east-west trending elongate tonalite porphyry stock, called the “Patton Porphyry” (PP), that intrudes mid-Cretaceous granitoids of the Dawson Range Batholith (WRGD) and Paleozoic schists and gneisses (YM) of the YTT. Emplacement of the Patton Porphyry tonalite stock into the older rocks caused brecciation of both the intrusive rocks and the surrounding country rocks along the northern, southern and eastern contacts of the stock. Brecciation is best developed in the eastern end of the stock, where the breccia zone can be up to 400 m wide in plan view. To the west, along the north and south contacts, the breccias narrow gradually to less than 100 m. Drilling done at the western end of the tonalite stock has revealed a late, post-mineralization explosive breccia (MX) which has obliterated the Patton Porphyry stock and any related contact breccia in this area. The late explosive breccia (diatreme) forms an elliptical body over 300 m across. It also forms narrow east – west dykes extending into the tonalite stock and surrounding granitoids and metamorphic rocks. The Patton Porphyry, intrusive breccias and late explosive breccias comprise the Casino Intrusive Complex, measuring 1.8 km by 1.0 km.

Patton Porphyry dykes extend west of the deposit for several kilometres. Locally, these dykes are associated with breccia zones developed along their margins, and may be mineralized with pyrite, chalcopyrite and molybdenite as disseminations, vein and fracture fillings.

On the northwest side of the Casino intrusive complex, a swarm of Patton Porphyry dykes and related breccias occurs. This dyke swarm is speculated to represent the upper emanation of a buried satellite stock of the main Patton Porphyry stock.

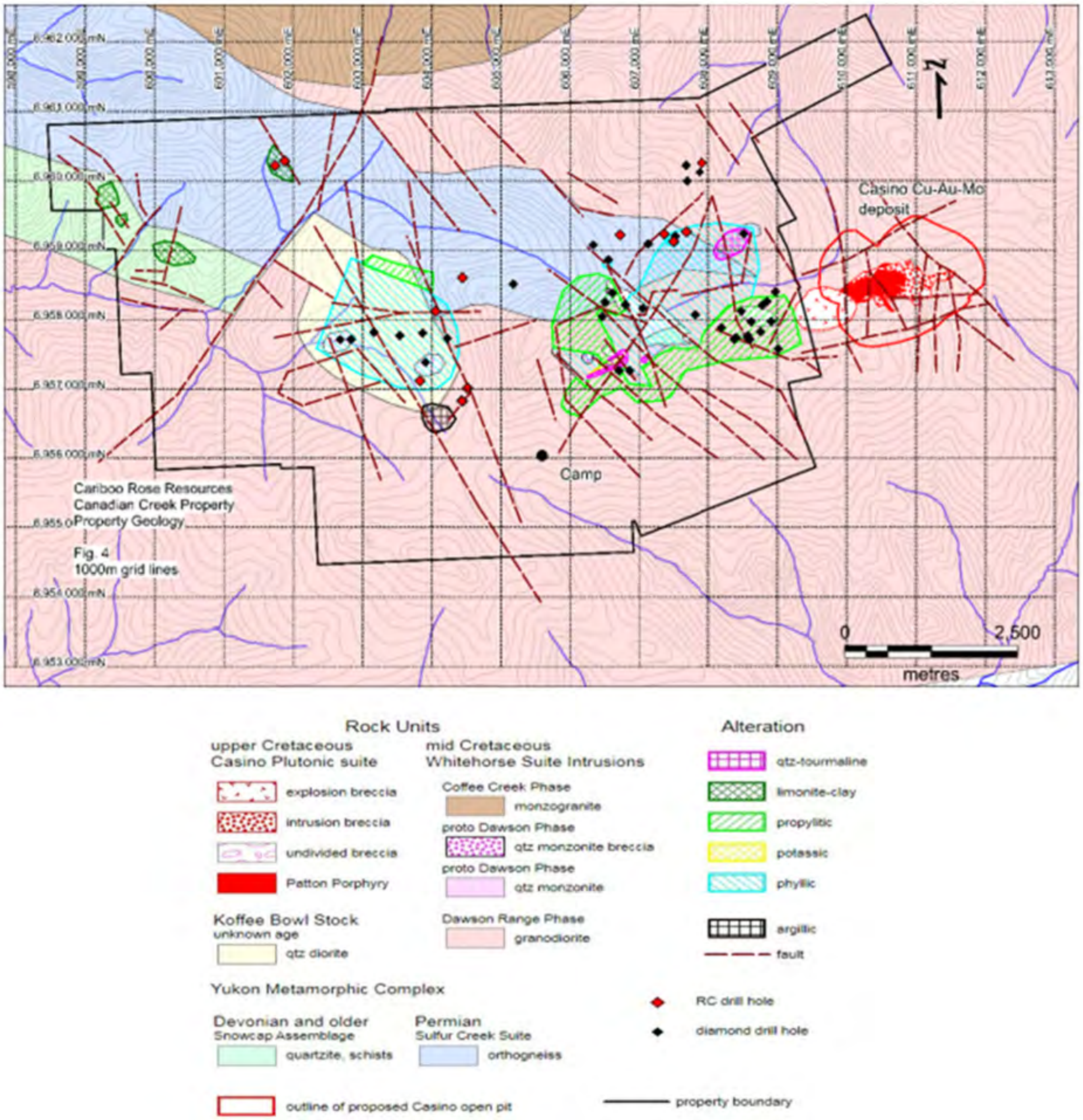


Figure 7-3: Property Geology (From R. Johnston, 2018)

7.3 MINERALIZATION

7.3.1 Hydrothermal Porphyry Alteration

Crystallization and exsolution of hydrothermal fluids from Patton Porphyry (PP) magmas produced porphyry style Cu-Mo-Au mineralization. Therefore, the Patton Porphyry, and associated Intrusive Breccia (IX), is genetically related to the Cu-Mo-Au mineralization of the deposit.

Hydrothermal alteration at the Casino property consists of a potassic core centered on and around the main Patton Porphyry body, in turn bordered by a contemporaneous, strongly developed and fracture controlled phyllic zone, a weakly developed propylitic zone, and a secondary discontinuous argillic overprint. Mineralized stockwork veins and breccias within the Casino Property are closely associated with the hydrothermal alteration.

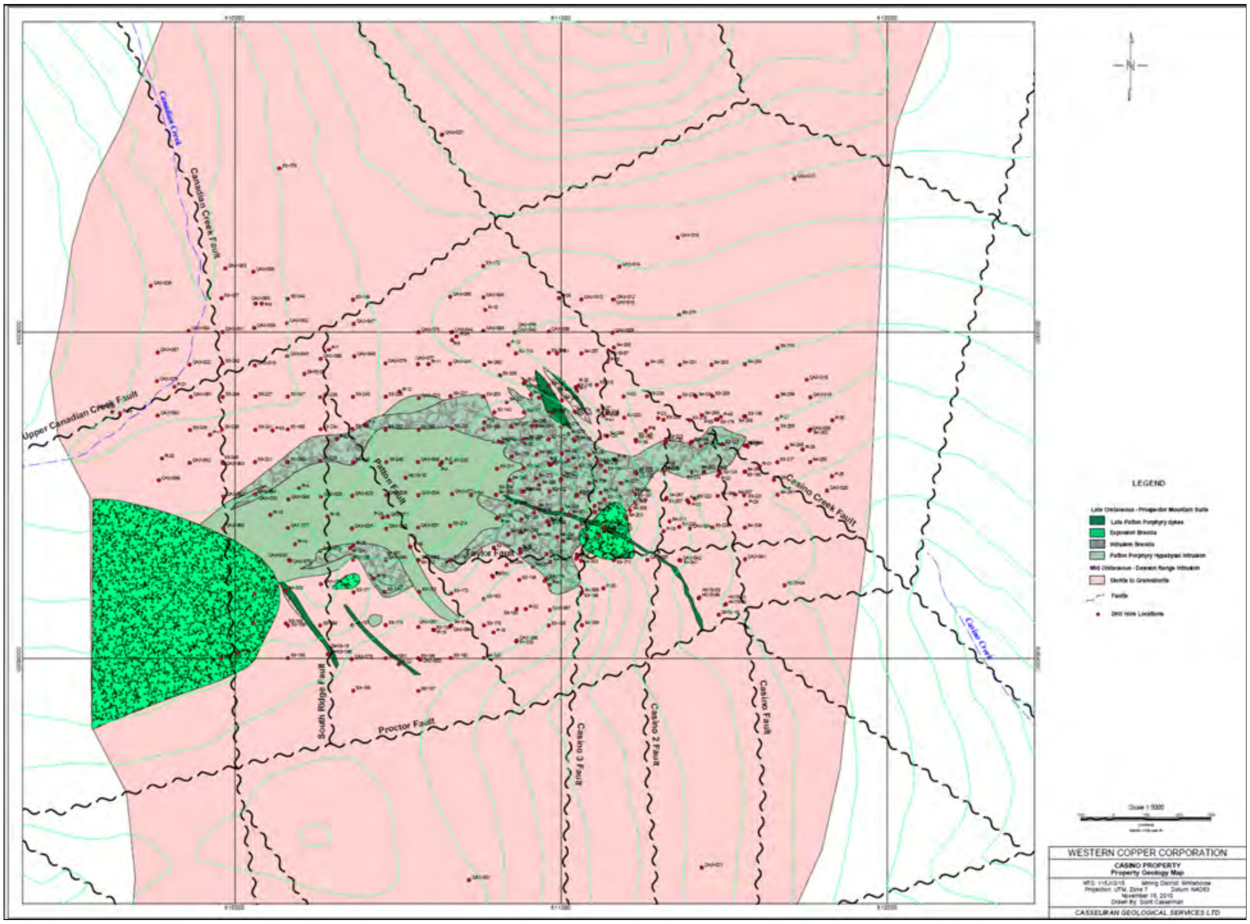


Figure 7-4: Geology of the Casino Deposit

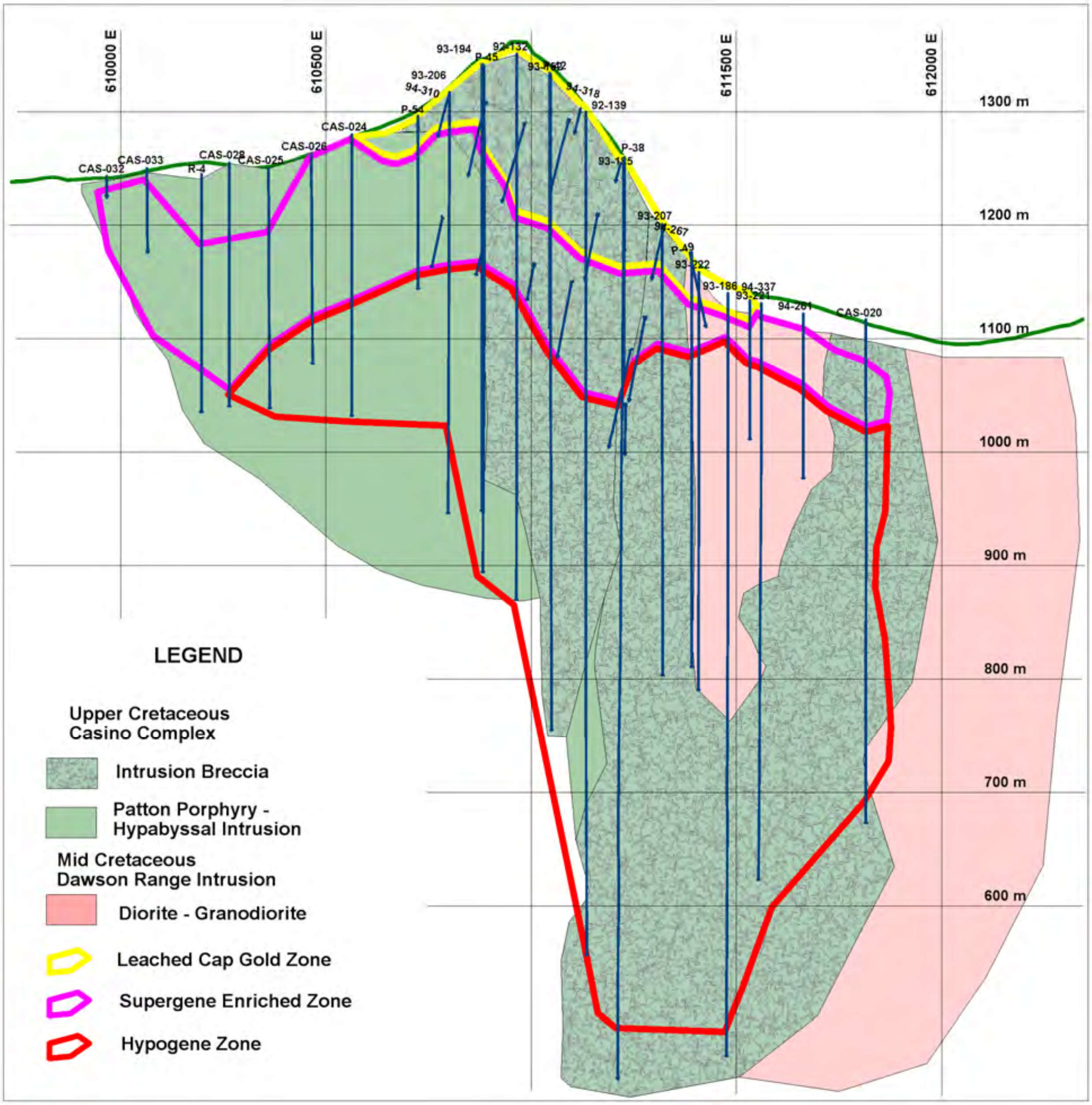


Figure 7-5: Casino Property Geology - Cross Section

Potassic alteration minerals include texturally destructive K-feldspar, biotite, magnetite and quartz with lesser hematite, purple anhydrite and gypsum. Biotite is generally felted and pseudomorphic after hornblende. Locally, magnetite forms braided veinlets. In drill core, potassic alteration is represented by dark brown to black biotite alteration and/or by pink potassium feldspar alteration.

The texturally destructive phyllic zone is found peripheral to, and locally overprinting, the potassic zone of alteration. It has a distinctive 'bleached' appearance and is locally structurally controlled. Phyllic alteration minerals include quartz, pyrite, sericite, muscovite (after biotite), and abundant tourmaline, as well as minor hematite and/or magnetite towards the potassic zone. Quartz and sericite are typically alteration minerals after potassic and plagioclase feldspars. Biotite alters to muscovite or titanite, and hornblende alters to chlorite, calcite, quartz and biotite. Tourmaline forms radiating disseminations and veinlets. Sulphide content is typically high, with pyrite ranging from 5-10% throughout, as disseminated blebs or cores to quartz "D" veins.

Where intense phyllic alteration overprints potassic alteration, relict textures are destroyed and minerals are recrystallized, commonly to equal portions of quartz, plagioclase, and K-feldspar, and including up to 10 percent biotite, trace apatite and titanite. Strongly zoned plagioclase and locally kinked biotite form subhedral lathes, surrounded by K-feldspar, locally strained quartz, and biotite. The overall colour is pale pink.

Propylitic alteration is rare on surface but forms a wide halo around the deposit in gradational contact with the inner potassic alteration. Alteration minerals include epidote, chlorite and calcite, with lesser carbonate, clay, sericite, pyrite and albite. Hornblende and biotite are completely chloritized, whereas feldspars are relatively fresh and textures are generally well-preserved.

In typical porphyry copper deposits, advanced argillic alteration occurs above the phyllic alteration. It appears that, on the Casino property, all evidence of advanced argillic alteration has been eroded or destroyed.

Secondary argillic alteration is closely associated with the supergene zone and may appear locally as patches or pockets within potassic and phyllic alteration zones. It is poorly developed, appears bleached or pale green, contains abundant clays (kaolinite, montmorillonite) and local chlorite and/or carbonate. In drill core, this unit may be recognized by distinctive "pock-marks" along the surface of the core.

7.3.2 Supergene Porphyry Mineralization

The Casino deposit is unusual among Canadian porphyry deposits as it has a substantially preserved outcropping gold-bearing oxidized "Leached Cap", an upper well-developed copper-gold enriched "Supergene Zone" and a lower copper-gold bearing "Hypogene Zone". The Supergene Zone is comprised of the Supergene Oxide (SOX) zone and the more extensive Supergene Sulphide (SUS) zone. Table 7-2 summarizes the main minerals identified in the Leached Cap and Supergene zones.

Leached Cap Mineralization (CAP)

The Leached Cap (oxide gold zone) is copper-depleted due to supergene alteration, mainly leaching, processes, and has a lower specific gravity relative to the other supergene zones. It averages 70 m thick and is characterized by boxwork textures filled with jarosite, limonite, goethite, and hematite. This weathering has completely destroyed rock textures and has replaced most primary minerals with clay. The resulting rock is pale gray to cream in colour and is friable to the touch, and the clay is commonly stained yellow, orange, and/or brown by iron oxides. The weathering is most intense at the surface and decreases with depth.

Supergene Oxide Mineralization (SOX)

The poorly defined Supergene Oxide zone (SOX) is copper enriched with trace molybdenite. It occurs as a few perched bodies within the leached cap, likely due to more recent fluctuations in the water table. This zone is thought to be related to present-day topography and is best developed where oxidation of earlier secondary copper sulphides occurs above the water table, typically on well drained slopes. Where present, the supergene oxide zone averages 10 m thick, and may locally contain chalcanthite, malachite and brocanthite, with minor azurite, tenorite, cuprite and neotocite. Where present, the supergene copper oxide zone grades into the better-defined supergene copper sulphide zone.

Supergene Sulphide Mineralization (SUS)

Supergene copper mineralization occurs in a weathered zone up to 200 m deep, below the leached cap and above the hypogene zone. It has an average thickness of 60 m and is positively correlated with high grade hypogene mineralization, high permeability and phyllic and/or outer potassic alteration. Grades of the Supergene sulphide zone vary widely, but are highest in fractured and strongly pyritic zones, due to their ability to promote leaching and chalcocite precipitation. Thus, secondary enrichment zones are thickest along contacts of the potassic and phyllic alteration halos; accordingly, the copper grades in the Supergene Sulphide zone are almost double the copper grades in the Hypogene zone (0.43% Cu versus 0.23% Cu). Grain borders and fractures in chalcopyrite, bornite and tetrahedrite may be altered to chalcocite, diginite and/or covellite. Chalcocite also locally coats pyrite grains and clusters, and locally extends along fractures deep into the hypogene zone. Molybdenite is largely unaffected by supergene processes, other than local alteration to ferrimolybdite.

In drill-core, the SUS zone is generally broken with decreasing clay alteration and weathering with depth and is 'stained' dark blue to gray.

Table 7-2: Leached Cap & Supergene Minerals

Zone	Minerals Present	Average Thickness
Leached Cap	jarosite, goethite, hematite, ferrimolybdite	70 m
Supergene Oxide	chalcanthite, brochantite, malachite, azurite, tenorite, cuprite, neotocite, copper WAD native copper, copper-bearing goethite	10 m
Supergene Sulphide	digenite, chalcocite, minor covellite, bornite, copper-bearing goethite	60 m

7.3.3 Hypogene Mineralization (HYP)

Mineralization of the Casino Cu-Au-Mo deposit occurs mainly in the steeply plunging, in-situ contact breccia surrounding the Patton Porphyry intrusive plug. It was formed by crystallization and exsolution of hydrothermal fluids from late Cretaceous magmas of the Casino Plutonic Suite. The breccia forms an ovoid band around the main porphyry body with dimensions up to 250 m and has an interior zone of potassic alteration surrounded by discontinuous phyllic alteration, typical of some porphyry deposits.

Hypogene mineralization occurs throughout the various alteration zones of the Casino Porphyry deposit as mineralized stockwork veins and breccias. Field relationships show that the potassic alteration occurred first as mineralized quartz veins of the phyllically altered zones, which cut those of the potassically altered zones. Re-Os age dating showed that the timing of the potassic and phyllic alteration are contemporaneous at around 74.4 ± 0.28 Ma. Significant Cu-Mo mineralization is related to the potassically-altered breccias surrounding the core Patton Porphyry, as well as in the adjacent phyllically-altered host rocks of the Dawson Range Batholith.

Mineralization in the potassic zone comprises mainly finely disseminated pyrite, chalcopyrite and molybdenite, as well as trace sphalerite and bornite. The phyllic alteration zones have increased gold, copper, molybdenite and tungsten values concentrated within disseminations and veins of pyrite, chalcopyrite and molybdenite along the inner part of the pyrite halo. The pyrite halo occurs within the phyllic alteration zone along the potassic-phyllic contact and discontinuously surrounds the main breccia body. It is host to the highest copper values on the property.

Chalcopyrite commonly occurs as veins, disseminations and irregular patches. In breccia zones and granodiorite west of the Casino Fault, disseminated chalcopyrite is more abundant than vein and veinlet-style chalcopyrite, whereas to the east of the fault, chalcopyrite is controlled by brittle deformation and occurs in fractures and open space fillings. Pyrite to chalcopyrite ratios range from less than 2:1 in the core of the deposit, to greater than 20:1 in the outer phyllic zones. Locally, coarse grained bornite and tetrahedrite are intergrown with chalcopyrite.

Molybdenite is not generally intergrown with other sulphides and occurs as selvages in early, high temperature, potassic quartz veins and as discrete flakes and disseminations.

Native gold can occur as free grains (50 to 70 microns) in quartz and as inclusions in pyrite and/or chalcopyrite grains (1 to 15 microns). High grade smoky quartz veins with numerous specks of visible gold have been reported to exist.

Late-stage, commonly vuggy, polymetallic veins (like those of the Bomber Vein) follow roughly parallel, steeply dipping fractures trending 150 to 170 degrees. Metallic mineralogy includes abundant sphalerite and galena, with less abundant tetrahedrite, chalcopyrite (commonly intergrown with tetrahedrite), and bismuth-bearing minerals, and are geochemically anomalous in any or all of Ag, As, Bi, Cu, Cd, Mn, Pb, Sb, Zn and locally W.

In drill-core, the hypogene zone is un-weathered and un-oxidized.

7.3.4 Structurally Hosted Gold Mineralization

Structurally controlled gold mineralization within the Canadian Creek portion of the Casino property occurs mostly in the northwestern part of the property. Drilling in 2009 and 2017 discovered widespread anomalous gold mineralization associated with clay altered-shears, sheeted pyrite veins and quartz-carbonate veins hosted in both intrusive and metamorphic rocks. To date, the identified structures are generally less than 3 m thick and of short strike length. Gold is accompanied by silver, arsenic, antimony, molybdenum, barium and bismuth.

8 DEPOSIT TYPES

The Casino deposit is best classified as a calc-alkalic porphyry type deposit associated with a tonalite intrusive stock (the Patton Porphyry). Primary copper, gold and molybdenum mineralization was deposited from hydrothermal fluids that exploited the contact breccias and fractured wall rocks. Higher grades occur in the contact breccias, and grades gradually decrease outwards away from the contact zone, both towards the centre of the stock and outward into the host granitoids and schists. A general zoning of the primary sulphides occurs, with chalcopyrite and molybdenite occurring in the core tonalite and breccias, grading outward into pyrite-dominated mineralization in the surrounding granitoids and schists. Alteration accompanying the sulphide mineralization consists of an earlier phase of potassic alteration and a later overprinting of phyllic alteration. The potassic alteration typically comprises secondary biotite and K-feldspar as pervasive replacement and includes veins and stockworks of quartz and anhydrite veinlets. Phyllic alteration consists of replacements and vein-style sericite and silicification.

The Casino Copper deposit is unusual amongst Canadian porphyry copper deposits in having a well-developed enriched secondary supergene blanket of copper mineralization similar to those found in deposits in Chile, including the Escondita deposit and the Morenci Deposit in the southwest United States. Unlike other porphyry deposits in Canada, the Casino deposit’s enriched supergene copper blanket was not eroded by the glacial action of ice sheets during the last ice age. At Casino, weathering during the Tertiary Period leached the copper from the upper 70 m of the deposit and re-deposited it lower in the deposit, forming the Supergene zone. This resulted in a layer-like sequence consisting of an upper leached zone up to 70 m thick, where all sulphide minerals have been oxidized and copper removed, leaving a bleached, iron oxide leached cap containing residual gold. Beneath the leached cap is a zone up to 100 m thick of secondary copper mineralization, consisting primarily of chalcocite and minor covellite, and a thin, discontinuous layer of copper oxide minerals at the upper contact with the leach cap. The copper grades of the enriched, blanket-like zone can be up to twice that of the underlying, unweathered hypogene zone hosting primary copper mineralization. Primary mineralization consists of pyrite, chalcopyrite and lesser molybdenite. The primary copper mineralization is persistent at depth, extending to more than 600 m, beneath the deepest drill holes completed to date.

9 EXPLORATION

9.1 EXPLORATION PROCEDURES

The history of exploration on the property includes prospecting, geological mapping, multi-element soil geochemical sampling, magnetic and induced polarization surveying, trenching and diamond and reverse-circulation drilling. Targets of early drilling on the Casino Deposit were based mainly on coincident copper and molybdenum-in-soil anomalies. Since 1993, with the exception of a Titan TM Survey, exploration in the vicinity of the Casino deposit has comprised drilling on a grid pattern using a core drill with a core diameter mainly of NQ and NTW thickness, with a smaller number of holes drilled with HQ diameter core. The earlier soil sampling and geophysical survey anomalies, in the vicinity of the Casino Deposit, have all been tested by drilling and shown to be caused by porphyry copper mineralization.

A Titan TM Geophysical survey was carried out by Quantec Geoscience Limited of Toronto, Ontario in 2009, to search for potential deeply buried porphyry mineralization beneath or peripheral to the Casino deposit. The survey utilized Titan-24 Galvanic Direct Current Resistivity and Induced Polarization (DC/IP) surveys as well as a Magnetotelluric Tensor Resistivity (MT) survey over the entire grid. Magnetotelluric Resistivity surveys result in high resolution and deep penetration (to 1 km), while the Titan DC Resistivity & Induced Polarization surveys provide reasonable depth coverage to 750 m. The survey grid, covering a 2.4 km by 2.4 km area, was centered on the Casino deposit. The grid consisted of nine (9) lines, spaced 300 m apart, each 2.4 km long and at an azimuth of approximately 64 degrees (perpendicular to the Casino Creek Fault). Results of the Titan survey were used by Quantec to identify a series of drill targets within the survey grid and adjacent to the known mineralization. A total of 10 holes, comprising 4,327 m, were drilled to test geophysical targets. With the exception of several distal Pb-Zn veins and arsenopyrite-rich veins intercepted during this drilling, no porphyry copper mineralization was found.

To the west of the Casino deposit, on the recently acquired Canadian Creek Property, exploration utilized grid soil sampling, ground magnetic and induced polarisation surveys to generate targets for trenching and drilling. Initially, the focus of the geochemical and geophysical surveys was to locate porphyry copper mineralization. Subsequent to 2016, the focus of this work switched to the identification of gold mineralization similar to that discovered at nearby Coffee Creek (Johnston, 2018).

Soil sampling surveys, to the west of the Casino Deposit, were carried out over the time period from the mid 1990s through to 2011. The soil surveys targeted mainly B horizon soils, but due to local talus cover and permafrost, sampling of the B horizon was not always possible. Soil samples underwent multi-element and gold analysis, mostly at Acme Analytical Labs Vancouver, using ICP methods and fire assay with atomic absorption finish for gold. The historical soil grids had sampling spacings that ranged from 25 to 75 m on 200 m spaced lines. Locally, infill sampling was done at a reduced spacing of 25 m stations on 100 m spaced lines within anomalies identified from previous wider spaced surveying, in order to better define the gold and arsenic anomalies. Results for copper are shown in Figure 9-1. The soil results show a coincident copper and gold-in-soil anomaly at the 50 ppm Cu and the 15 ppb Au threshold levels respectively, extending approximately 3 km west from the western limits of the Casino deposit. This coincident Cu-Au anomaly has been tested by 16 core holes. The holes closest to the Casino Deposit revealed moderate potassic alteration and strong propylitic alteration. The four closest holes intersected zones of leached cap or incipient leaching, weak supergene enrichment, and hypogene copper-gold-molybdenum mineralization typical of the outer edges of a porphyry copper – gold – molybdenum deposit. Copper grades are in the 300 – 700 ppm (0.03 to 0.07%) range, gold grades range from 0.1 to 0.3 g/t, and molybdenum values range from 20 – 40 ppm (0.002 to .004%). Further, there is a progressive increase in Cu, Au and Mo in the Casino B drill holes eastward towards the Casino deposit. These holes are defining the western limits of the Casino deposit system.

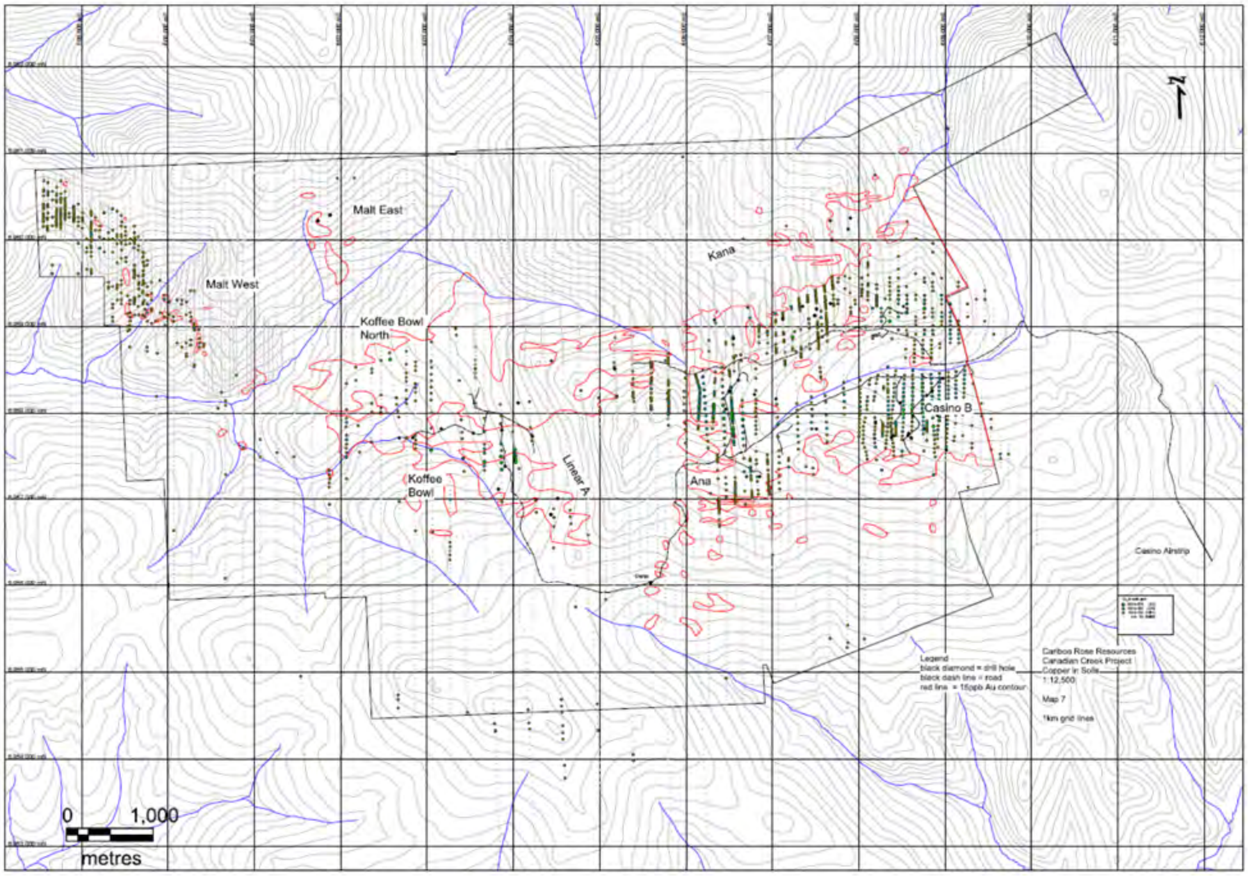


Figure 9-1: Copper and Gold in Soil Results (Johnston, 2018)

Elsewhere, the soil results identified a number of areas of anomalous gold, arsenic, and bismuth. These anomalies were further explored with trenching, core drilling and reverse circulation drilling. This work identified scattered narrow zones of gold mineralization associated with clay-altered shears, sheeted pyrite veins and quartz-carbonate veins, hosted both in intrusive and metamorphic rocks. With few exceptions, gold grades in the structures are sub-1.0 g/t (1,000 ppb). The structures identified to date are mainly less than 3 m thick and of short strike length.

Ground magnetic surveying at a line spacing of 100 m was undertaken over the Canadian Creek portion of the Casino Property in 2011 and 2017. The surveys detected a number of lineaments, oriented mostly northwest-southeast, though none obviously align with the soil geochemical anomalies. A plot of the un-levelled magnetic survey results of the property is shown in Figure 9-2.

The ground magnetic data shows a trend of magnetic high features extending from the Casino Deposit through the Ana Zone area to the Koffee Bowl area. This west-southwest trend follows the trend of Patton Porphyry dykes extending from the main intrusive complex.

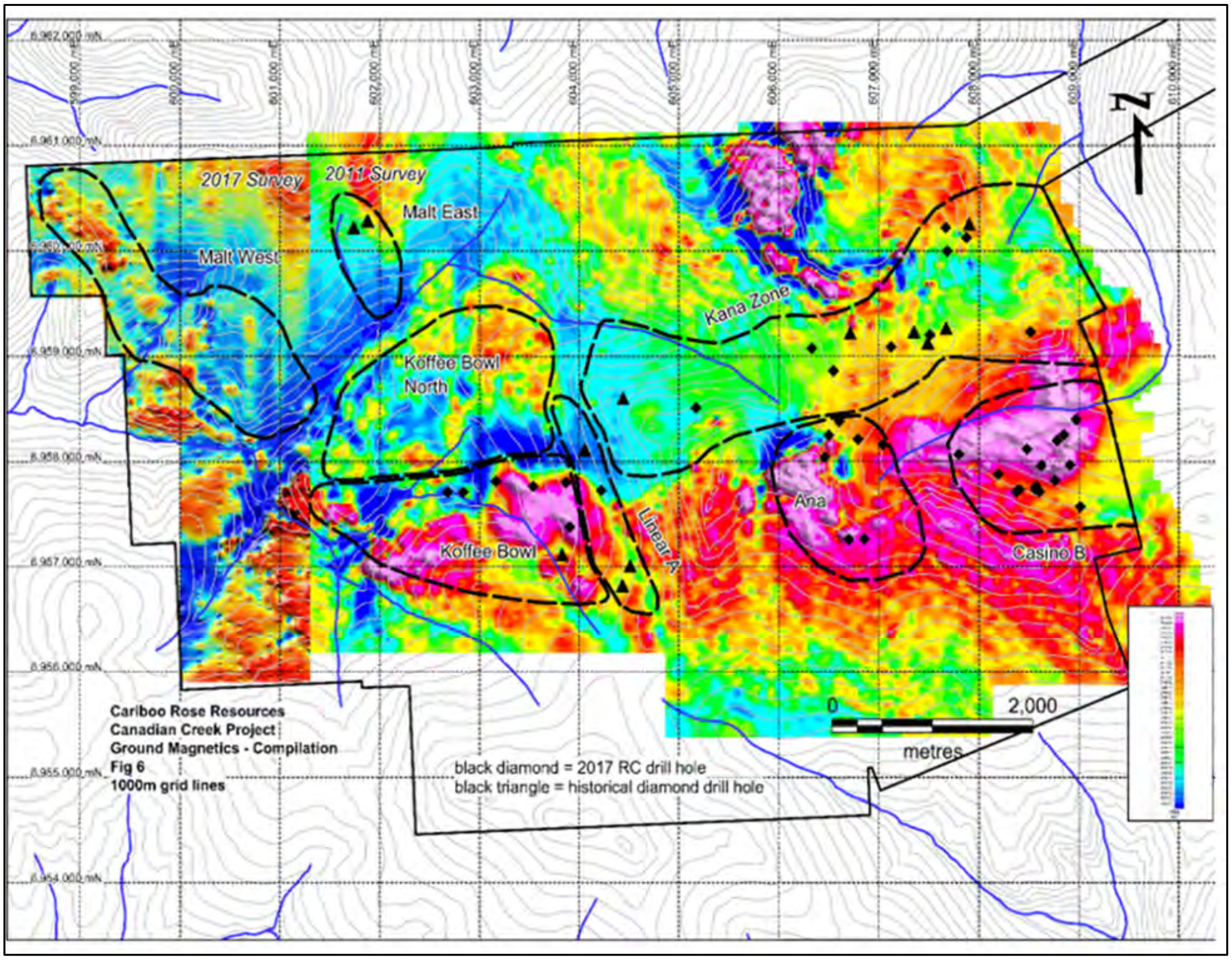


Figure 9-2: Magnetic Compilation (Johnston, 2018)

Induced polarization surveys were carried out in 1993, 1996, 2009 and 2011. The 1993 and 1996 surveys used a pole-dipole array with a spacing of 75 m and an N1 to N4 depth profile. The 2009 survey was a pole-dipole survey using an a spacing of 25 m and an N1 to N6 depth profile. The 2011 pole dipole survey used a spacing of 25 m and an N1 to N8 profile. A compilation of the results is shown in Figure 9-3.

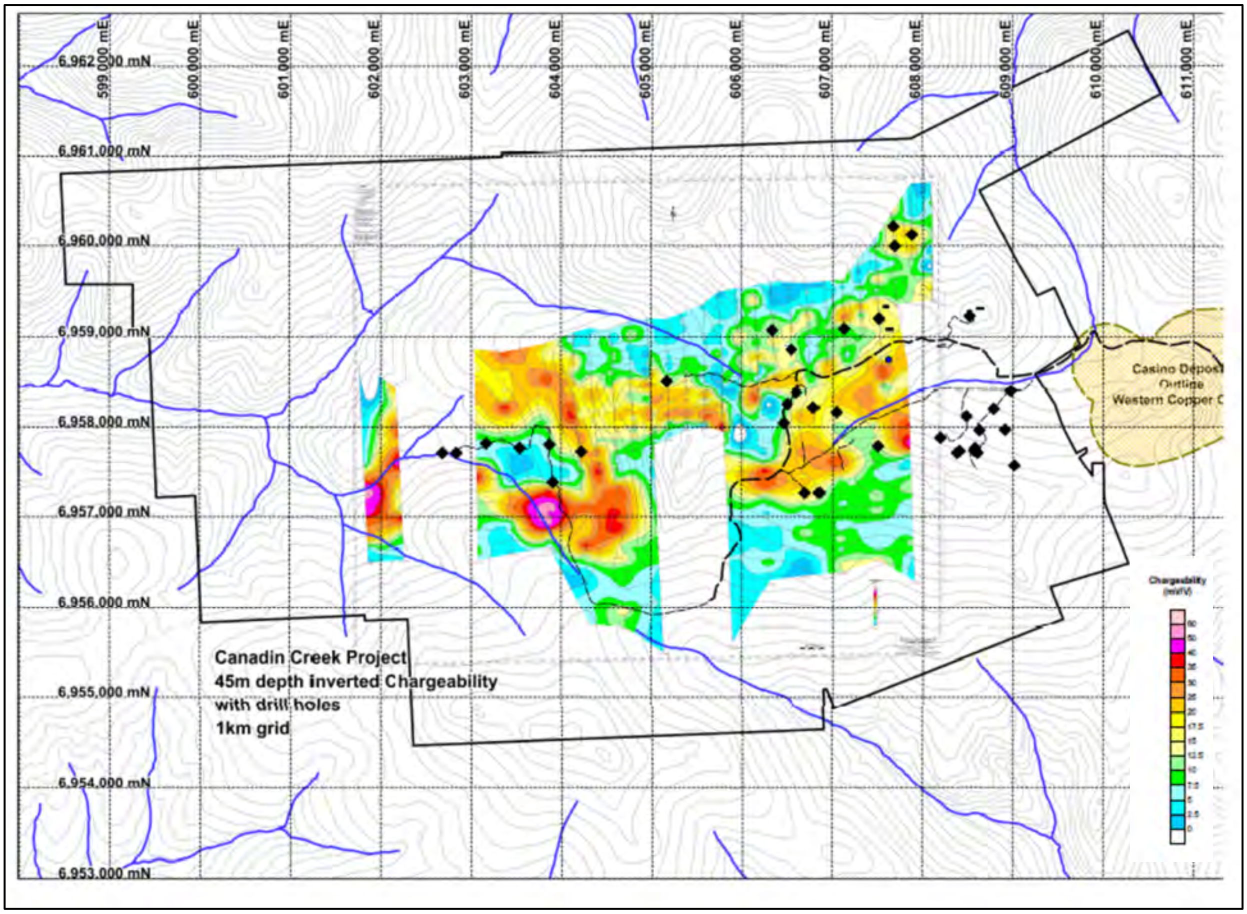


Figure 9-3: IP Compilation (Johnston, 2018)

In general, the surveys used small "n" spacings and have limited depth profiles. The survey identified a number of high chargeability anomalies which remain to be tested.

10 DRILLING

The following sections describe the various drilling programs developed on the Casino Property.

10.1 1992-1994 DRILLING PROGRAM

Drilling prior to 1992 (Figure 10-1) consisted of reverse circulation drilling and NQ-diameter diamond drilling. There is little documentation that specifically focused on this early drilling, its specifications or challenges. Following the acquisition of Casino Silver Mines Ltd. by Archer Cathro and Associates, then by PSG, the drilling is well documented.

During the intense campaigns from 1992 through 1994, (Figure 10-2) drilling was contracted to E. Caron Drilling Ltd. of Whitehorse, Yukon. Up to six diamond drills were utilized. The 1994 drilling program fulfilled a variety of purposes: infill and delineation drilling, and geotechnical, structural and waste rock characterization. Infill drilling involved a program of angle and vertical holes designed to outline and more fully define the Leached Cap (Oxide Gold zone) and Supergene copper zones. Delineation drilling to the north, northeast, east and southeast outlined the extent of the deposit area. Four oriented angle holes were drilled into the deposit area for geotechnical information, primarily rock strength and structural characteristics, and for geological information regarding vein-set orientations. Five vertical holes were drilled into the periphery of the deposit area for waste rock characterization studies. Seven vertical holes were drilled into the peripheral area of the deposit for geotechnical information. Eighteen vertical holes were drilled outside the deposit area for geotechnical and geological information regarding potential site development.

The combined drilling from 1992 through 1994 consisted of 71,437.59 m of NQ and HQ core in 236 holes.

Core recoveries were consistently in the 80% to 90% range in the Leached Cap and Supergene zones and 90% to 100% in the Hypogene zone.

Drilling can be carried out at Casino from March through November with minor logistical challenges, although conditions in the spring and fall require winter-type drilling equipment. The use of a water supply truck is necessary during very cold weather conditions, due to freezing of water lines. Three reliable water supply sites exist on the property and can all be utilized during multiple drill rig programs.

10.2 2008 TO 2012 DRILLING

The drilling for the 2008 to 2012 exploration programs was contracted to Kluane Drilling Ltd. from Whitehorse, Yukon. Up to three hydraulic diamond drills were utilized for these programs.

Water for the drilling was pumped from the Canadian Creek bend, at the location of the old placer camp, and from Casino Creek.

Drilling was carried out from March through November. Conditions in the late winter and fall required winter-type drilling equipment. The main challenges during the winter drilling were water supply due to the low water level in both creeks and the freezing of long water lines.

All drilling was done using "thin wall" drill rods. Holes CAS-001 to CAS-007 utilized HTW-size rods (core diameter 70.92 mm) and the remainder of the drilling was done utilizing primarily NTW core size (core diameter 56.00 mm). Deeper holes were collared using HTW rods and reduced to NTW rods typically from 200 to 300m of depth. In a few cases, holes were reduced further to BTW core size (core diameter 42.00 mm).

Core recoveries in the Leached Cap and Supergene zones were consistently in the 80% to 90% range and 90% to 100% in the Hypogene zone.

Down-hole orientation surveying was performed using an Icefield Tools MI3 Multishot Digital Borehole Survey Tool for holes CAS-002 to CAS-076. For holes CAS-077 to CAS-092, as well as the geotechnical and hydrogeological holes, a Reflex Instruments downhole survey instrument was used.

Western Copper and Gold Corp. (Western) continued the drilling pattern established by PSG, utilizing mainly a vertical drill hole orientation and a nominal 100 m grid spacing. Later in the program, Western drilled a series of inclined holes in the northern part of the deposit. Several inclined holes were also drilled in the western part of the deposit to establish contacts with the post-mineral explosion breccia (MX) and to confirm orientation of the interpreted N-S structure.

Geostatistics, done in 2010, have shown that the 100 m spacing was sufficient for delineation of supergene mineralization. The same studies have shown that the 100 m drill hole spacing is only marginally sufficient for delineation of hypogene copper mineralization.

10.3 2013 DRILLING

Drilling during the 2013 field season was contracted to Kluane Drilling Ltd. of Whitehorse, Yukon. Up to two hydraulic diamond drills were used for this program.

Drilling in 2013 was primarily for water wells and hydrogeological purposes. Each hole was fully logged by core loggers, but no samples were taken. Eleven holes (MW13-01D/S through MW13-06D/S) were drilled throughout June and another fifteen (DH13-01 through DH13-12) were completed during August. See Figure 10-4 and Figure 10-5 for detailed locations of drilling.

No diamond drilling was completed on the property from 2014 through to the end of 2018.

10.4 2019 DRILLING

Between May and October of 2019, Kluane Drilling Ltd. of Whitehorse, Yukon, drilled 72 core holes (DH 19-01 through DH 19-53, CRD 19-54 through CRD 19-59 and DH 19-60 through DH 19-69) on the Casino Property using up to two hydraulic diamond drills.

Water for the drilling was pumped from the Canadian Creek bend, from Casino Creek, and from several small ponds in the property area.

All drill holes in 2019 were of NTW core size (core diameter 56.00 mm) with the exception of some holes in difficult ground that were collared with HTW core size (core diameter 70.92 mm) and reduced to NTW when drilling conditions improved.

Core recoveries were consistently in the 75% to 80% range within the Leached Cap, 80% to 90% within the Supergene zones and 90% to 100% within the Hypogene zone.

Down-hole orientation surveying was performed using a DeviShot Magnetic Multishot survey tool. Each drillhole was surveyed on 30-50 m increments by the Kluane drilling team.

CAP Engineering, of Whitehorse, Yukon was on site for 2 days in late August to survey the drill hole collars. A team of two people used a Stonex GPS RTK Unit and a Topcon GPS RTK Unit to complete the surveys. See Figure 10-4 and Figure 10-5 for detailed locations of the drill holes.

The purpose of the 2019 drill program was to infill the previous drill hole spacing to upgrade the resource estimate for the project. All holes were logged, sampled and photographed by geologists on site before samples were sent to ALS

Global (ALS) in Whitehorse for analysis, with 20% of those pulps from ALS randomly selected and sent on to SGS Canada Inc. (SGS) in Vancouver for a QA/QC check analysis.

10.5 2020 DRILLING

Between June and September of 2020, Kluane Drilling Ltd. of Whitehorse, Yukon, drilled a total of 12,008 metres in 49 core holes (DDH 20-01 through DDH20-49) on the Casino Property utilizing up to three hydraulic drill rigs. Water for the drilling was pumped from the Canadian Creek bend, from Casino Creek, and from several small ponds in the property area.

The majority of holes in 2020 comprised NTW core (core diameter 56.00 mm) with the exception of some holes in difficult ground that were collared with HTW core (core diameter 70.92) and reduced to NTW when drilling conditions improved. Several NTW core holes were reduced in size to BTW core (core diameter 40.7 mm) where similarly difficult conditions were encountered.

Core recoveries were consistently in the 75% to 90% range within the Leached Cap, 80% to 100% within the Supergene zones and 90% to 100% within the Hypogene zone. Down-hole orientation surveying was performed using a DeviShot Magnetic Multishot survey tool. Each drillhole was surveyed on 30-50 m increments by the Kluane drilling team.

CAP Engineering, of Whitehorse, Yukon was on site for 2 days in late August 2020 to survey the drill hole collars. A team of two people used a Stonex GPS RTK Unit and a Topcon GPS RTK Unit to complete the surveys. See Figure 10-6 for a detailed location of the drill holes.

The 2019 program returned at least one high-grade intercept grading 55.1 g/t gold across 2.97 m from DDH 19-21 (WRN News Release Dec 19, 2019). By the end of 2019, three major zones were identified: the "Gold Zone", 1) an arcuate zone along the southern and western property boundaries hosting the majority of short high-grade gold values; 2) the "North Porphyry Zone", extending north and northwest of the main deposit; and 3) the "Canadian Zone" immediately west of the deposit. The goals of the 2020 drilling were: 1) to test for continuity of porphyry mineralization at the "North Zone", and to the west of the deposit; 2) to attempt delineation of a generally east-west oriented "Gold Zone" suggested by 2019 drill results, and; 3) to investigate the "Canadian Zone" immediately west of the deposit and within the Canadian Creek property (Schulze, 2021).

All holes were logged, sampled and photographed by geologists on site before samples were sent to ALS Global (ALS) in Whitehorse for analysis, with 5% of those pulps from ALS randomly selected and sent on to Bureau Veritas Canada Inc. (BV) in Vancouver for a QA/QC check analysis.

Drilling results indicate the eastern area of the "Gold Zone" hosts an area of higher-grade Cu-Au mineralization, now termed the "System Core" (Williams, 2021). However, no further extremely high-grade intercepts were intersected in the "Gold Zone", which is no longer considered as a discrete mineralized horizon. Results also determined that any similar high-grade intercepts are not significant contributors to the mineral resource base at the Casino deposit (Schulze, 2021). Table 10-1 lists the summary of drilling results in each zone. The Canadian Zone has been re-named the "Casino West" zone.

Table 10-1: Summary of drill targets in 2020

Sector	2020 Drill holes
System Core (4 holes)	DH20-05, -08, -27, -45
"Gold Zone" (14 holes)	DH20-01, -02, -10, -12, -13, -17, -18, -20, -22, -24, -25, -29, -32, -35
Northern Porphyry (13 holes)	DH20-03, -04, -05, -07, -09, -11, -15, -42, -44, -45, -47, -48, -49
Western Sector (7 holes)	DH20-33, -34, -37, -38, -39, -41, -43
Casino West (8 holes)	DH20-14, -16, -19, -21, -23, -26, -28, -30
Ana Zone (3 holes)	DH20-31, -36, -40

Source: Williams, 2021

The 2020 results were not included in the resource estimate included in this PEA.

10.6 CANADIAN CREEK DRILLING SUMMARY

Following acquisition of the Canadian Creek property by Western in 2019, all drilling data was transferred from Cariboo Rose Resources Ltd. and is summarised in Table 10-2. Since 1992, when exploration first began on the Canadian Creek property, soil sampling, trenching, geophysical surveys and drilling have focused on several areas of interest. A full history of the Canadian Creek property can be found in Section 6 of this report.

Table 10-2: Summary of Canadian Creek Drilling

Year	Drilling Summary (# holes)	Area	Type of Drilling
1970	2	Casino B	Diamond Drilling
1993	7	Ana, Koffee	Diamond Drilling
1994	4	Casino B	Diamond Drilling
2000	11	Ana, Casino B, Koffee	Diamond Drilling
2007	5	Casino B	Diamond Drilling
2009	10	Kana	Diamond Drilling
2017	24	Various	Reverse Circulation Drilling

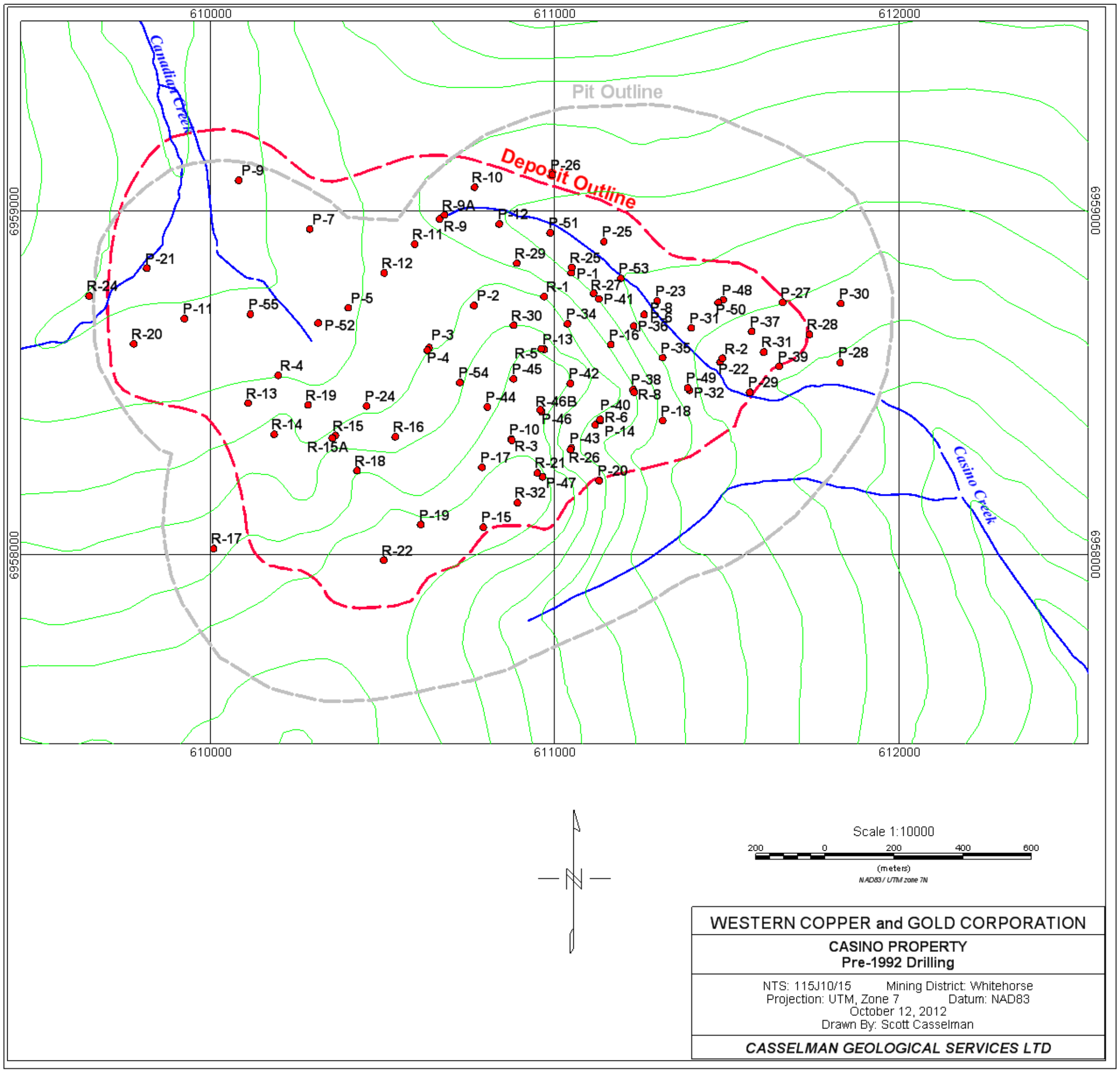


Figure 10-1: Casino Property Drilling Pre-1992

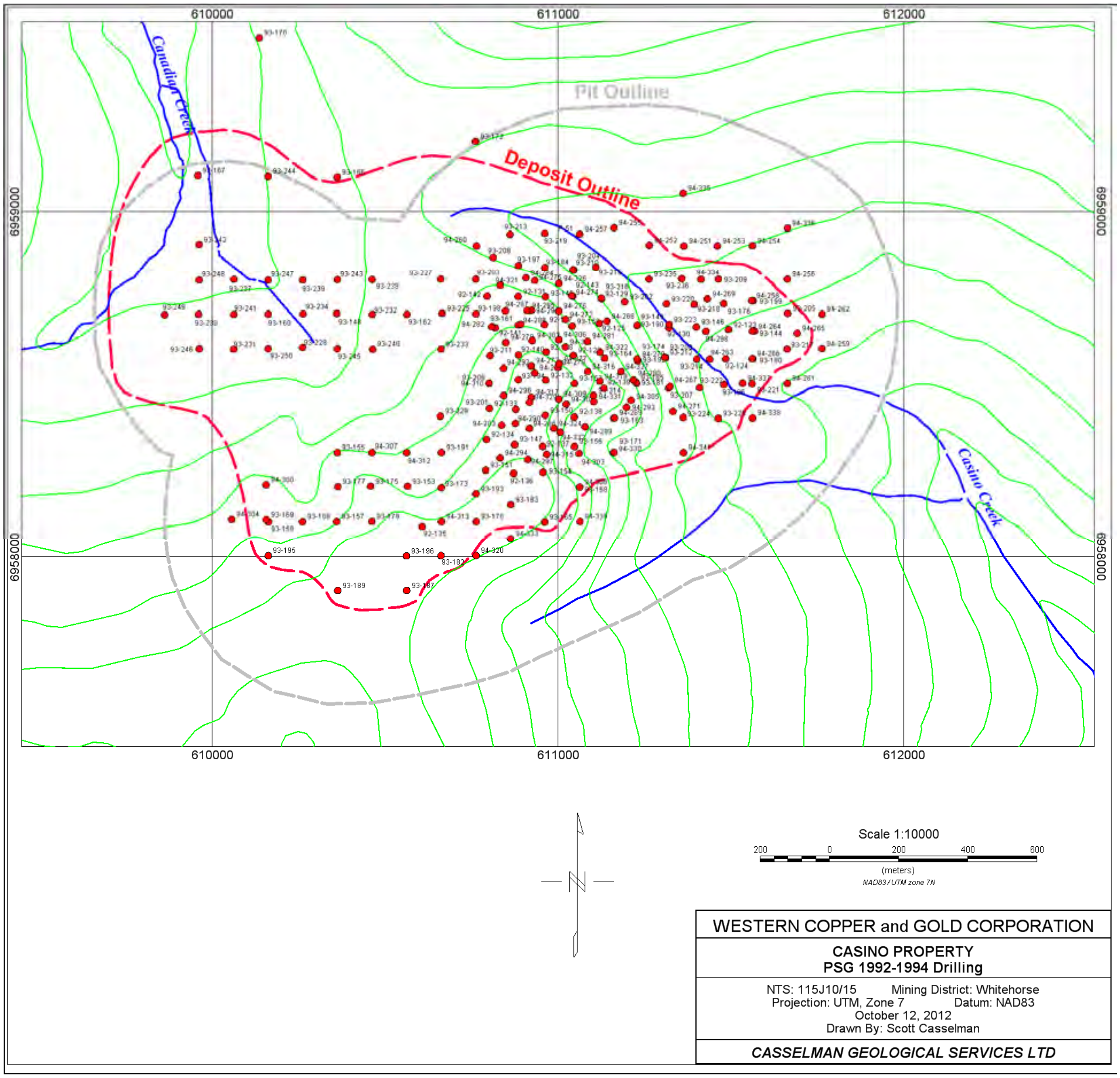


Figure 10-2: Casino Property Drilling 1992 to1994

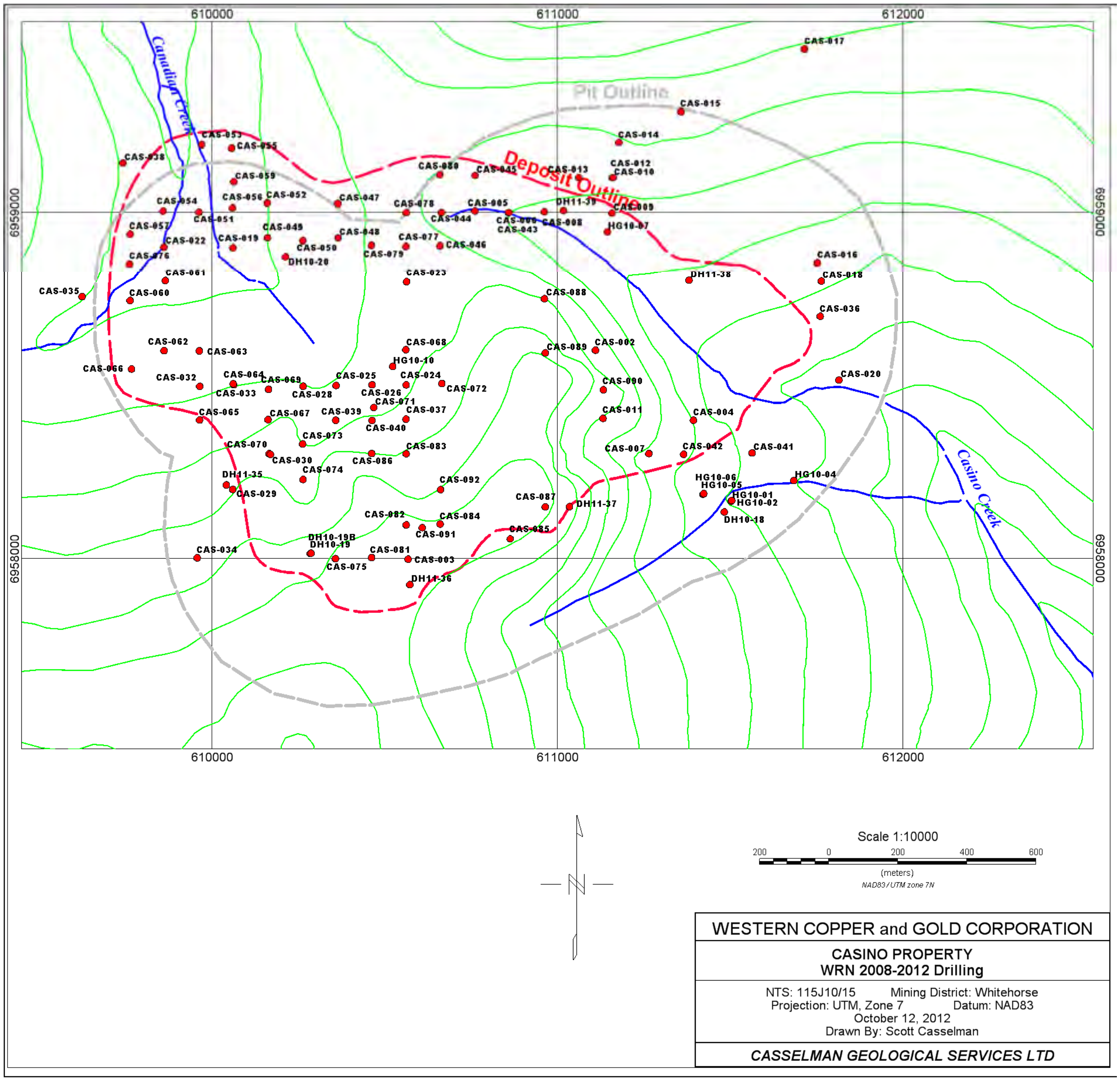


Figure 10-3: Casino Property Drilling from 2008 to 2012

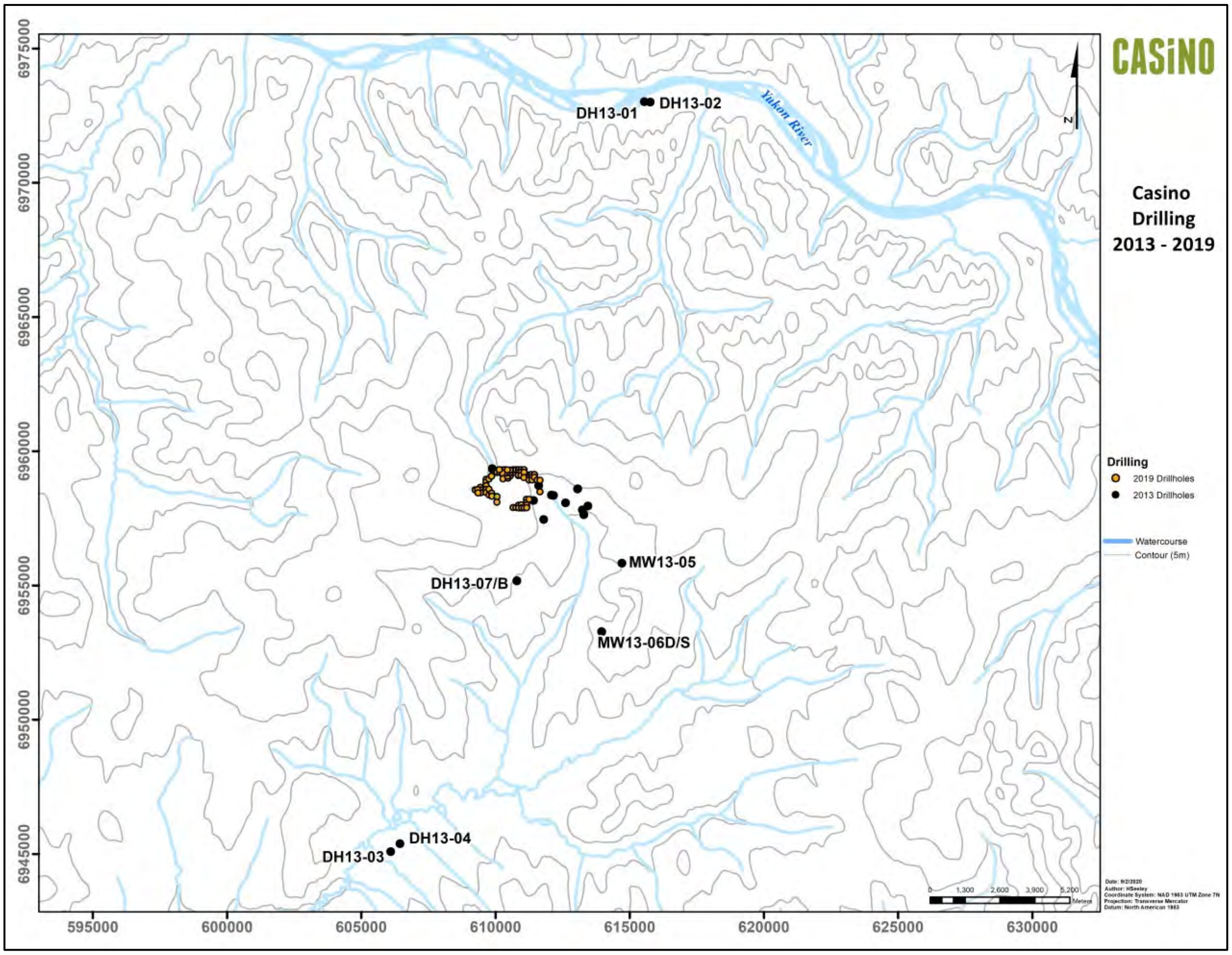


Figure 10-4: Casino Project Drilling from 2013 through 2019

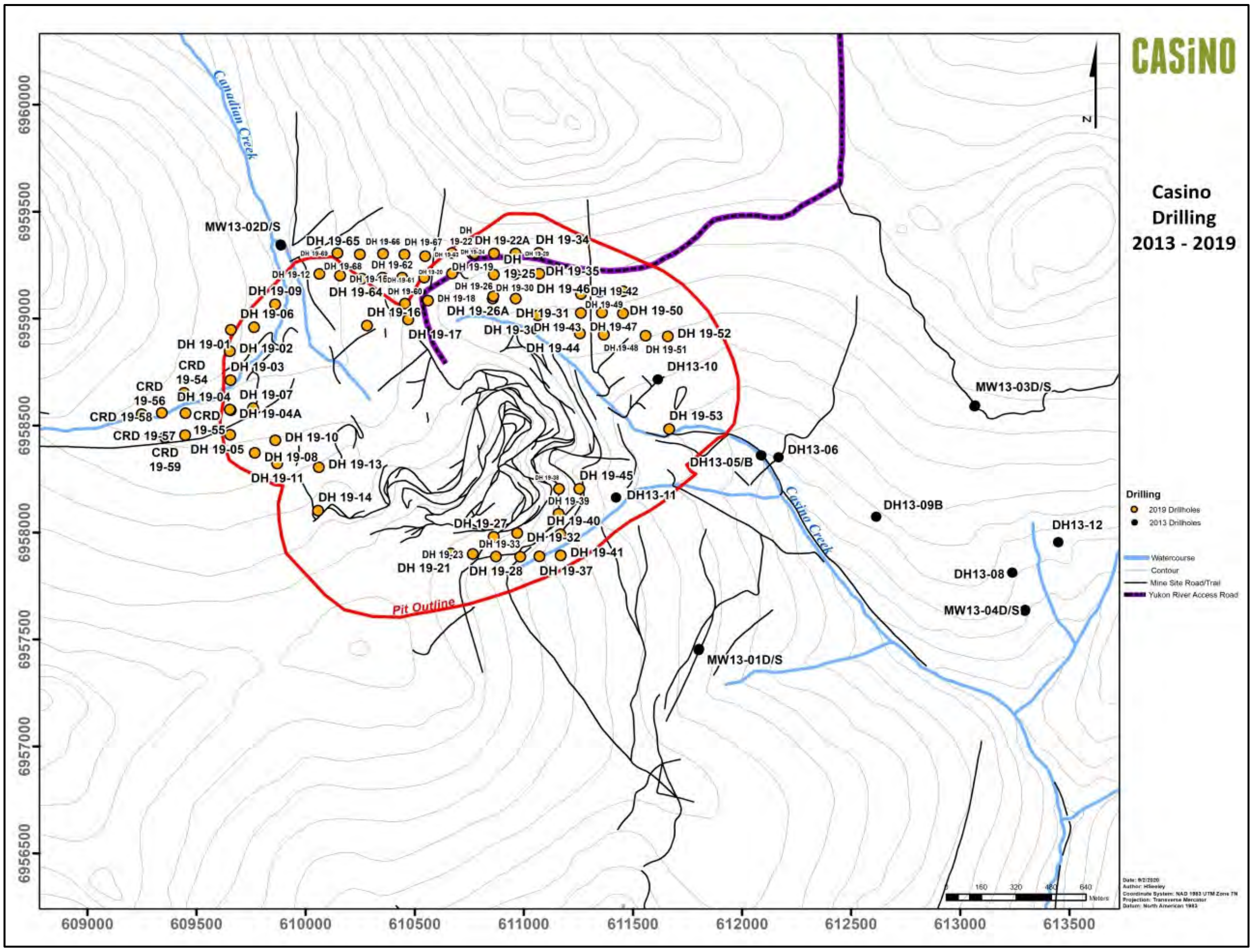


Figure 10-5: Casino Project Drilling from 2013 through 2019

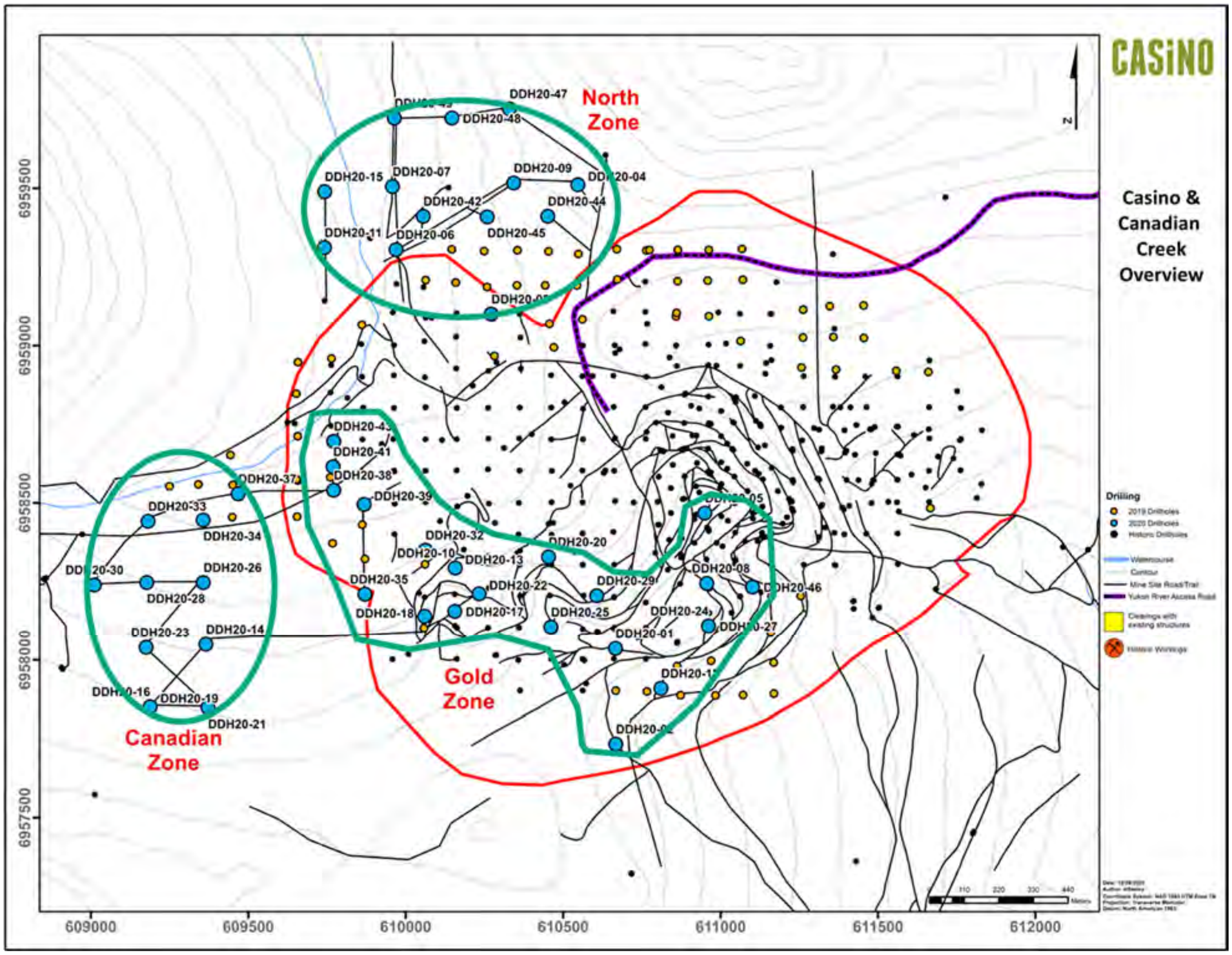


Figure 10-6: Location of 2020 Diamond Drill Holes (Schulze, 2021)

10.7 SENSITIVITY DATA PHOTOGRAMMETRY

In April 1993, McElhanney Consulting Services Ltd. of Vancouver, BC, produced a map of the Casino area based on 1985 air photos provided by the Department of Energy, Mines and Resources.

New aerial photography was conducted in July 1993, by Lamerton & Associates of Whitehorse. The area was mapped by Eagle Mapping Services Ltd. of Port Coquitlam, BC. Eagle Mapping utilized two government UTM co-ordinates systems, NAD83 and WGS84, in the derivation of the deposit grid co-ordinates at photo target station #11. The following transformation parameters were used to convert from UTM coordinates to Property Grid:

ROTATION:	-0° 00' 05"
SCALE:	1.000453652
TRANSLATION:	-6703701.92 N
	-499861.96 E
ELEVATION SHIFT:	-8.32 m

The contours on McElhanney and Eagle Mapping Services maps compare to within approximately 5 m and commonly closer. Generally, Eagle Mapping contours are smoother, having more gradual changes in direction.

10.8 COLLAR COORDINATES

The 1992 to 1994 collar co-ordinates (northing, easting and elevation) were surveyed using a total station Nikon C-100 system. Surveying of the 1992 and 1993 drill holes was undertaken by Lamerton & Associates. The 1994 holes were surveyed by Z. Peter, Surveyor from Burnaby, B.C. It should be stressed that all Pacific Sentinel's collar coordinates were surveyed in local grid coordinates.

The 2008-2012 drill collars were surveyed by Yukon Engineering Services from Whitehorse. The survey was completed using Differential GPS units and the results are reported in UTM NAD83, Zone 7 coordinates.

Twenty-eight (28) of Pacific Sentinel's drill hole collars were also re-surveyed by Yukon Engineering for comparison purposes. Those were entered into the data base with their new UTM NAD83 collar coordinates.

The 2013 and 2019 drill collars were surveyed by CAP Engineering from Whitehorse. CAP used a Stonex GPS RTK Unit and a Topcon GPS RTK Unit to complete the surveys These results were reported in UTM NAD83, Zone 7 coordinates.

10.9 SPERRY SUN SURVEYS

During the 1993 drilling program, all drill holes, including deepened 1992 holes, were down-hole surveyed by a Sperry Sun magnetic compass tool to determine azimuth and dip. In the 1994 drilling program, only angle holes were Sperry Sun surveyed. Tests were normally performed every 152 m (500 ft) down hole on vertical holes and every 76 m (250 ft) down hole on angle holes. In the shallower angle hole program of 1994, Sperry Sun tests were taken at the bottom of the hole as well as half-way up.

The Sperry Sun surveys, taken in the 123 vertical holes drilled or deepened in 1993, averaged a dip reading of 89.03°. Since the average deviation observed in the 123 vertical holes of the 1993 program was less than one degree, it was decided not to survey the vertical holes of the 1994 program.

10.10 LIGHT-LOG SURVEY SYSTEM

A Light-Log directional drill hole survey system, developed by H.J. Otte & Co., was used for sixteen angle holes at Casino, starting at hole 94-285 and continuing for most of the angle holes through the remainder of the 1994 drilling program. This system recorded, on film, the bending of the unit caused by the natural curvature in a drill hole. The instrument's timer activated the camera and advanced the film at pre-set time intervals, allowing time to lower the instrument between pictures (normally every 3 m). Upon completion of the survey, the film was developed. The values observed on the film were converted by a computer program to provide co-ordinates, dip and azimuth at every three-metre interval downhole.

10.11 ACID DIP TESTS

In the 1994 program, acid dip tests were performed in the vertical holes while Sperry Sun surveys were continued in the angle holes.

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

The following section summarizes the 2019 and 2020 sampling and assaying protocols that have been utilized at the Casino Project site.

11.1 SAMPLING METHOD AND APPROACH

11.1.1 Core Processing

At the drill site, core was placed into wooden core boxes directly upon emptying of the core tube. A wooden block marked with the depth, both in feet and metres, was placed in the core box upon completion of each drill run. Under good ground conditions, each run comprises 10 ft (3.05 m) of core. Core boxes were stored at the core logging facility adjacent to the Casino Airstrip. As core came in from the rig, each hole was stacked separately and clearly labelled outside of the core shack. Once the core was ready to be logged, it was laid out in sequence on elevated tables in the core shack.

Core boxes were labelled with black felt tip pens and embossed steel tags containing hole number, box number and interval of core within the box. Geotechnical data including core recovery, rock quality designation (RQD), hardness and natural breaks were recorded for each drill run, as marked by the wooden core run blocks. This information was recorded on paper by the geologist or geotechnical logger, supervised by the lead geologist. Logging of the geotechnical data followed codes and format outlined in a project-specific manual prepared by Knight Piésold.

The geologist recorded key geologic information including lithology, zone, mineralization and alteration. The data was entered onto paper. The codes and logging forms followed, as closely as possible, the format used by Western during the 2010-2012 drilling programs. The lithology codes, copper mineralization zone codes and alteration codes utilized in the 2013 to 2020 drilling programs were all initially developed by Pacific Sentinel and modified by Western.

Core was photographed after the geology log was completed and after the sample intervals were marked.

The core processing protocol for 2020 was similar to that for 2019. Core boxes were laid out outside of the core shacks, then cleaned utilizing spray bottles in preparation for geological summary logging. All core then underwent calculation of box intervals, core recoveries and “Rock Quality Designation” (RQD), prior to more detailed geotechnical logging comprising core hardness, nature of core fractures and “breaks” and type of fracture-filling material, if any. The core was then logged for lithology, alteration, mineralization, colour, structural characteristics and metallurgical zones (“Met zones”). Select intervals were also analyzed by an XRF device for Au, Cu, Mo and Ag. Sample intervals were also laid out, and were typically 3.0 m in length, although shorter if a lithological contact or significant structural zone was encountered. Following logging, all core was digitally photographed, utilizing the same location to minimize effects of changing light conditions. Two boxes were photographed at once, including a “white board” showing the box numbers and core intervals.

All data was entered into a Geospark database, uploaded every evening to a “Share Point” portal managed by Wolfbear Geological Consulting (Wolfbear). All data was subsequently managed by Wolfbear and paired with analytical results upon their receipt.

11.1.2 Core Sampling

Sampling and analytical protocols in use prior to the PSG diamond drill programs are not well documented. In June 1992, core from 22 previous holes was re-sampled by Archer Cathro. The new assay results for all metals were compared to the originals. The results indicated 14 holes (64%) had essentially identical results, while five holes (23%) had higher re-assays and three (13%) were inconclusive. When results of the old holes were compared with those of new holes drilled in the same locations, the results were similar to the re-sampling tests. Archer Cathro surmised that the higher gold results in the new holes were due to a combination of improved drilling techniques that resulted in better core recovery, and advanced laboratory techniques that provided lower analytical detection limits.

The PSG core sampling followed rigorous procedures that were well documented and standardized throughout the drilling programs. In the 1992, 1993 and 1994 programs, exploration targets were sampled by HQ (63.5 mm diameter) core drilling; occasionally this was reduced to NQ (47.6 mm). The boxed core samples were transported by truck less than 5 km to a core logging facility adjacent to the Casino Airstrip for geotechnical logging, sample selection quality control designation and sampling by PSG personnel. The average core recovery for all PSG core was 94%, with Hypogene core averaging 96%, Supergene 92% and the Leached Cap (Oxide Gold zone) averaging 89%. Sample intervals were marked on the core by the geologist generally at 3-metre-long intervals or at geological contacts. Core intervals were sampled by mechanical splitting. The remaining half core was returned to the boxes and stored in racks at the site. The average lengthwise half-split provided 10 to 15 kg of material, which was transported by charter aircraft (primarily DC-3) directly from the core sampling facility to Whitehorse and then by commercial air freight to Vancouver for delivery to the sample preparation laboratory.

In 2008, all samples were split using a conventional core splitter. In 2009, about 150 samples were split with a core splitter at the beginning of the program. From then on, in 2009 through 2012, all samples were cut with a core saw. All samples were split or cut on site and placed in individually labelled plastic sample bags with the unique sample number selected by the geologist logging the hole. The core samples were split lengthwise with half of the core placed in the sample bag, and the other half returned to the core box. The samples were sent to ALS Chemex Labs in North Vancouver for analysis.

In 2013 no core was sampled, but all other core logging protocols were followed as per 2012. The 2019 drill program followed the protocols established in 2012. Core was split in half lengthwise with a core saw and half of the core was placed, with a sample tag, in plastic bags with the corresponding sample ID noted on the outside of the bag. Metal tags marking the sample intervals (in metres) and with the sample ID matching the tag book were added at the applicable locations in the core boxes. The remaining half of the core was placed back into the core box, stacked outside the core cutting shack and then moved to the core storage yard where each hole was stored either in stacks, securely covered by tarps and labelled as per hole, or directly within the core racks. Bagged and labelled samples were then placed in larger white rice bags, each labelled with a unique batch letter and the address of the receiving lab. A running list of each batch was maintained in Excel spreadsheet form, including the samples per bag and the dates they were sent out by plane to ALS Global in Whitehorse.

In 2020, the core was then sawn lengthwise utilizing rock saws for an even cut, so that half of the core was sent to the laboratory, and the other half remained in the core box. The sample was placed in a pre-labelled poly bag, with a tag having a unique assay number placed within it, and the same number written with a “Sharpie” on the bag. All samples were sealed with a cable tie (“Zap Strap”), and placed into rice bags, with the sample numbers written on the bags. Shipments typically comprised 20 rice bags, each sealed with a security tag with a unique ID number. Samples were shipped in batches of 20 by fixed wing aircraft, and a spreadsheet with the bag number (per batch), number of samples, sample IDs and weights was included with the Sample Shipment Form in Bag 20.

The samples were shipped to Small’s Expediting of Whitehorse, Yukon, where they were driven to the ALS Geochemistry (ALS) lab in Whitehorse, Yukon. The ALS lab performed sample preparation and analysis.

In 2008, 422 drill core samples were collected and shipped; in 2009, 3,832 drill core samples were shipped; in 2010, 4,768 drill core samples were shipped; in 2011, 387 drill core samples were shipped; and in 2012, 533 drill core samples were shipped. In 2013, no samples were collected. In 2019, 4,939 core samples were collected, shipped and analysed. In 2020, 4,069 samples were collected, shipped and analyzed.

11.2 SAMPLE PREPARATION

11.2.1 2019 Sample Preparation

All original samples in 2019 were sent to ALS Global Labs in Whitehorse for analysis. The standard analytical request for all samples was for preparation by procedure Prep-31A. This process involved logging the sample into the tracking system, weighing, drying and crushing the entire sample to better than 70% passing through a 2 mm screen. A 250-gram split of the crushed material was then collected by riffle splitter and was pulverized to better than 85% passing 75 microns. The resultant pulp was analysed by the ALS lab in Whitehorse.

Sample "standards", provided by CDN Resource Labs and inserted in the sample stream at site, arrived at ALS Global in pulp form and went straight to analysis. Blank samples inserted into the sample stream at site arrived as rock and went through the same preparation and analytical processes as the core samples. Duplicate samples were sent to ALS in separate batches, arriving at a later date than the original samples. These also underwent the same preparation and analytical processes as the original core samples.

Check pulp samples were sent from ALS in Whitehorse to SGS Canada Inc. (SGS) in Burnaby, British Columbia (BC). At SGS, the pulps were checked for weight and fineness before a full geochemical assay was run. This involved logging the sample into the tracking system (confirming the samples received matched the electronic list of samples sent by Western staff), weighing and then checking that the pulps were of appropriate fineness.

11.2.2 2020 Sample Preparation

The standard analytical request for all samples was for preparation by procedure Prep-31A. This process involved logging the sample into the tracking system, weighing, drying and crushing the entire sample so that more than 70% of the material could pass through a 2 mm screen. A 250-gram split of the crushed material was then collected by riffle splitter and was pulverized so that a minimum of 85% of the material could pass through a 75-micron screen. The resultant pulp was then sent for analysis within the ALS lab in Whitehorse.

"Standard" reference material, provided by CDN Resource Labs and inserted in the sample stream at site, arrived at ALS Global in pulp form and went straight to analysis. Blank samples arrived as rock and went through the same process as the core samples. Duplicate samples were sent to ALS in separate batches, arriving at a later date than the original samples and then undergoing the same process as the original core samples.

"Check" pulp samples were sent from the Whitehorse ALS lab to the Bureau Veritas Minerals lab in Vancouver, BC. At Bureau Veritas, the pulps were checked for weight and fineness before a full geochemical assay was run. This involved logging the sample into the tracking system and checking to confirm that the samples received matched the electronic list of samples sent by Western Copper staff. Samples were then weighed and checked to ensure the pulp was of appropriate fineness.

11.3 ASSAY ANALYSIS

Chemex Labs analysed all 1992-1994 regular (mainstream) samples, 1992-1993 selected duplicate samples and 1994 random half-core duplicate samples. Immediately prior to selecting each pulp's analytical aliquot, each pulp sample was passed through a 20-mesh screen to eliminate lumps of agglomerated clay minerals. Gold (Au) was analysed by fire assay with atomic absorption finish. Silver (Ag) values were reported in g/t and Cu and MoS_2 values were reported as percentages. Chemex also performed 32-element ICP analysis for: Ag, Al, As, Ba, Be, Bi, Ca, Cd, Co, Cr, Cu, Fe, Ga, Hg, K, La, Mg, Mn, Mo, Na, Ni, P, Pb, Sb, Sc, Sr, Ti, Tl, U, V, W and Zn. Mineral Environments (Min-En) Laboratories, of North Vancouver, BC, analysed the selected duplicate samples from 1992 and 1993, and random duplicate samples from 1994. Gold was analysed by fire assay and reported in g/t. Values for Cu and MoS_2 were reported as percentages. The analytical procedures utilized prior to 1992 are unknown.

The analytical processes used at ALS Global and for the sample duplicates at Acme Analytical Laboratories were similar. The processes used by ALS Global in Whitehorse in 2019 and those used by SGS Canada Inc. in Burnaby were also similar. The processes used by ALS Global and Bureau Veritas in 2020 were also similar.

11.3.1 Gold Analysis

At ALS Global gold assays were run using 30-gram sample of the pulp with fire assay and AA finish to a 0.005 ppm detection limit according to procedure Au-AA23. Results were reported in parts per million (ppm).

At SGS gold assays were run by using a 30-gram sample of the pulp with fire assay and AAS finish to a 5-ppb detection limit according to procedure GE_FAA30V5. Results were reported in parts per billion (ppb). Note that 5 ppb = 0.005 ppm.

These analytical processes were employed by Western in 2019, as well as from 2008 through 2012.

In 2020, following preparation of the sample at the Whitehorse ALS lab, a full ICP suite was run on the resultant pulps at the Vancouver ALS lab using four-acid digestion with ICP-AES finish. Gold analyses were run according to procedure Au-AA23 on the pulps using a 30-gram sub-sample by fire assay and atomic absorption spectroscopy (AAS) finish to a 0.005 ppm detection limit. Results were reported in parts per million (ppm) or g/t. Ore grade analyses for gold was run by fire assay and gravimetric finish. At Bureau Veritas, gold assays were run using a 30-gram sample of the pulp with fire assay and AAS finish to a 0.005 ppm detection limit according to procedure FA530. Results were reported in parts per million (ppm).

11.3.2 Copper, Molybdenum and Silver Assay

Samples that returned over-limits for copper, molybdenum or silver in the ICP analysis were assayed by process OG62 at ALS Global. This process involved four-acid digestion and analysis by Inductively Coupled Plasma-Atomic Emission Spectroscopy (ICP-AES) or Inductively Coupled Plasma-Atomic Absorption Spectroscopy (ICP-AAS). Results were reported in percent (%). At SGS a similar process was followed for any over-limit results for copper, molybdenum or silver involving sodium peroxide fusion with ICP-AES using method GO ICP90Q.

These analytical processes were employed by Western in 2019, as well as from 2008 through 2012. In 2020, the same analytical processes were employed at ALS Global, and a similar process was employed at Bureau Veritas for any over-limit results for copper, molybdenum or silver. This process comprised a four-acid digestion (0.5 g / 200 ml) with AAS finish using the MA404 method. Results were reported in percent (%).

11.3.3 ICP Analysis

Samples sent to ALS Global were analysed for multiple elements, including copper, molybdenum and silver by process ME-ICP61. This process involved a four acid “Near Total” digestion of 1.0 grams of sample pulp with Mass Emission-Inductively Coupled Plasma Spectroscopy (ICP-MS) for the analysis. This process returned results for: Ag (ppm), Al (%), As (ppm), Ba (ppm), Be (ppm), Bi (ppm), Ca (%), Cd (ppm), Co (ppm), Cr (ppm), Cu (ppm), Fe (%), Ga (ppm), K (%), La (ppm), Mg (%), Mn (ppm), Mo (ppm), Na (%), Ni (ppm), P (ppm), Pb (ppm), S (%), Sb (ppm), Sc (ppm), Sr (ppm), Th (ppm), Ti (%), Tl (ppm), U (ppm), V (ppm), W (ppm), and, Zn (ppm).

Samples sent to SGS were analysed for 56 elements, including copper, molybdenum and silver by method GO_ICP90Q100. This process involved a mineralized material grade sodium peroxide fusion with ICP-AES. This process returned results for: Ag (ppm), Al (%), As (ppm), Ba (ppm), Be (ppm), Bi (ppm), Ca (%), Cd (ppm), Ce (ppm), Co (ppm), Cr (%), Cs (ppm), Cu (ppm), Dy (ppm), Er (ppm), Eu (ppm), Fe (%), Ga (ppm), Gd (ppm), Ge (ppm), Hf (ppm), Ho (ppm), In (ppm), K (%), La (ppm), Li (%), Lu (ppm), Mg (%), Mn (ppm), Mo (ppm), Nb (ppm), Nd (ppm), Ni (ppm), P (%), Pb (ppm), Pr (ppm), Rb (ppm), Sb (ppm), Sc (ppm), Si (%), Sm (ppm), Sn (ppm), Sr (ppm), Ta (ppm), Tb (ppm), Th (ppm), Ti (%), Tl (ppm), Tm (ppm), U (ppm), V (ppm), W (ppm), Y (ppm), Yb (ppm), Zn (ppm) and Zr (ppm).

These analytical processes were employed by Western in 2019, as well as from 2008 through 2012. In 2020, samples sent to ALS Global also underwent the same analytical processes. However, in 2020 "check" pulp samples were sent to the Bureau Veritas Commodities lab in Vancouver, British Columbia, where they underwent analysis for 35 elements, including copper, molybdenum and silver by method MA300. This process involved trace analysis using a multi-acid digestion and ICP-ES finish; it returned results for: Ag (ppm), Al (%), As (ppm), Ba (ppm), Be (ppm), Bi (ppm), Ca (%), Cd (ppm), Co (ppm), Cr (ppm), Cu (ppm), Fe (%), K (%), La (ppm), Mg (%), Mn (ppm), Mo (ppm), Na (%), Nb (ppm), Ni (ppm), P (%), Pb (ppm), S(%), Sb (ppm), Sc (ppm), Sn (ppm), Sr (ppm), Th (ppm), Ti (%), U (ppm), V (ppm), W (ppm), Y (ppm), Zn (ppm) and Zr (ppm).

11.3.4 Acid Soluble Copper Analysis

In 2008 and 2009, following receipt of the copper analyses, samples were selected for "non-sulphide" or "acid soluble" copper analysis. The criteria for "non-sulphide" selection was any sample that contained >100 ppm Cu in the Leached Cap, Supergene Zone, or top 50 m of the Hypogene Zone. A list of these samples was presented to ALS Chemex. ALS Chemex then retrieved the pulps and analysed it by 5% sulphuric acid leach and AAS finish (procedure Cu-AA05).

In 2010 to 2012, selected samples for "acid soluble" copper analyses were identified by the geologist logging the core and the request for this analysis was submitted when the samples were originally sent to the lab. The samples identified by the geologist were generally from the top of the hole down through the top 50 m of the hypogene zone. On a few occasions, after receiving the geochemical results, additional samples were identified for "non-sulphide" copper analyses and ALS Chemex was requested to pull these sample pulps and perform the analysis.

In 2019, once initial ICP assays were returned from ALS Global on the original samples, another group of sample pulps were sent for further assay by Cu-AA05 to identify non-sulphide copper. All pulps returning higher than 100 ppm copper that were also within the Leached Cap, Supergene Oxide, Supergene Sulphide and the initial 50 m of the Hypogene zone were pulled by ALS Global for this analysis. The process for 2020 was the same. The pulps were assayed by method Cu-AA05a (3% sulfuric acid leach and AAS) to test for non-sulphide copper; this was the identical method as employed in 2019.

11.3.5 Cyanide Soluble Copper Analysis

In 2010, a large number of samples from the 2008, 2009 and 2010 programs were identified for cyanide-soluble copper analyses. These samples were selected to aid with identification of the Supergene Sulphide – Hypogene metallurgical boundary. The selected samples were analysed by cyanide leach with AAS finish (ALS Chemex procedure Cu-AA17a). For samples that had already been received and processed at the lab, ALS Chemex retrieved the pulps and analysed this material. For samples not yet sent to the lab, the geologist would identify the Supergene Sulphide – Hypogene boundary visually, and samples 30 m on either side of the boundary were identified for cyanide leach copper analysis. On a few occasions, after receiving the geochemical results additional samples were identified for cyanide soluble copper analyses and ALS Chemex was requested to pull these sample pulps and perform the analysis.

In 2019, the senior geologist used the core logging results to choose samples for cyanide-soluble copper analysis using method Cu-AA17a at ALS Chemex; all sample pulps from 30 m on either side of the Supergene Sulphide and Hypogene boundary were sent for this type of assay. The process utilized in 2020 was the same, and method Cu-AA17, as indicated for 2020, was identical to method Cu-AA17a for 2019.

11.3.6 Security

During the historic pre-1992 drilling campaigns at Casino the rigours of "chain of custody" were not as stringent as presently required. The remoteness of the Casino site provided a large degree of security as air traffic into the project was closely monitored. Further, the Casino gold grades were low and any metal contamination or grade enhancement would be quickly and easily identified. However, good sample handling procedures were in place during the 1992 – 1994 PSG programs. Geologists supervised the sampling process and the samples were kept in a secure impoundment prior to shipping. The best vigilance on the samples was the attention to results, and in that regard, PSG maintained a thorough QA/QC program.

Samples were shipped in rice bags with uniquely numbered, non-re-sealable security tags. Each sample shipment was transported from the Casino Property via air to Whitehorse. The samples were received at the airport by the project expediter and shipped to the appropriate lab from there. In 2008 and early 2009, all shipments were sent by Byers Transport to the ALS Chemex lab in North Vancouver. Later in 2009 and early 2010, samples for ALS Chemex were shipped by Byers Transport to the ALS Chemex preparation facility in Terrace, BC, where they were crushed and pulverized. The pulps were then shipped by ALS Chemex to North Vancouver for analysis. In May of 2010, ALS Chemex opened a preparation facility in Whitehorse. From then on, all samples were delivered to the Whitehorse preparation lab by the project expediter. The samples were crushed and pulverized in Whitehorse and the pulps were shipped to North Vancouver for analysis.

In 2019, ALS Chemex had changed its name to ALS Global, and installed an analytical lab in Whitehorse so that samples could be both prepared and analysed there. This eliminated the problems that could occur with further transport. Samples were shipped from the Casino site by Alkan Air to their base in Whitehorse where the project expediter picked up the samples upon arrival and delivered them directly to ALS Global. Rice bags were organized in batches of 20, with unique identifiers on each bag and sealed with a uniquely numbered non-resealable security strap. Each 20th bag contained the sample submittal form and a list of all the samples that should be included in that particular batch. Upon receipt, ALS would confirm via email with the project manager/senior geologist exactly which samples had been received. The same security regimen was in place in 2020. Upon landing at the Alkan airbase, the samples were picked up by Small's Expediting of Whitehorse, Yukon, and driven to the ALS lab in Whitehorse.

If a shipment was received with a broken security tag, the lab would notify the project manager to determine if the shipment had been tampered with, or if the tag was accidentally damaged during shipping. Any broken sample bags were also brought to the attention of the project manager. In 2020, the project manager was not notified of any broken security tags or sample bags.

11.3.7 Quality Assurance and Quality Control

Exploration sampling and analysis prior to 1992 was not subjected to the rigours required of modern regulatory requirements, but work conducted by major companies, like Quintana and Teck Corporation generally followed industry standard best practices.

However, details of the sampling and analytical methodology are unknown. Moreover, analytical quality, particularly with respect to the determination of gold in the sub-1.000 g/t range, has improved considerably since the pre-1992 work was done. It is for these reasons that the assay results from these old holes were not used in this study.

During the 1993 and 1994 Pacific Sentinel Gold drilling programs, standards, reject duplicates, and half-core replicates were assayed at regular intervals in order to check the security of the samples, as well as the quality and accuracy of the laboratory analyses. Further, in-house laboratory standards, duplicates and blanks were also run and reported as normal assays on certificates.

Figure 11-1 and Figure 11-2 are flow charts illustrating the processing of drill core and quality control procedures from 1992 to 1994.

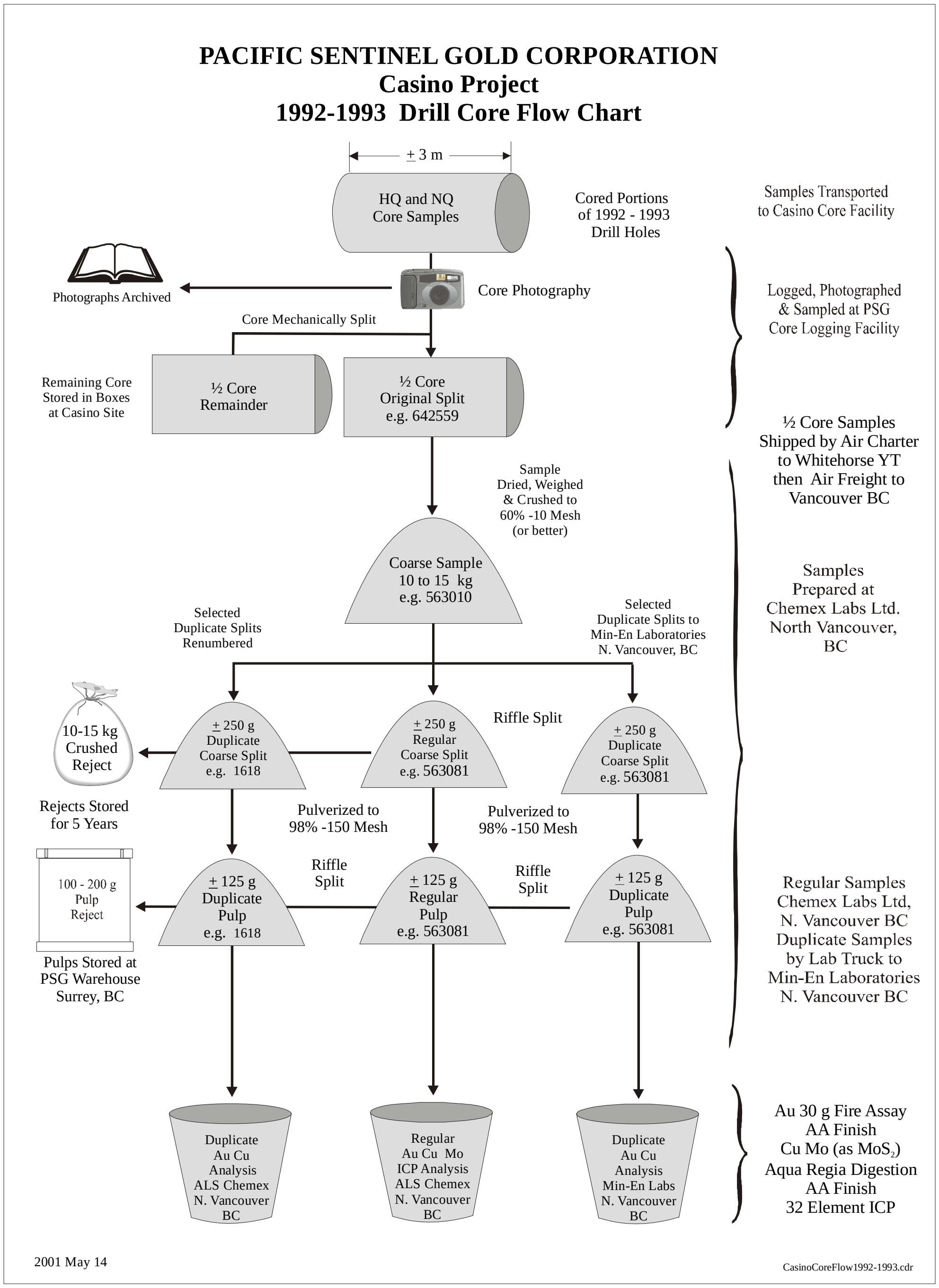


Figure 11-1: Casino Drill Core Processing and Quality Control Procedures, 1992-93

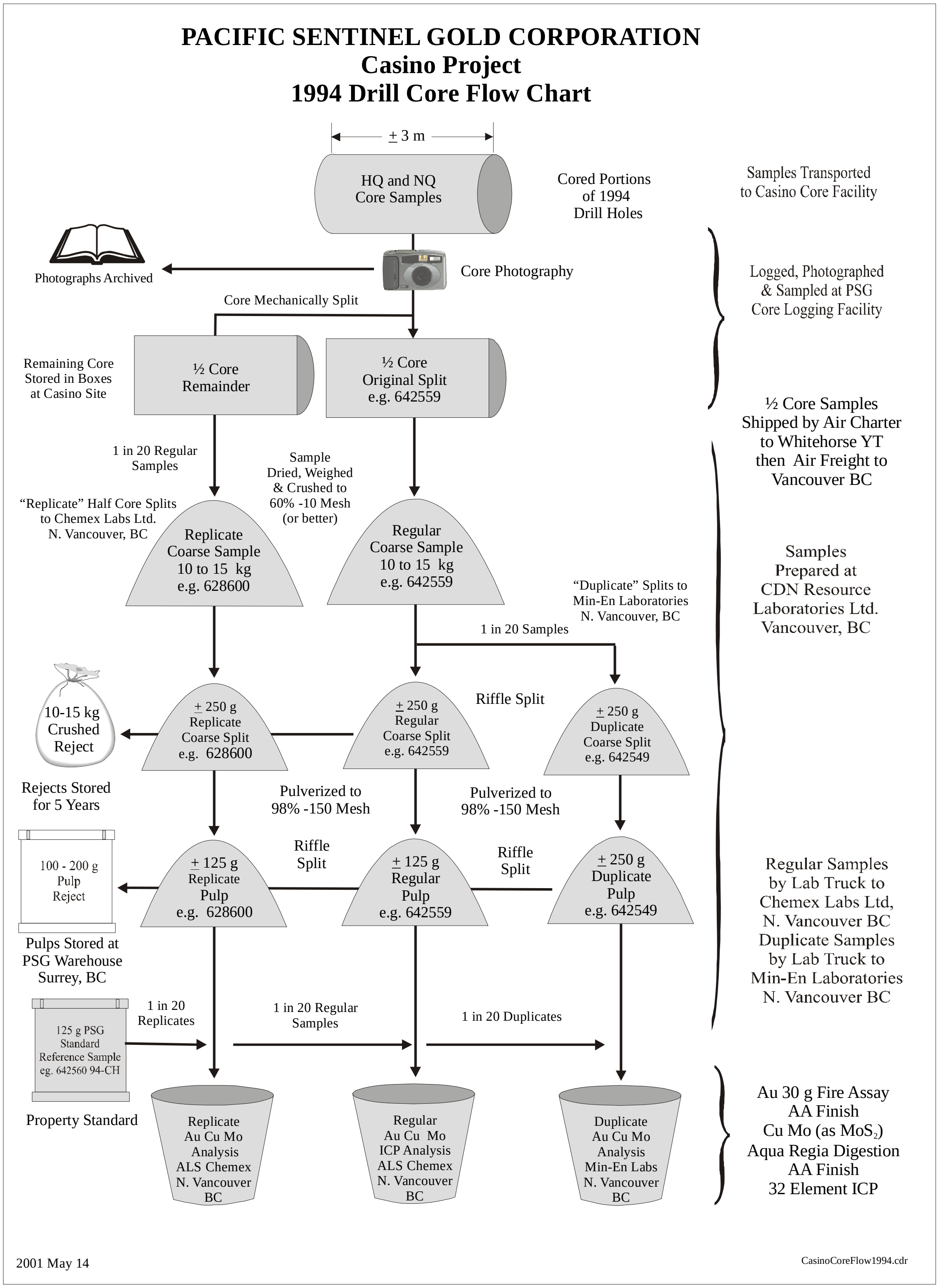


Figure 11-2: Casino Drill Core Processing and Quality Control Procedures 1994

During the 2008 through 2012 drilling programs at Casino, reference material "standards" of known metal content, "blanks", with background metal values, and half-core duplicates were assayed at regular intervals in order to check the security of the samples, as well as the quality and accuracy of the laboratory analyses. The standards and blanks were prepared by CDN Resource Laboratories Ltd. of Delta, BC.

In 2019, standards, quarter-core duplicates and blanks were assayed at regular intervals within the sample stream by the primary lab, ALS Global. One of each (standard, blank, duplicate) were randomly inserted within every 20 core samples. The standards were prepared by WCM Minerals in Burnaby, BC. The same density and protocol of random insertion of quarter-core duplicates, "standards" and blank samples were utilized in 2020. Blank material was comprised of dolomite pebbles commonly sold as garden stone.

11.3.8 Sample Standards

11.3.8.1 2008 through 2010

The standard samples used in 2008, 2009 and 2010 were prepared by CDN Resource Laboratories Ltd. of Delta, BC. The standard was a gold-copper-molybdenum standard, CDN-CM-4. It was certified by Duncan Sanderson, Licensed BC Assayer with independent certification by Dr. Barry Smee, Ph.D., geochemist. Round-robin assaying for the standard was performed at 12 independent laboratories. CDN reports the recommended values and the "Between Lab" Two Standard Deviations of the standard values as:

Gold: 1.18 + 0.12 g/t
Copper: 0.508 + 0.025 %
Molybdenum: 0.032 + 0.004 %

In 2008, 8 standard samples were submitted regularly with the sample shipments; in 2009, 81 standards samples were submitted; and in 2010, 86 standard samples were submitted (approximately 1 per 50 core sample). ALS Chemex analysed the standards along with the drill core samples by gold, copper and molybdenum assay, as well as multi-element ICP as described above.

The results from sample standard CDN-CM-4 for 2008, 2009 and 2010, for gold, copper and molybdenum analyses are plotted below.

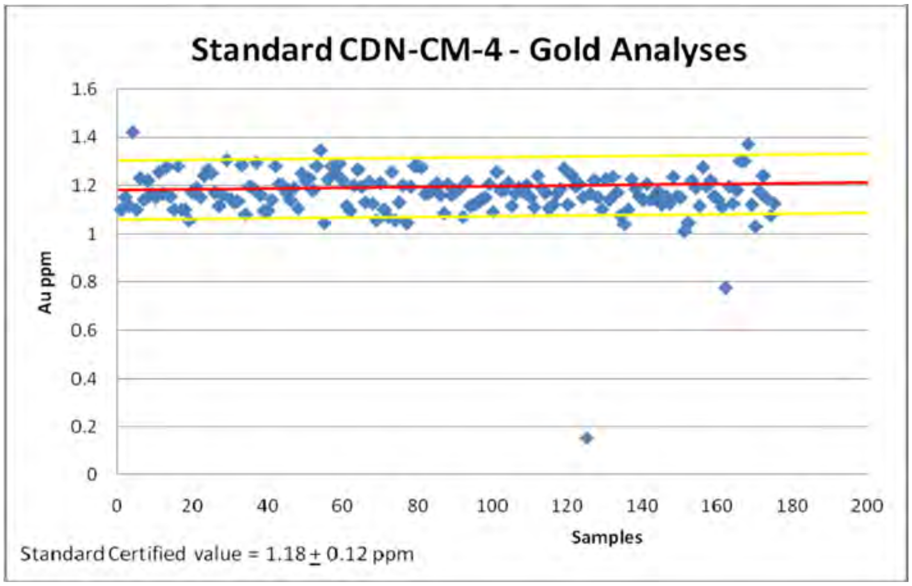


Figure 11-3: Sample Standard CDN-CM-4 Gold Assay Results

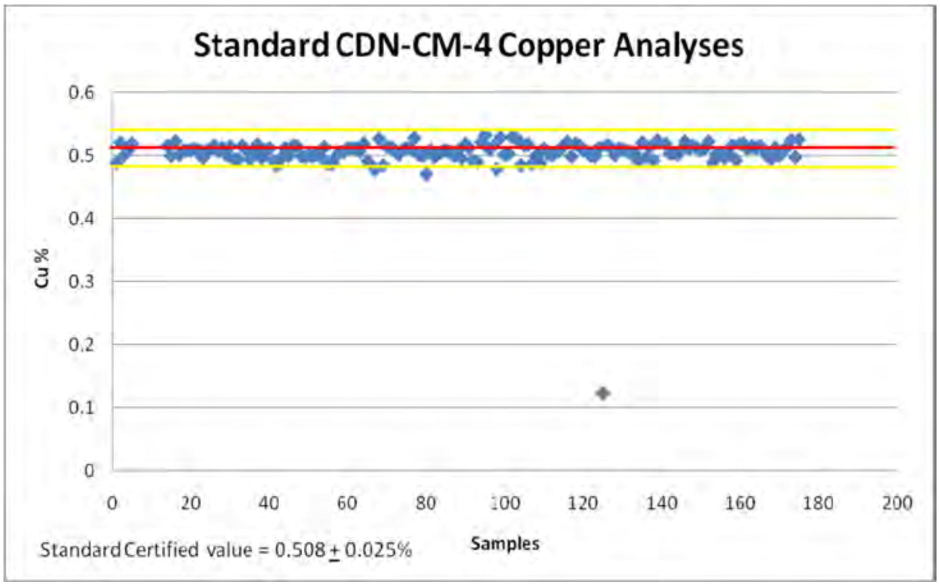


Figure 11-4: Sample Standard CDN-CM-4 Copper Assay Results

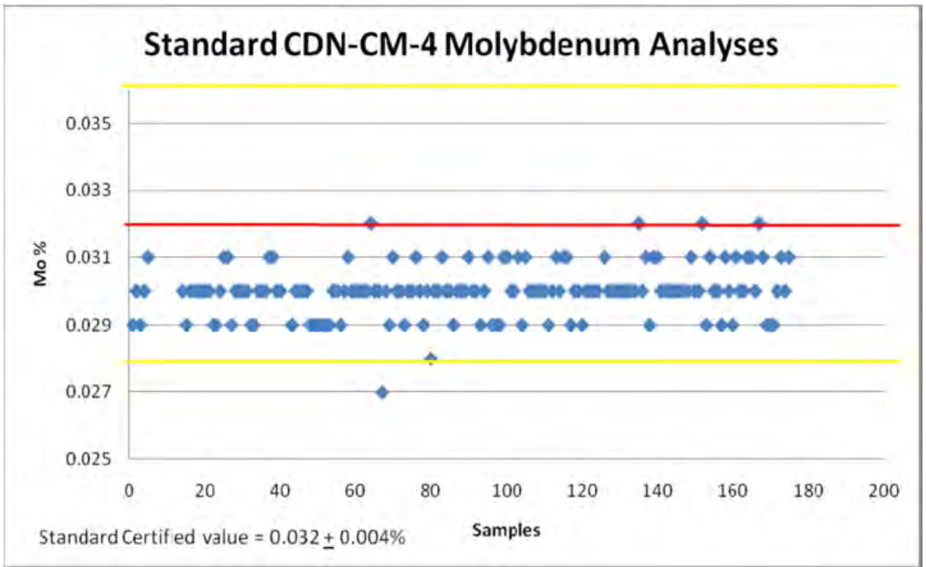


Figure 11-5: Sample Standard CDN-CM-4 Molybdenum Assay Results

The three plots demonstrate that with very few exceptions (9 exceptions for gold, 2 for copper, and one for molybdenum), the values plot within the acceptable range of the certified standard. The plots also demonstrate that there is a reasonable spread of values within the recommended value range of 2 standard deviations as provided by CDN Resource Laboratories Ltd. There does not appear to be any systematic bias.

Later in 2010, a second sample standard (CDN-CM-7) was purchased from CDN Resource Laboratories Ltd. because they had run out of standard CDN-CM-4. This sample is also certified by Duncan Sanderson and Dr. Barry Smee. CDN reports the recommended values and the "Between Lab" Two Standard Deviations of this standard as:

- Gold: 0.427 + 0.042 g/t
- Copper: 0.445 + 0.027 %
- Molybdenum: 0.027 + 0.002 %

Fifteen of these standards were submitted in 2010. ALS Chemex analysed these standards in the same manner as standard CDN-CM-4, described above.

The results from sample standard CDN-CM-7 for 2010, for gold, copper and molybdenum analyses are plotted below:

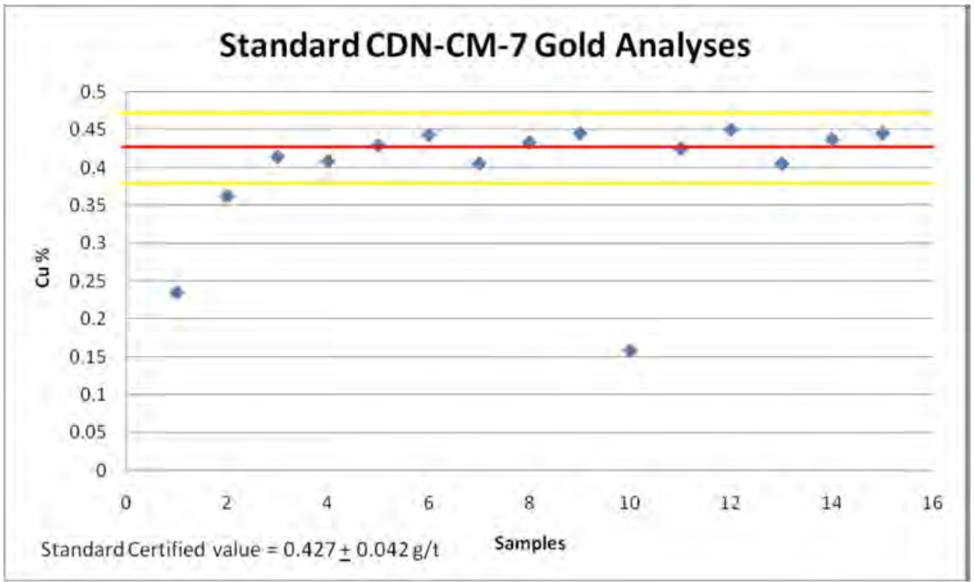


Figure 11-6: Sample Standard CDN-CM-7-Gold Assay Results

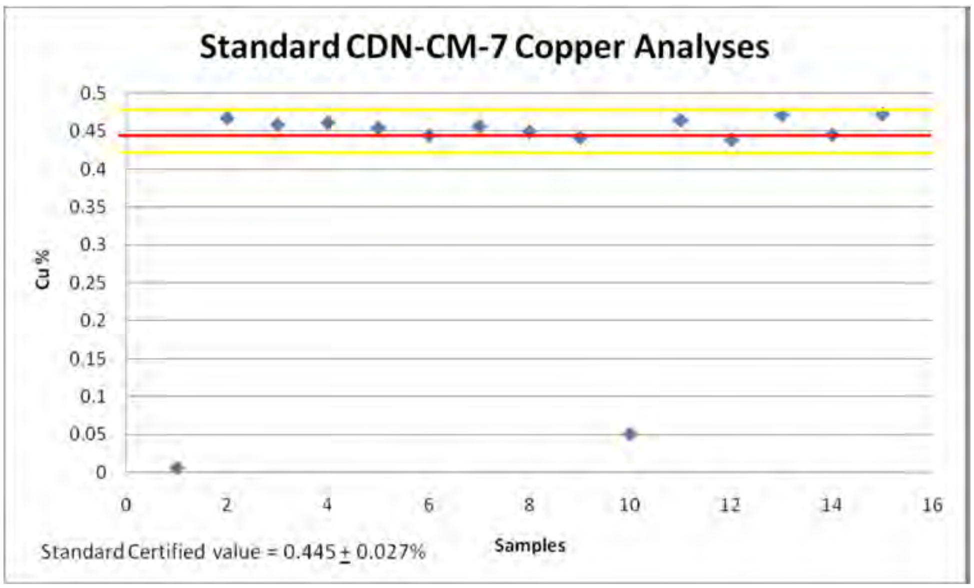


Figure 11-7: Sample Standard CDN-CM-7-Copper Assay Results

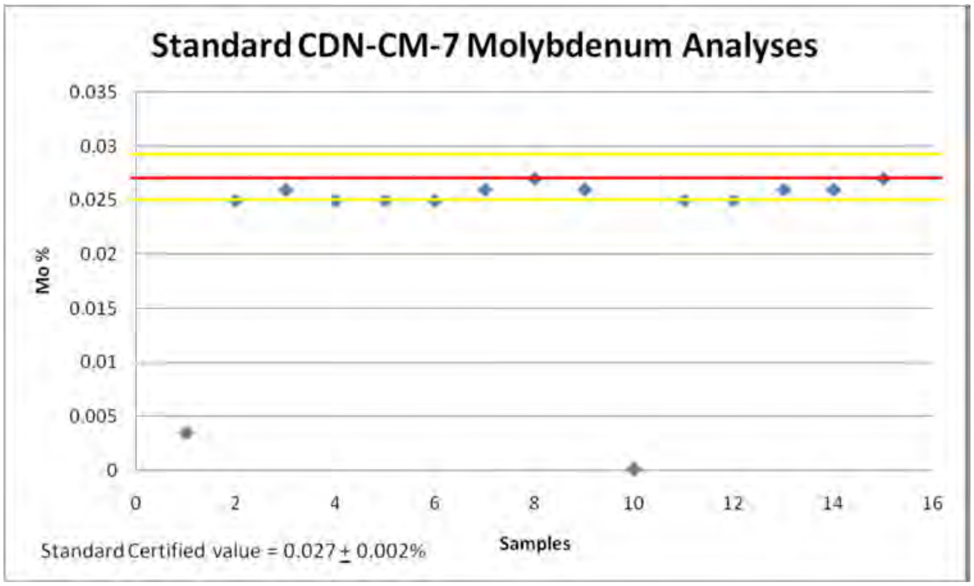


Figure 11-8: Sample Standard CDN-CM-7 Molybdenum Assay Results

The three plots show good precision with the exception of samples 1 and 10 which are well below the expected values as certified by CDN. After checking the ALS Chemex internal standards and the duplicates from these batches, there did not appear to be a systemic error in the batches. The error may have occurred when the sample standards were inserted in the field, or when the standards were originally placed in the geochemical run at the lab. These anomalous errors are not considered significant considering that the great majority of standards were within the expected range.

11.3.8.2 2019

The sample standards used in 2019 were prepared by WCM Minerals in Burnaby, BC. Details of the standards are outlined in Table 11-1 below. Both standards were certified by Lloyd Twaites and Glen Armanini, who are both Registered Assayers in British Columbia.

Table 11-1: 2019 Standard Reference Materials from WCM Minerals

Standard	Copper (%)	Standard Deviation Cu	Molybdenum (%)	Standard Deviation Mo	Silver (g/t)	Standard Deviation Ag	Gold (g/t)	Standard Deviation Au
CU-185	0.400	0.0093	0.035	0.0019	15	0.6242	0.62	0.0217
CU-188	0.179	0.0068	0.018	0.0009	15	0.7883	0.4	0.0199

In 2019, 273 standard samples (1 standard within every 20 samples) were submitted regularly with the sample shipments; 154 of which were of CU-188 and 119 of which were of CU-185. ALS Global analysed the standards along with the drill core samples by gold, copper and molybdenum assay, as well as multi-element ICP as described above.

The results from sample standards CU-185 and CU-188 for gold, silver, copper and molybdenum analyses are plotted below in Figure 11-9 through Figure 11-16.

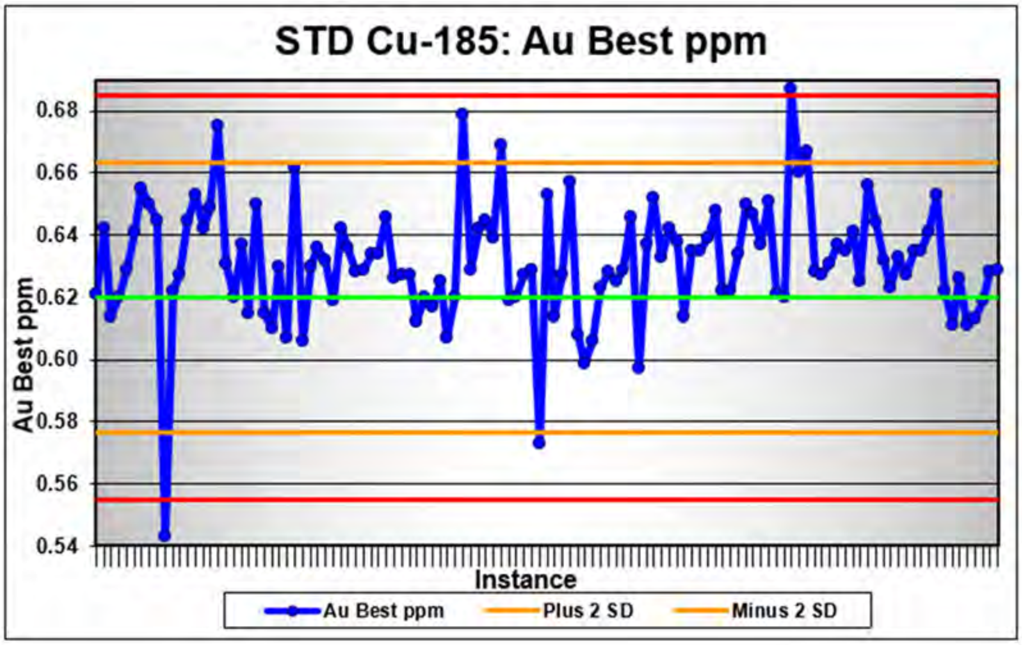


Figure 11-9: Sample Standard CU-185 Gold Assay Results

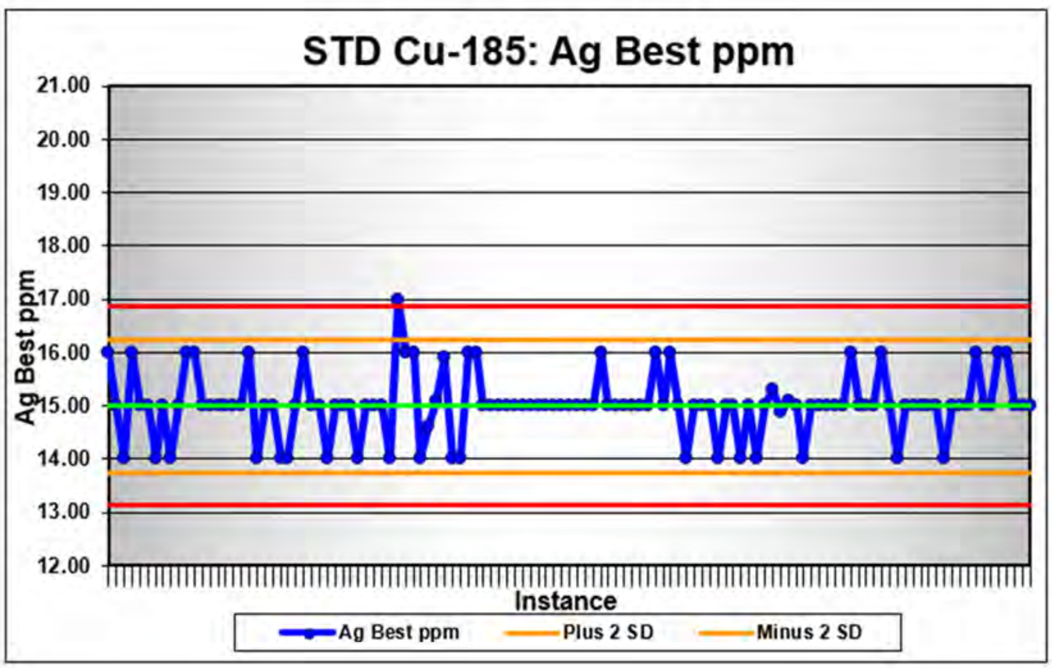


Figure 11-10: Sample Standard CU-185 Silver Assay Results

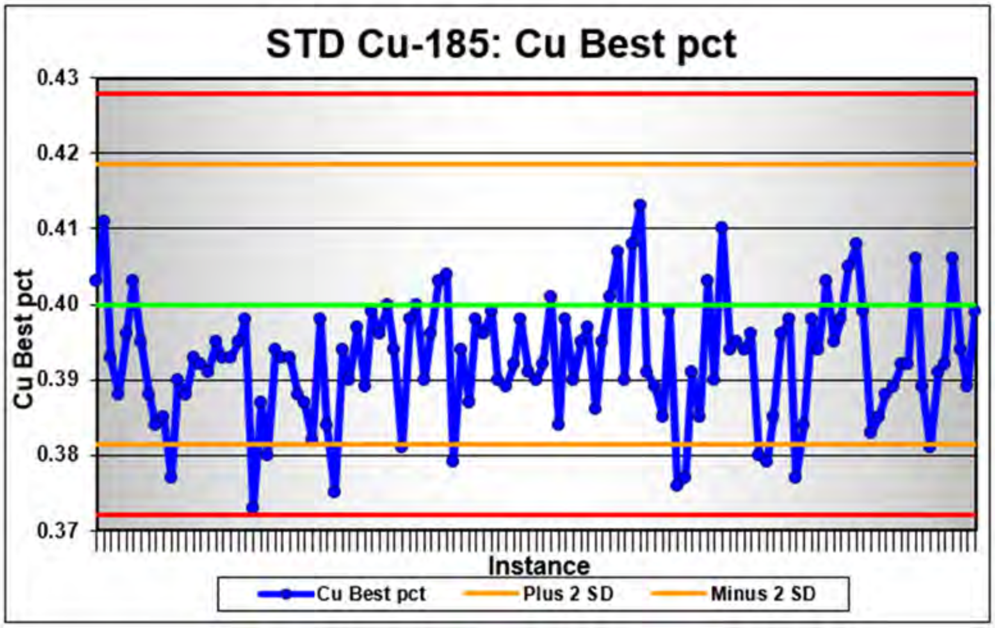


Figure 11-11: Sample Standard CU-185 Copper Assay Results

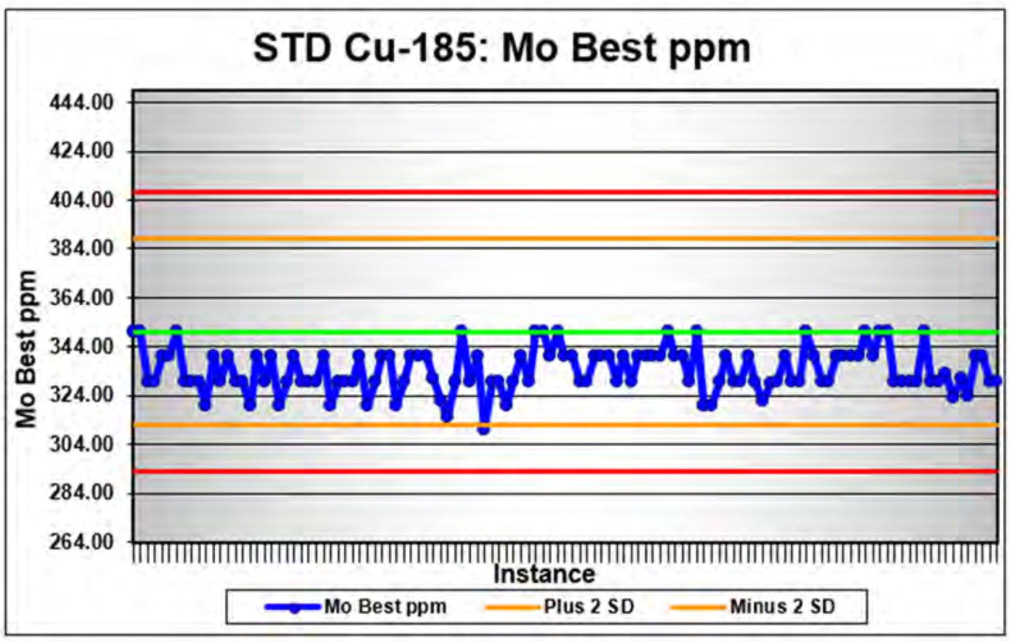


Figure 11-12: Sample Standard CU-185 Molybdenum Assay Results

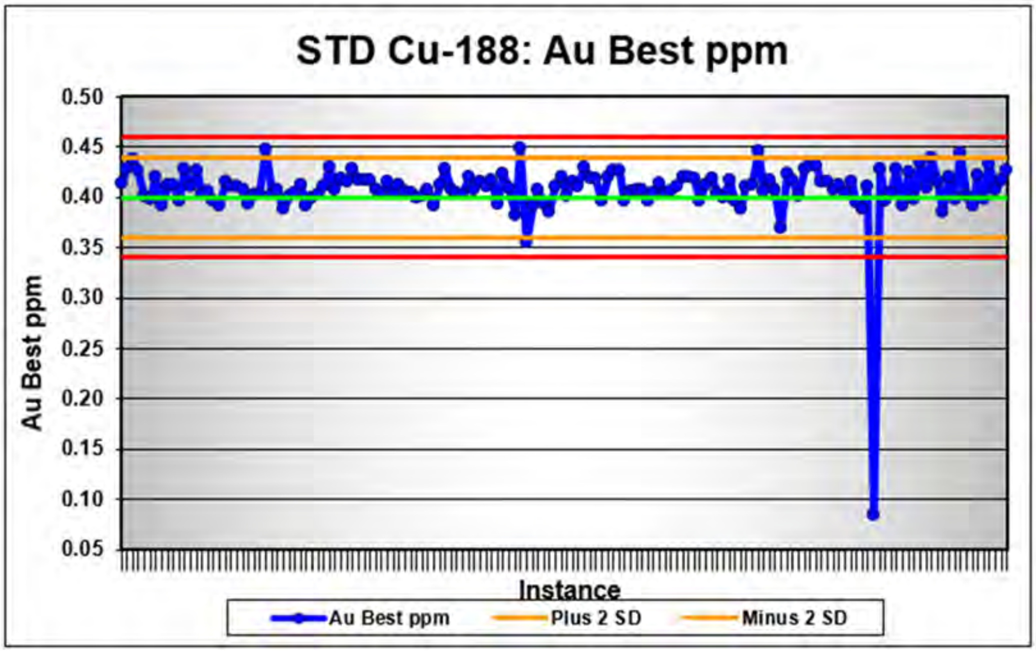


Figure 11-13: Sample Standard CU-188 Gold Assay Results

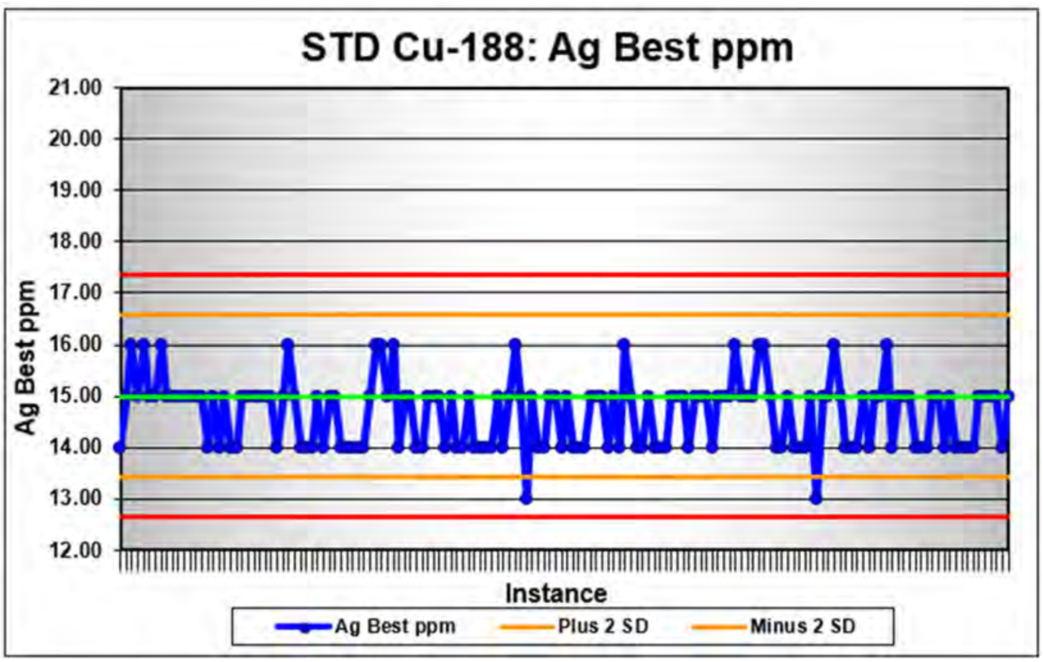


Figure 11-14: Sample Standard CU-188 Silver Assay Results

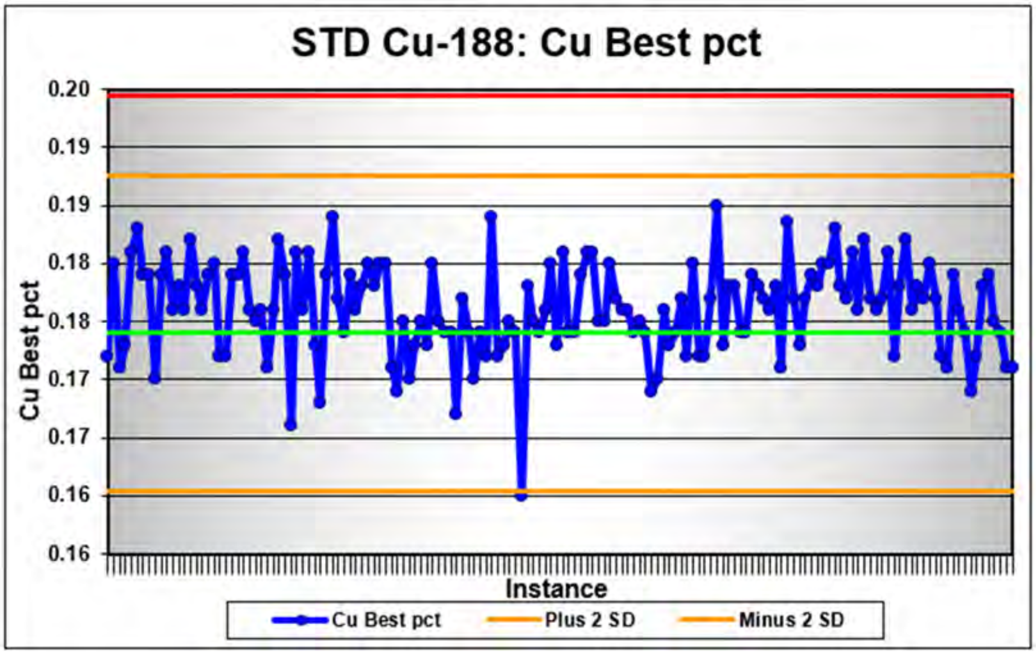


Figure 11-15: Sample Standard CU-188 Copper Assay Results

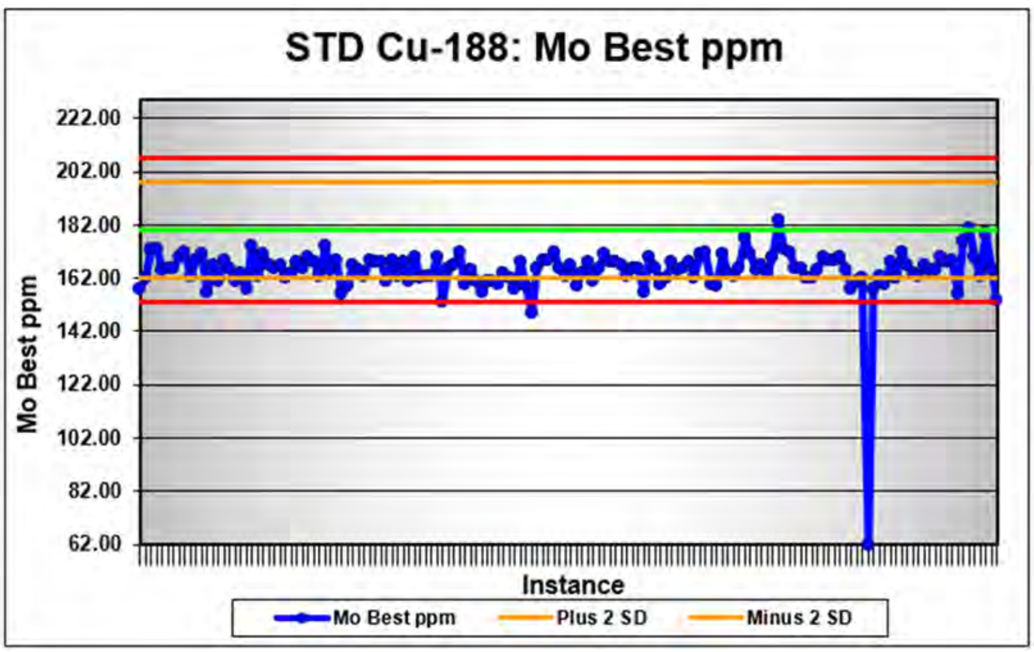


Figure 11-16: Sample Standard CU-188 Molybdenum Assay Results

Standard CU-185 performed well for all elements of interest in 2019; all elements had higher than 90% passing rates within both two and three standard deviations of the mean expected values. In general, both copper and molybdenum values fell below the expected mean for CU-185, but still within an acceptable range. Silver showed good variation both above and below the mean value and gold values generally plotted slightly above the mean value. Table 11-2 summarizes the results for CU-185.

Table 11-2: Performance of Standard CU-185 During 2019 Drill Program Sampling

Element	# Failures within 2 Standard Deviations	% Passing within 2 Standard Deviations	# Failures within 3 Standard Deviations	% Passing within 3 Standard Deviations
Au	7	94	2	98
Ag	1	99	1	99
Cu	12	90	0	100
Mo	1	99	0	100

Standard CU-188 also performed well for all elements of interest in 2019; all elements, except Molybdenum (Mo) had higher than 90% passing rates within both two and three standard deviations of the mean expected values. In the case of the 32 standards that fell outside of the range of 2 standard deviations for Mo, the chart shows that, overall, this standard returned assay results below the expected mean value for Mo, as did those of CU-185. This indicates that both standards should be reassessed in a round robin process, and that the assay method ALS Global uses may tend toward a low bias for Mo. Even with the 32 Mo failures, 79.2% of the samples fell within 2 standard deviations and the range of values was acceptable. One sample, A0612554, failed outright for both Mo and Au. It is possible this sample became contaminated, as ALS Global had notified the project manager that this sample arrived with a torn plastic bag and had to be dried. Table 11-3 summarizes the results for CU-188.

Table 11-3: Performance of Standard CU-188 During 2019 Drill Program Sampling

Element	# Failures within 2 Standard Deviations	% Passing within 2 Standard Deviations	# Failures within 3 Standard Deviations	% Passing within 3 Standard Deviations
Au	7	95	1	99
Ag	2	98.7	0	100
Cu	1	99	0	100
Mo	32	79.2	2	98.7

11.3.8.3 2020

The sample reference materials "standards" used in 2020 were prepared by WCM Minerals in Burnaby, BC. Details of the standards are outlined in Table 11-4 below. Both standards were certified by Lloyd Twaites and Glen Armanini, who are both Registered Assayers in British Columbia.

Table 11-4: Reference material "Standards", utilized in 2020 (WCM Minerals)

Standard	Copper (%)	Standard Deviation Cu	Molybdenum (%)	Standard Deviation Mo	Silver (g/t)	Standard Deviation Ag	Gold (g/t)	Standard Deviation Au
CU-190	0.65	0.0188	0.032	0.0013	9	0.7580	0.68	0.0279
CU-188	0.179	0.0068	0.018	0.0009	15	0.7883	0.4	0.0199

In 2020, 250 sample standards (1 standard within every 20 samples) were submitted regularly with the sample shipments. Of these, 178 were CU-188 and 72 of which were CU-190. ALS Global Geochemistry (ALS) analyzed the standards along with the drill core samples by gold, copper and molybdenum assay, as well as multi-element ICP.

The results from sample standards CU-188 and CU-190 for gold, silver, copper and molybdenum analyses are plotted below in Figure 11-17 through Figure 11-24.

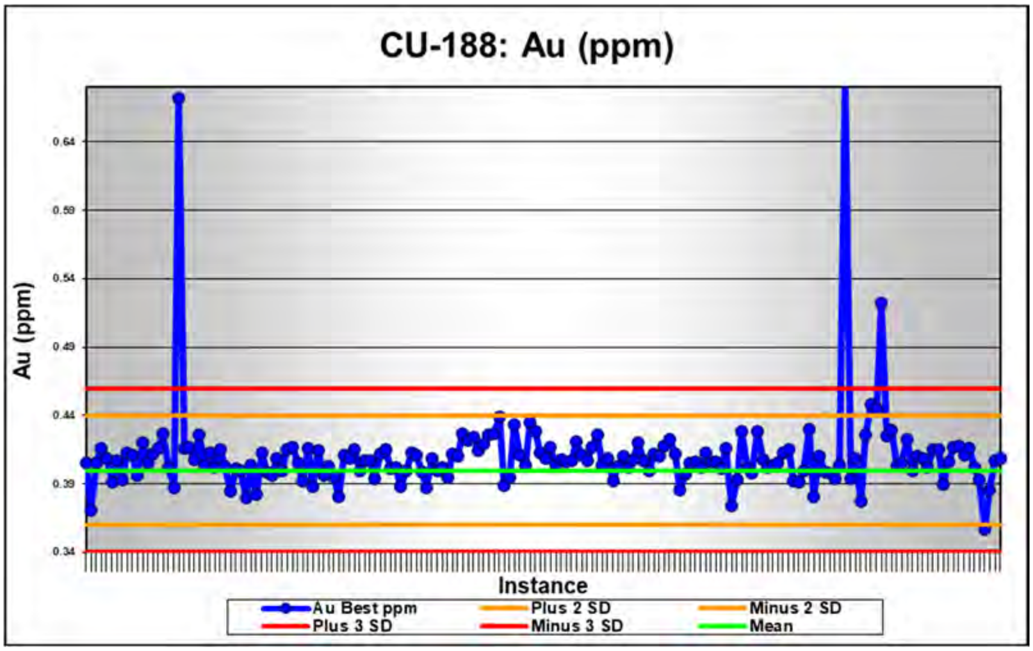


Figure 11-17: Standard CU-188 gold assay results

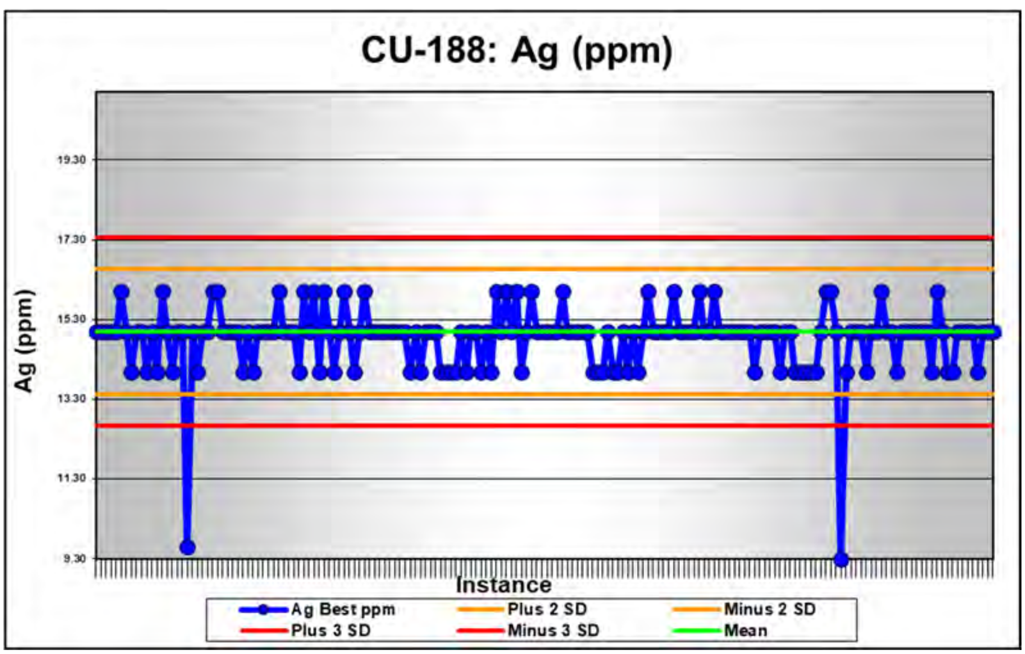


Figure 11-18: Standard Cu-188 silver assay results

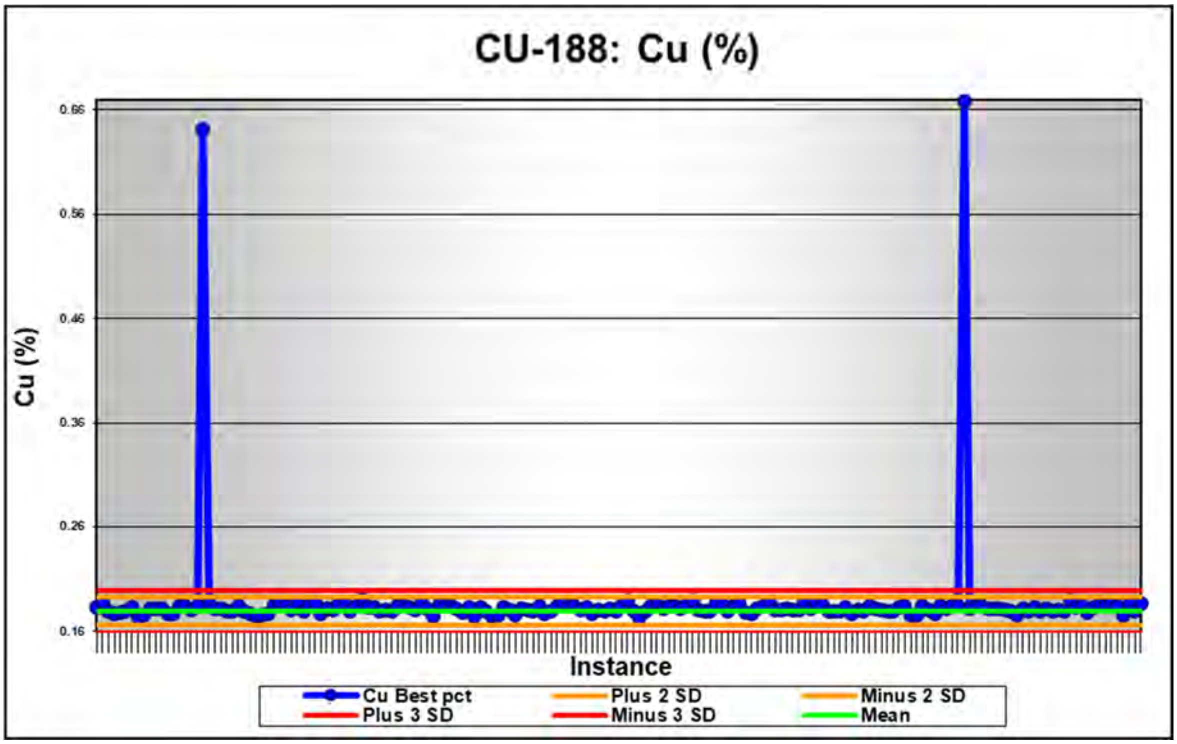


Figure 11-19: Standard Cu-188 copper assay results

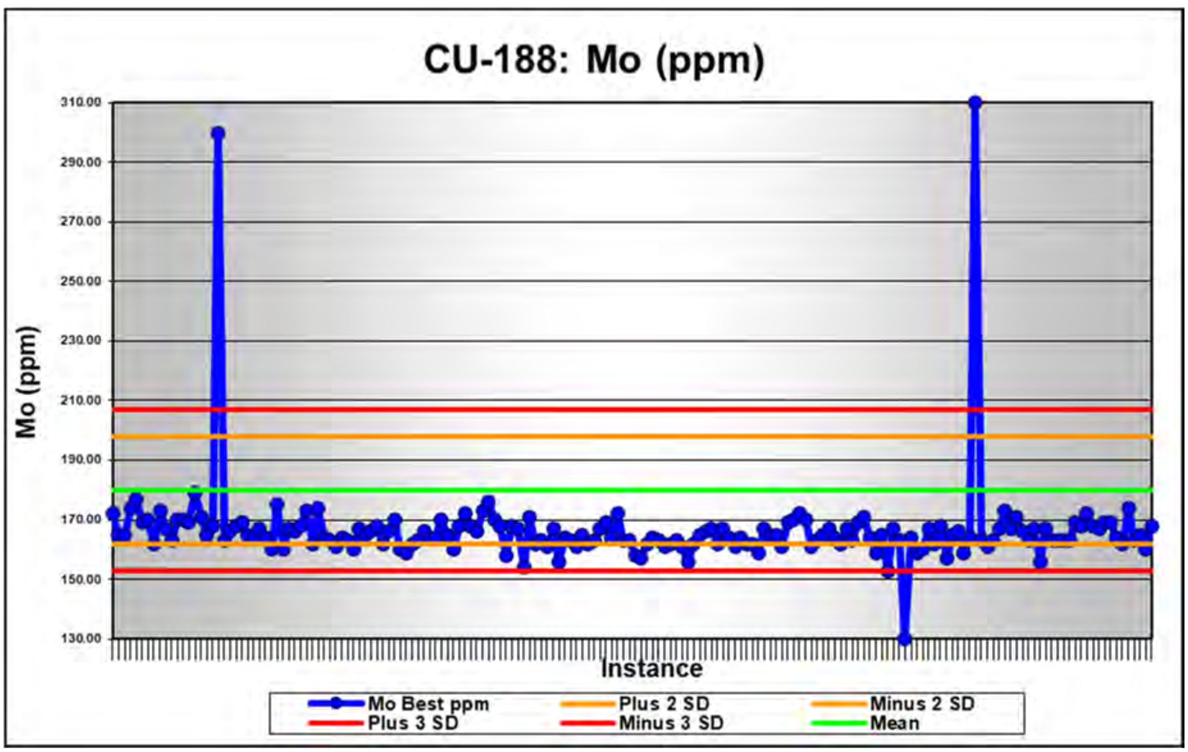


Figure 11-20: Standard Cu-188 molybdenum assay results

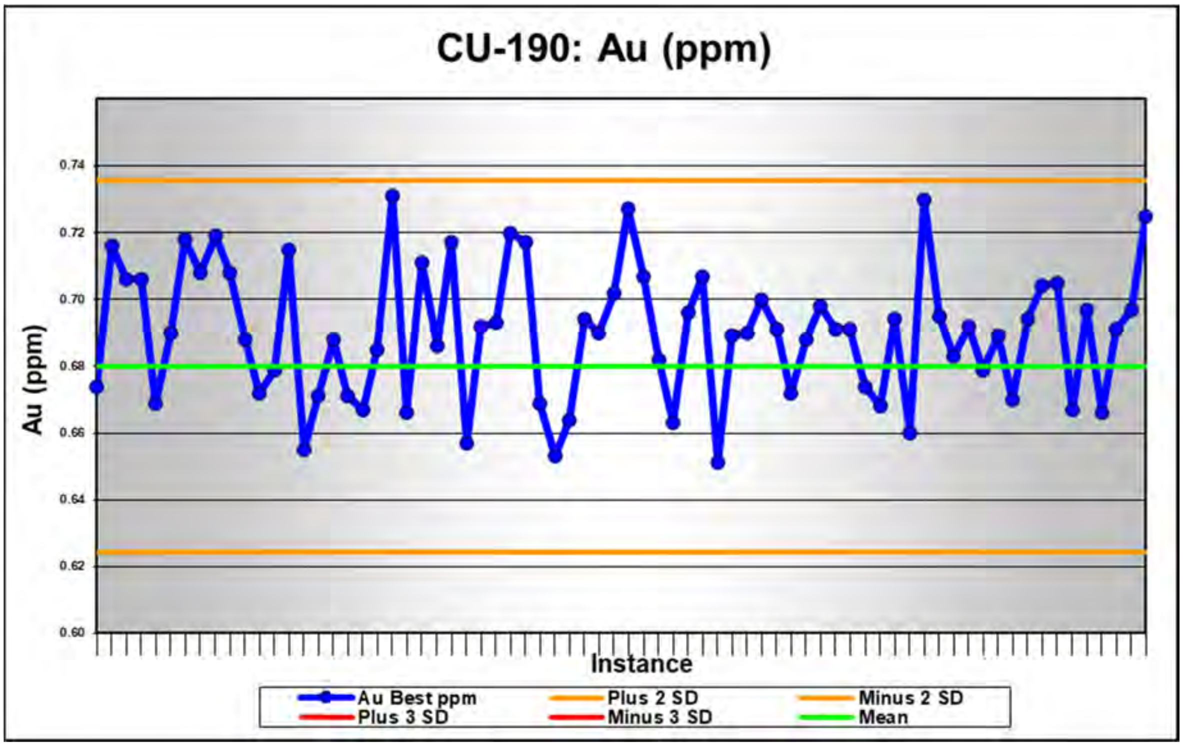


Figure 11-21: Standard Cu-190 gold assay results

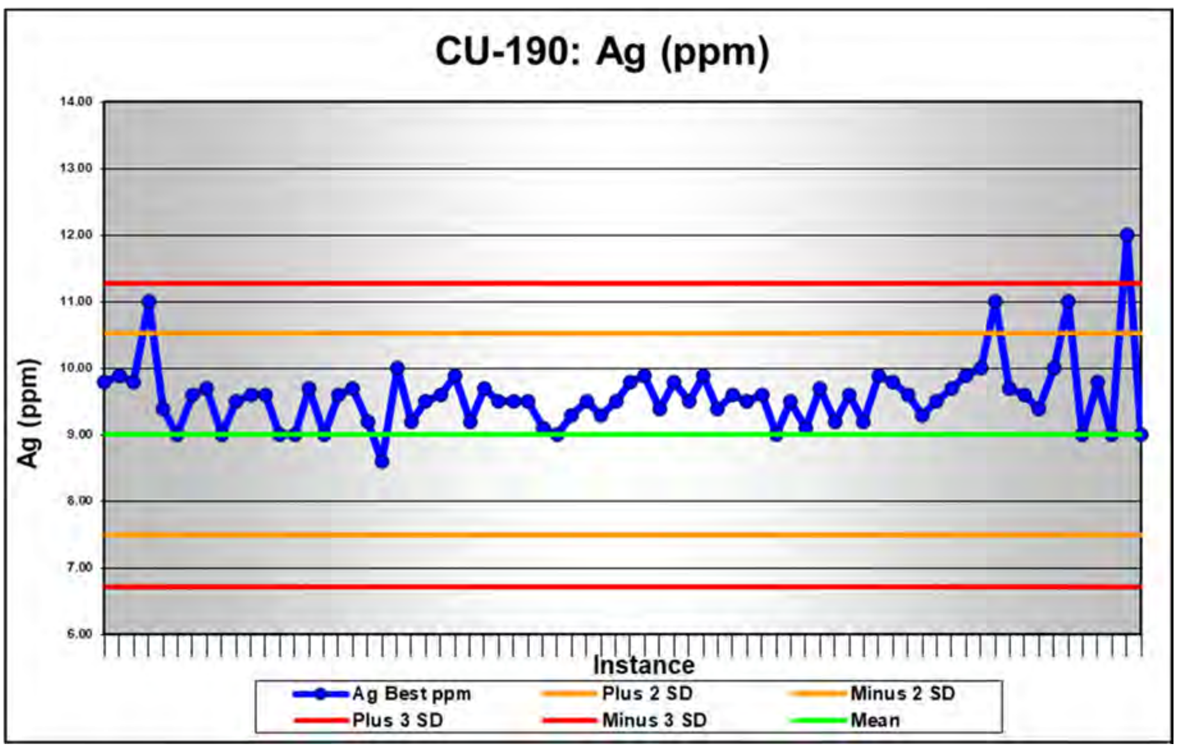


Figure 11-22: Standard Cu-190 silver assay results

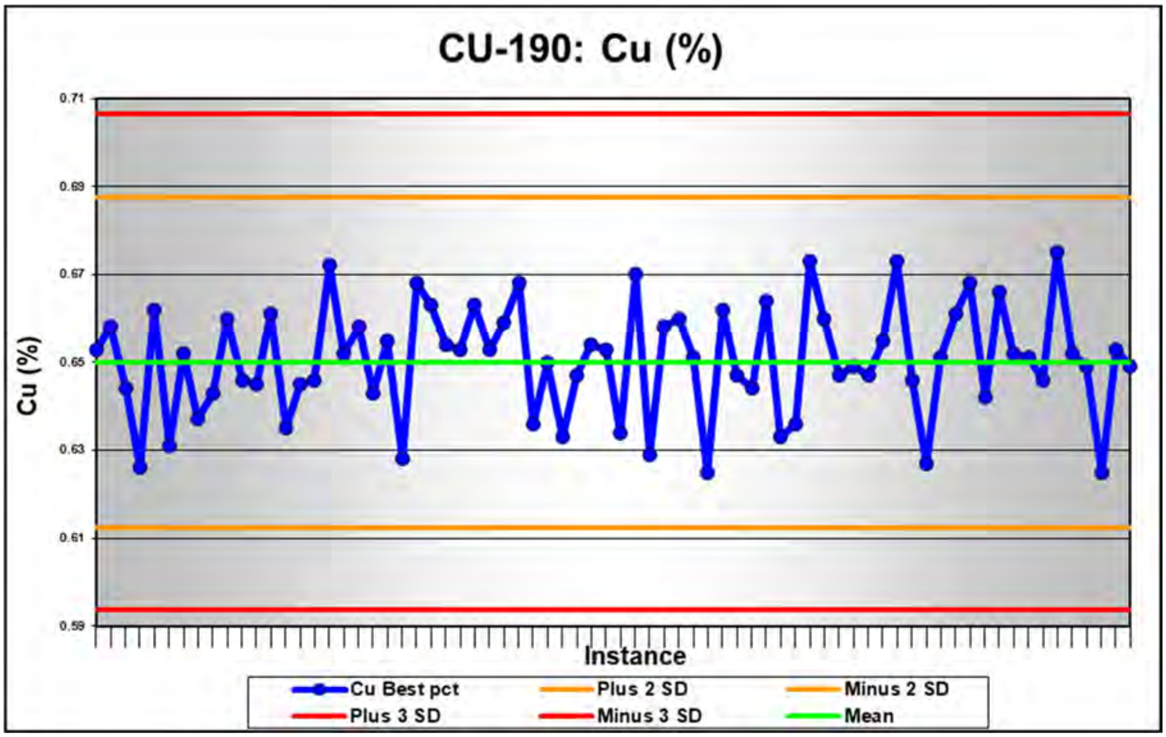


Figure 11-23: Standard CU-190 copper assay results

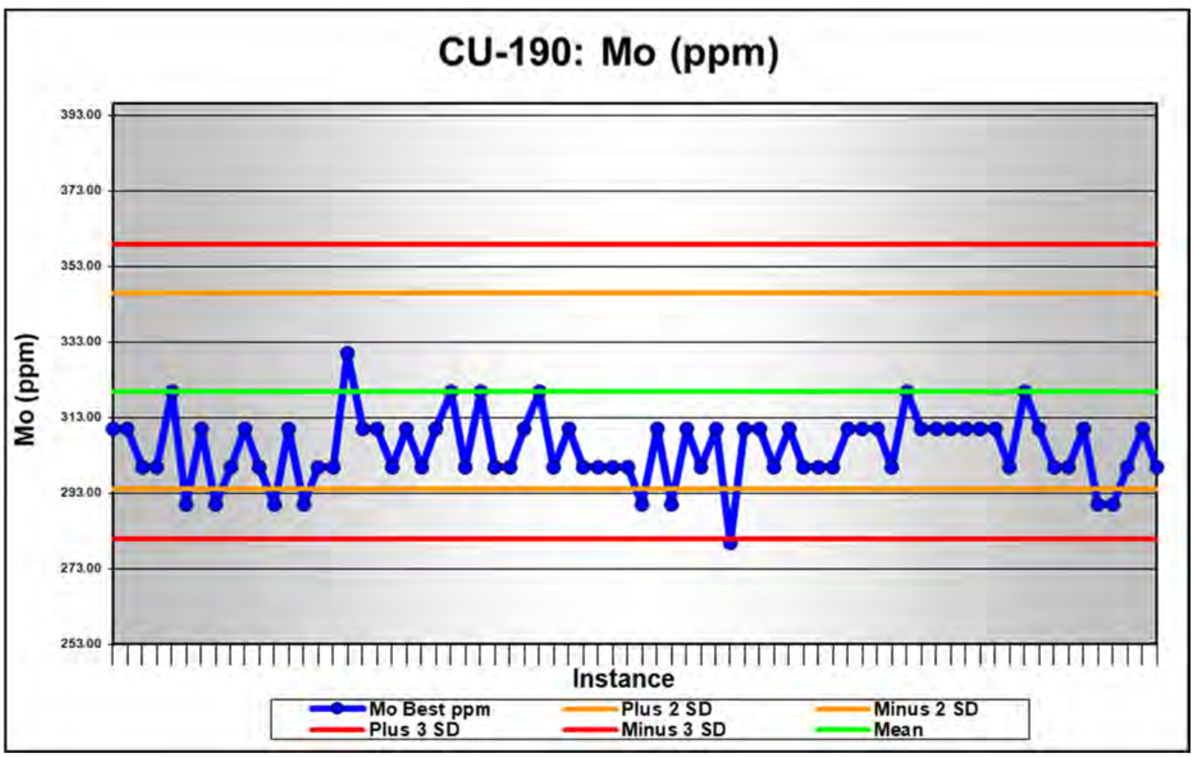


Figure 11-24: Standard CU-190 molybdenum assay results

Standard reference material CU-188 performed well for gold, silver and copper; all had higher than 96% passing rates within both two and three standard deviations of the mean expected values. CU-188 did not perform as well for molybdenum, but results were still acceptable, with 82% passing within two standard deviations and 98% within three standard deviations. Molybdenum values in general fell below the expected mean for CU-188, with an average closer to 166 ppm Mo, 14 ppm below the expected mean of 180 ppm. This also occurred in 2019 with CU-188 and indicates the standard itself should be reassessed by a Round Robin analysis. Two samples that failed for all the elements of interest (B660215 and A0610430) likely represent data entry errors as their values correspond to those of CU-190. Table 11-5 summarizes the results for CU-188.

Table 11-5: Performance of Standard CU-188 during 2020 drill program

Element	# Failures within 2 Standard Deviations	% Passing within 2 Standard Deviations	# Failures within 3 Standard Deviations	% Passing within 3 Standard Deviations
Au	6	97	3	98
Ag	2	99	2	99
Cu	2	99	2	99
Mo	33	82	3	98

Standard reference material CU-190 also performed well for all elements of interest in 2020 (Table 11-6); all elements, except molybdenum (Mo) had higher than 93% passing rates within both two and three standard deviations of the mean expected values. As with CU-188, the overall average Mo value returned for CU-190 is 305 ppm, 15 ppm lower than the expected certified value of 320 ppm Mo. Over 81% of CU-190 sample values were within two standard deviations, which is acceptable, but similar to results for CU-188. CU-190 likely requires an updated Round Robin analysis to determine the accuracy of the certified mean value for molybdenum.

Table 11-6: Performance of Standard CU-190 during 2020 drill program

Element	# Failures within 2 Standard Deviations	% Passing within 2 Standard Deviations	# Failures within 3 Standard Deviations	% Passing within 3 Standard Deviations
Au	0	100	0	100
Ag	4	94	1	99
Cu	0	100	0	100
Mo	9	87.5	1	99

11.3.9 Blanks

11.3.9.1 2010-2012

Commencing in 2010, sample blanks were regularly inserted into the sample stream. Blanks are included as a check of the lower limit of the analytical range and to ensure that, at all stages in the process, the equipment and instruments are thoroughly cleaned prior to running subsequent samples. This is particularly important for precious metals. A total of 75 blanks were submitted during the 2010 program, nominally one every 50 samples.

The blank samples were also prepared by CDN Resource Laboratories Ltd (CDN-BL-6). They were certified for gold, platinum and palladium. The recommended values for these elements are:

- Gold: <0.01 g/t

- Platinum: <0.01 g/t
- Palladium: <0.01 g/t

Since the reported recommended gold values by CDN are less than detection it is not included in a plot. The gold values of the blanks analysed ranged from below detection (<0.005 g/t) to a maximum of 0.046 g/t. The silver values ranged from <0.5 to 0.8 ppm.

11.3.9.2 2019

During the 2019 drill program, a landscape aggregate that was readily available in Whitehorse was used as blank material. It was sent to 4 different labs for a Round Robin analysis and the following values were calculated from those Round Robin results:

- Gold: 0.002 ppm
- Silver: 0.2 ppm
- Copper: 0.00045 %
- Molybdenum: 0.39 ppm

Approximately 100g of blank material was placed in each sample bag and 1 blank sample was inserted randomly within every 20 core samples. A total of 277 blank samples were inserted into the sample stream in 2019.

The results from blank material for gold, silver, copper and molybdenum analyses are plotted below in Figure 11-25 through Figure 11-28.

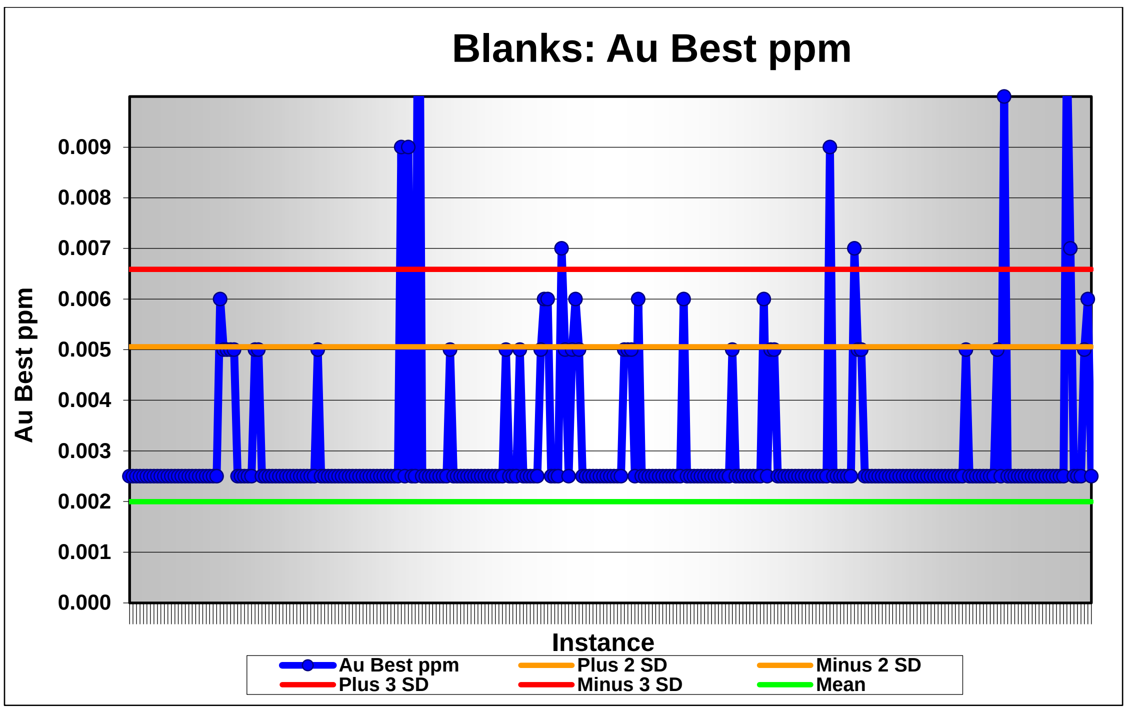


Figure 11-25: Blank Material Gold Assay Results

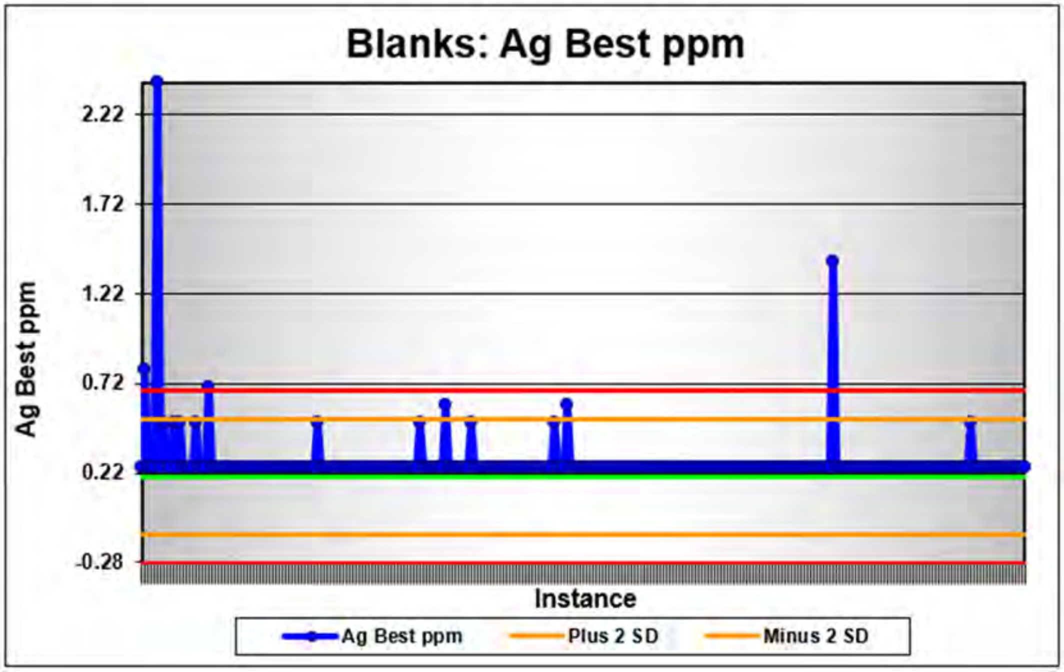


Figure 11-26: Blank Material Silver Assay Results

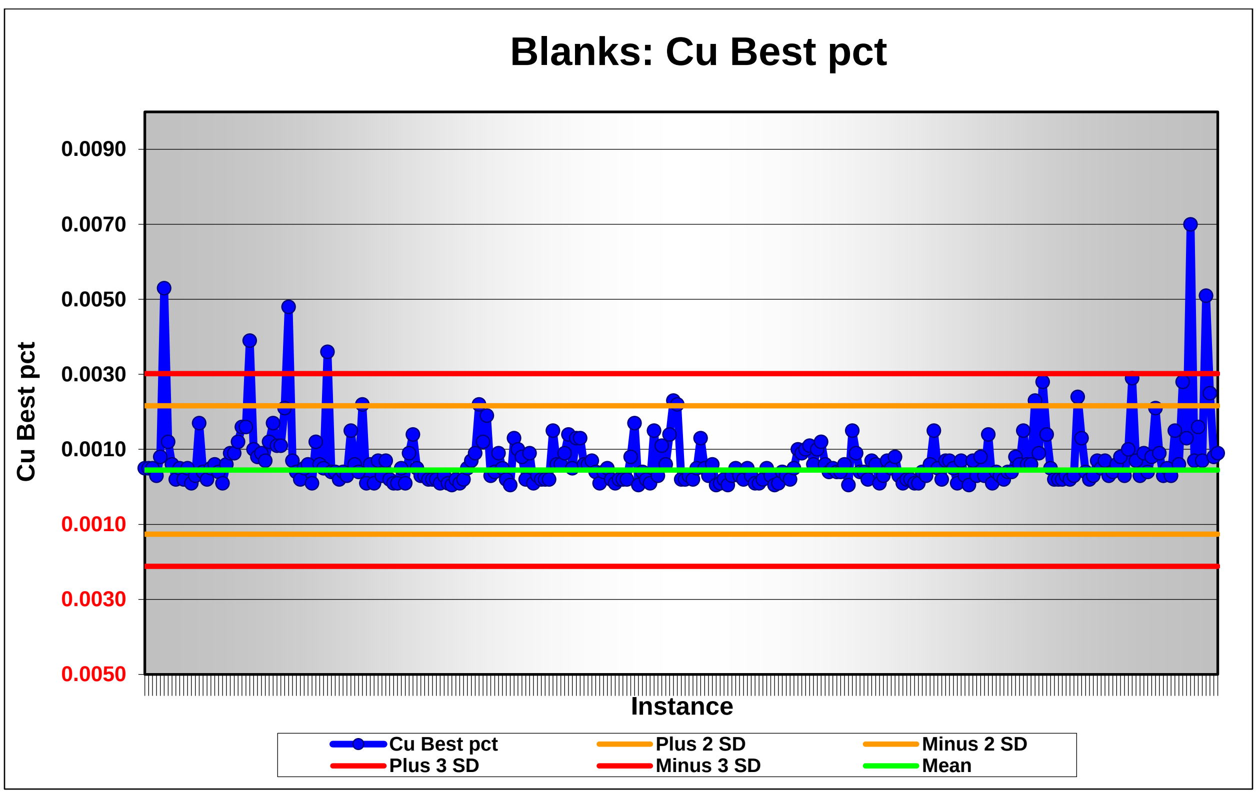


Figure 11-27: Blank Material Copper Assay Results

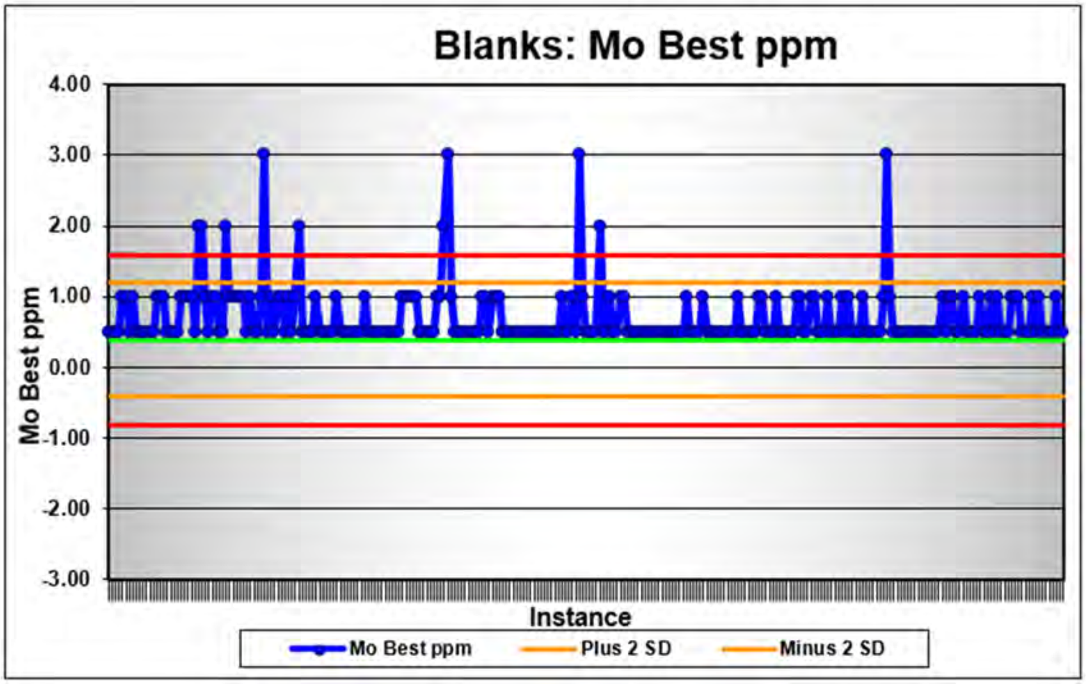


Figure 11-28: Blank Material Molybdenum Assay Results

The blank material performed well for all elements of interest in 2019; all elements had higher than 90% passing rates within both two and three standard deviations of the mean expected values. On average the gold plotted well above the expected mean, but as Figure 11-17 shows, the detection limit for gold at ALS Global limits the lowest assay value to 0.0025 ppm, which is above the expected mean of 0.002 ppm for the blank material. The detection limits for silver and molybdenum are also higher than the expected mean of the blank material for those elements. Even with the detection limit cut-offs, the passing rates are still acceptable. Figure 11-17 and Figure 11-18, for Gold and Silver respectively, do show a few off-chart potentially high-value failures. Upon investigation for gold the two samples with the greatest variation from the expected mean of 0.002 ppm Au varied by only 10-13%. The samples prior to these blanks returned 0.159 ppm Au and 0.283 ppm Au respectively, indicating the likelihood of some minor smear during the assaying. The failures for silver are somewhat less certain as there is no indication of high-grade material prior to the failed silver values. Table 11-4 below summarizes the overall performance of the Blank Material in 2019.

Table 11-7: Performance of Blank Material During 2019 Drill Program Sampling

Element	# Failures within 2 Standard Deviations	% Passing within 2 Standard Deviations	# Failures within 3 Standard Deviations	% Passing within 3 Standard Deviations
Au	17	94	9	97
Ag	6	98	4	99
Cu	16	94	6	98
Mo	10	96	10	96

11.3.9.3 2020

During the 2020 drill program, a landscape aggregate (dolostone) that was readily available in Whitehorse was used as Blank Material. It was sent to 4 different labs for a Round Robin analysis in 2019 and the following values were calculated:

- Gold: 0.002 ppm
- Silver: 0.2 ppm
- Copper: 0.00045 %
- Molybdenum: 0.39 ppm

Each "blank" sample was comprised of approximately 200 g of material placed into a sample bag. One blank sample was inserted randomly within every 20 core samples. A total of 251 Blanks were inserted into the sample stream in 2020.

The results from blank material for gold, silver, copper and molybdenum analyses are plotted below in Figure 11-29 through Figure 11-32.

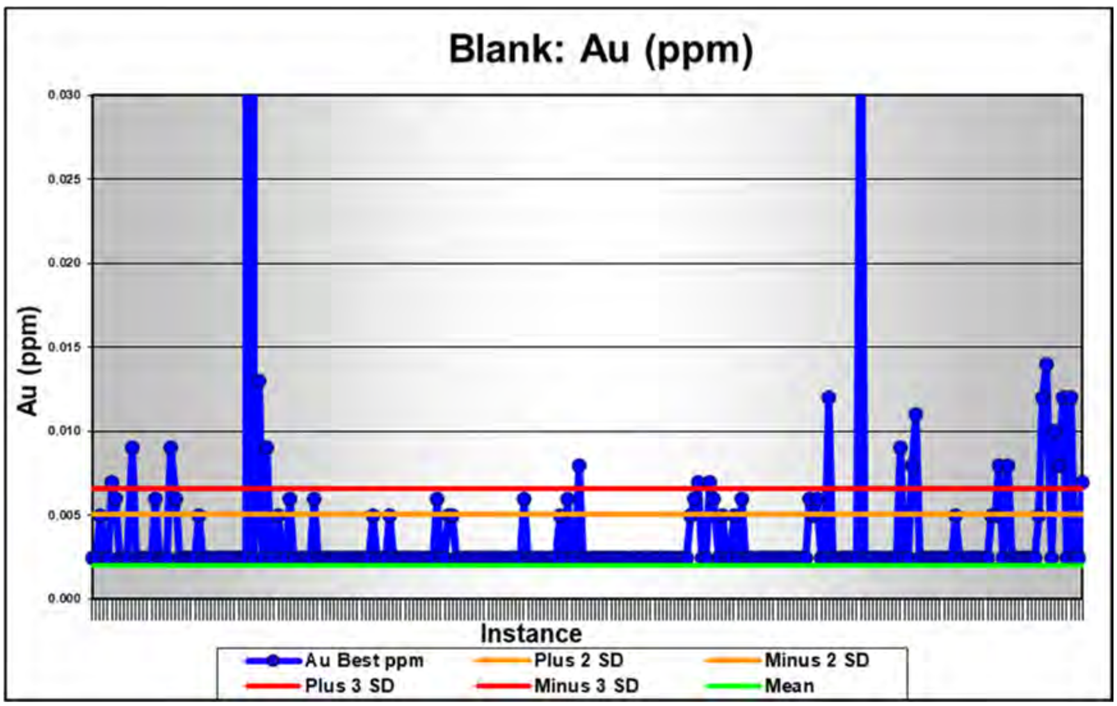


Figure 11-29: Gold assay results, blank material

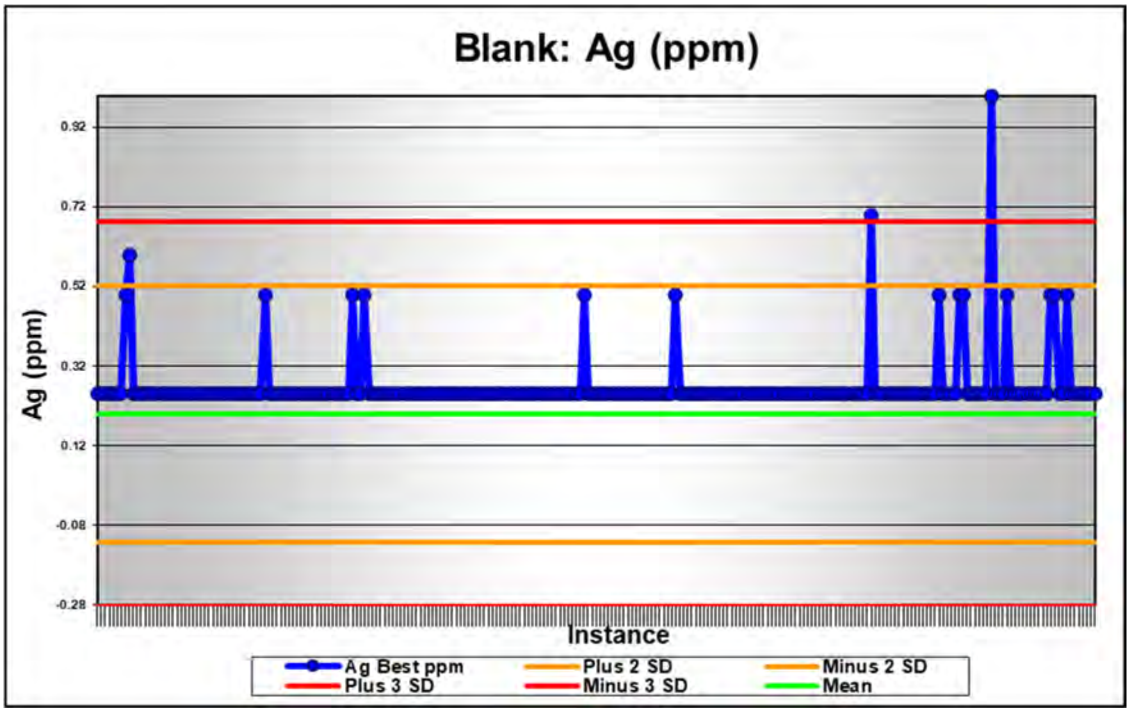


Figure 11-30: Silver assay results, blank material

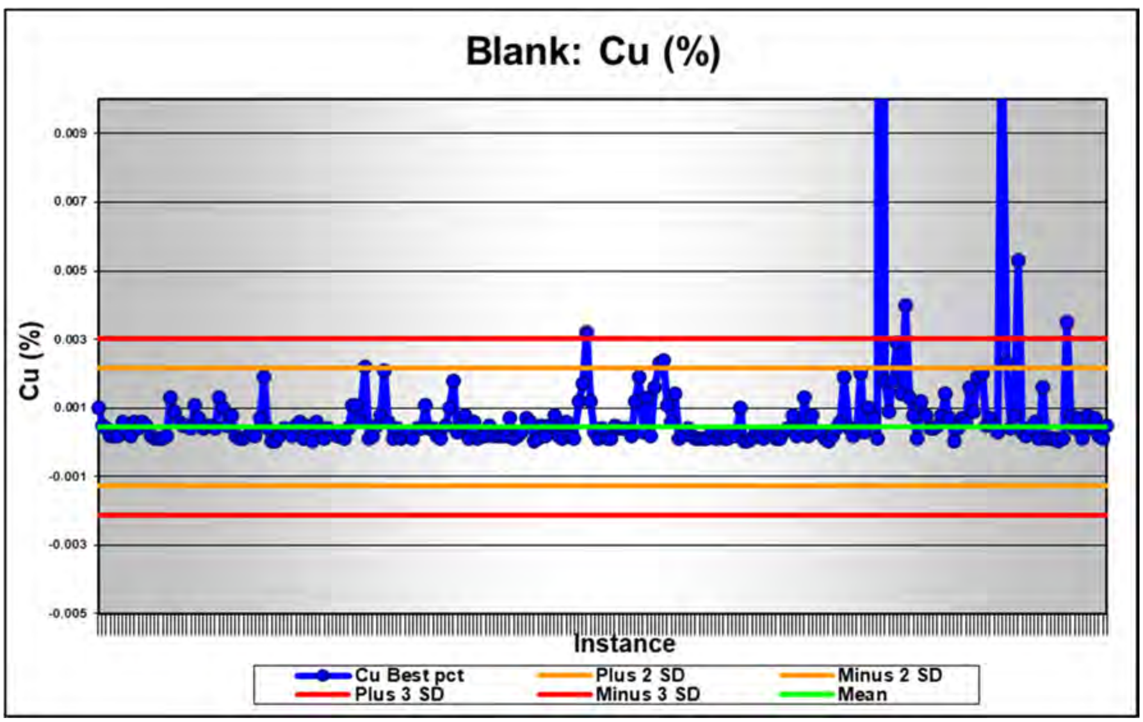


Figure 11-31: Copper assay results, blank material

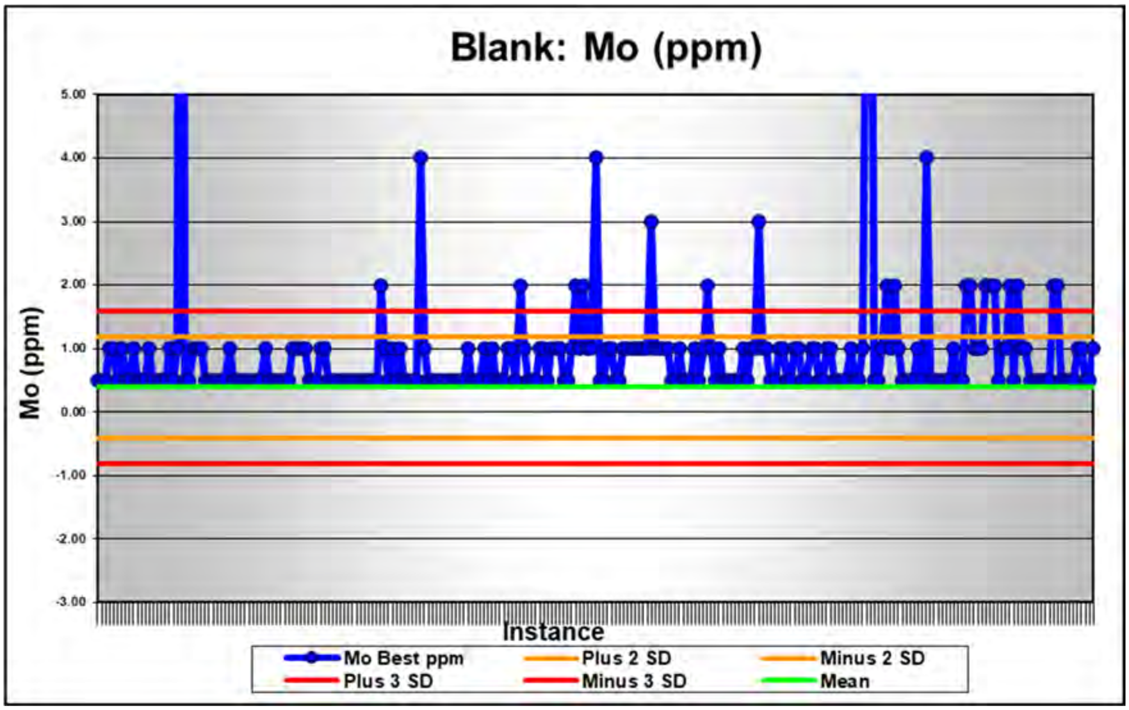


Figure 11-32: Molybdenum assay results, blank material

The blank material performed well for all elements of interest in 2020; all elements had higher than 85% passing rates within both two and three standard deviations of the mean expected values. On average, gold values plotted well above the expected mean, but, as Figure 19 shows, the detection limit for gold at ALS limits the lowest assay value to 0.0025 ppm, which is above the expected mean of 0.002 ppm for the blank material. The detection limits for silver and molybdenum are also higher than the expected means of the blank material for those elements. Even with the detection limit cut-offs, the passing rates are still acceptable. Figure 11-29 through Figure 11-32 do show a few off-chart, high value fails. Table 11-8 below summarizes the overall performance of the blank material in 2020.

Upon review of gold values, the sample with the greatest variation is sample B661435, at 2.35 ppm Au. This sample is only anomalous for gold as the other elements are all within the acceptable values for blank material. This sample was placed within a zone of anomalous gold values, but this zone is significantly less than 2 ppm Au. It's likely this is a lab error for gold. Another sample, B660416, returned 0.044 ppm Au, over twenty times the expected value; it was not placed in a zone of higher-grade material. The geochemical analysis for this sample shows fail values for all elements of interest; therefore, this is likely a data entry or sampling error. The remaining fail values for gold are all within 10 x the expected value of 0.002 ppm (e.g. B660460 is 0.012 ppm Au). They are all located either within zones of higher-grade material, or immediately following a "Standard" sample. The likely cause is laboratory equipment smear.

Of the three samples returning fail values for silver, one, as noted above for gold, is B660416, which likely represents a data entry error. The causes of the other two failures are somewhat less certain, as there are no indications of higher-grade material prior to these samples, and the other elements were within acceptable ranges.

The majority of the copper fail values were on the border of either two or three standard deviations, allowing for some leeway in accepting these as actual fail values. These samples were all within zones of anomalous Cu grades. The remainder of the fails that exceeded three standard deviations account for only 2% of all the blanks within the 2020 season. One of these samples is B660416, which has been deemed a data entry error, and the other four are within higher grade zones and can be attributed to minor smear effects.

Table 11-8: Performance of blank material during 2020 drill program

Element	# Failures within 2 Standard Deviations	% Passing within 2 Standard Deviations	# Failures within 3 Standard Deviations	% Passing within 3 Standard Deviations
Au	36	85.7	23	90.8
Ag	3	98.8	2	99.2
Cu	11	95.6	6	97.6
Mo	24	90.4	24	90.4

11.3.10 Field Duplicate Drill Core Analysis

11.3.10.1 2008 through 2010

Field duplicates are separate samples taken in the same manner and at the same core interval as the original sample. They are utilized to measure inherent variability in metal content from a single location and sample medium and give an idea of sample reproducibility in the field. Core duplicates were collected from the half-core that remained following the collection of the original sample. The duplicate was collected by sawing the half-core in half longitudinally, so that one quarter of the original core was collected. Duplicates were collected nominally for every 20th sample. Where duplicates were collected, only one quarter of the core remains stored in the core box on the property.

In 2008, 21 core duplicate pairs were collected; in 2009, 199 core duplicate pairs were collected; in 2010, 245 core duplicate pairs were collected. The original half-core samples were shipped to ALS Chemex and assayed for gold, copper and molybdenum, as well as multi-element ICP analysis as described above. The duplicate quarter-core samples were shipped to Acme Labs for gold, copper and molybdenum assay, as well as multi-element ICP analysis in a manner identical to that performed at ALS Chemex, as described above. The results for the duplicate analyses for gold, silver, copper and molybdenum are demonstrated in comparison plots between the Acme and ALS Chemex values below:

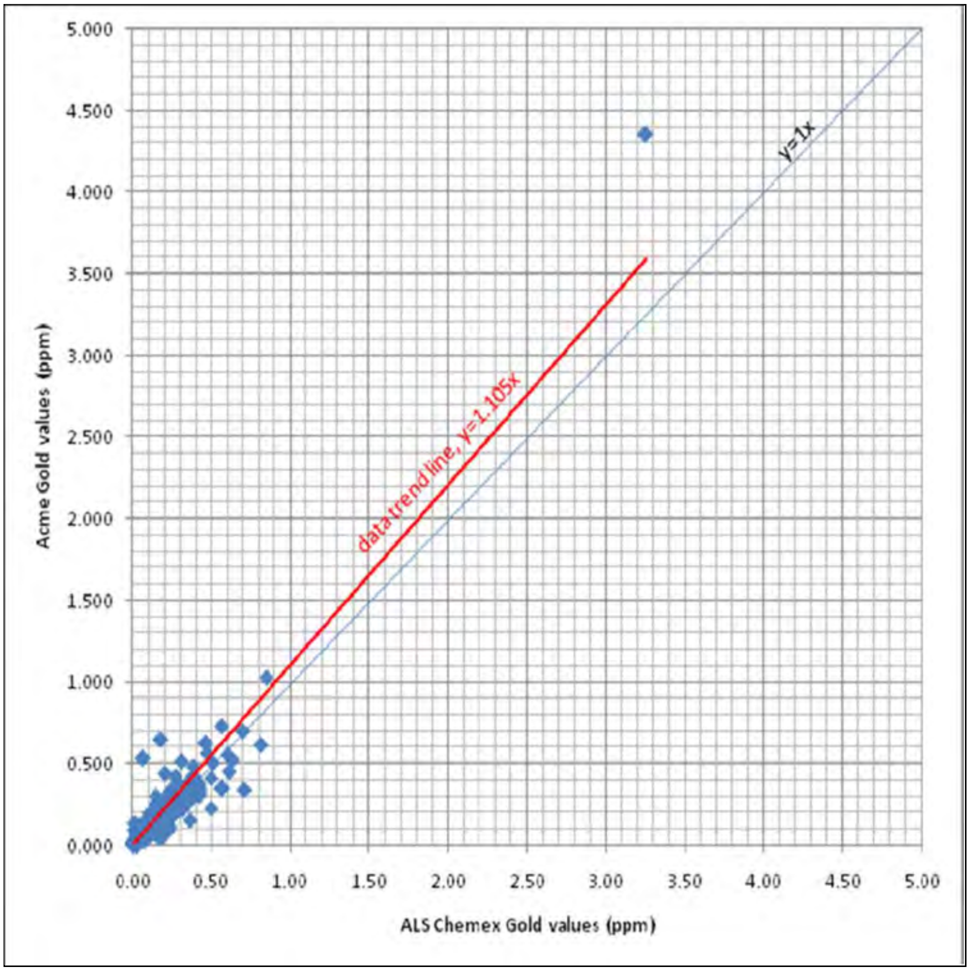


Figure 11-33: Plot of ALS Chemex Gold Assay Versus Acme Labs Gold Assay for Field Duplicate Samples (2008, 2009 and 2010 Data)

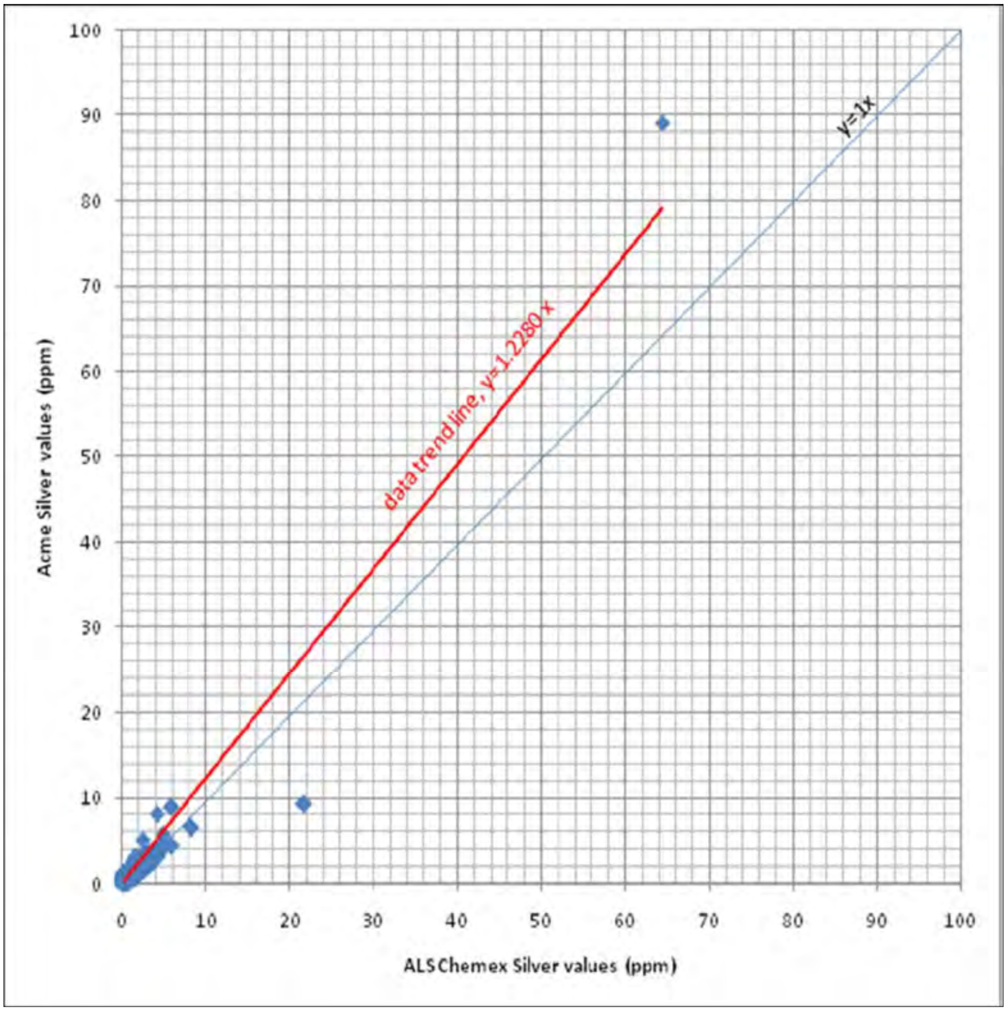


Figure 11-34: Plot of ALS Chemex Silver Analyses Versus Acme Labs Silver Analyses for Field Duplicate Samples (2008, 2009 and 2010 Data)

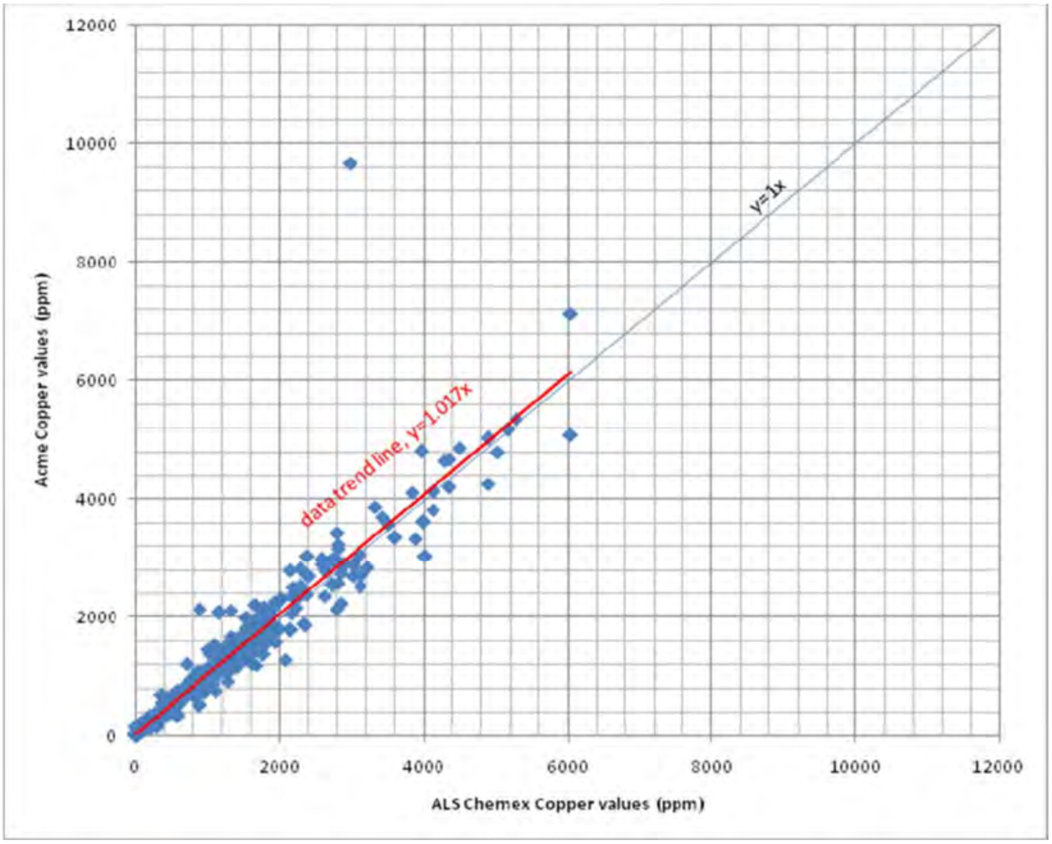


Figure 11-35: Plot of ALS Chemex Copper Assay Versus Acme Labs Copper Assay for Field Duplicate Samples (2008, 2009 and 2010 Data).

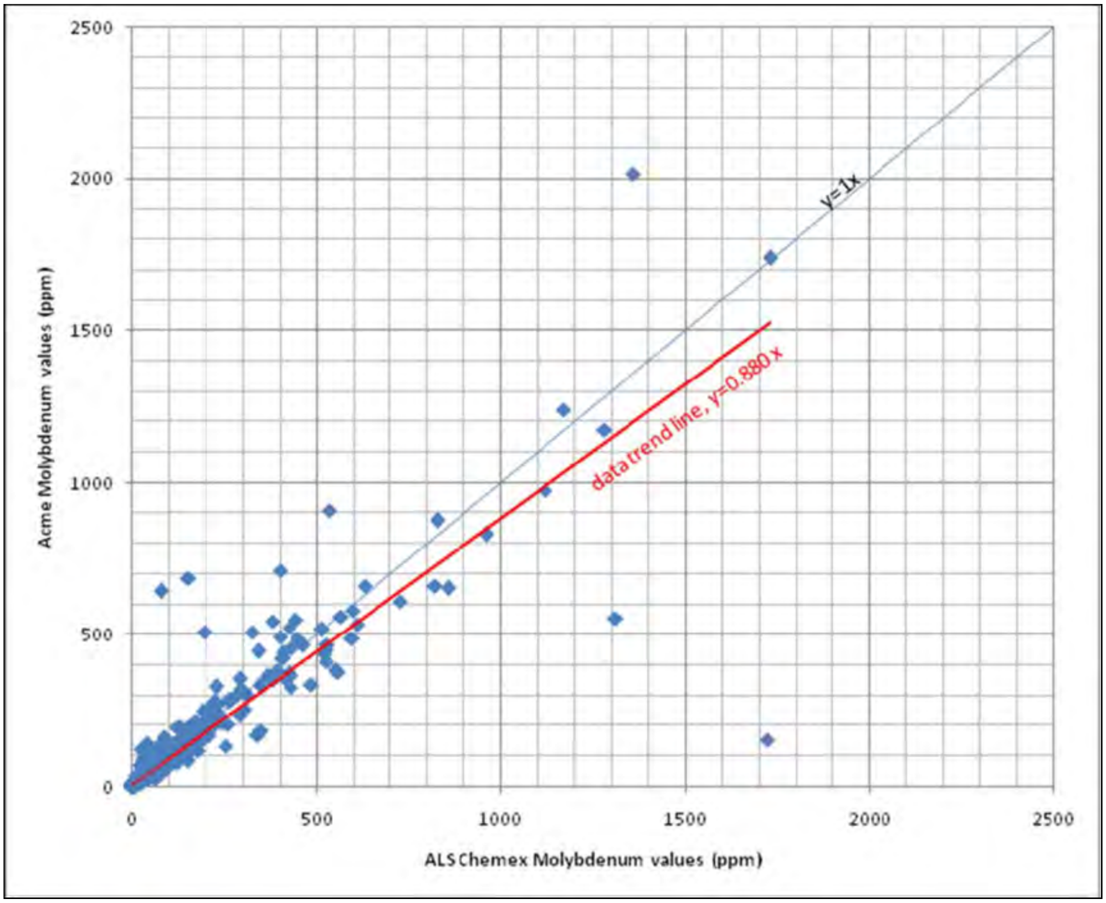


Figure 11-36: Plot of ALS Chemex Molybdenum Assay Versus Acme Labs Molybdenum Assay for Field Duplicate Samples (2008, 2009 and 2010 Data).

The plots generally show good correlation between ALS Chemex and Acme Labs for all four elements of interest.

Often the “nugget effect” associated with gold and silver content will produce widely divergent values, which would plot as highly scattered data points. However, the gold and silver results from the duplicate samples show good correlations.

Ideally, a trend line of y=1x would show perfect reproducibility. This is rarely, if ever, the case due to the difference of mineral content between duplicate samples. The data trend line for gold returned y=1.105x. This demonstrates that Acme Lab results, as a whole, are 10.5% higher than ALS Chemex results. All samples cluster in close proximity to the trend line which indicates no strong “nugget effect” and good reproducibility.

The data trend line for silver is y=1.228x. This demonstrates that Acme Lab analytical results, as a whole, are 22.8% higher than ALS Chemex values. In general, the points cluster well around the trend line with the exception of one sample. This also demonstrates good reproducibility.

The results for duplicate analyses for copper demonstrate excellent reproducibility. The data trend line returned y=1.017x. The copper data clusters tightly around trend line with the exception of one value. In general, the Acme results are very slightly higher (1.7%) than the ALS Chemex results.

The molybdenum plot demonstrates slightly more scattered results with 8 points plotting far off the trend line (y=0.880x). The trend line indicates that, in general, the Acme results for molybdenum are 12% lower than ALS Chemex results. Overall, the duplicate results show good correlation. Molybdenite mineralization was observed in quartz veins in the drill core and it is possible that the 8 erratic values are reflecting a molybdenum "nugget effect", where there is a variability of molybdenite concentration between samples.

The results of analyses from the sample standards, blanks and duplicates provide for acceptable Quality Assurance and Quality Control (QA/QC) for the geochemical programs at Casino from 2008 through 2010. The results also indicate that there is no evidence of tampering during the sample collection process, shipping or at the laboratory. There is also no evidence of systemic errors in the sample preparation and analytical processes.

11.3.10.2 2019

In 2019, insertion of both field duplicates and pulp check duplicates were part of the overall sampling protocol at Casino.

Field Duplicates

Similar to standards and blanks, 1 field duplicate was inserted randomly within every 20 samples. The duplicate would be quarter-cored by the core cutter and placed in a separate bag from the original sample with its own sample tag. This duplicate quarter-core sample would be set aside in a bin to be sent to ALS Global for analysis in a separate batch at a later date than its corresponding original sample. The purpose of this kind of duplicate is to test the reproducibility of the lab's analytical methods.

Figure 11-37 through Figure 11-40 show the comparison between the original core sample results and the duplicate core sample results for Gold, Silver, Copper and Molybdenum.

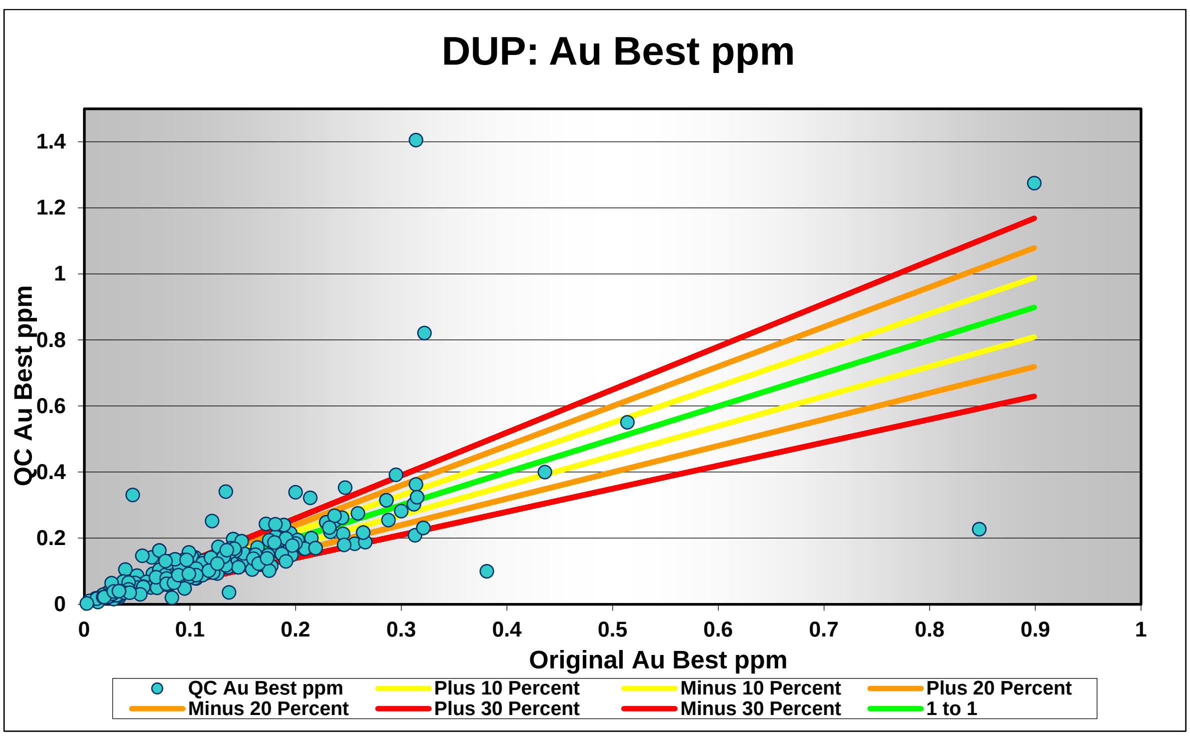


Figure 11-37: Comparison Plot Between Original Gold Values and Duplicate Gold Values

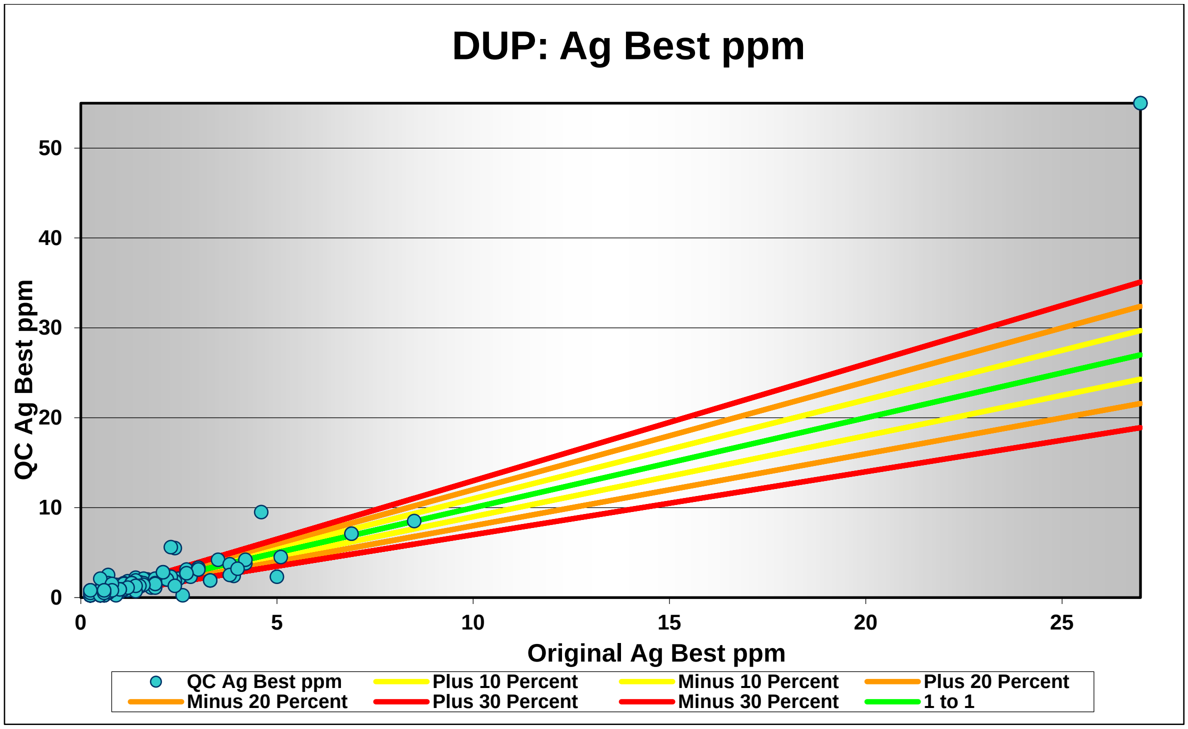


Figure 11-38: Comparison Plot Between Original Silver Values and Duplicate Silver Values

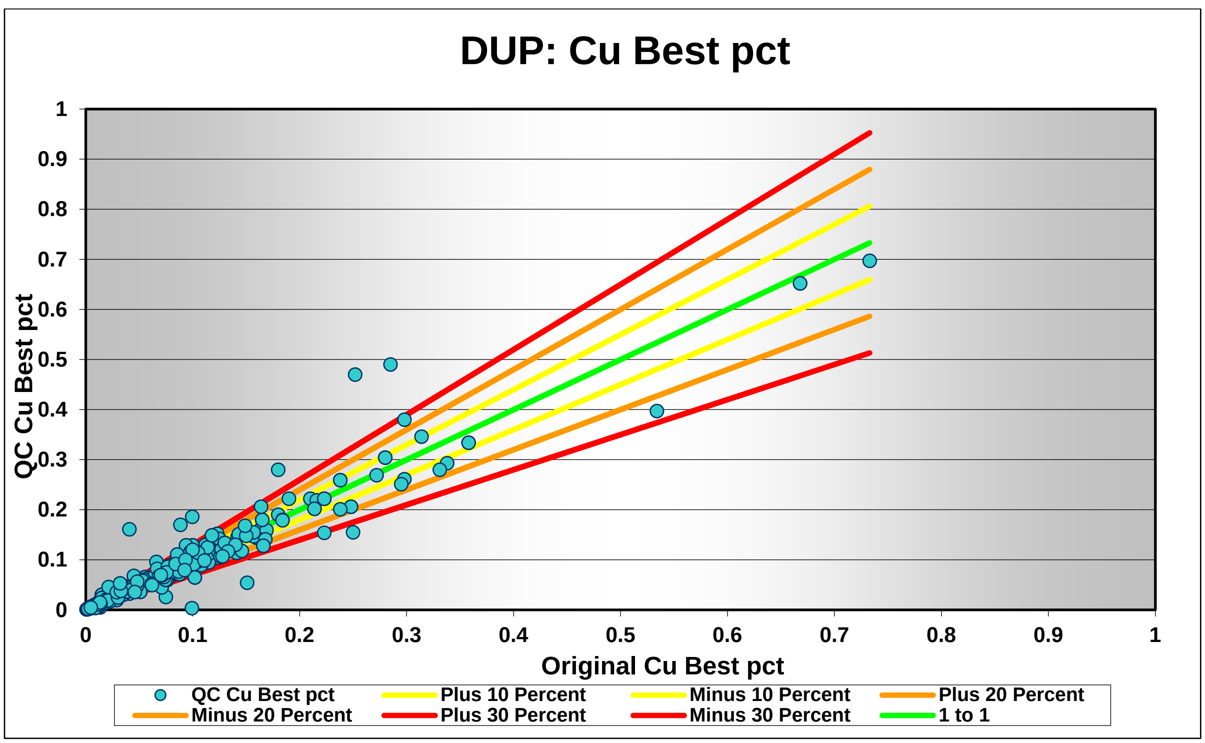


Figure 11-39: Comparison Plot Between Original Copper Values and Duplicate Copper Values

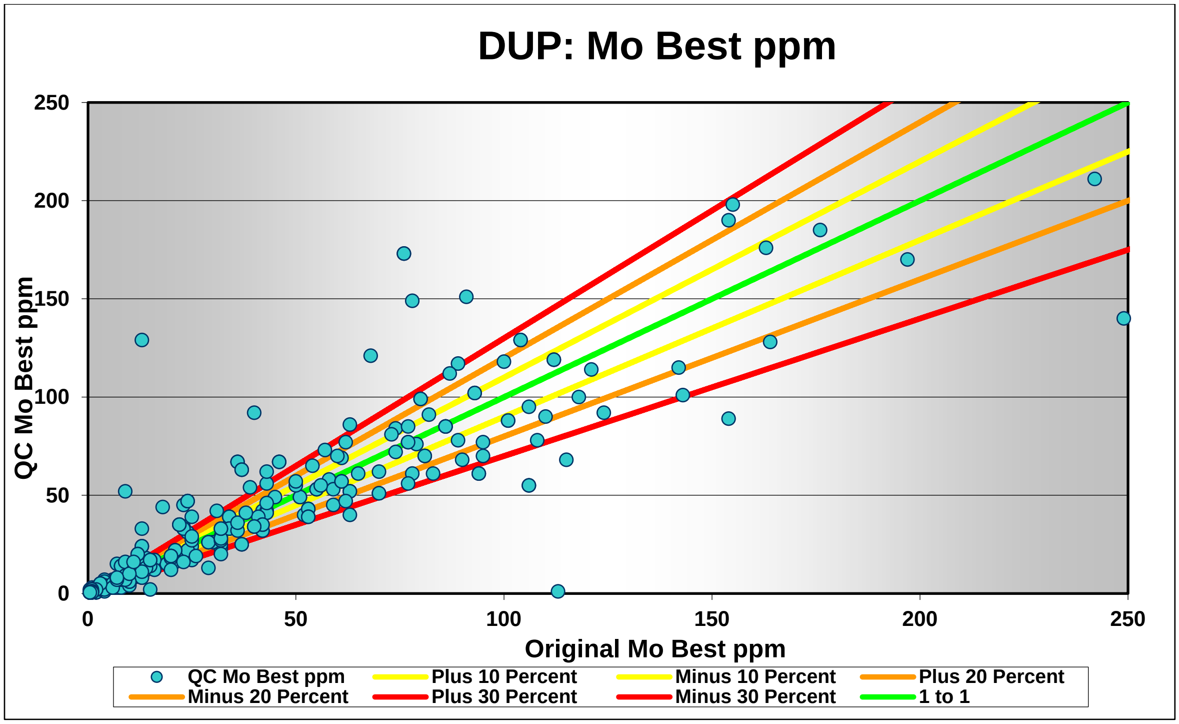


Figure 11-40: Comparison Plot Between Original Copper Values and Duplicate Copper Values

Field duplicates for 2019 performed well, but not without issue. The problem with field duplicates in this type of deposit is the difficulty to accurately cut a piece of core into two identical quarters. While it is a good method to test the reproducibility of a lab, it might be better served to have the primary lab split the pulp after the preparation process and set aside one split to process at a later date. An alternate, and perhaps better method, is to have the project manager send a list of pulps to the primary lab to re-assess as duplicates. In this way a pulp duplicate would more effectively test the reproducibility of results.

Table 11-9: Summary of Duplicate (Core) Pair Performance During 2019 Drill Program Sampling

Element	Duplicate Pairs Within 10% Difference	% total pairs within 10%	Duplicate Pairs Within 20% Difference	% total pairs within 20%	Duplicate Pairs Within 30% Difference	% total Duplicates within 30%
Au	120	34.9	211	61.3	269	78.2
Ag	141	41	214	62.2	240	69.8
Cu	181	52.6	276	80.2	308	89.5
Mo	109	31.7	166	48.3	215	62.5

<u>Check Duplicates</u>

Check samples were selected at random from the entire sample population once the primary lab, ALS Global, had reported all the final assay results for the 2019 Casino Project. A list of 973 sample numbers (using a random selection in Excel) was sent to ALS Global in Whitehorse from the project manager/senior geologist, requesting ALS to pull the pulps for the samples listed and send them directly to SGS Canada Inc. in Burnaby, BC for processing. This represents a little over 20% of the entire 2019 sample population. Once received by SGS, these pulps were logged into their system, re-homogenized non-mechanically, then dry-screened randomly (1/100 samples were checked) to various mesh sizes to verify fineness. No major issues were found regarding fineness, and SGS proceeded with the full assay protocol.

The purpose of this kind of duplicate/check is to test the methodology of the primary lab to ensure there is no bias or systemic errors, and that other labs using similar methods can reproduce their results within a predetermined degree of variance.

Figure 11-41 through Figure 11-44 show the comparison between the original core sample results from ALS Global in Whitehorse and the check pulp sample results from SGS in Burnaby for gold, silver, copper and molybdenum.

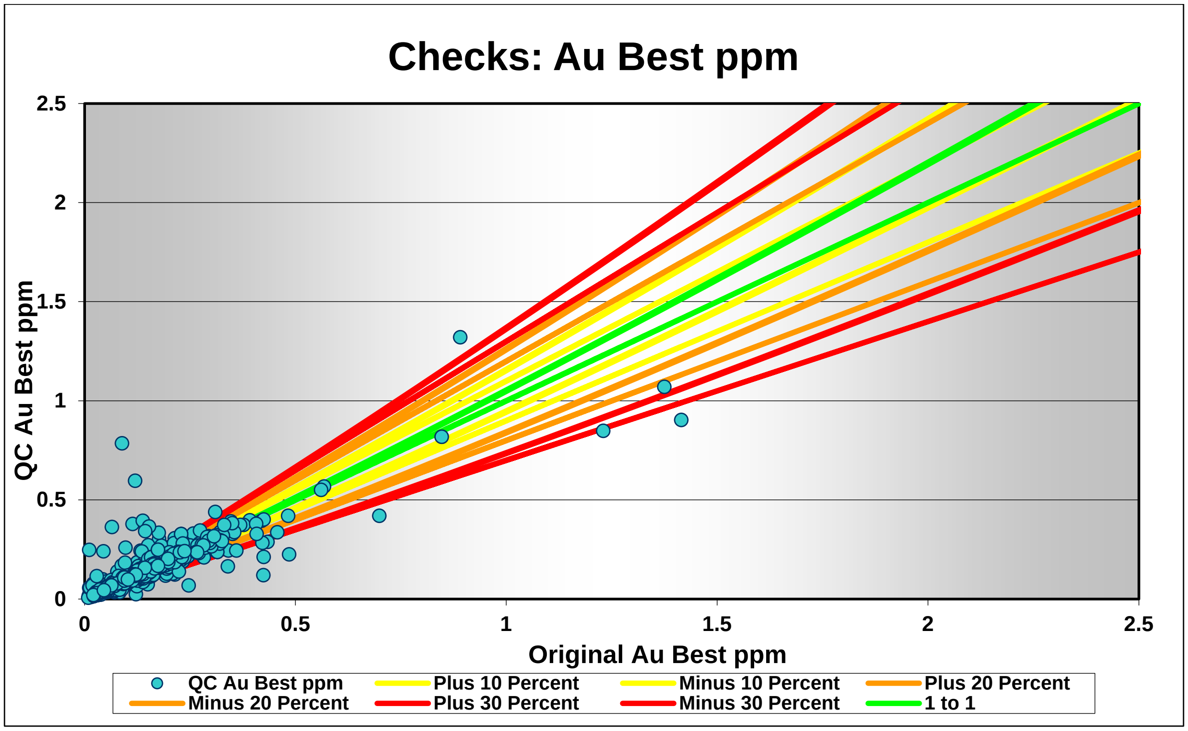


Figure 11-41: Comparison Plot Between Gold Values from ALS Chemex and Gold Values from SGS

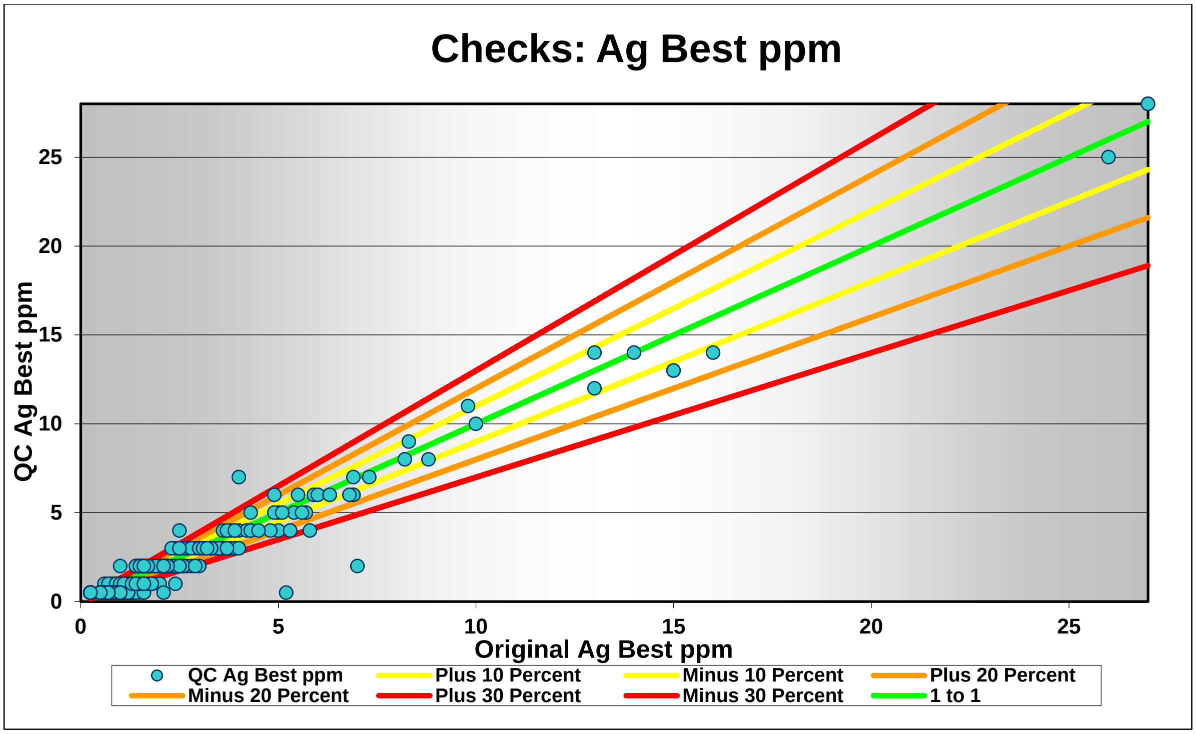


Figure 11-42: Comparison Plot Between Silver Values from ALS Global and Silver Values from SGS

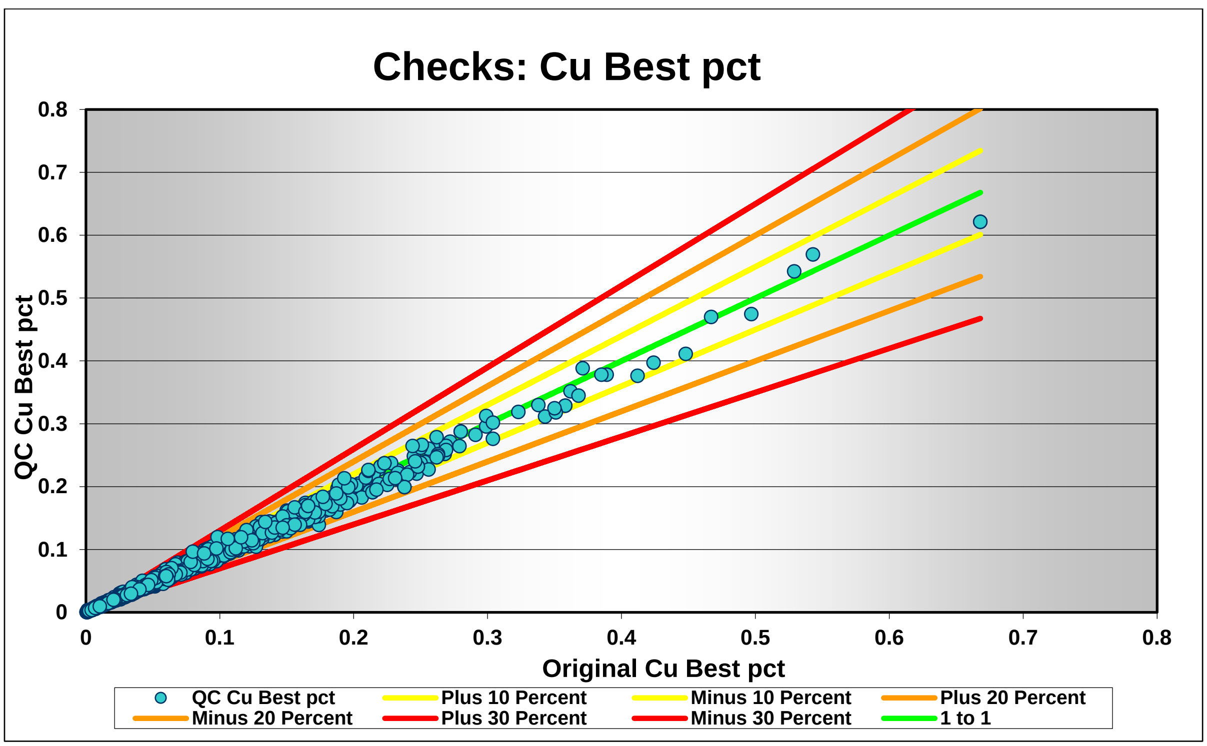


Figure 11-43: Comparison Plot Between Copper Values from ALS Global and Copper Values from SGS

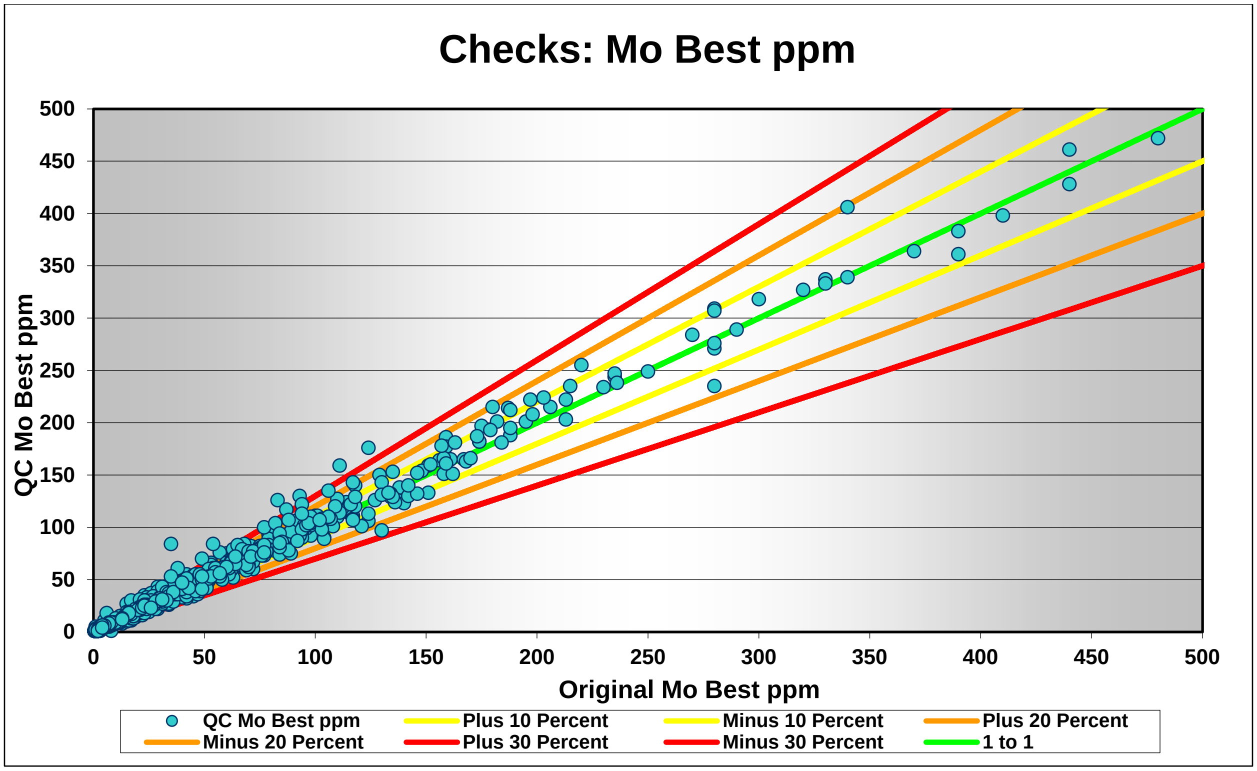


Figure 11-44: Comparison Plot Between Molybdenum Values from ALS Global and Molybdenum Values from SGS

Check samples for 2019 performed well, better than most duplicates overall, but still not without issue. In general, gold and silver showed lower correlation between pairs of pulps analysed for all elements of interest than did copper and molybdenum. Silver showed the worst correlation with less than 20% of samples having under a 10% variation. At more than 20%, the number of 2019 check samples sent to the secondary lab represented a much larger population of samples than is industry practice, which is about 5%. The high percentage delivered in 2019 was partially due to utilization of a new laboratory for check samples, and to proven past success.

Table 11-10: Summary of Check (Pulps) Pair Performance During 2019 Drill Program Sampling

Element	Check Pairs Within 10% Difference	% total pairs within 10%	Check Pairs Within 20% Difference	% total pairs within 20%	Check Pairs Within 30% Difference	% total Duplicates within 30%
Au	465	47.8	711	73.1	817	84
Ag	187	19.2	351	36	469	48.2
Cu	692	71.1	953	97.9	961	98.8
Mo	596	61.3	757	77.8	812	83.5

11.3.10.3 2020

In 2020, field duplicates and pulp check duplicates both comprised part of the overall sampling protocol at Casino.

Field Duplicates

Similar to standards and blanks, 1 field duplicate was inserted randomly within every 20 samples, equating to 252 duplicate samples in 2020. The duplicate sample was quarter-cored by the core cutter and placed in a separate bag, along with its own sample tag, from the original sample. This duplicate quarter-core sample was set aside in a bin to be sent to ALS for analysis in a separate batch that was sent at a later date than its corresponding original sample. The purpose of this is to test for the uniformity of metal content in the core, and for the reproducibility of the lab's analytical methods.

Figure 11-45 through Figure 11-48 show the comparison between the original core sample results and the duplicate core sample results for gold, silver, copper and molybdenum.

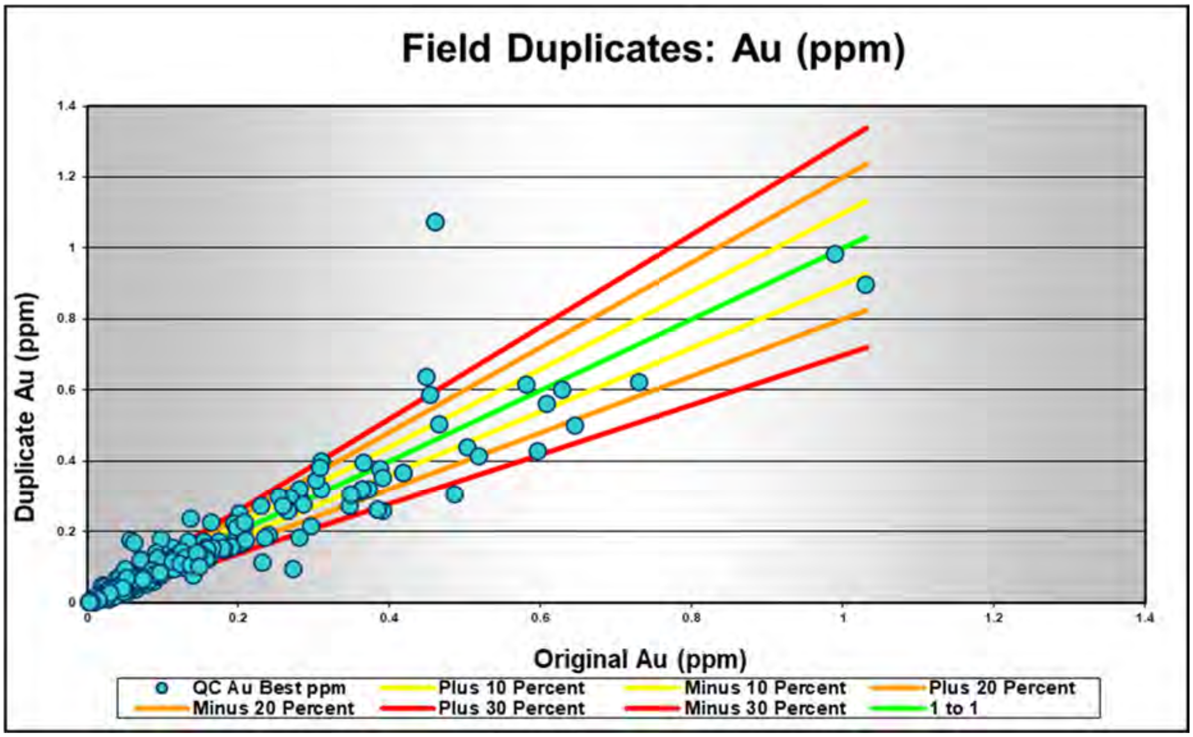


Figure 11-45: Comparison between original and duplicate gold values

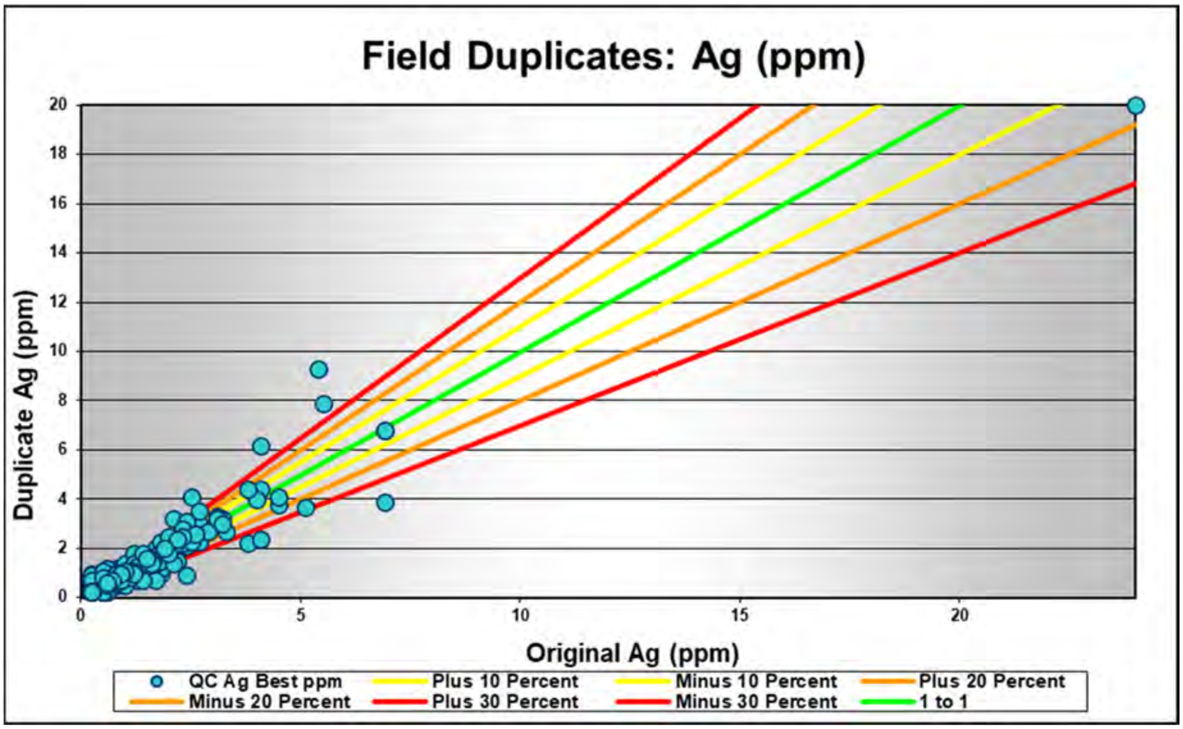


Figure 11-46: Comparison between original and duplicate silver values

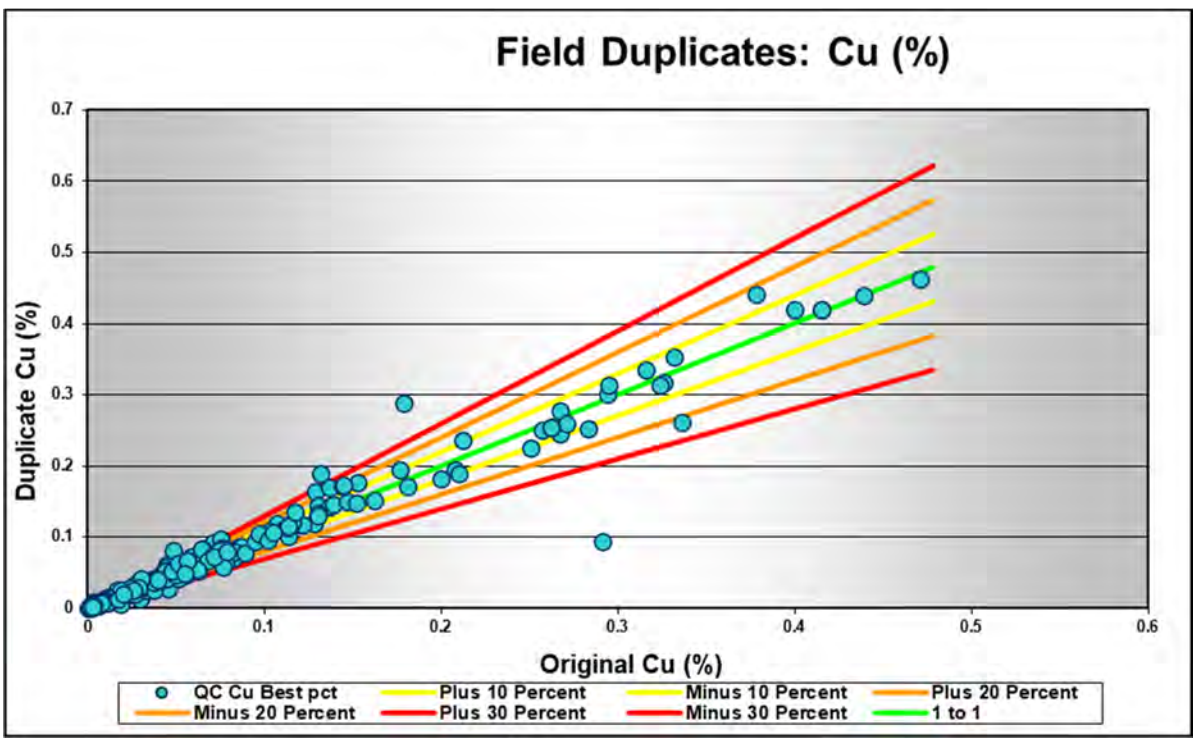


Figure 11-47: Comparison between original and duplicate copper values

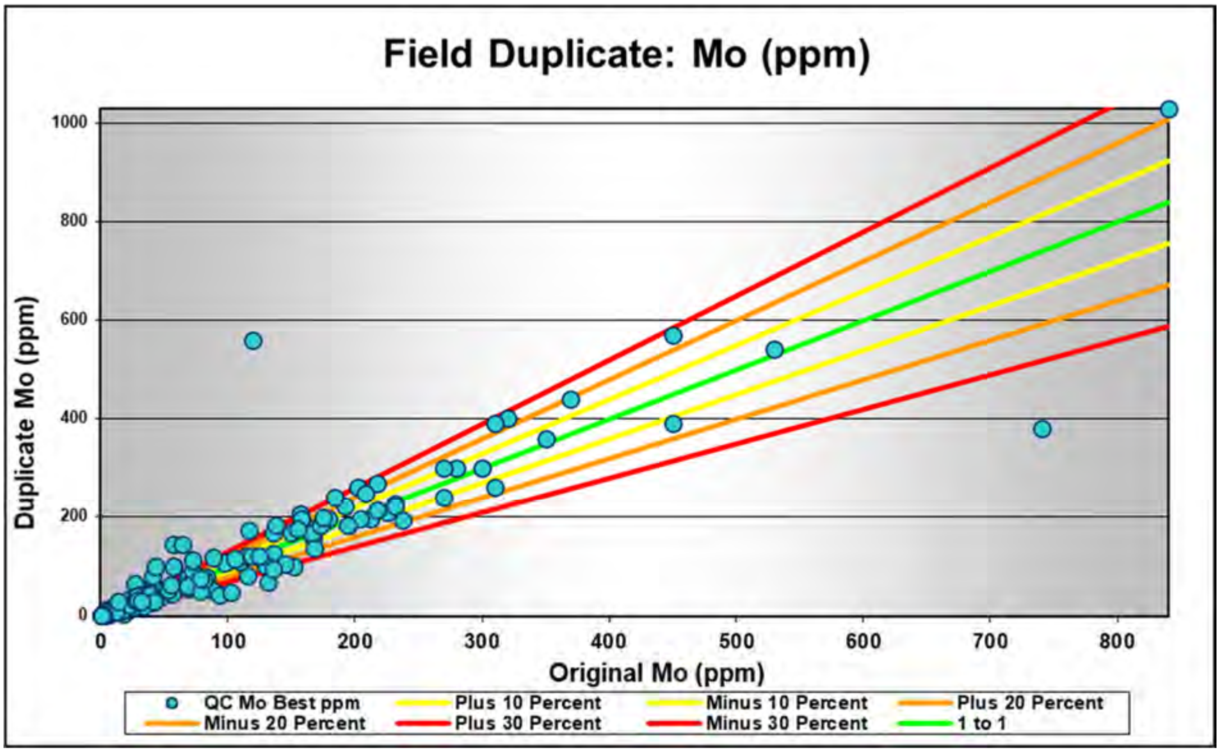


Figure 11-48: Comparison between original and duplicate molybdenum values

Field duplicates for 2020 performed well, but not excellently. The issue with field duplicates in this type of deposit is the difficulty to accurately cut the core into identical "quarters". While it is a good method to test the reproducibility of a lab, for this type of deposit, it may be preferable to have the primary lab split the pulp after the preparation process and set it aside to process at a later date. An even more preferable method would be for the project manager to send a list of pulps for the primary lab to re-assess as duplicates. In this way a pulp duplicate would test the labs reproducibility more effectively. Table 11-11 summarizes the duplicate performance for each element of interest in 2020.

Table 11-11: Summary of duplicate (core) pair performance during 2020 drill program

Element	Duplicate Pairs Within 10% Difference	% total pairs within 10%	Duplicate Pairs Within 20% Difference	% total pairs within 20%	Duplicate Pairs Within 30% Difference	% total Duplicates within 30%
Au	88	34.9	164	65.1	191	75.8
Ag	121	48	163	64.7	240	69.8
Cu	181	52.6	276	80.2	308	89.5
Mo	109	31.7	166	48.3	215	62.5

Some degree of "coarse gold (nugget) effect" is likely here. Both Au and Ag commonly occur as fine nuggets, as is evident by visible gold occurring in DDH 19-21. This may account for the lower reproducibility rates for these elements. Chalcocite veins are fairly common, which are prone to uneven distribution within core duplicates. The low rates for Mo may also be due to the sub-centimetre vein-associated nature of molybdenite mineralization, particularly prone to uneven distribution within core duplicates (Schulze, 2021).

Check Pulp Duplicates

Check samples were selected at random from the entire sample population once the primary lab, ALS, had reported all the final assay results for the 2020 Casino Project. A list of 200 sample numbers (using a random selection process in Microsoft Excel) was sent to ALS requesting them to pull the pulps for the samples listed and send them directly to Bureau Veritas Minerals (BV) in Vancouver, BC for analysis. This represents 5% of the entire 2020 sample population. Once received by BV, these pulps were logged into their system, re-homogenized non-mechanically, then dry-screened and sent through various mesh sizes on a random basis to verify fineness. No major issues were found regarding fineness and BV proceeded with the full assay protocol.

The purposes of these kind of check samples are: to test the methodologies of the primary lab to ensure there are no biases or systemic errors, and to verify that other labs using similar methods can reproduce the results within a certain degree of variance.

Figure 11-49 through Figure 11-56 show the comparison between the original core sample results from ALS and the check pulp sample results from BV for gold, silver, copper and molybdenum.

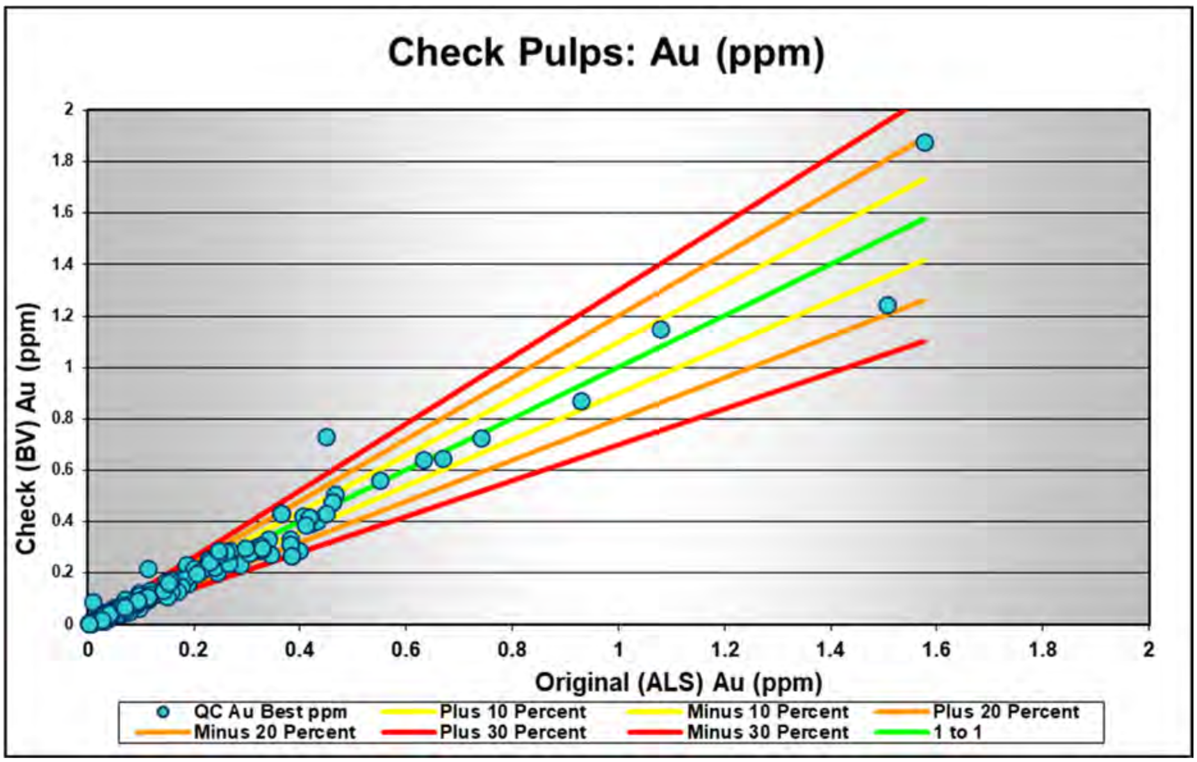


Figure 11-49: Comparison of gold values from ALS and BV

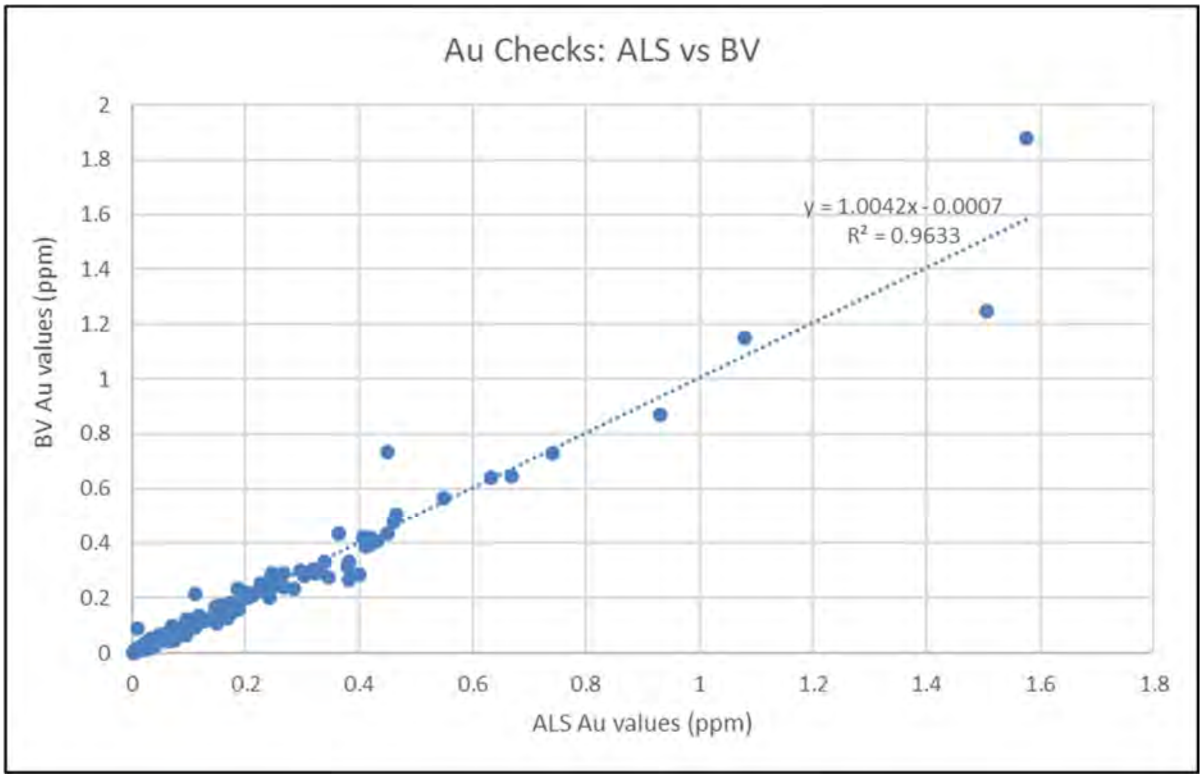


Figure 11-50: Correlation between gold values from ALS and BV

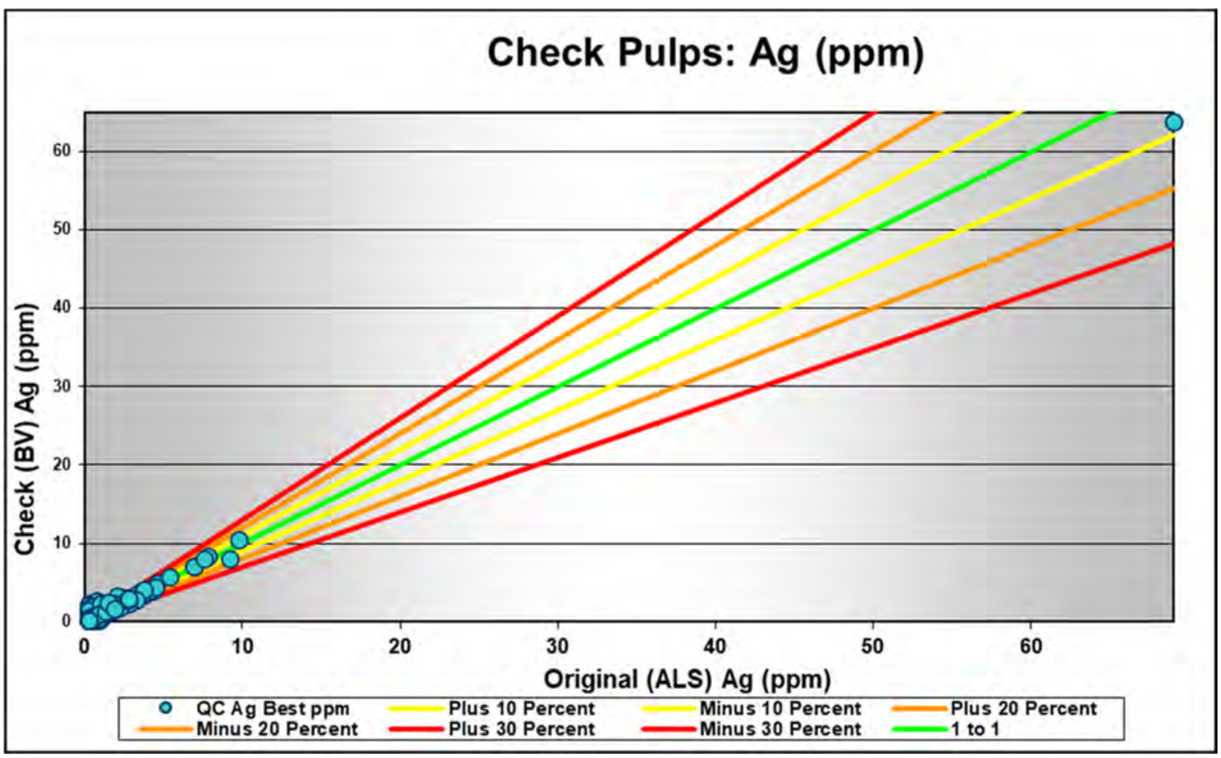


Figure 11-51: Comparison between silver values from ALS and BV

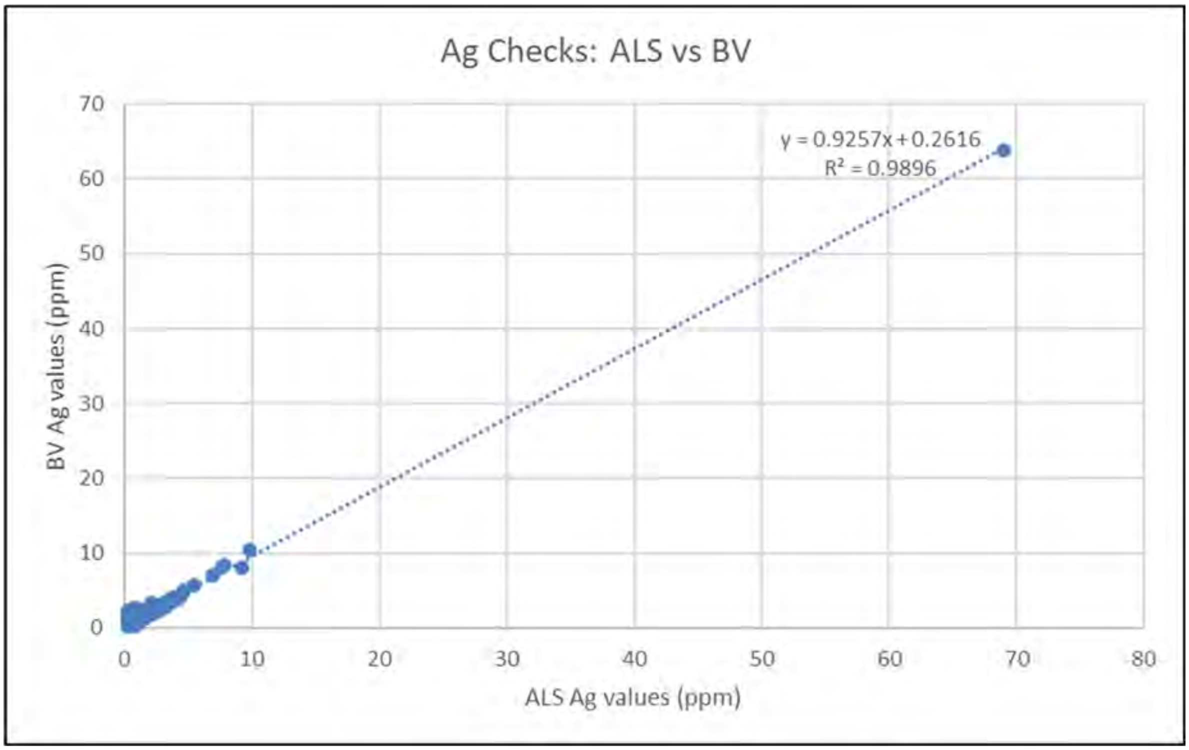


Figure 11-52: Correlation between silver values from ALS and BV

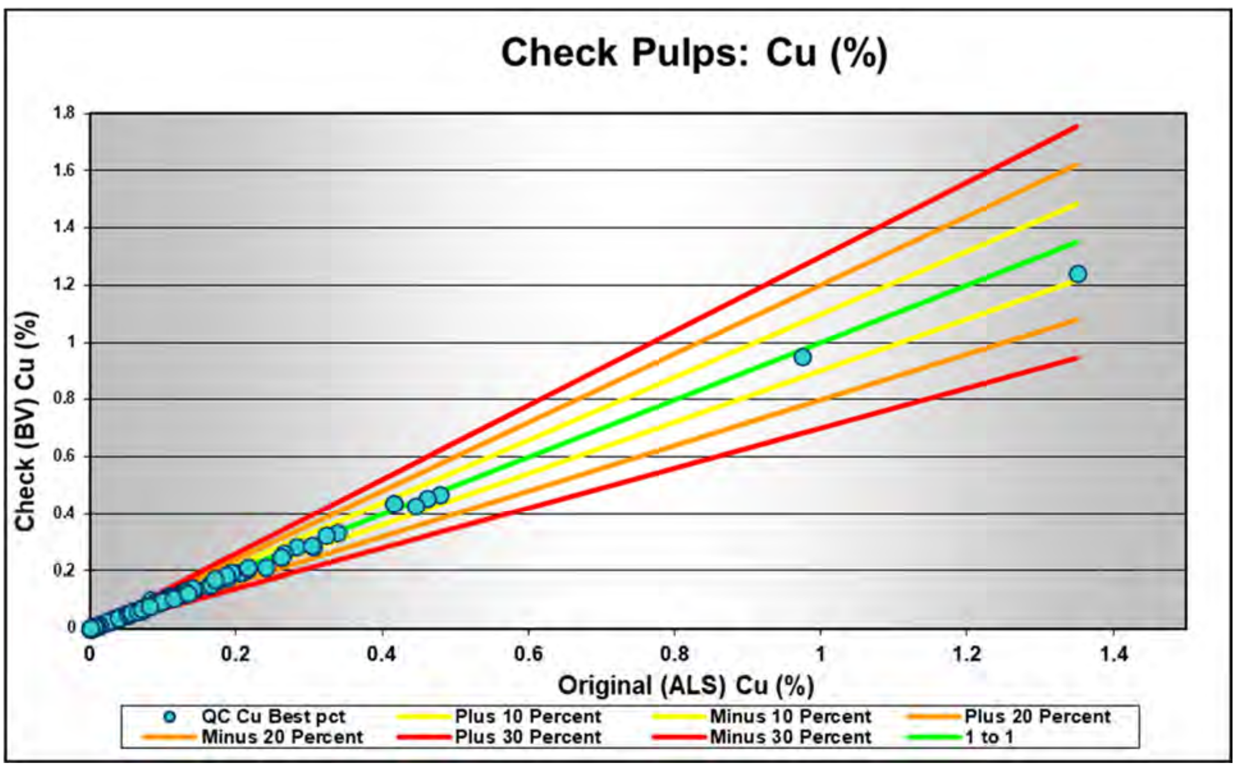


Figure 11-53: Comparison between copper values from ALS and BV

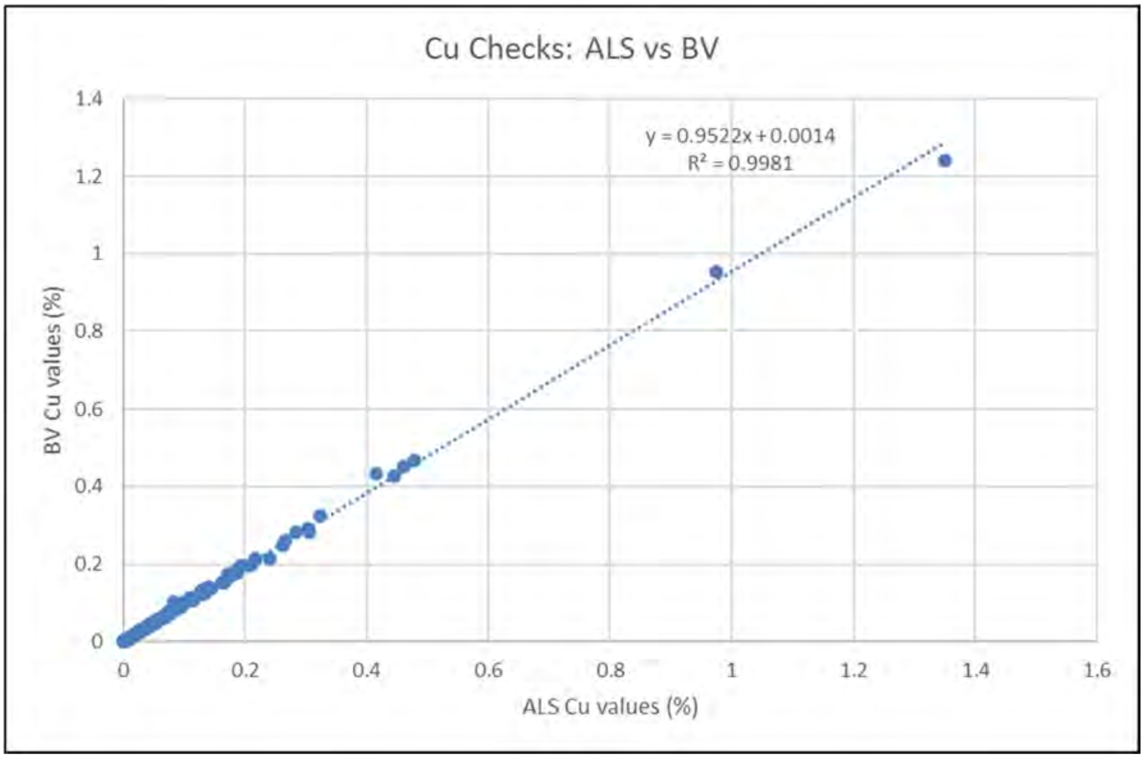


Figure 11-54: Correlation between copper values from ALS and BV

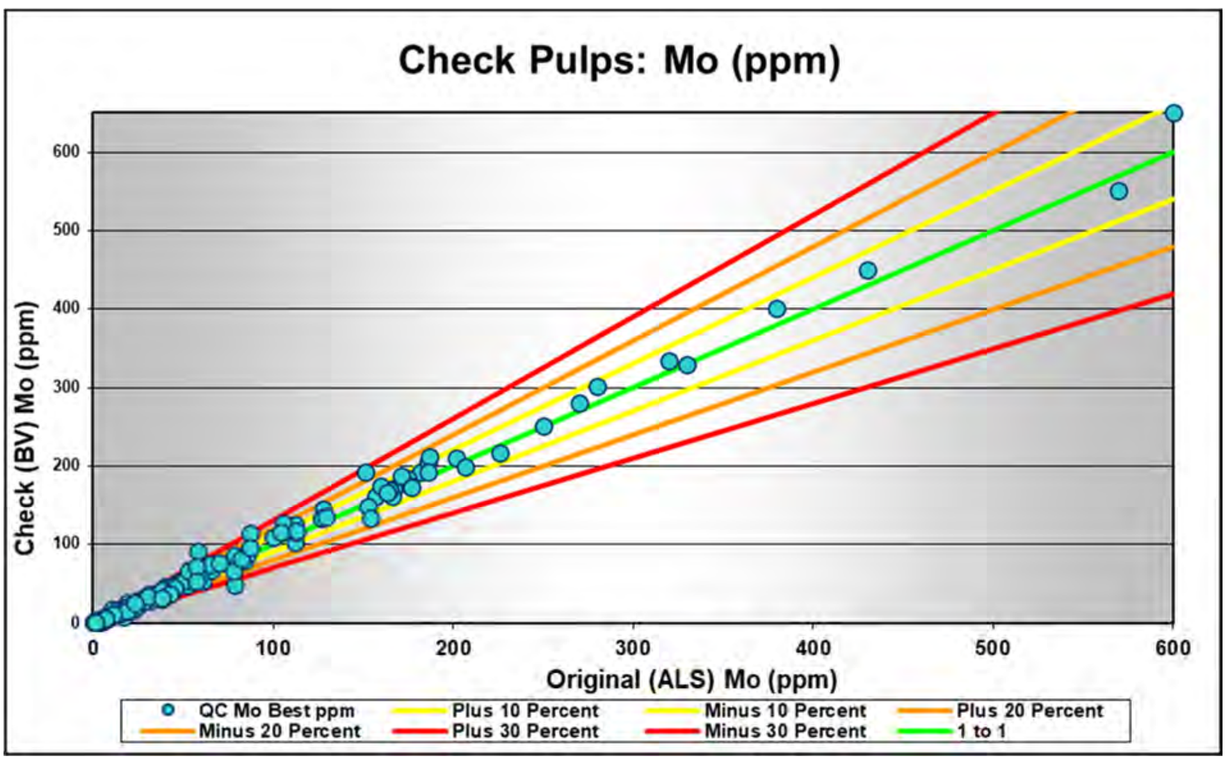


Figure 11-55: Comparison of molybdenum values from ALS and BV

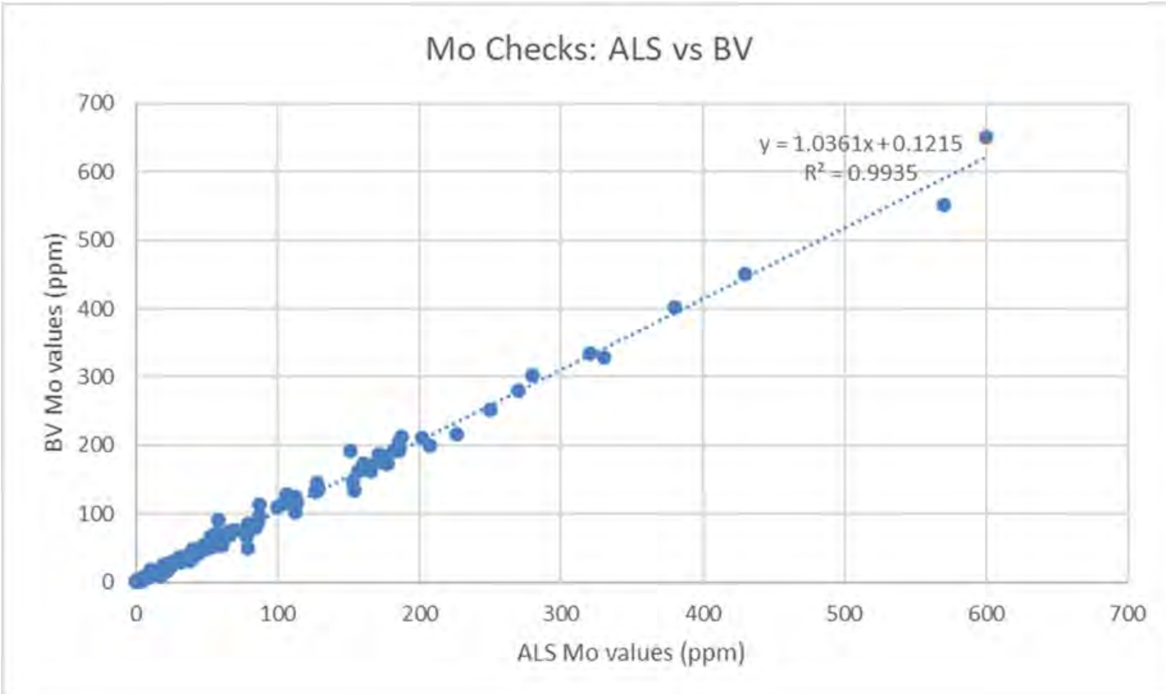


Figure 11-56: Correlation between molybdenum values from ALS and BV

Check Pulp Duplicates performed well in 2020 and better than Field Duplicates overall. Table 11-12 summarizes the percentages of check pairs that fall within 10, 20 and 30% variation bands. Figure 11-50, Figure 11-52, Figure 11-54 and Figure 11-56 all show a good correlation between the pairs of check samples. Ideally, a trend line on these figures of y=1x would show perfect correlation and reproducibility, but this is rarely the case due to the differences in mineral content between duplicate samples.

Gold showed good correlation with a data trend line of y=1.0042x, demonstrating that BV results for gold as a whole are only 0.4% higher than ALS results. The data trend line for silver is y=0.9257x, demonstrating that BV results for silver are 7.4% lower than ALS results. For copper the data trend line is y=0.9522x, which indicates that BV results for copper are about 4.8% lower than ALS. Finally, for molybdenum, the data trend line is y=1.0361x, showing that BV results are 3.6% higher than ALS.

A "nugget effect" associated with gold and silver content can often produce widely divergent values, plotting as highly scattered data points. Aside from a small handful of gold pairs, gold and silver results show good correlation, indicating well-mixed pulp material by the labs during the preparation stage, and no strong "nugget effect" within the samples.

Table 11-12: Summary of check (pulps) pair performance during 2020 drill program

Element	Check Pairs Within 10% Difference	% total pairs within 10%	Check Pairs Within 20% Difference	% total pairs within 20%	Check Pairs Within 30% Difference	% total Duplicates within 30%
Au	95	48.0	156	78.0	172	86.0
Ag	103	51.5	134	67.0	144	72.0
Cu	168	84	191	95.5	196	98.0
Mo	116	58.0	161	80.5	168	84

12 DATA VERIFICATION

This section contains a summary and review of the data entry and data verification related to the Casino Project. Several phases of exploration have taken place related to the project and the data entry and verification processes will be discussed for each phase. The phases of exploration included diamond drilling programs dated 1992 through 1994 performed by PSG Exploration, followed by exploration by Western dated 2008 through 2010, Under Western, a transition to a new data system was implemented in 2013 and has continued since. The most recent exploration phase relevant to this report is dated 2020, although the resource estimate excludes 2020 data.

Data entry and verification for the 1992 through 1994 programs was reviewed. Further, the author has reviewed a selection of original scanned drill logs and analytical certificates as compared with data in the current Casino Project database.

Data collection, entry and verification for the drilling programs conducted by Western from 2008 through 2010 has been reviewed. In addition, the author has reviewed original reports about the programs, and compared a selection of original drill logs and analytical results to the data within the current database.

A transition to an updated database system, GeoSpark Core, was implemented in 2013 by Western to streamline the data flow and provide automated data validation and checking. All data ranging from 1960's RC drill data to 2013 drill core logging details were merged and imported to the new database system. The author has performed a validation of the current database against a selection of original scanned drill logs and analytical certificates, allowing for confidence in the data entry and data validation.

In 2019 the database system, GeoSpark Core, was used to combine core logging and assay data during the field season. Following the field season, a full audit/verification was done of all 2019 Collar, Survey and Assay, Alteration, Metallurgical and Lithology data. The 2019 data validation effort has been reviewed and, for this report, the author has also performed a review on the data entry and data validation related to the 2019 drilling program through comparison of original reports, original scans of drill logs, and analytical results to the data in the database. The same validation effort was conducted for data from the 2020 program.

Ultimately this section of this report includes a comprehensive review of the data within the Casino Project database, thus confirming that the data has been generated using proper procedures, has been correctly entered digitally from the source files, and is suitable for use. Data validated to the database includes original scanned drill logs, and analytical certificates signed by an authorized individual.

12.1 DATA ENTRY

12.1.1 1992-1994

Original 1992 and 1993 field data were entered by Archer, Cathro and Assoc. and by Nowak and Assoc., both of Vancouver, B.C. Data was entered to a database on site and in the Vancouver office, by PSG personnel.

Assay, ICP, copper leach data, check assays and specific gravities were downloaded from the Chemex Labs computer-based data access system.

Pacific Sentinel Gold Corp. personnel entered the down hole surveys and the collar surveys and were responsible for making corrections from the data verification process.

12.1.2 2008-2012

For the 2008 through 2012 exploration programs, field data processing and reporting was contracted to Casselman Geological Services Ltd. of Whitehorse, YT by Western.

Drill hole logging, sampling and geotechnical data was entered directly by the geologist or geotechnical logger working on the core in a Microsoft Excel spreadsheet. Upon completion of each hole these files were submitted to the Project Manager for checking. Upon receipt of analytical data from the lab, the data was merged with the sample intervals by the Project Manager and the data was then verified.

All data was entered into Microsoft Excel spreadsheets organized into a standardized format. Once the data was checked it was posted on the Western FTP site. The data was then merged into Geosoft Target software for creation of drill plans, drill sections, and 3D modelling.

12.1.3 2013- 2019

A transition to an updated database system, GeoSpark Core, was implemented in 2013 by Western to streamline the data flow and provide automated data validation and checking. All data ranging from 1960’s RC drill data to 2013 drill core logging details were merged and imported to the new database system.

Similar procedures to those used from 2008 through 2010 were used to collect the hydrogeological and water well drill data in 2013.

In 2019, all data was initially transcribed onto paper. Each part of the data logging was captured on a different piece of paper formatted for that specific data. Sample interval data was written directly onto the portion of the sample tag books that does not go into the sample bags during cutting.

The completed sample books were then checked by the Project Manager and stored in a secure cabinet in the geology office. The core logger was responsible for collecting all the data sheets in a file folder and scanning to digital files upon completion of the hole. These digital files were then uploaded to the Western remote server. Original paper copies were then filed in a secure cabinet in the geology office at the Casino Project site. The Project Manager would then ensure that each file folder for each hole had all the required data sheets, including Downhole Survey forms submitted by the drillers.

Down hole survey information was recorded digitally by the DeviShot downhole survey tool and then downloaded directly from the digital recorder by the Project Manager. This data was checked by the Project Manager and the digital files were uploaded to the Western server. Collar surveying was performed by surveyors from CAP Engineering and this data was provided to the Project Manager for addition to the main Casino Project database.

Upon completion of the field season, all 2019 data was entered into GeoSpark Core using the digital scans of the original core logging data. GeoSpark Core contains built in checks to ensure a clean and usable dataset. Libraries of all data (e.g. lithology codes) link to each portion of the data entry, so that only codes that are checked and used by the project are accepted. Digital survey files that are downloaded from the downhole survey tool can be directly imported into GeoSpark. Assay files submitted directly from the lab were also imported directly without manipulation.

During the 2020 field season, all field data was entered directly into GeoSpark Core, using the same built-in checks and library codes as in 2019 to ensure a clean and usable dataset. The field geologists provided semi-weekly exports from GeoSpark to the Database Manager, who then compiled all the core logging data in the central database using GeoSpark. Digital survey files were downloaded from the downhole survey tool and also provided to the Database Manager for import into GeoSpark. Assay files received directly from the lab were imported by the Database Manager to GeoSpark.

12.2 DATA VERIFICATION

For the purposes of this report, the author has visually verified five percent of original scanned paper drill logs and original drill hole sample assay certificates, and compared these to the digital data used in the resource assessment. This verification amounted to a review of 26 original drill logs and one excel file (containing re-logged primary lithologies and alterations), as well as 32 original signed assay certificates containing gold, silver, copper, and molybdenum analytical results. These were compared to the digital data within the project database.

The author has found no errors in the data transcription. This infers that the errors mentioned below have been addressed and provides further confidence in the data within the database.

12.2.1 1992-1994

The data verification process was performed under the supervision of a geologist familiar with the site logging procedures. In teams of two, one person read the original certificate, information sheet or logging form out loud while the other visually scanned the database printouts. Differences between the two were noted and corrected on the printout and the digital database. When required, a second pass was done on selected data.

The procedure for correcting errors was to highlight the value in question and to write the correct value beside it. Occasionally, the verification of field logs was followed up by a geologist familiar with logging and sampling techniques.

In addition, validations occurred throughout the exploration programs with ongoing monitoring and validation of field logs and analytical results, during the entire PSG Exploration endeavors.

12.2.2 2008-2013

The data verification process was performed under the supervision of the Project Manager. When errors were observed in geological, geotechnical, or sample intervals, the Project Manager and geologist or technician would go back to the core and/or original notes or sample tag booklets, sort out the error and make necessary corrections.

Data verification was performed on an ongoing basis. At times where data were first recorded on paper, original copies of the hand notes were kept for future reference.

During verification of field logs, when it was unclear which value was correct, a decision was made by a geologist familiar with logging and sampling techniques.

12.2.3 2019

The data verification process was performed under the supervision of the Project Manager/Senior Geologist on site. Digital scans of all the original core logs and related data were used to compare directly to the data that had been entered into GeoSpark Core software. The core log data was split up into sections (e.g. Assay results comprised one section and Lithology was another) and assigned to two separate people to verify. Each person was also given a full Excel export of the master database, with which they would be comparing the original log scans. After each section was verified, the person that performed the work would submit a memo outlining the errors encountered and possible solutions to the Database Manager. The Database Manager would then work through the errors and make changes, when warranted. A complete review (100% of the records) was done for Assays, Alteration, Metallurgy, Lithology, Collars and Surveys. Assays were compared directly to the original assay certificates. Collar data was compared directly to the surveys performed by CAP Engineering and the logging forms and Downhole Survey data was compared directly to the exports from the DeviShot survey tool. All other data was compared directly to the core log scans. A partial (approximately 20%) review was then completed by the database manager for Geotechnical and Specific Gravity data by comparing the master database records to the original log scans.

Upon completion of 2019 data verification, the Project Manager reviewed the errors found and made changes where warranted. In cases where it was unclear what data was correct, the Project Manager would review related information (e.g. notes/comments on the logs and core photos) and make a final decision based on that related data and on extensive knowledge of the project itself. Overall, the data was in good shape, with occasional missing records or incorrect codes (e.g. POT instead of PRO for an alteration interval) entered during the first phase of data transcribing from original logs to GeoSpark. There were very few errors found during the partial review of the Geotechnical and Specific Gravity data. A complete review (100%) of this data was not conducted at the time, due to the initial 20% pass revealing so few errors, and because a 20% verification rate is considered acceptable by the manager for this data.

12.2.4 2020

Verification of the 2020 data comprised a three-step process that remains ongoing as of the date of this report. As in 2019, Step One in 2020 utilized the "GeoSpark Core" program in the field to collect core logging data. Assay results were imported directly from the laboratory into the program. As the data was collected and the logging of each hole was completed, geologists in the field would visually review the digital core log for accuracy and use the validation tools within "GeoSpark Core" to complete Step One of verification. The database manager compiled all core logging data received from the field, imported assay results received directly from the lab and visually verified the data during this process.

The second stage of verification was completed by two consulting geologists who did a complete audit of the 2020 Collar, Survey, Assay, Alteration, Metallurgical, Lithology, Structural and Geotechnical data. As in 2019, after each section was verified, the person who performed the work would submit a memo outlining the errors encountered, and possible solutions to these errors, to the Database Manager. The Database Manager would then work through the errors and make changes, where warranted. A 20% verification of assay data was completed by comparing the original assay certificate to the values imported into GeoSpark.

Upon completion of 2020 data verification, the Database Manager reviewed the errors found and made changes where warranted. In cases where it was unclear what data was correct, the Database Manager would review related information (e.g. notes/comments on the logs and core photos) and make a final decision based on that related data and on extensive knowledge of the project itself. Overall, the data was in good shape, with occasional missing records due to issues during import and typos during data entry (e.g. CAP litho code entered in the middle of 100 m of WRGD). Any missing data was located and imported, and typos were corrected.

The final stage of verification will take place while the lithology, metallurgy and alteration surfaces are being updated in a 3D model and compared with historic data. Any logged data from 2020 that does not appear to visually match up with historic logged data will be investigated further.

12.3 VERIFICATION ERRORS

For the purposes of this report, the author has verified five percent of original drill logs and drill sample assays to the data used in the resource assessment. This involved visually comparing the analytical results for gold, silver, copper, and molybdenum within the original scans of signed assay certificates with assay data in the database. It also involved visual comparison of scanned paper drill logs, focusing primarily on lithology and alteration data and the corresponding intervals, to the data in the database, as well as review of re-logged data where applicable. There were no discrepancies found during this verification. This infers that the errors noted below that are related to earlier reviews have been addressed.

12.3.1 1992-1994

The geological logs had some errors introduced when the data entry personnel were unclear of the recording method the geologist was using. Additionally, changes in definitions of many of the lithology types required re-logging of many of the holes in the 1992 and 1993 programs. The process of combining the information from the old and new logs introduced some errors into the database. Due to the number of discrepancies encountered in the Geology data of the 1992 and 1993 programs, a second verification of lithologies and alteration was performed after the errors detected in the first pass were corrected. During the re-logging of the historic core in 2010, these errors associated with geology, mineralogy or alteration have been eliminated.

12.3.2 2008-2012

There were very few errors in the database. The most common errors occurred during recording of geological or sample intervals, where the “To” recording of a previous sample did not match the “From” recording of the subsequent sample. These were generally easy to sort out by the geologist or geotechnical logger.

Discrepancies with the assay, ICP and copper leach data involved values below the detection limit. Occasionally, “less than” signs (<) were misplaced for the lower detection limit values. Anomalously high ICP values were occasionally rounded off differently in the assay certificates than in the assay data downloaded from the computer bulletin board.

The geotechnical logs were checked by the computer to detect any intervals with combinations of parameters that were suspect. These intervals were extracted from the database and the suspect values were checked against the originals and against other available information, such as core photos, to determine if they were in error. A large majority of the extracted parameters were correct and considered to be caused by normal variance of geotechnical characteristics.

Errors detected in the field data of the geological logs, geotechnical logs, synoptic logs, specific gravity logs and down-hole survey data were often a result of human error during recording of the original logs, or during transcription. Wherever possible, computer checks were done on the data; several types of errors were detected this way.

Errors found in the specific gravity data were due to the geotechnician assigning the wrong sample number to the interval from which the specific gravity was taken. These errors were detected by a computer check and confirmed by the data verification personnel.

12.3.3 2019

A complete data audit took place following the 2019 exploration program at the Casino Project. The data audit included a 100% audit of: Assay Data, Alteration Data, Metallurgical Data, Lithography Data, Survey Data and Collar Locations.

The audit was performed using exported data from the Geospark Core database.

There were two main types of errors encountered during the verification process: missing records that had not been imported or entered, and incorrect codes/typos. Overall, there were very few errors in the data entry, and all could be easily corrected by the project manager. Missing data was imported in the case of assay certificates or entered from original logs in the case of logging information.

The GeoSpark Core catches the inherent errors that crop up from manual entry into the Excel or Access programs. As a result, the 2019 dataset was ready for import into other software for maps, cross sections and 3D modeling directly after the audit and verification process.

12.3.4 2020

A complete data audit for the 2020 exploration program at the Casino Project is in progress, with 100% of core logging data and at least 20% of assay data audited to date. The data audit will include a 100% audit of: Assay Data, Alteration Data, Metallurgical Data, Lithography Data, Survey Data and Collar Locations.

The audit is being performed using exported data from the Geospark Core database.

As in 2019, the GeoSpark Core catches the inherent errors that crop up from manual entry into the Excel or Access programs. As a result, the 2020 dataset will be ready for import into other software for maps, cross sections and 3D modeling directly after the audit and verification process.

12.4 OPINION OF QUALIFIED PERSON

It is the author's opinion, as the Qualified Person responsible for this section of this report, that the data for the Casino Project meets NI 43-101 standards and is adequate for the purposes of resource estimation and for use in this technical report.

All parts of the data collection process from drilling, sampling, and logging to shipping, assaying and verification have been reviewed by the author or supporting workers. It is the author's opinion that the Casino Project database has been maintained at high quality.

In addition, the author has performed a five percent verification on scanned, original drill logs and signed, original assay certificates compared to the data in the Casino Project database; the author has found no errors in the data transcription.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

The Casino Project will produce copper flotation concentrates with contained gold and silver values, and molybdenite flotation concentrates. Gold in the form of doré, and a high-grade copper sulphide product will also be produced from an oxide mineralized material heap leach. All products will be shipped offsite for sale or further processing.

13.1 METALLURGICAL SAMPLES

In the test work commissioned by Pacific Sentinel Gold in the mid 90's, all of the samples used were assay rejects that were nominally minus 10 mesh in particle size. These assay rejects were combined to prepare a number of composites that were sent to Lakefield Research for flotation and other testing under the direction of Melis Engineering, Ltd., to Brenda Process Technology for flotation testing, and to Kappes, Cassiday and Associates for copper and gold leaching.

The source of samples for all the 2008 work was split HQ core that was retrieved from site in September 2007. The core had been at site since it was drilled in 1993 and 1994 but was stored under cover.

Samples for the G&T Metallurgical Services (G&T) test program reported in early 2011 were split from fresh core from the 2010 drill program.

Samples for the comminution testing performed by Starkey and Associates (Starkey), and comminution and flotation testing by G&T reported in early 2012 were retrieved from the 1993 to 2010 drill programs and consisted of split core.

A drill program to retrieve fresh hypogene core was completed in early 2012 and split core from this drilling campaign was used for the flotation tests reported by G&T in December 2012.

In June 2013, bulk samples of different lithologies were taken from just below the surface of the deposit using an excavator and were used for heap leaching studies performed by SGS E&S Engineering Solutions Inc. (SGS) reported in October 2014.

13.2 COMMINUTION TESTING

In 2008 SGS Lakefield, under the direction of SGS MinnovEX, performed a comprehensive comminution study. Fifty (50) split drill core samples, representing the first six (6) years of production were sent to SGS and subjected to several tests.

A summary of the grinding results is presented in Table 13-1. As SGS reports, the samples tested were characterized as medium in hardness from the perspective of semi-autogenous milling and of medium in hardness with respect to ball milling.

Table 13-1: Summary of Comminution Results

Test Name	CEET CI	SPI (Min)	RWI (kWh/t)	BWI (kWh/t)	MBWI (kWh/t)	AI (g)
Average	29.2	52.9	9.9	14.5	14.3	0.265
Std. Dev.	13.9	20.8	5.6	2.6	1.6	0.046
Rel. Std. Dev.	47.5	39.3	56.5	18.1	11.3	17.0
Minimum	13.5	12.6	0.0	11.2	11.4	0.226
10th Percentile	15.3	31.4	4.4	12.1	12.5	0.232
25th Percentile	19.1	37.4	11.1	13.3	13.0	0.242
Median	24.1	50.3	12.5	14.1	14.1	0.252
75th Percentile	38.0	63.4	13.0	15.9	15.6	0.275
90th Percentile	52.3	82.5	13.0	17.3	16.3	0.309
Maximum	66.9	114.1	13.0	18.2	18.3	0.332

Additional comminution testing was performed in 2012 under the direction of FLSmidth and Starkey and Associates at FLSmidth laboratories and G&T Metallurgical Services. This program tested 11 composites of mineralized material representing a combination of different zones, lithologies and alterations. The 11 composites represent over 80% of the material that will be processed through the mill.

Mineralized material composite types that were not tested were mapped to similar composites that were tested by CMC geologists.

The 11 comminution composites were subjected to a series of tests at G&T's laboratory and FLSmidth's laboratory. The test results are summarised in the following Table 13-2 and Table 13-3.

Table 13-2: Summary of G&T SAG Mill Comminution (SMC) Test Results

Sample ID	DWi, kWh/m^3	DWi, %	Mia, kWh/t	Mih, kWh/t	Mic, kWh/t	A	B	SG	t_a
Composite 1	4.90	39	15.5	10.8	5.6	56.7	0.95	2.64	0.53
Composite 2	4.35	32	14.1	9.6	5.0	56.3	1.07	2.63	0.59
Composite 3	6.05	55	18.6	13.5	7.0	61.8	0.70	2.60	0.43
Composite 4	6.62	63	19.8	14.6	7.6	62.3	0.64	2.63	0.39
Composite 5	6.69	64	19.9	14.7	7.6	63.4	0.62	2.64	0.39
Composite 6	3.92	26	13.3	8.8	4.6	62.9	1.05	2.58	0.66
Composite 7	5.75	51	18.1	13	6.7	66.4	0.67	2.58	0.45
Composite 8	5.60	49	16.9	12.1	6.2	64.1	0.75	2.69	0.46
Composite 9	5.00	40	16.1	11.3	5.8	67.9	0.76	2.58	0.52
Composite 10	9.63	90	26.3	20.9	10.8	91.3	0.30	2.67	0.27
Composite 11	5.69	50	17.6	12.6	6.5	66.4	0.69	2.62	0.46
Average	5.84	51	17.8	12.9	6.7	65.4	0.75	2.62	0.47

Table 13-3: Summary of SAG Design Results and Crushed Bond Test Results

Sample ID	DML SAG Design Test Results			G&T Crushed Bond Test Results			
	Relative Density	Calc W_{SAG} to 1.7 mm (kWh/t)	SAG Dis. Bond BWi (kWh/t)	BWi (kWh/t)	RWi (kWh/t)	Ai (g)	CWi
Composite 1	2.66	8.19	16.18	13.5	12.9	0.162	9.41
Composite 2	2.60	6.78	17.26	14.1	12.3	0.176	10.00
Composite 3	2.66	9.39	15.70	14.1	14.5	0.198	13.62
Composite 4	2.72	12.41	18.36	15.5	15.5	0.199	13.84
Composite 5	2.64	9.56	18.26	15.3	14.6	0.156	11.20

Sample ID	DML SAG Design Test Results			G&T Crushed Bond Test Results			
	Relative Density	Calc W_{SAG} to 1.7 mm (kWh/t)	SAG Dis. Bond BWi (kWh/t)	BWi (kWh/t)	RWi (kWh/t)	Ai (g)	CWi
Composite 6	2.67	5.05	16.37	13.7	10.4	0.118	10.22
Composite 7	2.69	7.45	16.12	13.4	12.4	0.155	14.57
Composite 8	2.82	7.71	17.82	15.2	14.1	0.170	12.27
Composite 9	2.57	6.48	14.35	12.9	11.4	0.158	11.03
Composite 10	2.71	11.68	18.93	16.6	14.9	0.161	13.23
Composite 11	2.67	8.50	17.23	15.1	13.5	0.170	10.33
Average	2.67	8.47	16.96	14.5	13.3	0.166	11.79

A circuit consisting of one 40 ft diameter (12.2 m) SAG mill and two 28 ft diameter (8.5 m) ball mills in closed circuit with three pebble crushers was selected, based on discussions with M3 and FLSmidth, as a circuit that would likely meet the design tonnage. This circuit was modelled by FLSmidth using the parameters developed by SGS, G&T, and FLSmidth. The results of this exercise are shown in Table 13-4.

Table 13-4: Predicted Production Rate

Project Sample Number	Client Sample Information	BWi	Production Rate (t/d)
		G&T (kWh/t)	
1	Composite 1	13.5	133,805
2	Composite 2	14.1	128,064
3	Composite 3	14.1	128,064
4	Composite 4	15.5	116,582
5	Composite 5	15.3	118,018
6	Composite 6	13.7	131,818
7	Composite 7	13.4	134,798
8	Composite 8	15.2	118,790
9	Composite 9	12.9	139,987
10	Composite 10	16.6	108,854
11	Composite 11	15.1	119,674
Average		14.5	125,314

13.3 FLOTATION

13.3.1 2008 G&T Metallurgical Work

In 2008, Western and G&T Metallurgical reviewed the previous metallurgical work and developed an updated flotation program. To prevent oxidation, the program used split drill core rather than assay rejects as it had been done for the previous work.

The work focused on two composites at two different levels of oxide copper – an “oxide composite” and a “sulphide composite”. The composites were prepared to be close to the average grade of mineralized material received for the first 5 years. Assays for these composites are shown in Table 13-5.

Table 13-5: G&T Flotation Composite Assays

Composite	Cu (%)			Mo (%)		Fe	Au
	Total	WAS*	CNS**	Total	AS	(%)	(g/t)
Oxide Composite	0.275	0.132	0.042	0.019	0.006	3.225	0.345
Sulphide Composite	0.260	0.016	0.032	0.021	0.002	3.525	0.255

* Weak Acid Soluble
** Agent that affects the Central Nervous System

13.3.1.1 Oxide Composite

Copper recovery and grade from the oxide composite was very poor. Various combinations of sulphidizing the mineralized material, changing grind size, using different reagents were attempted. Based on the poor performance of the oxide flotation, no further testing on the oxide composite was performed.

13.3.1.2 Sulphide Composite

Copper recovery from the sulphide composite was much better than that achieved for the oxide composite. Copper concentrate grades greater than 28% were routinely achieved.

Copper recoveries of 70-82% were obtained into concentrates grading from 26.8 to 32.2% copper in cleaner tests. Good recovery of copper was obtained with both a primary grind with K80's of 147 and 121 µm and regrinds with K80's less than 22 µm. A coarser grind with a K80 of 209 µm was examined in rougher tests and shown to be less favorable than the finer particle sizes selected for cleaner testing.

13.3.1.3 Locked Cycle Tests

Duplicate locked cycle tests at both primary grind K80's of 121 µm and 147 µm were performed as well as one locked cycle at a primary K80 of 209 µm. The results from these tests indicate that a grind with a K80 of 147 µm, 85.6% copper can be recovered into a 28.5% copper concentrate. Molybdenum recovery was variable and ranged from 26.5% to 69.4%. Gold recovery was more consistent and averaged 64.0%.

13.3.1.4 Variability Testing

A total of 63 individual split drill core intervals were tested for variability. These samples were chosen to primarily represent the first six years of production and covered a broad range of total copper, acid soluble copper, molybdenum and gold values. Each of these samples was individually ground and floated in a cleaner test with regrind under the conditions determined from the locked cycle tests.

13.3.2 2009-2011 G&T Metallurgical Work

13.3.2.1 2009 Fresh Core Tests

The 2009 drilling campaign included two holes in the middle of the deposit – CAS-002 and CAS-003. A composite from CAS-002 had 92% copper recovery into a concentrate grading about 28% copper in cleaner tests. Similarly, a composite from CAS-003 had 87% of the copper in the feed recovered into a concentrate grading 26% copper. Moly recoveries were high in both tests at approximately 90%.

13.3.2.2 2010 Supergene Sulphide Composite Tests

The material tested in the 2010 test program (reported at the beginning of 2011) was a composite of supergene material that was obtained from the drilling campaigns in 2009 and 2010. This material represented mineralized material that will be fed to the mill in the later years of the operation. The feed grade averaged 0.30% copper and 0.037% molybdenum.

One of the main objectives of the 2010 test program was to evaluate coarser grinds than were tested in the 2008 test program. Results of this evaluation indicate that copper flotation response is virtually unaffected by primary grind size between 142 and 253 µm for this composite. Molybdenum flotation recovery to the bulk rougher concentrate was lower at grinds coarser than 179 µm. Molybdenum recovery was also reduced at elevated pH levels.

13.3.2.3 2010 Supergene Sulphide Composite Locked Cycle Tests

Locked cycle tests at primary grind K80's of 142 µm and 222 µm were performed. The results from these tests are presented in Table 13-6. The effect of regrind size on bulk concentrate copper grade and the effect of primary grind and regrind size on moly recovery are indicated in the table below.

Table 13-6: 2010 Supergene Sulphide Composite Locked Cycle Test Results

Test	P. Grind K80 µm	Regrind K80 µm	Cycle	Assay Cu (%)	Mo (%)	Fe (%)	Au (g/t)	Distribution - percent Cu	Mo	Fe	Au
KM2721-33	222	19	IV	30.8	1.6	23.6	20.2	82.9	34.7	5.2	71.9
KM2721-33	222	19	V	28.2	1.4	26.5	19.9	81.6	34.2	6.2	69.7
KM2721-34	222	20	IV	26.1	1.6	25.7	17.8	88.6	48.8	7.0	68.4
KM2721-34	222	20	V	25.7	1.5	26.6	19.9	86.6	45.1	7.5	64.7
KM2721-35	142	19	IV	26.3	1.9	27.8	18.6	87.3	57.1	7.1	71.4
KM2721-35	142	19	V	25.1	1.7	27.6	16.1	86.4	54.3	7.6	66.1
KM2721-36	222	37	IV	17.8	1.4	31.1	13.1	81.7	55.7	9.6	67.0
KM2721-36	222	37	V	18.8	1.4	30.4	10.1	82.8	51.2	10.9	61.5
KM2721-37	222	31	IV	21.2	1.4	31.1	11.7	83.2	54.1	9.2	62.4
KM2721-37	222	31	V	20.8	1.7	31.3	11.7	83.9	59.7	9.9	65.7

13.3.2.4 Pyrite Flotation

Pyrite flotation was examined as a process to produce tailings samples that had low levels of residual sulphur, and thus could be deemed non-acid generating (NAG).

The locked cycle tests outlined in Table 13-6 included a pyrite rougher to reduce the sulphide concentration of the tailings. Pyrite flotation tailings from these tests obtained tailings averaging less than 0.08% sulphur.

13.3.3 2011-2012 G&T Metallurgical Work

Western retained International Metallurgical and Environmental to assist in the metallurgical testing and continued to perform the testing at G&T Metallurgical Services (name changed to ALS Metallurgy in late 2012).

13.3.3.1 Flowsheet Development

In previous testing campaigns, to achieve acceptable recoveries from the conventional copper flotation flowsheet's tested, 15-20% of the feed material needed to be reground. The focus of the flowsheet development was to reduce the material sent to the regrind mills.

The flowsheet development centered on a flowsheet where rougher concentrate was sent to the first cleaning stage prior to regrinding, the first cleaner concentrate went to regrinding and the second and third cleaner tails were returned to the first cleaner. By utilizing this flowsheet, the amount of feed material that needed to be reground dropped from 15-20% to 3 to 5%.

Locked cycle test results from the composites tested using this flowsheet are shown in Table 13-7. The results show similar recoveries to previous test work using a conventional copper flotation flowsheet.

Table 13-7: Flowsheet Development Locked Cycle Test Results

Composite	Tests	P. Grind K80 µm	Regrind K80 µm	Assay				Distribution (%)			
				Cu (%)	Mo (%)	S (%)	Au (g/t)	Cu	Mo	S	Au
HYP1	38, 42	218	19	26.0	1.98	33.1	24.6	82.1	64.9	24.9	61.1
HYP2	39, 43	216	16	26.3	1.31	32.8	23.9	81.7	37.1	14.6	56.1
SUS1	44, 46	192	17.5	21.8	1.77	33.9	23.6	77.7	59.5	24.9	75.9
SUS2	47	190	14	24.1	0.85	38.1	28.3	62.8	32.8	20.4	64.4

13.3.3.2 Tests using Fresh Core

While supergene flotation tests were performed on fresh core obtained during the 2010 campaign, no flotation tests had been performed on fresh hypogene core except for a limited number of tests performed in 2009.

In 2012, a drilling campaign was executed to obtain fresh hypogene core from the first years of mining that represented the predominate mineralization that would be fed to the mill. In total, five holes were drilled (CAS-088 to CAS-093), and from these five holes, three composites were made representing lithologies: Patton porphyry (PP), Intrusion breccia (IX), and Dawson range batholith (WR).

Table 13-8: Hypogene Composites

	Cu (%)			Mo	Fe	Au
	Total	WAS	CNS	(%)	(%)	(g/t)
PP Composite	0.14	0.004	0.008	0.030	2.95	0.22
IX Composite	0.17	0.006	0.012	0.071	2.39	0.22
WR Composite	0.19	0.005	0.013	0.019	2.50	0.18

Locked cycle recoveries using these fresh composites were significantly better than previous testing on oxidized core and are shown in Table 13-9. Note that the primary grind size for these tests was also higher than the target of 200 µm, in some cases significantly, so it would be expected that actual plant recovery would be better than these tests indicate.

Table 13-9: Locked Cycle Test Results

Composite	Tests	P. Grind K80 µm	Regrind K80 µm	Assay - percent or Au g/t				Distribution (%)			
				Cu (%)	Mo (%)	Ag (g/t)	Au (g/t)	Cu	Mo	Ag	Au
PP	23	234	31	18.6	7.5	126	15.6	89.9	77.9	46.5	57.3
IX	24	254	32	24.6	4.3	107	24.3	87.2	78.6	46.0	55.4
WR	25	211	31	17.5	1.50	82	13.5	91.9	89.4	53.8	67.2

13.3.3.3 Pilot Plant Testing and Copper/Molybdenum Separation

A pilot plant was performed on hypogene and supergene composites taken from the drilling campaign to produce representative tailings for environmental testing, geotechnical testing and thickener testing and to produce sufficient copper/molybdenum concentrate for copper moly separation tests. Unfortunately, there was not sufficient feed material to obtain operating information from the pilot plant.

Although suitable copper/molybdenum concentrate was produced to perform several copper/molybdenum separation tests, only one cleaner test was performed. The results from this test were sufficiently good to warrant no further testing. The results from this test are shown in Table 13-10.

Table 13-10: Copper/Molybdenum Separation Cleaner Test

Cumulative	Cum. Weight		Assay				Distribution (%)			
Product	%	grams	Cu (%)	Mo (%)	Fe (%)	S (%)	Cu	Mo	Fe	S
Final Conc.	3.1	31.2	0.39	57.4	0.8	37.9	0.1	94.1	0.1	2.6
Second Conc.	3.5	35.5	2.38	51.3	3.8	37.6	0.5	95.7	0.4	3.0
Rougher Conc.	6.1	62.5	9.04	29.7	15.3	38.0	3.5	97.4	2.9	5.3
Tails	93.9	953.8	16.5	0.05	33.9	44.3	96.5	2.6	97.1	94.7
Feed	100.0	1016.3	16.0	1.87	32.8	43.9	100	100	100	100

13.3.4 Interpretation of Flotation Test Results

The 2012 at ALS Metallurgy has shown good copper recovery to copper concentrates that routinely achieve 28% or greater for various drill core samples from the deposit using the reagent scheme developed. The conclusions from this work are unambiguous and will be used as the basis of this study.

13.3.4.1 Supergene – Copper

It was difficult to achieve good copper concentrate grades from supergene oxide material that had copper oxide concentrations greater than 25-30% of the total copper. For this reason, during operation of the mill, supergene oxide mineralized material will need to be blended in with the other mineralized material to achieve an oxide copper percentage less than 25%.

The supergene mineralized material contains a certain percentage of oxide copper minerals (this is what defines it as being supergene material). Oxide copper minerals are poorly recovered by the flotation process; therefore, in the interpretation of the results, it is important to examine the recovery of sulphide copper to a copper concentrate. Sulphide copper can be calculated by subtracting the concentration of oxide copper from the total copper. Supergene mineralization at Casino has been assayed for weak acid soluble copper (WAS), which is approximately equal to the amount of oxide copper in the sample assayed but may under or over represent the amount of oxide copper present depending on the specifics of the mineralization.

Sulphide copper recovery as a function of total copper grade and sulphide copper grade is shown in Table 13-11 for the supergene locked cycle tests by G&T Metallurgical. Recovery appears to be consistent.

Table 13-11: Supergene Locked Cycle Recoveries to Concentrate

	Feed Assays					Recovery to Concentrate			
	Cu (%)			Au	Mo	Total	Sulphide		
Test	Total	WAS	Sulphide	(g/t)	(%)	Cu	Cu	Au	Mo
KM2721									
33	0.3	0.03	0.27	0.25	0.036	82.3	91.4	70.7	34.5
34	0.3	0.03	0.27	0.25	0.036	87.6	97.3	66.4	46.9
35	0.3	0.03	0.27	0.25	0.036	86.8	96.4	68.8	55.7
36	0.3	0.03	0.27	0.25	0.036	82.3	91.4	64.4	53.3
37	0.3	0.03	0.27	0.25	0.036	83.5	92.8	64.1	57
KM3134									
44	0.3	0.056	0.244	0.37	0.022	79.9	98.2	75.5	64.6
46	0.3	0.056	0.244	0.47	0.022	75.6	93.0	76.1	54.6
47	0.3	0.094	0.206	0.47	0.028	62.8	91.5	64.4	32.8

Averaging the locked cycle tests results indicates that an average of 94% of the sulphide copper was recovered to a copper concentrate. This result also closely mirrors the variability results. Thus, the overall copper recovery for the supergene material will be:

Cu Recovery = 94 x $(Cu_{total} – Cu_{WAS})/(Cu_{total})$

13.3.4.2 Supergene – Gold

Averaging the gold recovery from Table 13-11, an average gold recovery of 69% to copper concentrate is obtained:

Au Recovery = 69%

13.3.4.3 Supergene Molybdenum

In the majority of the tests, no attempt was made to optimise the molybdenum recovery. For this reason, the molybdenum recovery is quite variable.

Examining the locked cycle tests in Table 13-10, an average molybdenum recovery of 55% to copper concentrate was chosen, which represents the average molybdenum recovery when the two low outliers are removed.

Recovery of molybdenum from the copper-molybdenum concentrate to a molybdenum concentrate was not specifically tested for the supergene material, but it is expected to be similar to that obtained in hypogene tests that achieved approximately 95% molybdenum recovery to a molybdenum concentrate. Molybdenum recovery throughout the plant is equal to the recovery to the copper-molybdenum concentrate multiplied by recovery to a molybdenum concentrate and is shown below:

Mo Recovery = 52.3%

13.3.4.4 Supergene – Silver

Unfortunately, silver recovery was not determined in all test programs. The 2011 test program followed silver recovery. Averaging the silver recovery from these locked cycle tests indicates that a silver recovery of 60% should be achievable:

Ag Recovery = 60%

13.3.4.5 Hypogene

Hypogene recoveries are based on the December 2012 flotation work performed by ALS Metallurgy on “fresh” core that had been drilled earlier specifically for flotation test work.

Table 13-12 shows cleaner circuit recoveries for both copper and molybdenum for all three locked cycle tests with hypogene material. Copper concentrate grades have been corrected to reflect the removal of molybdenum and represent final concentrate grades in terms of copper.

Table 13-12: Cleaner Circuit Recoveries for Locked Cycle Test Results

Test and Cycle No.	Cu Con Grade %Cu	Cu Recovery %	Mo Recovery %	Au Recovery %
WR Composite				
Cycle 4	17.8	96.4	95.0	86.0
Cycle 5	17.9	96.9	95.6	88.0
IX Composite				
Cycle 4	22.8	96.9	81.7	83.3
Cycle 5	21.2	96.7	80.8	80.6
PP Composite				
Cycle 4	26.1	97.1	90.4	88.0
Cycle 5	26.5	97.1	91.1	84.9

Copper, molybdenum, and gold recovery, when a primary grind size of 200 to 220 µm is used, is summarised in Table 13-12 and is based on both locked cycle testing and open circuit rougher flotation tests. Molybdenum recovery was variable and the higher-grade molybdenum sample (IX) had the lowest molybdenum recovery, indicating that reagent conditions could possibly improve this recovery. Within the cleaning circuit copper and gold recoveries were similar, irrespective of the final copper concentrate grade.

Table 13-13: Predicted Recoveries to Copper/Molybdenum Concentrate

Process Stream	Cu Recovery, %	Mo Recovery, %	Au Recovery, %
Rougher Circuit Recovery	95	92	78
Cleaner Circuit Recovery	97	90	85
Metal Recovery	92	82.8	66

13.3.4.6 Hypogene – Copper Molybdenum Separation

One test was performed to determine how well molybdenum could be separated from a copper/molybdenum concentrate. The test indicated that approximately 95% molybdenum recovery could be achieved. Thus, the overall recovery of molybdenum will be equal to:

Mo Recovery = 78.7%

13.3.4.7 Hypogene – Silver Recovery

Hypogene silver recovery was followed in the last set of tests on fresh core. Reviewing these recoveries, a silver recovery of 50% was chosen.

Ag Recovery = 50.0%

13.3.4.8 Concentrate Quality

Estimates of the chemistry of the copper concentrate are summarised in Table 13-14. The table is comprised of the best estimates of analysis of concentrates produced in test work. Concentrate chemistry estimation is based on detailed analysis of test products, conducted at various metallurgical test facilities.

Table 13-14: Copper Concentrate Chemistry

Element	Average Expected Value	High Range	Low Range
Copper - %	28	30	25
Gold – g/t	25	30	15
Silver – g/t	120	180	80
Molybdenum -%	0.05	0.1	0.02
Iron - %	26	30	24
Sulphur - %	36	40	28
Arsenic – g/t	200	500	100
Antimony – g/t	250	400	100
Mercury – g/t	1	2	0.1
Cadmium – g/t	40	80	20
Fluorine – g/t	100	200	50
Silica - %	2	5	1

Key analytical results for the Casino Project molybdenum concentrate are summarised in Table 13-15. Limited test work allows for only an average chemistry estimate to be made for the molybdenum concentrate at this time.

Table 13-15: Molybdenum Concentrate Chemistry

Element	Average Expected Value
Molybdenum -%	56.0
Copper - %	0.25
Gold – g/t	1
Silver – g/t	10
Iron - %	1
Sulphur - %	38
Arsenic – g/t	1500
Antimony – g/t	100
Mercury – g/t	<1
Cadmium – g/t	30
Silica - %	1.5
Rhenium – g/t	130

13.4 DEWATERING TESTS

Flotation tailing from the 2008 test program piloting were submitted to Outotec for dynamic high-rate thickening tests. Results were favorable and a thickener underflow of over 55 percent solids was achieved. Flocculant addition was 22 g/t. The solids loading rate of 1.05 t/m^2h was demonstrated. Rheology on the thickened material was low.

13.5 LEACHING TESTS

13.5.1 Kappes, Cassiday and Associates (KCA)

KCA performed two studies in 1995 on the leaching of the oxide cap (oxide gold zone) and supergene (oxide copper) material. In the first study they leached a selection of oxide cap material with cyanide. In the second study they examined pre-leaching both oxide cap and supergene material with acid followed by cyanidation of the residue.

Gold extraction was affected by the quantity of copper leached during cyanidation and ranged from 10-97.4%. Average gold extraction was 79.9%.

Lime consumption during cyanidation averaged 3.9 kg/t without the acid pre-leach, and 4.1 kg/t with the acid pre-leach. Cyanide consumption was significant, averaging 5.5 kg/t without the acid pre-leach. There was not a significant difference between the lime consumption for the oxide gold composites and oxide copper composites.

13.5.2 SGS E&S Engineering Solutions Inc.

SGS E&S Engineering Solutions Inc. (at the time METCON) ran two column tests on a composite sample blended to create gold and copper concentrations similar to the average reserve concentrations in 2008.

The mineralized material was crushed coarsely to -3.8 cm (-1.5 inch), placed in 15 cm by 6-metre columns, and irrigated at 12 L/h/m^2. One column was leached "open cycle" – a 0.5 g/L NaCN solution was fed to the top of the column and the pregnant solution was collected and assayed. The second column was conducted as a "locked cycle" and solution was recycled. In the locked cycle column when the copper concentration in solution exceeded 50 mg/L, the solution was treated through a Sulphidization, Acidification, Recycling and Thickening (SART) pilot plant discussed in the next section, and the gold was recovered on activated carbon.

The gold, silver, and copper extractions from the open and locked cycle tests compare favorably. Although the gold extraction was slightly higher for the open cycle test, both tests produced good gold recovery considering the coarse crush size.

Cyanide consumptions were similar based on titrations and the amount of cyanide added to the system for the locked cycle column at approximately 0.5 kg/t. Lime consumptions were similar to the bottle roll test work at approximately 3 kg/t.

Table 13-16: Extractions and Reagent Consumptions from Open Cycle and Locked Cycle Cyanidation

	Assays (calculated head) (g/t)			Percent Extraction			Reagent Consumption (kg/t)		
	Au	Ag	Cu	Au	Ag	Cu	NaCN*	NaCN**	CaO
Open	0.47	1.92	693	69.52	25.14	17.4	0.39		2.83
Locked	0.42	1.61	654	65.79	27.31	18.2	0.48	0.54	3.06

*based on titrations
**based on additions

A second set of testing was performed in 2013, which investigated metal recovery as a function of lithology. Based on an examination of a mine plan developed in 2013, it was determined that the heap leach would be primarily composed of Granodiorite (WR), Intrusive Breccia (IX) and Patton Porphyry (PP) lithology types with argillic (ARG) alteration.

The breakdown of heap leach mineralized material tested by lithology type is as below:

Lithology Type	% of Mineralized Material
WR - Dawson Granodiorite	64%
IX - Intrusive Breccia	28%
PP - Patton Porphyry	8%

The mineralized material was crushed coarsely to -3.8 cm (-1.5 inch), placed in 15 cm by 3-metre columns, and irrigated at 9.78 L/h/m^2. Each column was run in duplicate. The columns were operated in "open cycle". Solution containing 0.75 g/L free NaCN and 300 mg/L Cu (added to approximate that steady state Cu concentration that would be used to leach the mineralized material in practice) was added to the top of the column to irrigate.

Table 13-17: Extractions and Reagent Consumptions from Column Tests Investigating Lithology

Mineralized Material Type	Head Assays (g/t)			Percent Extraction		Reagent Consumption (kg/t)	
	Au	Ag	Cu	Au	Ag	NaCN	CaO
WR	0.27	0.85	72.3	82.56	27.97	0.26	4.34
(dup)	0.27	0.85	72.3	81.90	27.78	0.20	4.18
average	-	-	-	82.2	27.9	0.23	4.3
IX	0.54	2.70	46.2	64.55	22.71	0.68	3.11
(dup)	0.54	2.70	46.2	62.10	16.63	0.44	3.10
average	-	-	-	63.3	19.7	0.56	3.1
PP	0.63	2.76	73.0	75.09	26.29	0.47	3.51
(dup)	0.63	2.77	73.0	73.28	26.01	0.19	3.38
average	-	-	-	74.2	26.2	0.33	3.4
Weighted average	-	-	-	76.3	25.4	0.33	3.3

Gold extraction for WR and PP lithologies are higher than the gold recoveries in previous testing, and gold recovery for IX lithology is higher indicating that there is some variability in gold extraction based on lithology. Cyanide and lime consumption are more variable but are similar to what was obtained in previous work.

As described in the Metallurgical Sample section above, the samples used in the latest SGS work are from the first five (5) metres of the deposit. Samples used in previous SGS testing (2008 report) were from drill core that was spatially distributed in the heap leach resource. There is a significant difference in the gold recovery between the locked cycle test completed with SART in the early test and the latest tests as shown in Table 13-18 below.

Table 13-18: Comparison of Recovery

	Recovery		Consumption	
	Au	Ag	NaCN	Lime
Locked cycle w/SART 2008	66	26	0.50	3.3
Weighted average 2013	76.3	25	0.33	3.9

13.6 SART COPPER RECOVERY

In this process, a cyanide solution containing copper is treated to remove copper—gold is not affected.

In the locked cycle test described previously, the pregnant leach solution from the column was treated using a SART pilot plant several times before removing the gold with carbon and recycling the treated fluid to the column. The SART results are summarised in Table 13-19.

Table 13-19: SART Results

Pregnant Solution				Barren Solution after SART & Carbon				Copper	Reagent Consumption (g/L solution treated)		
Free NaCN (g/L)	Cu (ppm)	Au (ppm)	Ag (ppm)	Free NaCN (g/L)	Cu (ppm)	Au (ppm)	Ag (ppm)	Removal (%)	S^{2-}	H_2SO_4	CaO
0.25	81	0.21	0.30	0.39	6.8	0.04	0.02	91.3	0.024	0.64	0.37

13.7 DETERMINATION OF RECOVERIES, REAGENT, AND OTHER CONSUMABLE CONSUMPTIONS

As described in the preceding sections, the recoveries, reagent, and other consumable consumptions shown in Table 13-20 and Table 13-21 will be used. Where values were unknown, typical values based on M3's experience are used.

Table 13-20: Flotation Operational Parameters

Parameter	Value	Units
Copper recovery		
Supergene	Recovery = 94 x (Cu_{total} – Cu_{WAS})/(Cu_{total})	percent
Hypogene	92	percent
Gold recovery		
Supergene	69	percent
Hypogene	66	percent
Molybdenum recovery (final conc)		
Supergene	52.3	percent
Hypogene	78.7	percent
Silver recovery		
Supergene	60	percent
Hypogene	50	percent
Bond work index	14.5	kWh/t
Primary grind size (P80)	200	µm
Regrind size (P80)	25	µm
Reagent and wear consumptions		
Lime (93% active)		
Supergene	2.7	kg/t Mineralized material
Hypogene	1.1	kg/t Mineralized material
Aerophine 3418A		
Supergene	8.4	g/t Mineralized material
Hypogene	4.0	g/t Mineralized material
Aerofloat 208		
Supergene	16.7	g/t Mineralized material
Hypogene	8.0	g/t Mineralized material
MIBC	10	g/t Mineralized material
Fuel Oil	7.0	g/t Mineralized material
PAX	40	g/t Mineralized material
NaSH	0.053	kg/t Mineralized material
Flocculant	25.4	g/t Mineralized material
SAG Mill – Liners	0.040	kg/t Mineralized material
Ball Mill – Liners	0.048	kg/t Mineralized material
SAG Mill – Balls	0.400	kg/t Mineralized material
Ball Mill – Balls	0.400	kg/t Mineralized material
Regrind – Balls	0.0410	kg/t Mineralized material

Table 13-21: Heap Leach Operational Parameters

Parameter	Value	Units
Gold recovery	70	Percent
Copper recovery	18	Percent
Silver recovery	26	Percent
Crush size	-1	inch
Irrigation rate	12	$L/h/m^2$
Lift height	8	M
Leach solution application, primary	60	days
Reagent and wear consumptions		
NaHS	0.025	kg/t Mineralized material
Sulfuric acid	0.328	kg/t Mineralized material
Hydrochloric acid	0.010	kg/t Mineralized material
Lime (CaO) (93% active)	3.516	kg/t Mineralized material
Sodium hydroxide	0.130	kg/t Mineralized material
Sodium cyanide (NaCN)	0.500	kg/t Mineralized material
Activated Carbon	0.011	kg/t Mineralized material
Antiscalant	0.003	kg/t Mineralized material
Flocculant	0.350	g/t Mineralized material
Primary crusher liners	0.040	kg/t
Secondary crusher liners	0.085	kg/t

14 MINERAL RESOURCE ESTIMATES

14.1 MINERAL RESOURCE

The Mineral Resource for the Casino Project includes Mineral Resources amenable to milling and flotation concentration methods (mill material) and Mineral Resource amenable to heap leach recovery methods (leach material). Table 14-1 presents the Mineral Resource for mill material. Mill material includes the supergene oxide (SOX), supergene sulphide (SUS) and hypogene sulphide (HYP) mineral zones. Measured and Indicated Mineral Resources amount to 2.17 billion tonnes at 0.16% total copper, 0.18 g/t gold, 0.017% moly and 1.4 g/t silver and contained metal amounts to 7.43 billion pounds of copper, 12.7 million ounces gold, 811.6 million pounds of moly and 100.2 million ounces of silver. Inferred Mineral Resource is an additional 1.43 billion tonnes at 0.10% total copper, 0.14 g/t gold, 0.010% moly and 1.2 g/t silver and contained metal amounts to 3.24 billion pounds of copper, 6.4 million ounces of gold, 322.8 million pounds moly and 53.5 million ounces of silver for the Inferred Mineral Resource in mill material.

Table 14-2 presents the Mineral Resource for leach material. Leach material is oxide dominant leach cap (LC) mineralization. The emphasis of leaching is the recovery of gold in the leach cap. Copper grades in the leach cap are low, but it is expected some metal will be recovered. Measured and Indicated Mineral Resources amount to 217.4 million tonnes at 0.03% total copper, 0.25 g/t gold and 1.9 g/t silver and contained metal amounts to 166.5 million pounds of copper, 1.8 million ounces gold and 13.3 million ounces of silver. Inferred Mineral Resource is an additional 31.1 million tonnes at 0.03% total copper, 0.17 g/t gold and 1.7 g/t silver and contained metal amounts to 17.2 million pounds of copper, 200,000 ounces of gold and 1.7 million ounces of silver for the Inferred Mineral Resource in leach material.

Table 14-3 presents the Mineral Resource for combined mill and leach material for copper, gold, and silver. Measured and Indicated Mineral Resources amount to 2.39 billion tonnes at 0.14% total copper, 0.19 g/t gold and 1.5 g/t silver. Contained metal amounts to 7.60 billion pounds copper, 14.5 million ounces gold and 113.5 million ounces of silver for Measured and Indicated Mineral Resources. Inferred Mineral Resource is an additional 1.46 billion tonnes at 0.10% total copper, 0.14 g/t gold and 1.2 g/t silver. Contained metal amounts to 3.26 billion pounds of copper, 6.6 million ounces of gold and 55.2 million ounces of silver for the Inferred Mineral Resource. The Mineral Resource for moly is as shown with mill material since it will not be recovered for leach material.

The Mineral Resources are based on a block model developed by IMC during June 2020. This updated model incorporated the 2019 Western drilling and updated geologic models. It also includes some 2010 through 2012 Western drilling that was not available for the previous Mineral Resource estimate done in 2010.

The Measured, Indicated, and Inferred Mineral Resources reported herein are contained within a floating cone pit shell to demonstrate "reasonable prospects for eventual economic extraction" to meet the definition of Mineral Resources in NI 43-101.

Figure 14-1 shows the constraining pit shell that is based on Measured, Indicated, and Inferred Mineral Resource.

Table 14-1: Mineral Resource for Mill Material at C$5.70 NSR Cutoff

Resource Class	Tonnes Mt	NSR (C$/t)	Copper (%)	Gold (g/t)	Moly (%)	Silver (g/t)	CuEq %	Copper (Mlbs)	Gold (Moz)	Moly (Mlbs)	Silver (Moz)
Measured	145.3	38.08	0.31	0.40	0.025	2.1	0.74	985.8	1.9	80.6	9.8
Indicated	2,028.0	19.10	0.14	0.17	0.016	1.4	0.33	6,448.5	10.9	731.0	90.4
M+I	2,173.3	20.37	0.16	0.18	0.017	1.4	0.36	7,434.3	12.7	811.6	100.2
Inferred	1,430.2	14.50	0.10	0.14	0.010	1.2	0.24	3,240.4	6.4	322.8	53.5

Table 14-2: Mineral Resource for Leach material at C$5.46 NSR Cutoff

Resource Class	Tonnes Mt	NSR (C$/t)	Copper (%)	Gold (g/t)	Silver (g/t)	AuEq (g/t)	Copper (Mlbs)	Gold (Moz)	Silver (Moz)
Measured	37.2	19.72	0.05	0.45	2.8	0.48	39.3	0.5	3.3
Indicated	180.2	9.54	0.03	0.21	1.7	0.23	127.2	1.2	10.0
M+I	217.4	11.28	0.03	0.25	1.9	0.27	166.5	1.8	13.3
Inferred	31.1	7.60	0.03	0.17	1.7	0.18	17.2	0.2	1.7

Table 14-3: Mineral Resource for Copper, Gold, and Silver (Mill and Leach)

Resource Class	Tonnes Mt	NSR (C$/t)	Copper (%)	Gold (g/t)	Silver (g/t)	Copper (Mlbs)	Gold (Moz)	Silver (Moz)
Measured	182.4	34.34	0.25	0.41	2.2	1,025.1	2.4	13.1
Indicated	2,208.3	18.32	0.14	0.17	1.4	6,575.6	12.1	100.5
M+I	2,390.7	19.54	0.14	0.19	1.5	7,600.7	14.5	113.5
Inferred	1,461.3	14.35	0.10	0.14	1.2	3,257.6	6.6	55.2

Notes:
1. The Mineral Resources have an effective date of 3 July 2020 and the estimate was prepared using the definitions in CIM Definition Standards (10 May 2014).
2. All figures are rounded to reflect the relative accuracy of the estimate and therefore numbers may not appear to add precisely.
3. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
4. Mineral Resources for leach material are based on prices of US$2.75/lb copper, US$1500/oz gold and US$18/oz silver.
5. Mineral Resources for mill material are based on prices of US$2.75/lb copper, US$1500/oz gold, US$18/oz silver, and US$11.00/lb moly.
6. Mineral Resources are based on NSR Cutoff of C$5.46/t for leach material and C$5.70/t for mill material.
7. NSR value for leach material is as follows:
   NSR (C$/t) = $12.65 x copper (%) + $41.55 x gold (g/t) + $0.191 x silver (g/t), based on copper recovery of 18%, gold recovery of 66% and silver recovery of 26%.
8. NSR value for hypogene sulphide mill material is:
   NSR (C$/t) = $60.18 x copper (%) + $41.01 x gold (g/t) + $214.94 x moly (%) + $0.355 x silver (g/t), based on recoveries of 92.2% copper, 66% gold, 50% silver and 78.6% moly.
9. NSR value for supergene (SOX and SUS) mill material is:
   NSR (C$/t) = $65.27 x recoverable copper (%) + $42.87 x gold (g/t) + $142.89 x moly (%) + $0.425 x silver (g/t), based on recoveries of 69% gold, 60% silver and 52.3% moly. Recoverable copper = 0.94 x (total copper – soluble copper).
10. Table 14-6 accompanies this PEA and shows all relevant parameters.
11. Mineral Resources are reported in relation to a conceptual constraining pit shell in order to demonstrate reasonable prospects for eventual economic extraction, as required by the definition of Mineral Resource in NI 43-101; mineralization lying outside of the pit shell is excluded from the Mineral Resource.
12. AuEq and CuEq values are based on prices of US$2.75/lb copper, US$1500/oz gold, US$18/oz silver, and US$11.00/lb moly, and account for all metal recoveries and smelting/refining charges.

14.2 SENSITIVITY TO NSR CUTOFF

Table 14-4 shows resources at varying NSR Cutoffs for mill material. All tabulations are contained by the constraining pit shell used for the base case Mineral Resource at C$5.70 per tonne (highlighted). Increasing the NSR Cutoff by 40% to C$8/t has only a modest effect on the size of the Mineral Resource amenable to milling, decreasing resource tonnes by 6% and the contained copper and gold by 1.6% and 2.6% respectively.

Table 14-5 shows resources at varying NSR Cutoffs for leach material. Again, all tabulations are contained by the constraining pit shell used for the base case Mineral Resource. The base case resource at an NSR Cutoff of C$5.46 per tonne is highlighted. Increasing the NSR Cutoff of leach material to C$8/t only reduces the contained gold by 20%.

Table 14-4: Mineral Resource – Mill Material by Various NSR Cutoffs (C$)

NSR Cog ($/t)	Resource Category	Tonnes Mt	NSR ($/t)	Copper (%)	Gold (g/t)	Moly (%)	Silver (g/t)	CuEq (%)	Copper (Mlbs)	Gold (Moz)	Moly (Mlbs)	Silver (Moz)
5.70	Measured	145.3	38.08	0.31	0.40	0.025	2.1	0.74	986.5	1.9	80.7	9.8
	Indicated	2,028.0	19.10	0.14	0.17	0.016	1.4	0.33	6,438.2	10.8	733.2	90.6
	M+I	2,173.3	20.37	0.15	0.18	0.017	1.4	0.36	7,424.7	12.7	813.9	100.4
	Inferred	1,430.2	14.50	0.10	0.14	0.010	1.2	0.24	3,247.6	6.4	324.8	53.3
8	Measured	144.6	38.22	0.31	0.40	0.025	2.1	0.74	985.2	1.9	80.7	9.7
	Indicated	1,898.4	19.93	0.15	0.17	0.017	1.4	0.34	6,319.6	10.5	724.0	87.3
	M+I	2,043.0	21.22	0.16	0.19	0.018	1.5	0.37	7,304.8	12.4	804.7	97.0
	Inferred	1,181.0	16.11	0.12	0.15	0.012	1.2	0.27	3,020.3	5.7	309.8	47.1
16	Measured	139.3	39.19	0.32	0.41	0.026	2.1	0.76	973.4	1.8	80.1	9.5
	Indicated	1,182.3	24.61	0.19	0.21	0.022	1.7	0.42	4,900.0	7.8	583.8	64.2
	M+I	1,321.5	26.15	0.20	0.23	0.023	1.7	0.46	5,873.4	9.6	664.0	73.8
	Inferred	390.0	24.95	0.19	0.21	0.021	1.6	0.42	1,625.0	2.6	180.6	20.6
30	Measured	101.3	44.77	0.36	0.47	0.030	2.3	0.87	799.4	1.5	67.2	7.6
	Indicated	229.6	36.14	0.28	0.31	0.032	2.3	0.62	1,402.1	2.3	163.0	16.9
	M+I	330.9	38.78	0.30	0.36	0.032	2.3	0.70	2,201.5	3.8	230.2	24.5
	Inferred	74.4	39.26	0.32	0.32	0.029	2.4	0.65	521.3	0.8	47.0	5.6

Table 14-5: Mineral Resource – Leach Material by Various NSR Cutoffs (C$)

NSR Cog ($/t)	Resource Category	Tonnes Mt	NSR ($/t)	Copper (%)	Gold (g/t)	Silver (g/t)	AuEq (g/t)	Copper (Mlbs)	Gold (Moz)	Silver (Moz)
5.46	Measured	37.2	19.72	0.05	0.45	2.8	0.48	39.3	0.53	3.29
	Indicated	180.2	9.54	0.03	0.21	1.7	0.23	127.2	1.23	10.03
	M+I	217.4	11.28	0.03	0.25	1.9	0.27	166.5	1.76	13.31
	Inferred	31.1	7.60	0.03	0.17	1.7	0.18	17.2	0.17	1.70
8	Measured	35.4	20.36	0.05	0.46	2.8	0.49	38.2	0.53	3.21
	Indicated	107.3	11.43	0.03	0.26	2.0	0.28	71.0	0.89	6.83
	M+I	142.7	13.64	0.03	0.31	2.2	0.33	109.2	1.41	10.04
	Inferred	10.6	9.84	0.02	0.22	2.3	0.24	4.7	0.08	0.79
12	Measured	29.5	22.45	0.05	0.51	3.0	0.54	33.8	0.48	2.88
	Indicated	36.3	14.76	0.03	0.34	2.4	0.36	24.0	0.39	2.83
	M+I	65.8	18.21	0.04	0.41	2.7	0.44	57.8	0.88	5.72
	Inferred	1.1	12.77	0.01	0.30	1.2	0.31	0.1	0.01	0.04
14	Measured	26.6	23.50	0.05	0.54	3.1	0.57	31.0	0.46	2.68
	Indicated	17.9	16.63	0.03	0.38	2.6	0.40	12.3	0.22	1.52
	M+I	44.5	20.73	0.04	0.47	2.9	0.50	43.3	0.68	4.20
	Inferred	0.0	0.00	0.00	0.00	0.0	0.00	0.0	0.00	0.00

14.3 MINERAL RESOURCE PARAMETERS

14.3.1 Metal Prices

Table 14-6 shows the economic and recovery parameters for the Mineral Resource estimate. Metal prices for the Mineral Resource estimate are US$2.75 per pound copper, US$1,500 per ounce gold, US$18 per ounce silver and US$11 per pound moly. A conversion of US$0.75 = C$1.00 was used to convert the prices to C$. IMC believes these prices to be reasonable based on the following: 1) historical 3-year trailing averages, 2) prices used by other companies for comparable projects, and 3) long range consensus price forecasts prepared by various bank economists.

14.3.2 Cost and Recovery Estimates

Mining Cost

The base mining cost of C$1.75 per total tonne was estimated by IMC. This estimate was based on likely production rates and equipment requirements and considered typical prices for fuel, blasting agents, equipment parts, and labor, etc.

Processing of Mill Material

Mill material refers to the supergene oxide, supergene sulphide, and hypogene sulphide zones of the mineral deposit. The processing will be in a conventional sulphide flotation plant that will produce copper and moly concentrates that will be sold to commercial copper smelters and moly roasting plants. The base unit costs for processing and G&A are estimated at C$5.33 and C$0.37 per tonne, respectively, provided by M3. The estimated plant recoveries for gold, moly, and silver in the supergene and hypogene zones are shown on Table 14-6. Copper recovery is estimated at 92.2% for hypogene sulphide material. The plant recovery for supergene material is estimated as follows:

Copper recovery = 94%(Cut% - Cuw%) / Cut%

Where,

Cut% = Total copper grade
Cuw% = Weak acid soluble copper grade

The copper, gold, and silver payable percentages shown on Table 14-6 are typical terms for copper concentrates, assuming a clean concentrate with a copper concentrate grade of 28% copper or greater. The off-site cost per pound of copper is estimated at US$0.437 or C$0.583. This is based on payment for 96.5% of the copper in concentrate, smelting cost at US$ 80 per tonne, refining at US$0.80 per pound, and concentrate freight of US$133 per tonne. The moisture content was estimated at 8.0% and 0.5% concentrate loss during shipping. Gold and silver refining is estimated at US$6.00 per ounce gold and US$0.50 per ounce silver which amounts to C$8.00 and C$0.667 respectively.

Note that the off-site cost for moly is assumed to be accounted in the 85% payable percentage for molybdenum in concentrate, i.e. this is assumed to be the net payable after treatment and transportation charges. This is applicable to a clean moly concentrate with a moly grade of about 50% or greater.

Processing of Leach Material

Leach material refers to the leach capping of the mineral deposit. Processing is by crushing and heap leaching with cyanide. Gold and silver from the heap leach will report to a typical doré which will be sent to a refinery. The SART process will be used to extract copper from the cyanide solution and produce a copper concentrate that can be sold to conventional copper smelters. Heap leach mineralized material processing is estimated at C$5.09 per tonne. The G&A cost of C$0.37 per tonne is also applied to leach material.

Heap leach recoveries are estimated at 18% for copper, 66% for gold, and 26% for silver. Typical terms for refining costs are shown on Table 14-6. The C$1.733 per ounce for gold and C$0.667 for silver are based on US$1.30 and US$0.50 respectively. The payable percentage is estimated at 98% for gold and silver.

It is also assumed that the SART process will produce a copper concentrate with a grade of about 60% copper. Smelting and refining terms are assumed the same as for the flotation concentrate. This results in a smelting, refining, and freight charge of about US$0.260 per pound copper or C$0.346 per pound.

14.3.3 NSR Calculations

Due to multiple mineral products and also the variable recovery for copper in the supergene zones, NSR values, in Canadian Dollars, were calculated for each model block to use to classify blocks into potential resource and waste. For the leach material:

NSR_au = ($2000 – $1.733) x 0.66 x 0.98 x gold(g/t) / 31.103 = C$41.55 x gold (g/t)

NSR_cu = ($3.67 - $0.346) x 0.18 x 0.965 x 0.995 x copper(%) x 22.046
= C$12.65 x copper (%)

NSR_ag = ($24.00 - $0.667) x 0.26 x 0.98 x silver (g/t) / 31.103 = C$0.191 x silver (g/t)

NSR = NSR_au + NSR_cu + NSR_ag

The internal NSR cutoff for leach material is the processing + G&A cost of C$5.46 per tonne since all the recoveries and refining costs are accounted for in the NSR calculation. Internal cutoff grade applies to blocks that have to be removed from the pit, so the mining cost is a sunk cost. Internal cutoff is also generally the minimum cutoff that would be evaluated for mine scheduling. The Mineral Resource tabulation for leach material on Table 14-2 is based on the internal cutoff. The breakeven NSR cutoff grade for leach material is C$7.21 per tonne (mining plus processing and G&A).

For processing of hypogene sulphide material the NSR values are calculated as:

NSR_cu = ($3.67 – $0.583) x 0.922 x 0.965 x 0.995 x copper(%) x 22.046
= C$60.18 x copper(%)

NSR_au = ($2000 – $8.00) x 0.66 x 0.975 x 0.995 x gold(g/t) / 31.103
= C$41.01 x gold (g/t)

NSR_mo = $14.67 x 0.786 x 0.85 x 0.995 x moly(%) x 22.046 = C$214.94 x moly(%)

NSR_ag = ($24.00 - $0.667) x 0.50 x 0.95 x 0.995 x silver(g/t) / 31.103
= C$0.355 x silver(g/t)

NSR = NSR_cu + NSR_au + NSR_mo + NSR_ag

For processing of supergene material, the NSR values are calculated as:

NSR_cu = ($3.67 – $0.583) x 0.965 x 0.995 x rec_cu(%) x 22.046
= C$65.27 x rec_cu(%)

NSR_au = ($2000 – $8.00) x 0.69 x 0.975 x 0.995 x gold(g/t) / 31.103
= C$42.87 x gold (g/t)

NSR_mo = $14.67 x 0.523 x 0.85 x 0.995 x moly(%) x 22.046 = C$142.89 x moly(%)

NSR_ag = ($24.00 - $0.667) x 0.60 x 0.95 x 0.995 x silver(g/t) / 31.103
= C$0.425 x silver(g/t)

NSR = NSR_cu + NSR_au + NSR_mo + NSR_ag

where,

rec_cu = 0.94 x (Cut% – Cuw%)

The internal NSR cutoff for flotation is the processing plus G&A cost of C$5.70. Breakeven NSR cutoff is C$7.45. The stockpile re-handle cutoff grade is estimated at C$7.00 per tonne which covers processing plus G&A costs plus mining re-handle estimated at about C$1.30 per tonne.

Table 14-6: Economic Parameters for Mineral Resource (C$)

Parameter	Units	Mill Material SOX	Mill Material SUS	Mill Material HYP	Heap Leach
Commodity Prices and Exchange Rate:					
Copper Price Per Pound (US$)	(US$)	2.75	2.75	2.75	2.75
Gold Price Per Ounce (US$)	(US$)	1500.00	1500.00	1500.00	1500.00
Silver Price Per Ounce (US$)	(US$)	18.00	18.00	18.00	18.00
Molybdenum Price Per Pound (US$)	(US$)	11.00	11.00	11.00	11.00
Exchange Rate (CAD to $US)	(none)	0.75	0.75	0.75	0.75
Copper Price Per Pound (C$)	(C$)	3.67	3.67	3.67	3.67
Gold Price Per Ounce (C$)	(C$)	2000.00	2000.00	2000.00	2000.00
Silver Price Per Ounce (C$)	(C$)	24.00	24.00	24.00	24.00
Molybdenum Price Per Pound (C$)	(C$)	14.67	14.67	14.67	14.67
Mining Cost Per Total Tonne:					
Base Mining Cost	(C$)	1.750	1.750	1.750	1.750
Sustaining Capital Allowance	(C$)	0.000	0.000	0.000	0.000
Total Mining Cost	(C$)	1.750	1.750	1.750	1.750
Processing and G&A Per Ore Tonne					
Processing	(C$)	5.330	5.330	5.330	5.090
G&A	(C$)	0.370	0.370	0.370	0.370
Total Processing and G&A	(C$)	5.700	5.700	5.700	5.460
Average Plant Recoveries:					
Copper Recovery (Note 1)	(%)	61.4%	80.9%	92.2%	18.0%
Gold Recovery	(%)	69.0%	69.0%	66.0%	66.0%
Silver Recovery	(%)	60.0%	60.0%	50.0%	26.0%
Moly Recovery	(%)	52.3%	52.3%	78.6%	N.A.
Refinery Payables:					
Copper Payable	(%)	96.5%	96.5%	96.5%	96.5%
Gold Payable	(%)	97.5%	97.5%	97.5%	98.0%
Silver Payable	(%)	95.0%	95.0%	95.0%	98.0%
Molybdenum Payable	(%)	85.0%	85.0%	85.0%	N.A.
Payable Concentrate (0.5% Conc Loss)	(%)	99.5%	99.5%	99.5%	Cu Only
Offsite Costs:					
Copper SRF Cost Per Pound	(C$)	0.583	0.583	0.583	0.346
Gold Refining Per Ounce	(C$)	8.000	8.000	8.000	1.733
Silver Refining Per Ounce	(C$)	0.667	0.667	0.667	0.667
Molybdenum Freight/Treatment Per Pound	(C$)	Note 2	Note 2	Note 2	N.A.
NSR Factors:					
Copper Factor (Note 3)	(C$/t)	40.08	52.81	60.18	12.65
Gold Factor (Note 3)	(C$/t)	42.87	42.87	41.01	41.55
Silver Factor (Note 3)	(C$/t)	0.425	0.425	0.355	0.191
Moly Factor (Note 3)	(C$/t)	142.89	142.89	214.94	N.A.
Equivalency Factors:		CuEq	CuEq	CuEq	AuEq
Copper		1.00	1.00	1.00	0.304
Gold		1.070	0.812	0.681	1.00
Silver		0.0106	0.0081	0.0059	0.0046
Moly		3.565	2.706	3.572	N.A.
NSR Cutoff Grades:					
Breakeven Cutoff (C$/t)	(C$/t)	7.45	7.45	7.45	7.21
Internal Cutoff (C$/t)	(C$/t)	5.70	5.70	5.70	5.46
Stockpile Cutoff (C$/t) ($1.30 Rehandle)	(C$/t)	7.00	7.00	7.00	N.A.
Note 1: Average Recovery based on Recovery = 94% x (Cutotal – CuWAS)/(Cutotal) for SOX and SUS					
Note 2: Moly offsite costs are accounted in payable percentage					
Note 3: NSR factors are applied to model grades, copper factor for SOX and SUS is based on average recovery.					

The copper and gold equivalent grades on the tables account for all metal recoveries and smelting/refining charges. The equivalency factors shown on Table 14-6 are derived from the NSR factors as follows for hypogene sulphide mill material:

CuEq% = copper(%) + (41.01/60.18) x gold(g/t) + (214.94/60.18) x moly(%) + (0.355/60.18) x silver(g/t)
CuEq% = copper(%) + 0.681 x gold(g/t) + 3.572 x moly(%) + 0.0059 x silver (g/t)

The calculations are similar for the other material types.

14.3.4 Slope Angles

Slope angles recommendations were developed by Knight Piésold Ltd. (KP) and documented in the report “Open Pit Geotechnical Design”, dated October 12, 2012.

Forty-five-degree inter-ramp angles were recommended for most of the slope sectors. The north sectors of the main pit and west pit were recommended to be designed at 42-degree inter-ramp angles. For the small amount of overburden on the north wall the recommended angle was 27 degrees. The slope angle recommendations also specified that there be no more than 200m of vertical wall at the inter-ramp angle without an extra wide catch bench (16 m instead of 8 m).

IMC used an overall slope angle of 41 degrees in the floating cone runs to approximate overall slope angles with the KP inter-ramp angles.

14.4 ADDITIONAL INFORMATION

The Mineral Resources are classified in accordance with the May 2014 Canadian Institute of Mining, Metallurgy and Petroleum (“CIM”) “CIM Definition Standards – For Mineral Resources and Mineral Reserves” adopted by the CIM Council (as amended, the “CIM Definition Standards”) in accordance with the requirements of NI 43-101. Mineral Reserve and Mineral Resource estimates reflect the reasonable expectation that all necessary permits and approvals will be obtained and maintained.

There is no guarantee that any of the Mineral Resources will be converted to Mineral Reserve. The Inferred Mineral Resources included in this Technical Report meet the current definition of Inferred Mineral Resources. The quantity and grade of Inferred Mineral Resources are uncertain in nature and there has been insufficient exploration to define these inferred Mineral Resources as an Indicated Mineral Resource. It is, however, expected that the majority of Inferred Mineral Resource could be upgraded to Indicated Mineral Resource with continued exploration.

IMC does not believe that there are significant risks to the Mineral Resource estimates based on environmental, permitting, legal, title, taxation, socio-economic, marketing, or political factors. The Project is in a jurisdiction friendly to mining. The most significant risks to the Mineral Resource are related to economic parameters such as prices lower than forecast, recoveries lower than forecast, or costs higher than the current estimates.

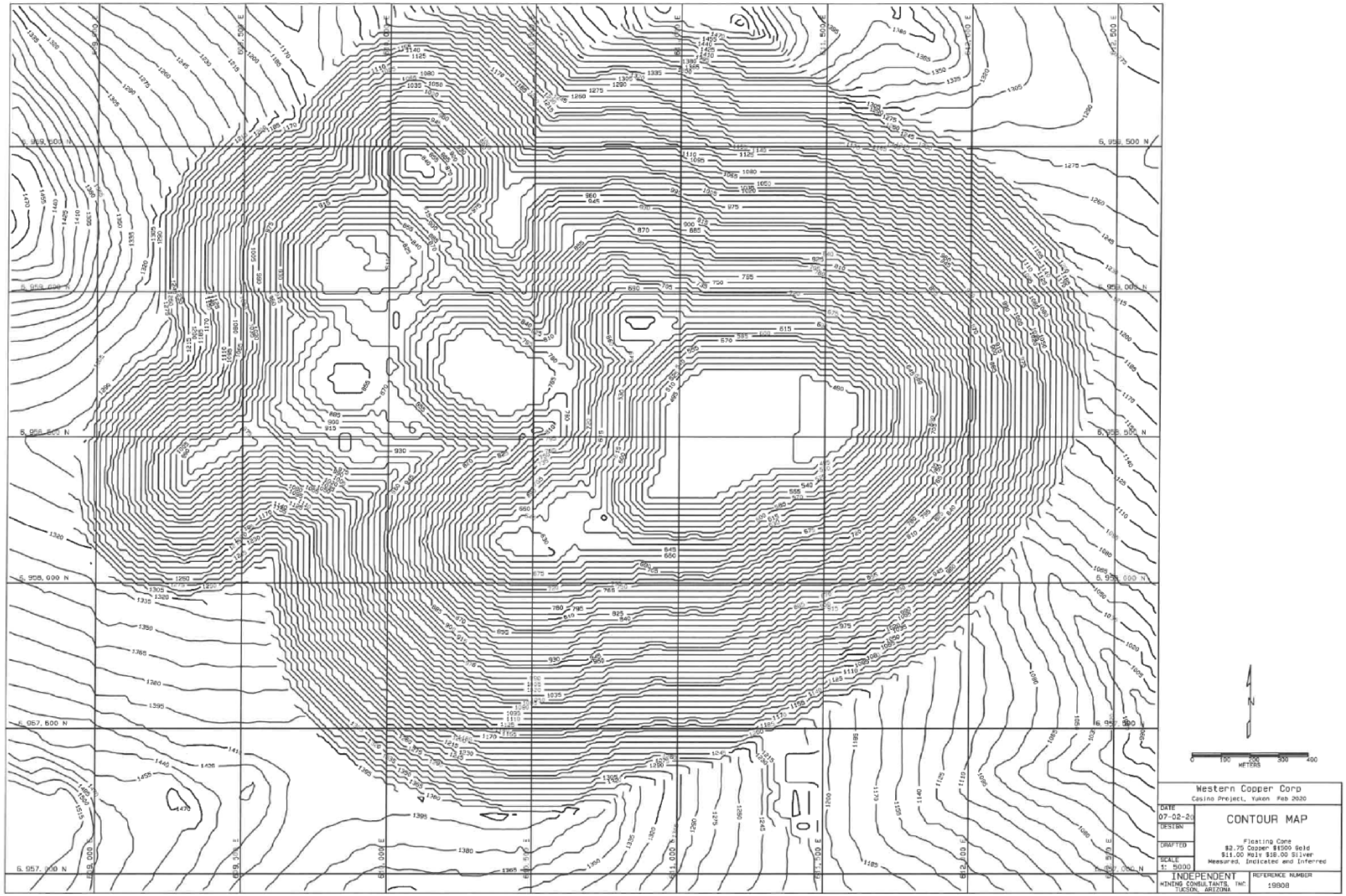


Figure 14-1: Floating Cone Shell for Mineral Resource

14.5 DESCRIPTION OF THE BLOCK MODEL

14.5.1 General

A 3D block model was developed by IMC during June 2020. The block model is based on 20 m by 20 m by 15 m high blocks. The model is not rotated. The previous resource model of record was developed by G. Giroux during 2010 and was the model used for the most recent Technical Report for the project, dated January 25, 2013.

14.5.2 Drilling Data

The drillhole database provided to IMC included 420 holes that represented 116,447 metres of drilling. Table 14-7 summarizes the drilling by date and company.

Table 14-7: Casino Drilling by Date and Company

Years	Company	No. of Holes	Metres
1992-1994	Pacific Sentinel Gold Corp.	236	73,085
2008-2012	Western Copper and Gold	112	29,775
2019	Western Copper and Gold	72	13,587
TOTAL		420	116,447

Figure 14-2 shows the hole locations and also the location of cross sections that will be presented for this report. The breakout of the data is slightly different on Figure 14-2 than the table. It is reported to IMC that the 2010 resource model was based on 305 holes and 95,655 metres of drilling. These are the holes marked in blue and termed the "historical" holes. The holes marked in red include geotechnical drilling conducted during 2011 and 2012 and also some 2010 drilling that did not make the cutoff date for the resource model. The holes in green are the new holes added to the database during 2019. It is also noted that the database includes 29 holes and 1,690 metres of drilling that are outside the model limits and not shown on Figure 14-2. These are mostly geotechnical drilling for the foundations of the tailings embankment, the plant, the leach pad and various stockpiles.

The analyses of interest for the study included total copper, weak acid soluble copper, gold, moly, and silver. Also available in the database is a complete suite of multi-element analyses. IMC's scope of work did not include a detailed review of the drilling data.

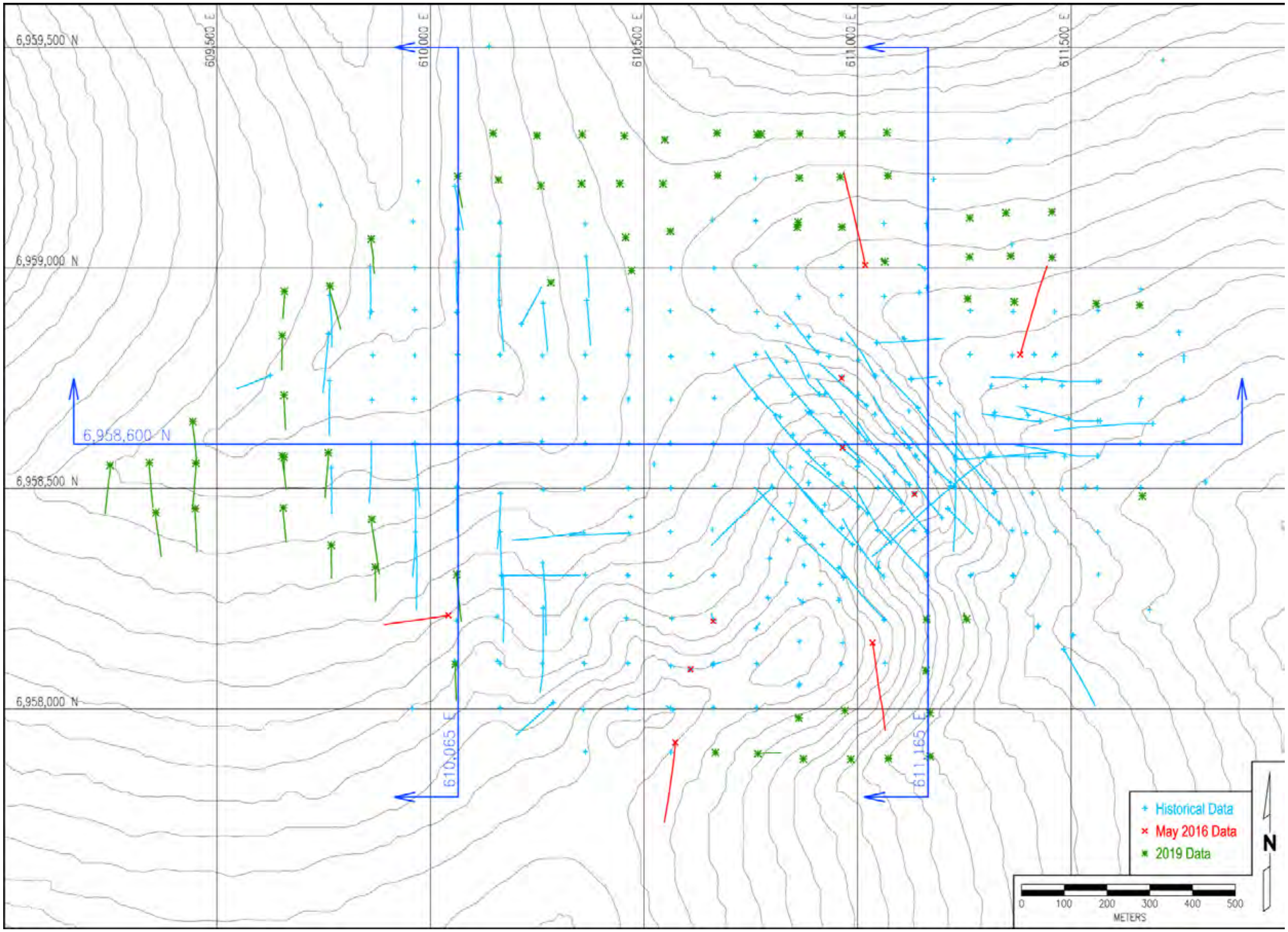


Figure 14-2: Hole Location Map (Source: IMC, 2020)

14.5.3 Geologic Controls

Oxidation Zone Types

The most important geologic control, particularly for copper mineralization, is the oxidation zones. Table 14-8 shows the zone names, codes used for modeling, and a description. The overburden is a relatively thin, highly weathered zone, near the top of current topography. There are some mineralized intervals in the overburden. The leach cap (LC) is a highly oxidized domain where the copper mineralization has largely been dissolved in acids over time and transported to the underlying supergene zones. The gold, silver, and molybdenum mineralization was not subject to the dissolution, at least to any significant degree; in particular there are significant gold values in the LC. The supergene domains have been divided into oxide dominant supergene oxide (SOX) and sulphide dominant supergene sulfide (SUS). Copper from the LC has been deposited in those zones, elevating the copper grade compared to the other domains. The hypogene sulphide (HYP) zone underlies the LC, SOX, and SUS zones. Mineralization is sulphidic in nature; percent of oxidation is very low, typically less than 10%.

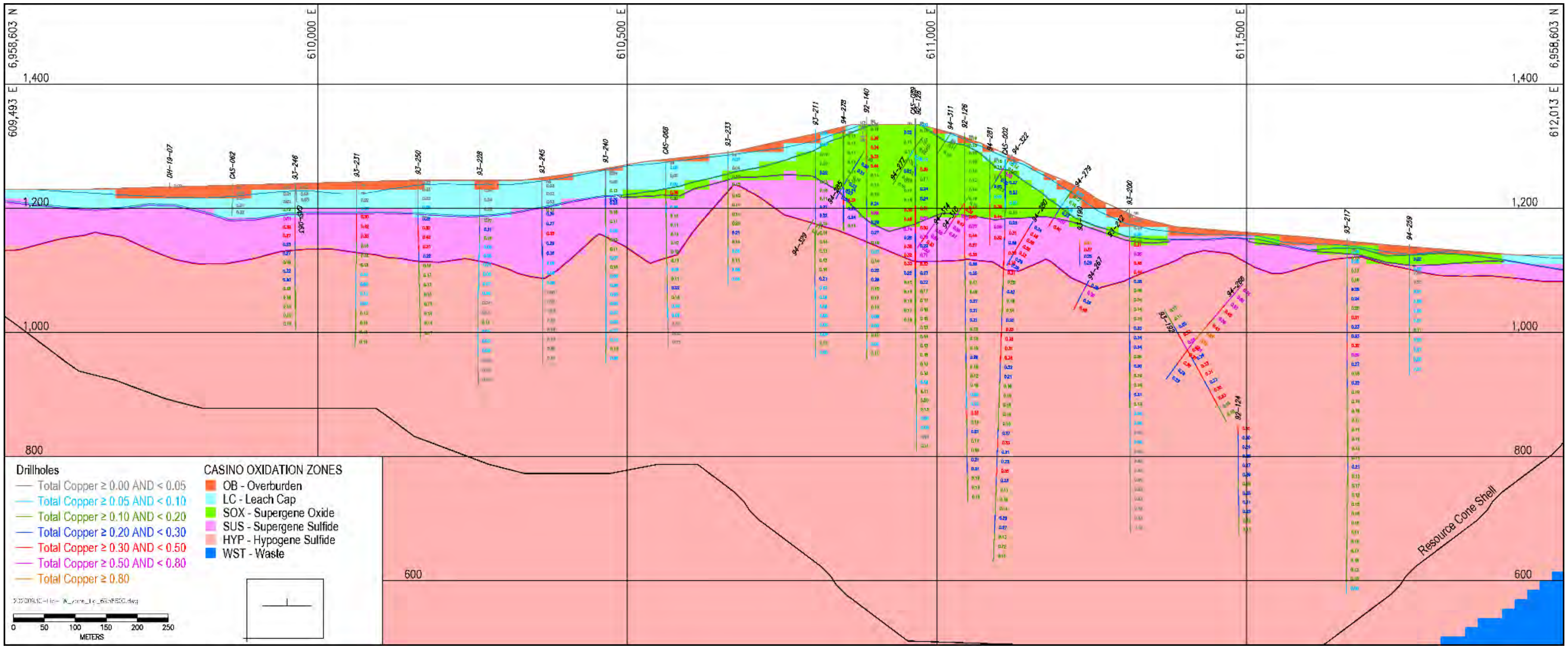
Western personnel provided IMC with solids to represent the LC, SOX, SUS, and HYP domains. IMC used these solids to assign oxidation zone types to model blocks. Code 6, waste, was used to denote blocks outside the provided solids. A surface was provided to denote the bottom of overburden. IMC assigned blocks above the leach cap as overburden.

Table 14-8: Oxidation Zone Types

Zone	Code	Description
OVB	1	Overburden
LC	2	Leach Cap
SOX	3	Supergene Oxide
SUS	4	Supergene Sulphide
HYP	5	Hypogene Sulphide
WST	6	Waste – Peripheral to Above Solids

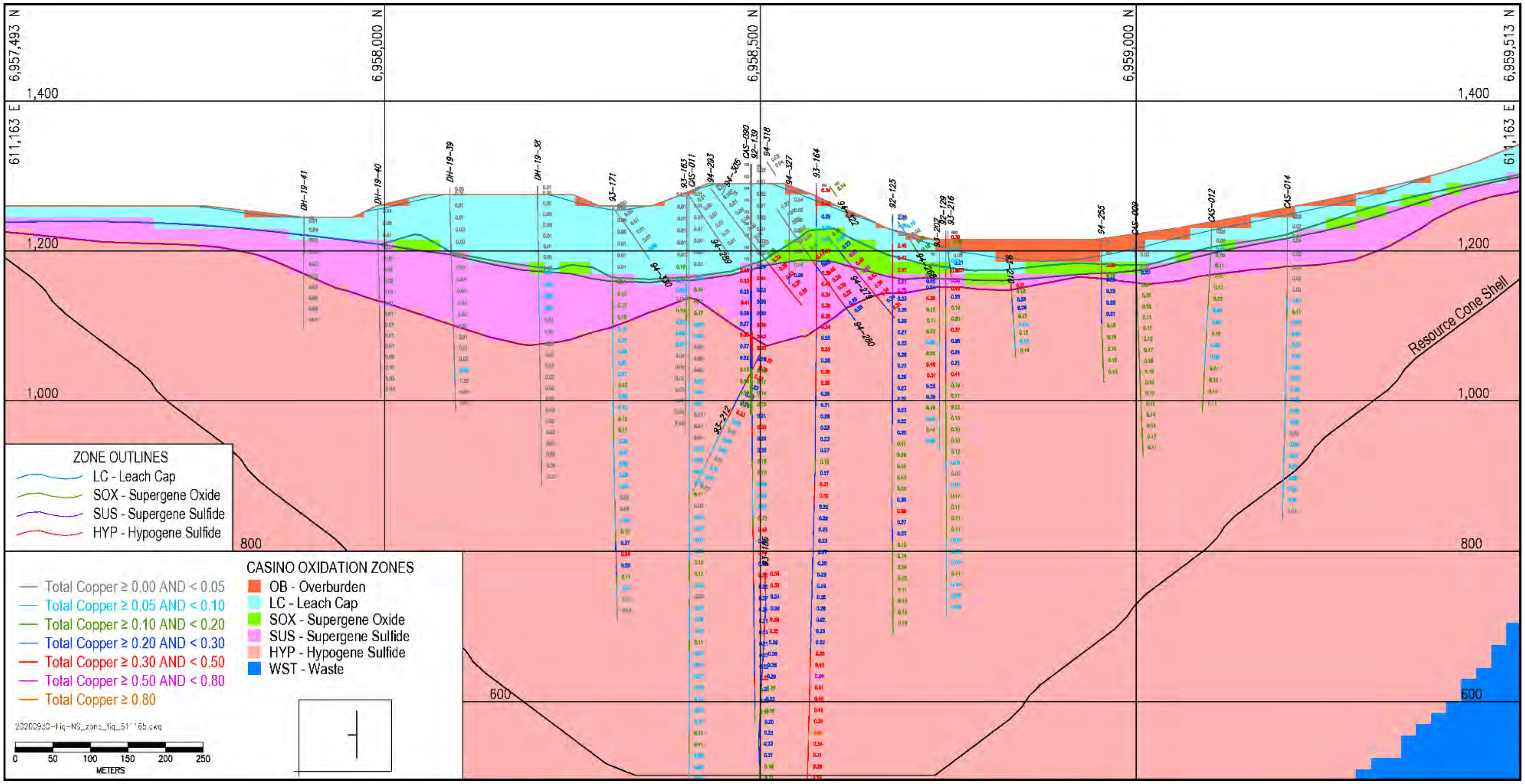
IMC also used the solids to back-assign the oxide domain codes to the assay database. It is noted that the assay database did include an oxide domain assignment from logging, but IMC used the back-assigned values for modeling so assay intervals would be consistent with the domains they are located.

Figure 14-3 and Figure 14-4 show the oxide zones on east-west and north-south cross sections, respectively. It can be seen that most of the Mineral Resource is in hypogene sulphide material.



(Source: IMC, 2020)

Figure 14-3: Oxidation Domains on East-West Section 6,958,600N



(Source: IMC, 2020)

Figure 14-4: Oxidation Domains on North-South Section 611,165E

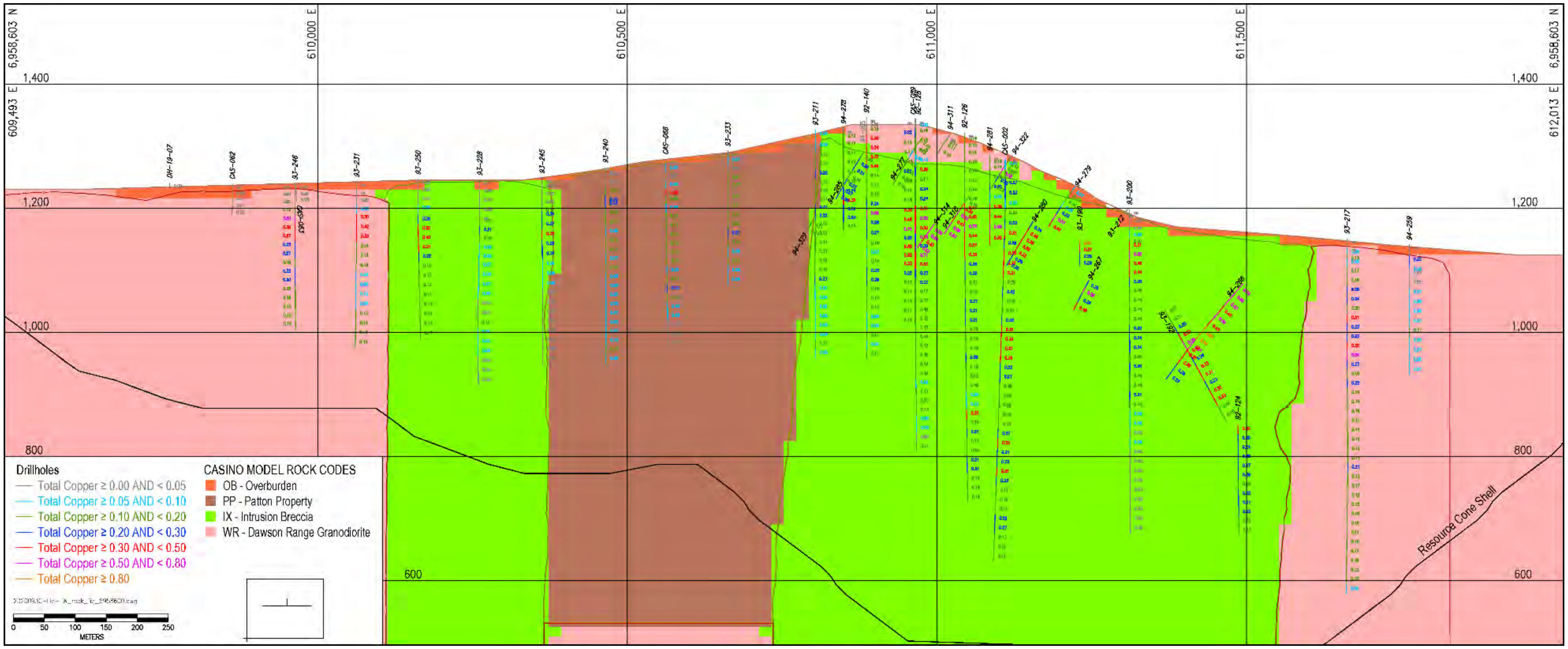
Rock Types

Rock type interpretations for four major rock types plus the overburden have been developed as 3D solids or a surface for the overburden. Table 14-9 shows the rock types. Figure 14-5 shows the rock types on east-west cross section 6,958,600N. It can be seen that the main host rock is the Dawson Range Granodiorite which has been intruded by the Intrusion Breccia and the Patton Porphyry. The third intrusion, the Post Mineral Explosive Breccia (MX) to the southwest of the pit, is post mineral in character.

IMC used the solids to assign rock codes to the model blocks. Rock codes were also assigned to the assay database by back-assignment from the solids. Note that there were rock type designations in the assay database, but the back-assigned values were used for the resource model so the assay assignments would be consistent with the block they were located.

Table 14-9: Model Rock Types

Rock	Code	Description
OVB	1	Overburden
PP	2	Patton Porphyry
IX	3	Intrusion Breccia
WR	4	Dawson Range Granodiorite
MX	5	Post Mineral Explosive Breccia



(Source: IMC, 2020)

Figure 14-5: Rock Types on East-West Section 6,958,600N

14.5.4 Cap Grades and Compositing

IMC reviewed the database to determine cap grades for the various minerals. The distribution of the length of sample intervals, when copper is assayed, is approximately as follows:

- About 24% are less than 3 m in length,
- About 69% are 3 m or 3.05 m (10 US ft), and
- About 7% are longer than 3.05 m.

IMC considers that a relatively consistent 3 m sample interval was used for the drilling and that cap grades may reasonably be applied to the assays.

IMC examined probability plots and sorted lists of the higher-grade assay intervals for copper, gold, moly, and silver by oxidation zones to determine cap grades. Table 14-10 shows the cap grades in the upper portion of the table and number of assays capped in the lower portion of the table. It can be seen that relatively small numbers of assays were capped for each metal in each population. The cap grades generally correspond to the upper 99.8 to 99.9 percentile of the populations.

The assay database was composited to nominal 7.5 m downhole composites, respecting the oxidation zones. It is noted this is one-half of the 15 m bench height used for the model. The smaller composite length allows capturing some of the narrowing zones and also tends to result in less grade smoothing during block grade estimation. Composited values included the total copper, weak acid soluble copper, gold, moly, and silver assays, the soluble copper to total copper ratio, and the rock type and oxidation zone codes.

The interpretation of nominal 7.5 m composites is described next. As noted, the composites do not cross oxidation zone boundaries. Composites within a zone are divided into equal length composites as close as possible to the target length. For example, a 28 m zone of supergene sulfide is composited into four 7 m composites. With this algorithm 93% of the composites are between 7 m and 8 m in length and 97.4% of the composites are between 6.5 m and 8.5 m in length; IMC does not consider the slight difference in the lengths of the composite's material for grade estimation purposes.

Table 14-10: Cap Grades and Number of Assays Capped

Metal	Units	OB	LC	SOX	SUS	HYP	WST
Copper	(%)	none	0.70	1.60	2.00	1.70	none
Gold	(g/t)	none	2.00	2.10	3.20	3.75	1.50
Moly	(%)	none	0.20	0.17	0.70	0.26	0.10
Silver	(g/t)	none	35.0	25.0	25.0	95.0	33.0
Number of Assays Capped							
Metal	Units	OB	LC	SOX	SUS	HYP	WST
Copper	(none)	0	5	5	3	12	0
Gold	(none)	0	8	8	7	11	4
Moly	(none)	0	9	8	6	9	1
Silver	(none)	0	11	8	5	12	4

14.5.5 Descriptive Statistics

Table 14-11 shows descriptive statistics for total copper, gold, moly and silver for the assay intervals. The table shows values by the oxidation zones. The left side of the table shows uncapped values and the right side shows capped values. For copper it can be seen that values in the overburden and leach cap are very low, values in the SOX and SUS are somewhat elevated, and values in the hypogene tend to be lower than the supergene. Gold, moly, and silver do not have the corresponding depletion of values in the leach cap. Mean gold and moly grades are slightly elevated in the SOX compared to SUS. The table includes only non-zero values for each population, though many have placeholders for below detection limit values.

Table 14-12 shows descriptive statistics for the 7.5 m composites. The table includes only non-zero values, but zero value assays are incorporated into the composites.

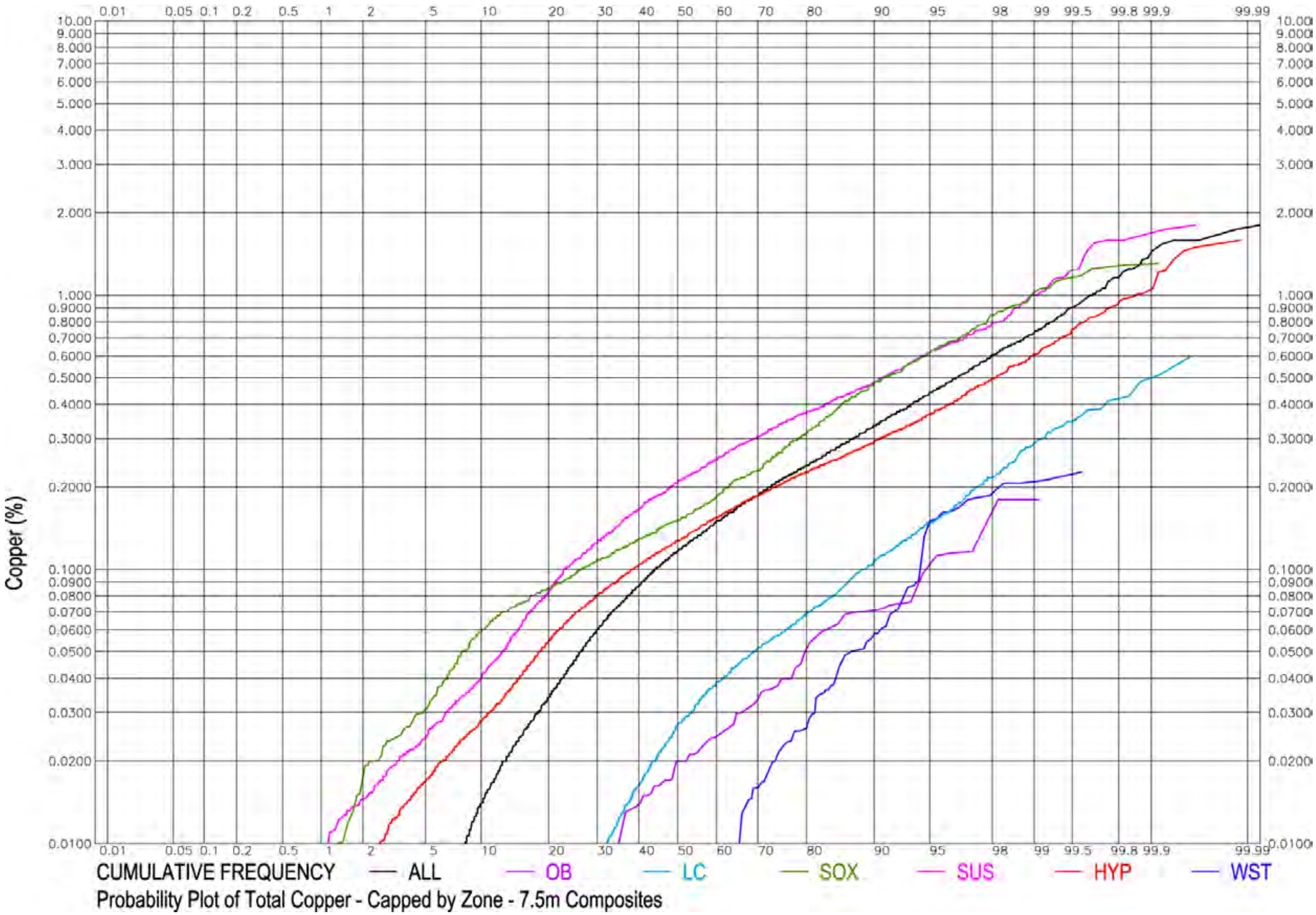
Figure 14-6 shows a probability plot of total copper grades for the composites for the various oxidation type domains. Figure 14-7 shows the probably plot for gold.

Table 14-11: Summary Statistics of Assays

Metal/Domain	Not Capped					Capped				
Metal/Domain	**No. of Samples**	**Mean (%)**	**Std Dev (%)**	**Max (%)**	**Min (%)**	**No. of Samples**	**Mean (%)**	**Std Dev (%)**	**Max (%)**	**Min (%)**
Copper:										
All Samples	38,968	0.154	0.175	5.63	0.000	38,968	0.154	0.170	2.00	0.000
Overburden	343	0.035	0.048	0.34	0.001	343	0.035	0.048	0.34	0.001
Leach Cap	6,246	0.045	0.068	1.36	0.000	6,246	0.045	0.065	0.70	0.000
Supergene Oxide	3,088	0.216	0.226	2.90	0.001	3,088	0.215	0.220	1.60	0.001
Supergene Sulfide	7,308	0.245	0.228	2.70	0.000	7,308	0.245	0.226	2.00	0.000
Hypogene	21,333	0.152	0.148	5.63	0.000	21,333	0.151	0.138	1.70	0.000
Waste	650	0.022	0.046	0.41	0.000	650	0.022	0.046	0.41	0.000
Metal/Domain	**No. of Samples**	**Mean (g/t)**	**Std Dev (g/t)**	**Max (g/t)**	**Min (g/t)**	**No. of Samples**	**Mean (g/t)**	**Std Dev (g/t)**	**Max (g/t)**	**Min (g/t)**
Gold:										
All Samples	38,744	0.233	0.696	99.96	0.003	38,744	0.227	0.246	3.75	0.003
Overburden	343	0.203	0.237	1.85	0.011	343	0.203	0.237	1.85	0.011
Leach Cap	6,238	0.293	1.290	99.96	0.003	6,238	0.277	0.265	2.00	0.003
Supergene Oxide	3,088	0.380	0.329	2.64	0.003	3,088	0.380	0.326	2.10	0.003
Supergene Sulfide	7,305	0.248	0.356	18.79	0.003	7,305	0.244	0.259	3.20	0.003
Hypogene	21,206	0.194	0.572	55.10	0.003	21,206	0.188	0.208	3.75	0.003
Waste	564	0.089	0.256	3.29	0.003	564	0.082	0.194	1.50	0.003
Metal/Domain	**No. of Samples**	**Mean (%)**	**Std Dev (%)**	**Max (%)**	**Min (%)**	**No. of Samples**	**Mean (%)**	**Std Dev (%)**	**Max (%)**	**Min (%)**
Moly:										
All Samples	38,588	0.0175	0.0293	1.240	0.0001	38,588	0.0174	0.0274	0.700	0.0001
Overburden	343	0.0124	0.0167	0.110	0.0001	343	0.0124	0.0167	0.110	0.0001
Leach Cap	6,230	0.0157	0.0224	0.363	0.0001	6,230	0.0156	0.0210	0.200	0.0001
Supergene Oxide	3,086	0.0218	0.0285	0.815	0.0001	3,086	0.0214	0.0231	0.170	0.0001
Supergene Sulfide	7,302	0.0190	0.0450	1.240	0.0001	7,302	0.0188	0.0420	0.700	0.0001
Hypogene	21,113	0.0174	0.0242	0.707	0.0001	21,113	0.0173	0.0233	0.260	0.0001
Waste	514	0.0035	0.0108	0.157	0.0001	514	0.0034	0.0095	0.100	0.0001
Metal/Domain	**No. of Samples**	**Mean (g/t)**	**Std Dev (g/t)**	**Max (g/t)**	**Min (g/t)**	**No. of Samples**	**Mean (g/t)**	**Std Dev (g/t)**	**Max (g/t)**	**Min (g/t)**
Silver:										
All Samples	38,552	1.92	51.09	9999.9	0.10	38,552	1.63	3.18	95.0	0.10
Overburden	344	1.25	1.47	18.0	0.10	344	1.25	1.47	18.0	0.10
Leach Cap	6,235	2.03	4.84	200.0	0.10	6,235	1.94	2.47	35.0	0.10
Supergene Oxide	3,086	1.96	2.91	70.2	0.10	3,086	1.91	2.20	25.0	0.10
Supergene Sulfide	7,298	1.63	2.31	116.0	0.10	7,298	1.61	1.79	25.0	0.10
Hypogene	21,060	1.99	69.02	9999.9	0.10	21,060	1.51	3.79	95.0	0.10
Waste	529	1.91	13.57	290.0	0.10	529	1.32	4.16	33.0	0.10

Table 14-12:Summary Statistics of 7.5m Composites

	Not Capped					Capped				
Metal/Domain	No. of Samples	Mean (%)	Std Dev (%)	Max (%)	Min (%)	No. of Samples	Mean (%)	Std Dev (%)	Max (%)	Min (%)
Copper:										
All Samples	14,910	0.155	0.162	3.70	0.000	14,910	0.155	0.157	1.94	0.000
Overburden	109	0.033	0.042	0.30	0.001	109	0.033	0.042	0.30	0.001
Leach Cap	2,425	0.045	0.060	0.82	0.001	2,425	0.045	0.058	0.63	0.001
Supergene Oxide	1,163	0.219	0.201	1.61	0.002	1,163	0.218	0.198	1.35	0.002
Supergene Sulfide	2,738	0.249	0.209	2.26	0.000	2,738	0.249	0.207	1.94	0.000
Hypogene	8,237	0.153	0.135	3.70	0.000	8,237	0.153	0.127	1.59	0.000
Waste	238	0.024	0.047	0.33	0.000	238	0.024	0.047	0.33	0.000
Metal/Domain	No. of Samples	Mean (g/t)	Std Dev (g/t)	Max (g/t)	Min (g/t)	No. of Samples	Mean (g/t)	Std Dev (g/t)	Max (g/t)	Min (g/t)
Gold:										
All Samples	14,879	0.233	0.348	24.24	0.000	14,879	0.228	0.213	3.16	0.000
Overburden	109	0.197	0.232	1.41	0.015	109	0.197	0.232	1.41	0.015
Leach Cap	2,424	0.286	0.541	24.24	0.003	2,424	0.276	0.236	1.74	0.003
Supergene Oxide	1,163	0.385	0.296	1.99	0.005	1,163	0.384	0.294	1.99	0.005
Supergene Sulfide	2,737	0.253	0.286	8.80	0.011	2,737	0.249	0.227	2.53	0.011
Hypogene	8,216	0.194	0.292	14.41	0.000	8,216	0.189	0.168	3.16	0.000
Waste	230	0.079	0.180	1.64	0.001	230	0.072	0.139	1.00	0.001
Metal/Domain	No. of Samples	Mean (%)	Std Dev (%)	Max (%)	Min (%)	No. of Samples	Mean (%)	Std Dev (%)	Max (%)	Min (%)
Moly:										
All Samples	14,833	0.0177	0.0257	0.715	0.0000	14,833	0.0176	0.0247	0.630	0.0000
Overburden	109	0.0131	0.0177	0.078	0.0001	109	0.0131	0.0177	0.078	0.0001
Leach Cap	2,422	0.0156	0.0207	0.316	0.0001	2,422	0.0154	0.0194	0.200	0.0001
Supergene Oxide	1,163	0.0219	0.0234	0.349	0.0001	1,163	0.0215	0.0205	0.160	0.0001
Supergene Sulfide	2,737	0.0195	0.0403	0.715	0.0001	2,737	0.0193	0.0388	0.630	0.0001
Hypogene	8,190	0.0176	0.0208	0.317	0.0001	8,190	0.0176	0.0203	0.237	0.0001
Waste	212	0.0031	0.0074	0.066	0.0000	212	0.0031	0.0069	0.053	0.0000
Metal/Domain	No. of Samples	Mean (g/t)	Std Dev (g/t)	Max (g/t)	Min (g/t)	No. of Samples	Mean (g/t)	Std Dev (g/t)	Max (g/t)	Min (g/t)
Silver:										
All Samples	14,807	1.78	16.40	1962.5	0.00	14,807	1.62	2.38	91.8	0.00
Overburden	109	1.31	1.39	9.8	0.20	109	1.31	1.39	9.8	0.20
Leach Cap	2,424	2.03	4.25	175.7	0.10	2,424	1.94	2.16	35.0	0.10
Supergene Oxide	1,163	1.97	2.26	35.2	0.10	1,163	1.92	1.79	18.0	0.10
Supergene Sulfide	2,734	1.64	1.62	40.7	0.10	2,734	1.62	1.40	17.8	0.10
Hypogene	8,161	1.75	21.93	1962.5	0.10	8,161	1.50	2.76	91.8	0.10
Waste	216	1.12	3.19	29.3	0.00	216	0.92	1.83	13.6	0.00



(Source: IMC, 2020)

Figure 14-6: Probability Plot of Total Copper Composites by Oxidation Type

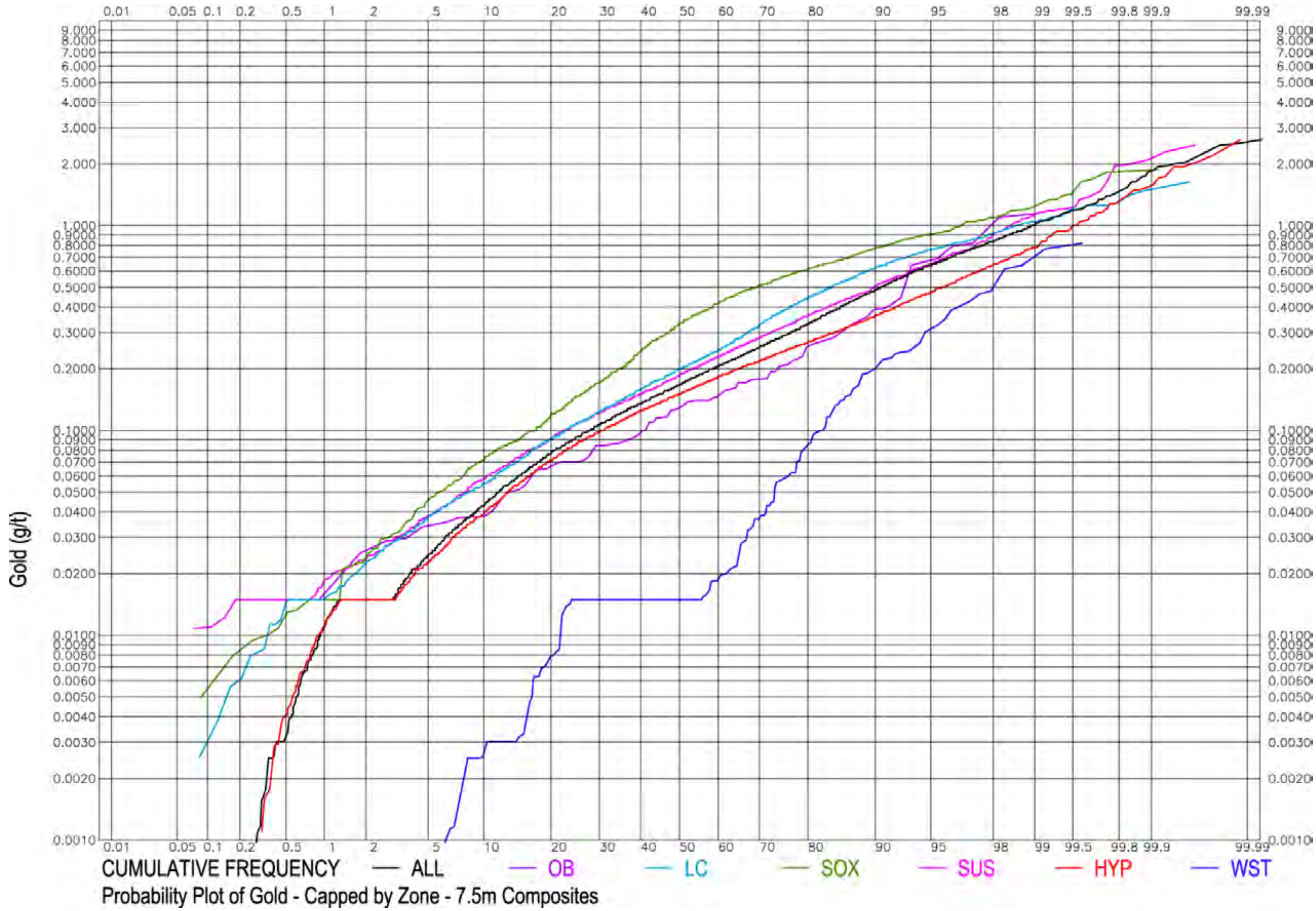


Figure 14-7: Probability Plot of Gold Composites by Oxidation Type

14.5.6 Variogram Analysis

Copper

IMC conducted variogram analyses of total copper by oxidation type domains. The analysis was based on the 7.5m composites. The leach cap, supergene oxide, and supergene sulphide domains are relatively flat lying and the distribution of copper mineralization appears to not vary much by direction. Figure 14-8 and Figure 14-9 show variograms for supergene oxide and supergene sulphide respectively. These variograms are calculated as the average of all horizontal directions which is consistent with the relatively flat lying mineralization in these domains. The ranges of the first variogram structures are 172 m for supergene oxide and 263 m for supergene sulphide. The variograms are of good clarity.

For the hypogene sulphide, IMC ran variograms in many directions. The various directional variograms tended to be similar, indicating a somewhat isotropic distribution of copper mineralization. Figure 14-10 shows the variogram for hypogene sulphide copper calculated as the average in all directions. The variogram has good clarity and the range of the first structure is 293 m. Geologic inference suggests that the range of influence should be slightly longer in the east-west direction than the north-south direction. This is indicated in the variograms. Figure 14-11 and Figure 14-12 show variograms for hypogene sulphide in the east-west and north-south directions respectively.

The variograms were calculated with the pairwise relative variogram method. The variogram values shown on the graphs would be multiplied by the mean squared to convert them to % total copper units.

Gold, Moly and Silver

Variograms were also calculated for gold, moly and silver. For these metals there no evidence of significant grade changes across oxidation domain boundaries, so the calculations combined the data for all zones. As with copper, mineralization tends to be somewhat isotropic. Figure 14-13 shows the variogram for gold. The variogram has good clarity and the range of influence of the first structure is over 200 m. Though not shown, the moly and silver variograms are also of good clarity and reasonably long ranges.

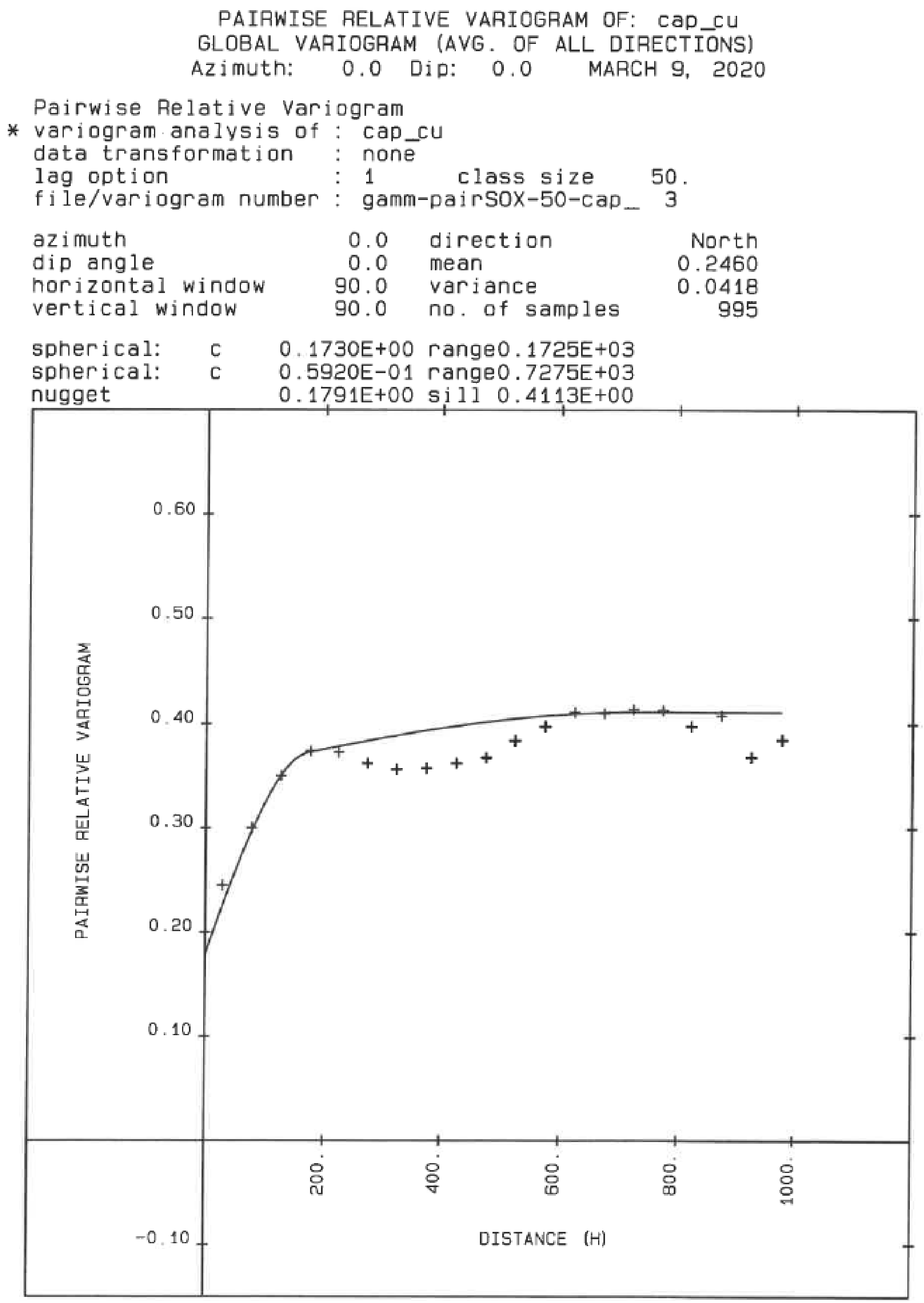


Figure 14-8: Total Copper Variogram – Supergene Oxide

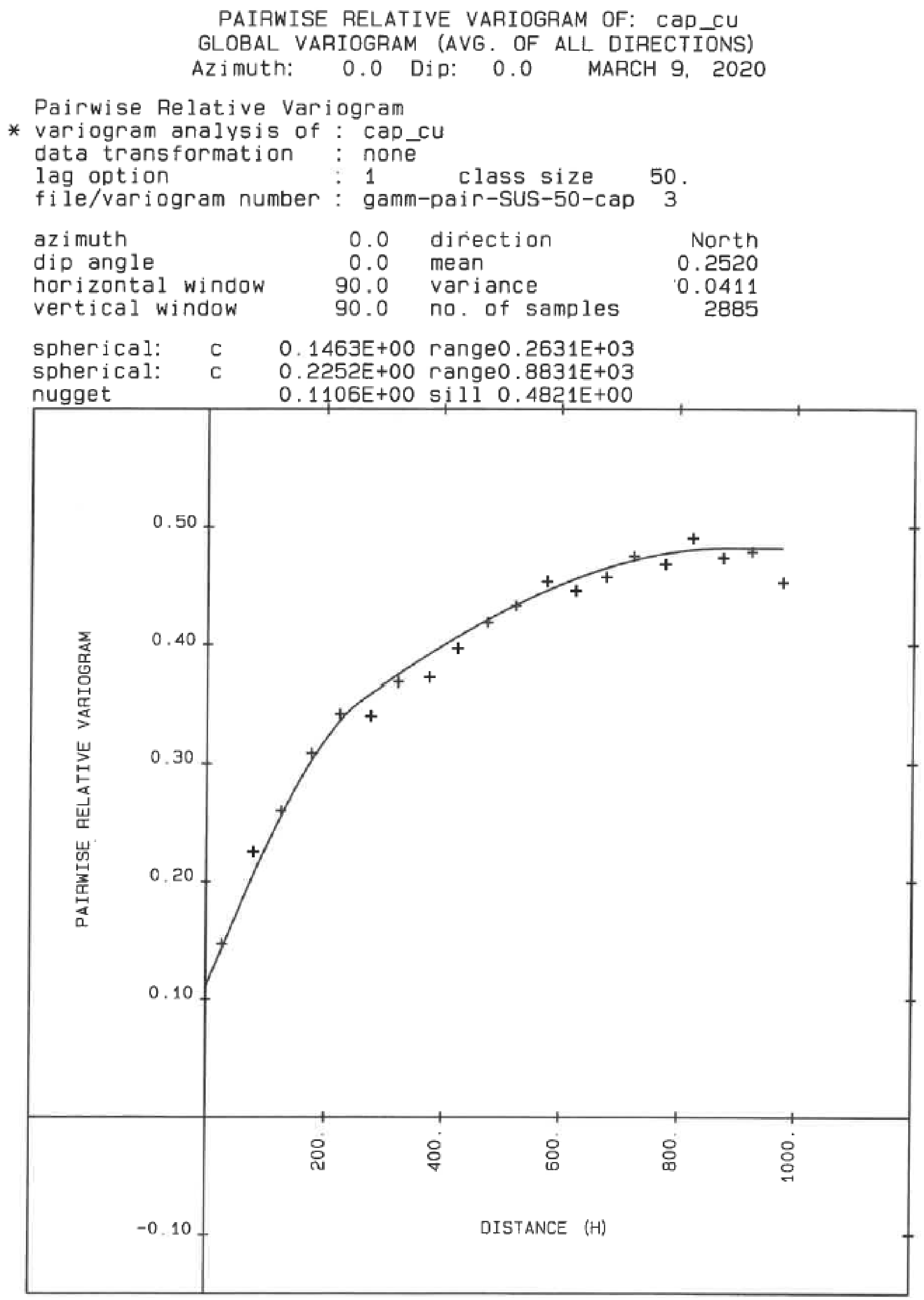


Figure 14-9: Total Copper Variogram – Supergene Sulphide

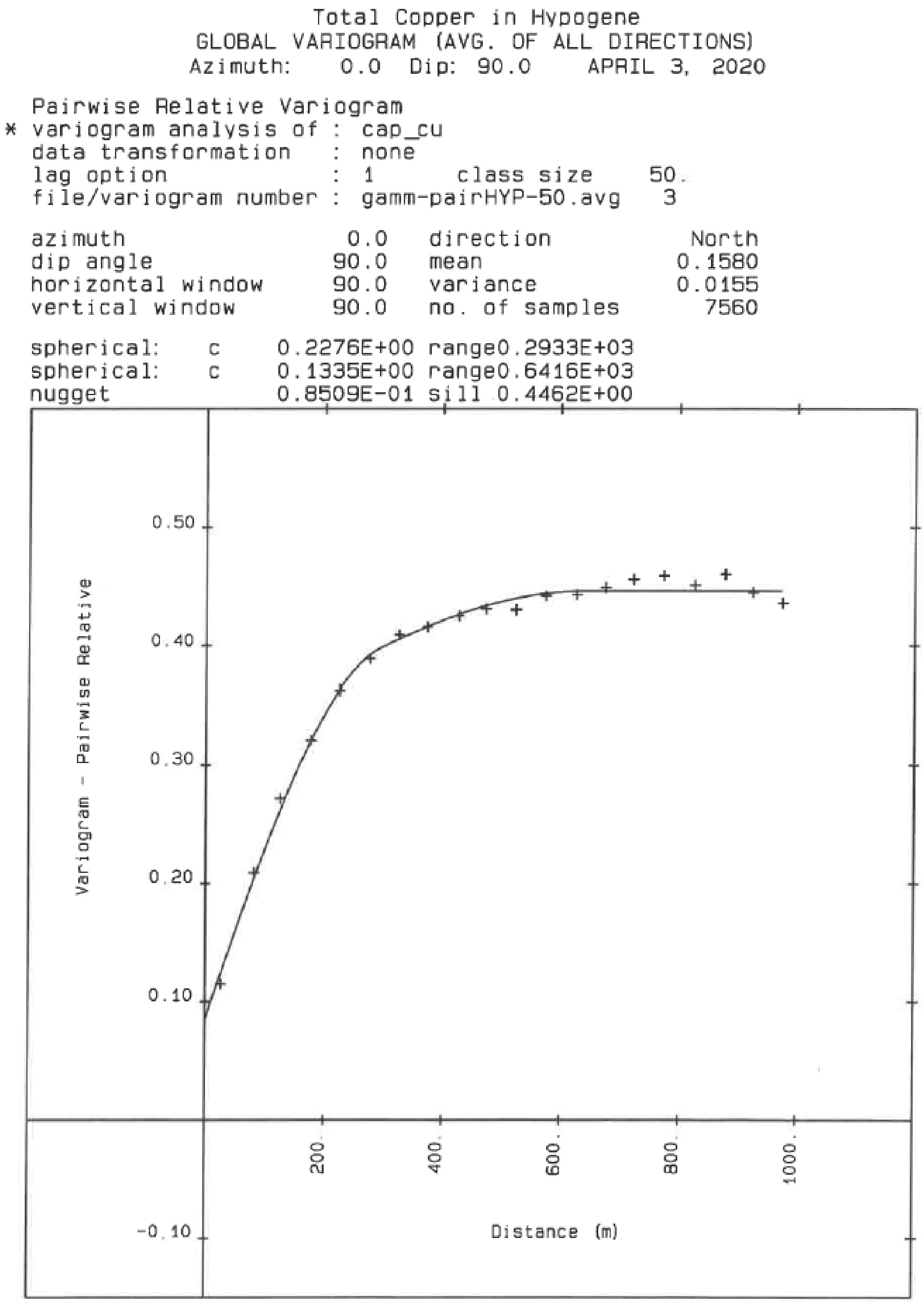


Figure 14-10: Total Copper Variogram – Hypogene Sulphide - Global

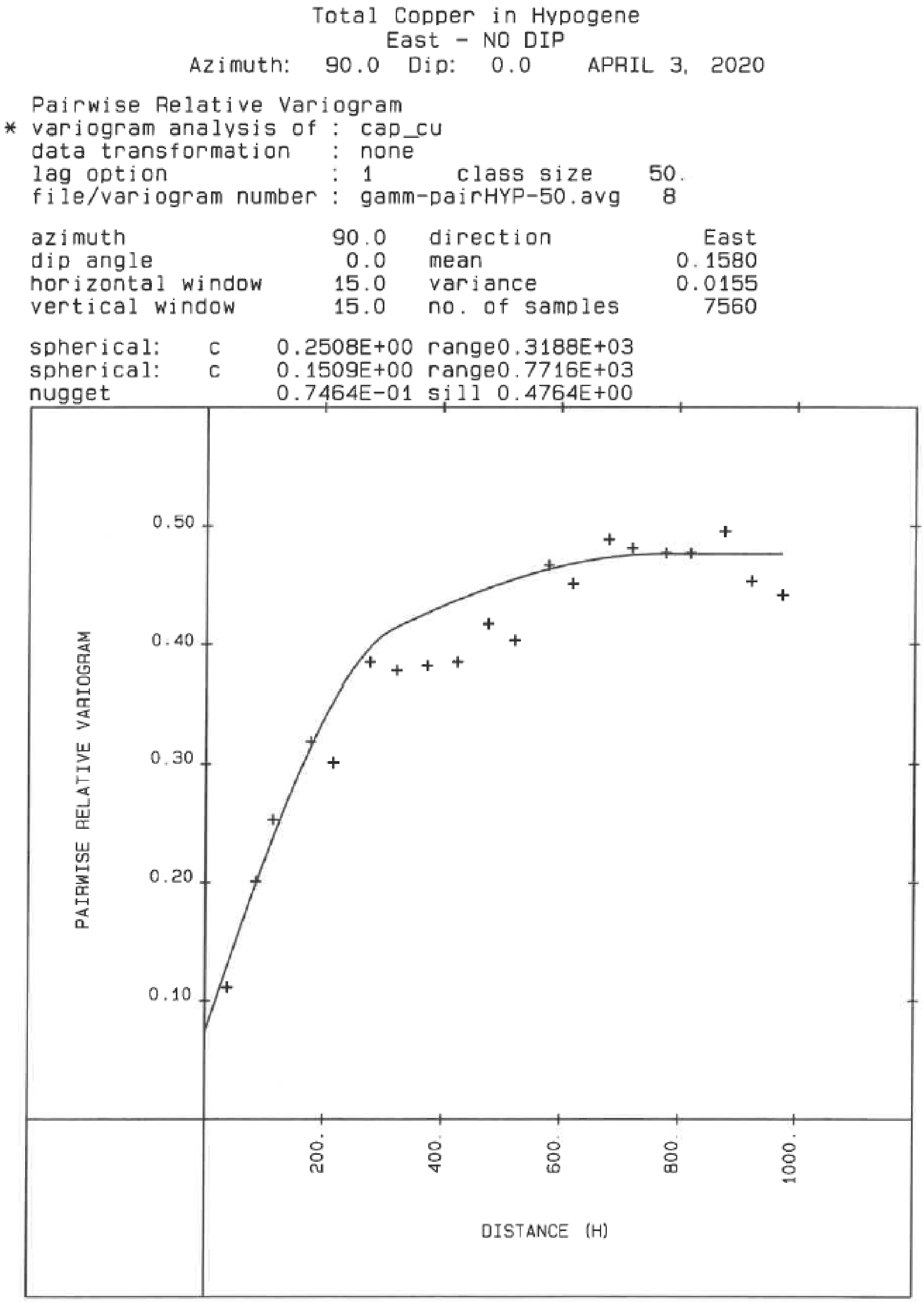


Figure 14-11: Total Copper Variogram – Hypogene Sulphide – East-West

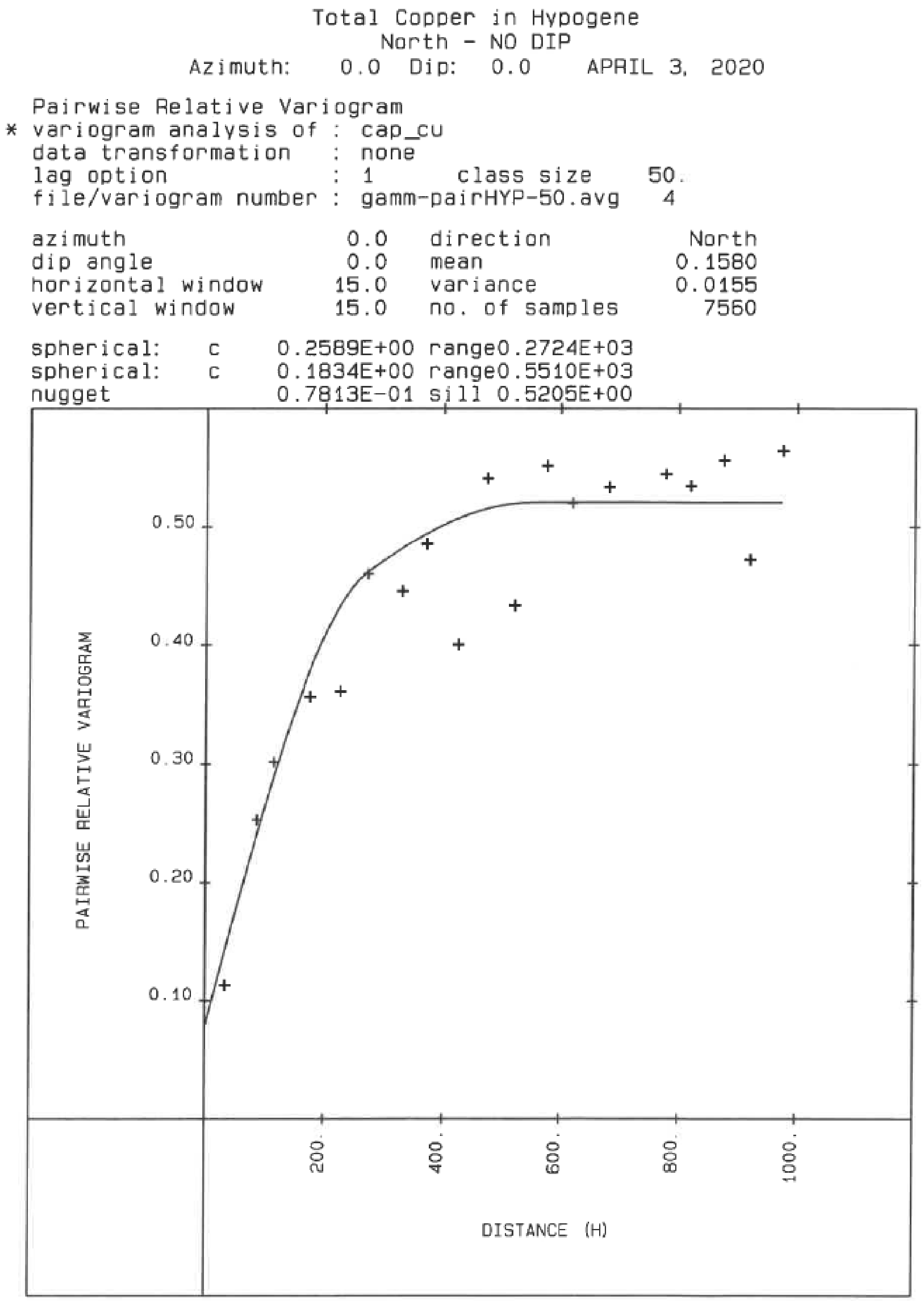


Figure 14-12: Total Copper Variogram – Hypogene Sulphide – North-South

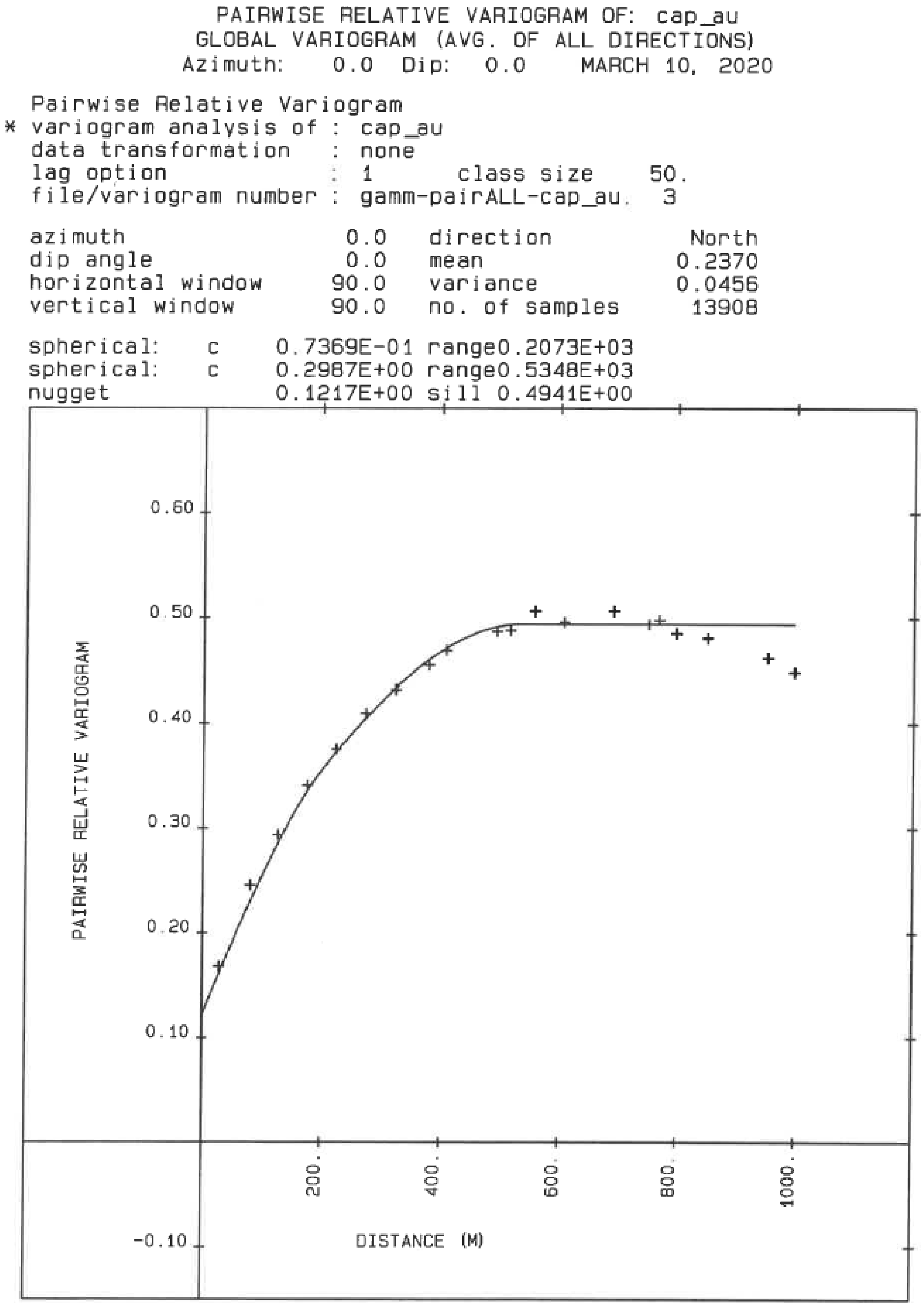


Figure 14-13: Gold Variogram

14.5.7 Block Grade Estimation

Block grades for total copper, weak acid soluble copper, gold, moly and silver were estimated with inverse distance with a power weight of 2 (ID2). The ID2 method was chosen because it generally results in less grade smoothing (smearing) than ordinary kriging (OK). Estimates were also done by OK, inverse distance with a power weight of 3 (ID3), and nearest neighbour (NN) for comparison purposes. The ID2, OK, and ID3 estimates were done with the 7.5 m composites. 15 m composites were used for the NN estimate.

Total and Soluble Copper

The leach cap, supergene oxide, supergene sulphide, and hypogene sulphide oxidation type boundaries were all considered hard boundaries for the estimation of total copper and weak acid soluble copper. The waste domain was not estimated; it is well outside the drilling data. Grades were also not estimated for overburden blocks. For the 2010 model supergene oxide and supergene sulphide were lumped into a single domain for total and soluble copper based on consideration of total copper grades. IMC believes this assumption was reasonable for total copper, but there are differences in soluble copper in the domains that indicate they should not be combined.

In terms of rock types, the Post Mineral Explosive Breccia was considered a separate domain, but the Patton Porphyry, Intrusion Breccia, and Dawson Range Granodiorite were combined into a single population. This was also the assumption for the 2010 work, and IMC agrees with it.

For leach cap, supergene oxide, and supergene sulphide the search radii for the estimations were 200 m (circular) in the horizontal direction and 30 m in the vertical direction. These search radii are well within the variogram ranges and are adequate to fill in the block grades. A maximum of 15 composites, a minimum of one composite, and a maximum of three composites per hole were used.

For hypogene sulphide the search radii were 220 m in the east-west direction, 180 m in the north-south direction, and 100 m in the vertical direction. A maximum of 24 composites, a minimum of two composites, and a maximum of six composites per hole were used.

Figure 14-14 and Figure 14-15 show copper grades on an east-west and north-south cross section respectively.

Soluble copper block grade estimates were also conducted for the leach cap, supergene oxide, and supergene sulphide domains. Soluble copper estimates were not done for hypogene sulphide. Soluble copper assays were generally not done for hypogene material. There were some slight adjustments to the database for the soluble copper estimates. There were 97 assay intervals where soluble copper exceeded total copper; these were capped at the total copper grade. After estimation there were 213 blocks with the soluble copper estimate higher than total copper; these were capped at the total copper grade.

Gold, Moly, and Silver

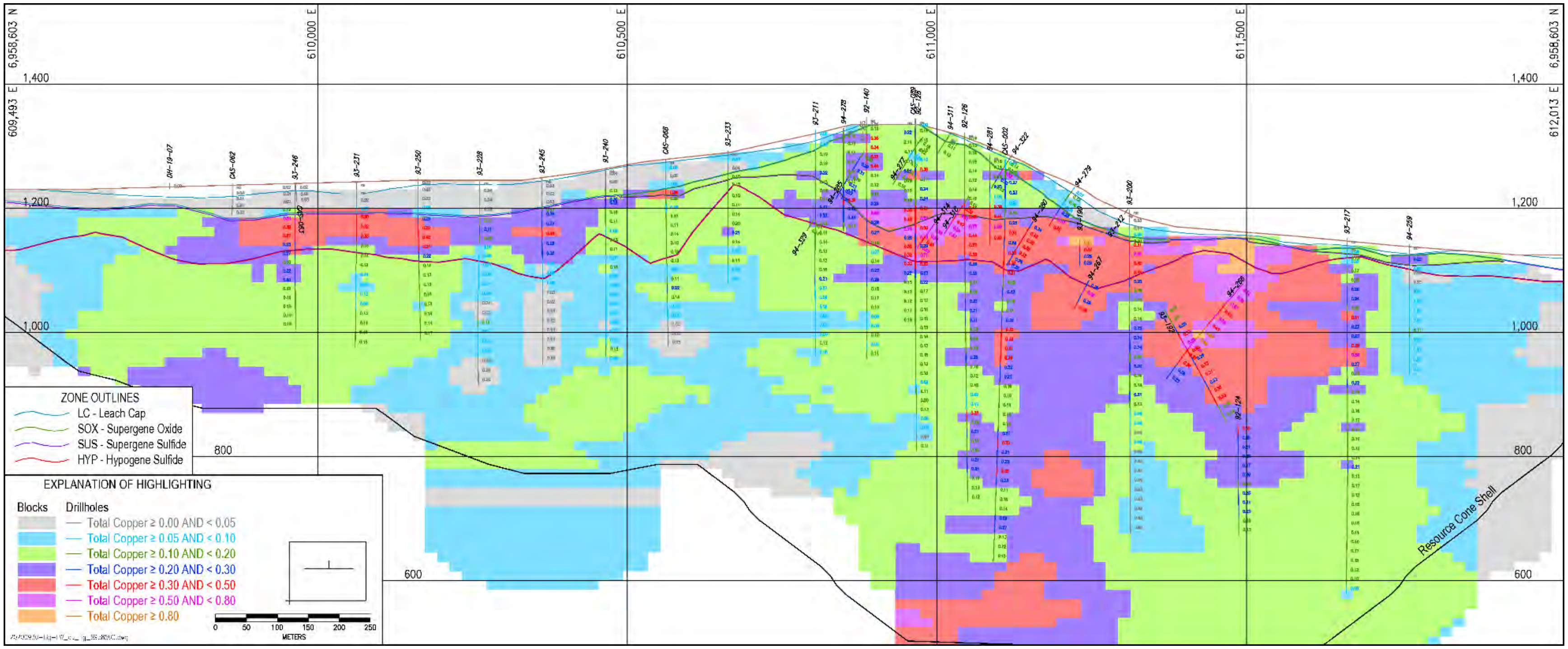
For gold, moly and silver the oxidation type boundaries were not considered hard boundaries for estimation. There is no evidence of significant changes in grade across the boundaries. The rock type populations were the same as for copper, the Post Mineral Explosive Breccia was considered a separate domain from the other rock types.

Search radii parameters were the same as for copper. For leach cap, supergene oxide, and supergene sulphide the search radii were 200 m (circular) in the horizontal direction and 30 m in the vertical direction. A maximum of 15 composites, a minimum of one composite, and a maximum of three composites per hole were used. For hypogene sulphide the search radii were 220 m in the east-west direction, 180 m in the north-south direction, and 100 m in the vertical direction and maximum of 24 composites, a minimum of two composite, and a maximum of six composites per hole were used.

Figure 14-16 and Figure 14-17 show gold grades on an east-west and north-south section respectively.

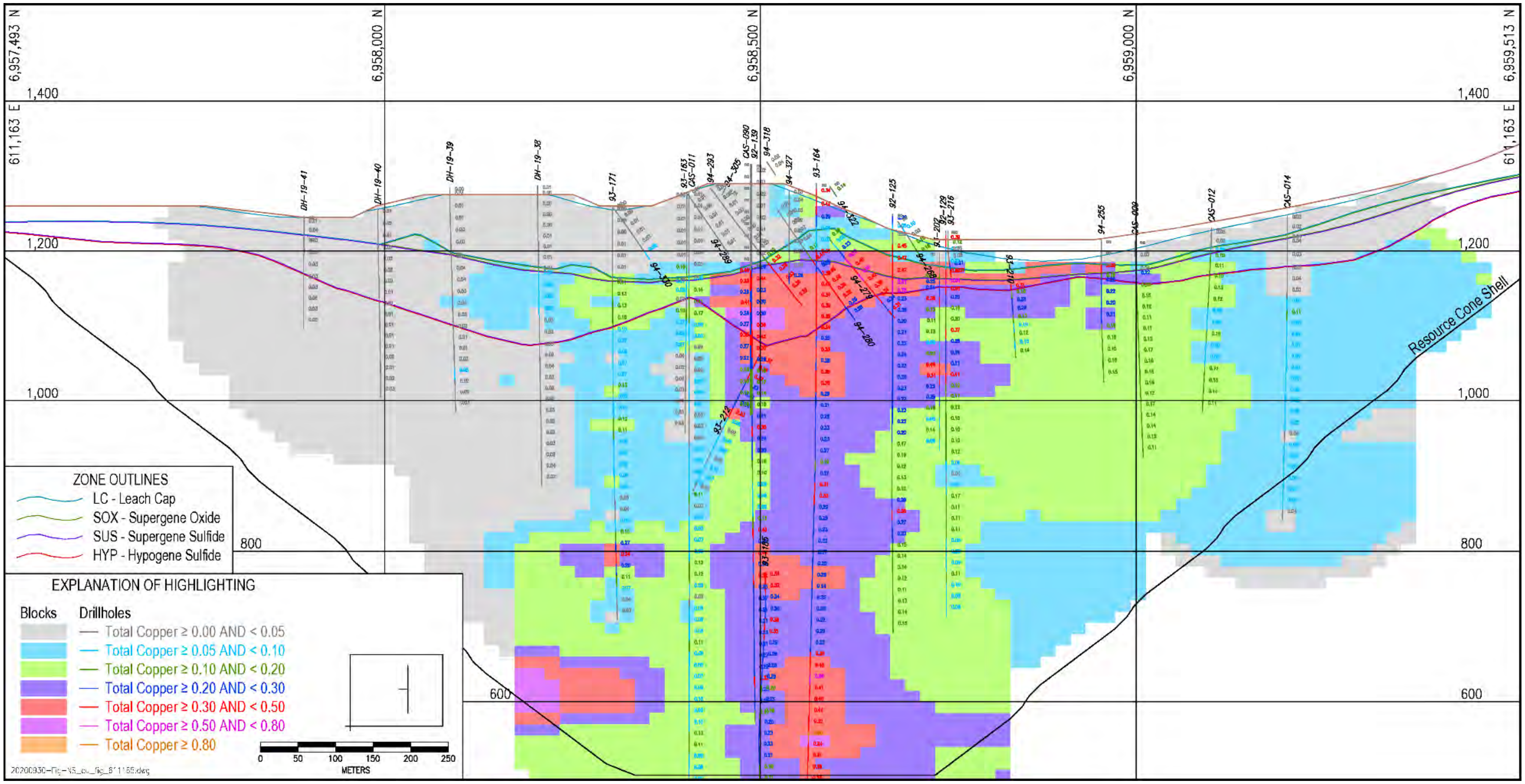
Arsenic, Antimony, and Bismuth

Block grade estimates were also conducted for arsenic, antimony, and bismuth. The methodology and search parameters were the same as for gold, moly, and silver, i.e. there is no indication of significant changes in mineralization across the oxidation type domains. The Post Mineral Explosive Breccia was considered as a separate domain from the other rock types. Minor capping of the assays was conducted. Arsenic was capped at 3,000 ppm, antimony at 1,000 ppm, and bismuth at 400 ppm.



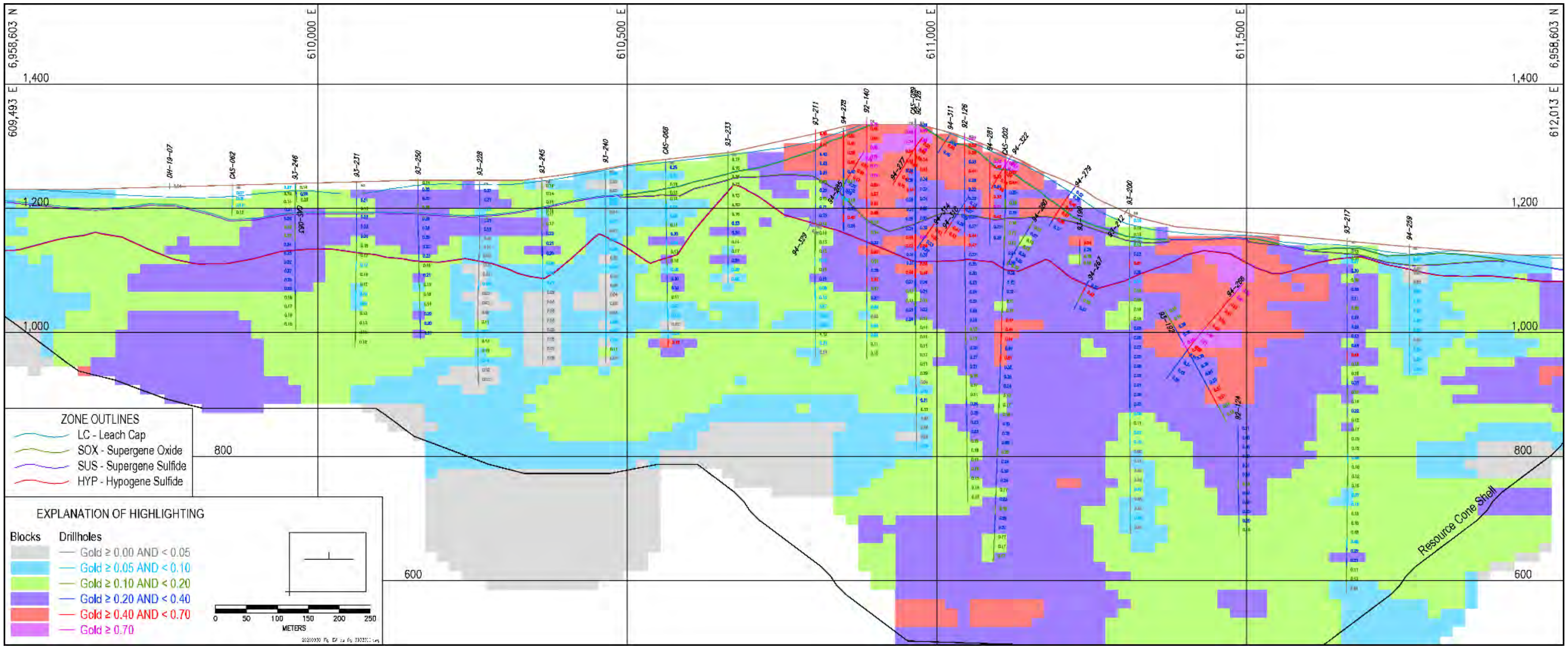
(Source: IMC, 2020)

Figure 14-14: Total Copper Grades on East-West Cross Section 6,958,600N



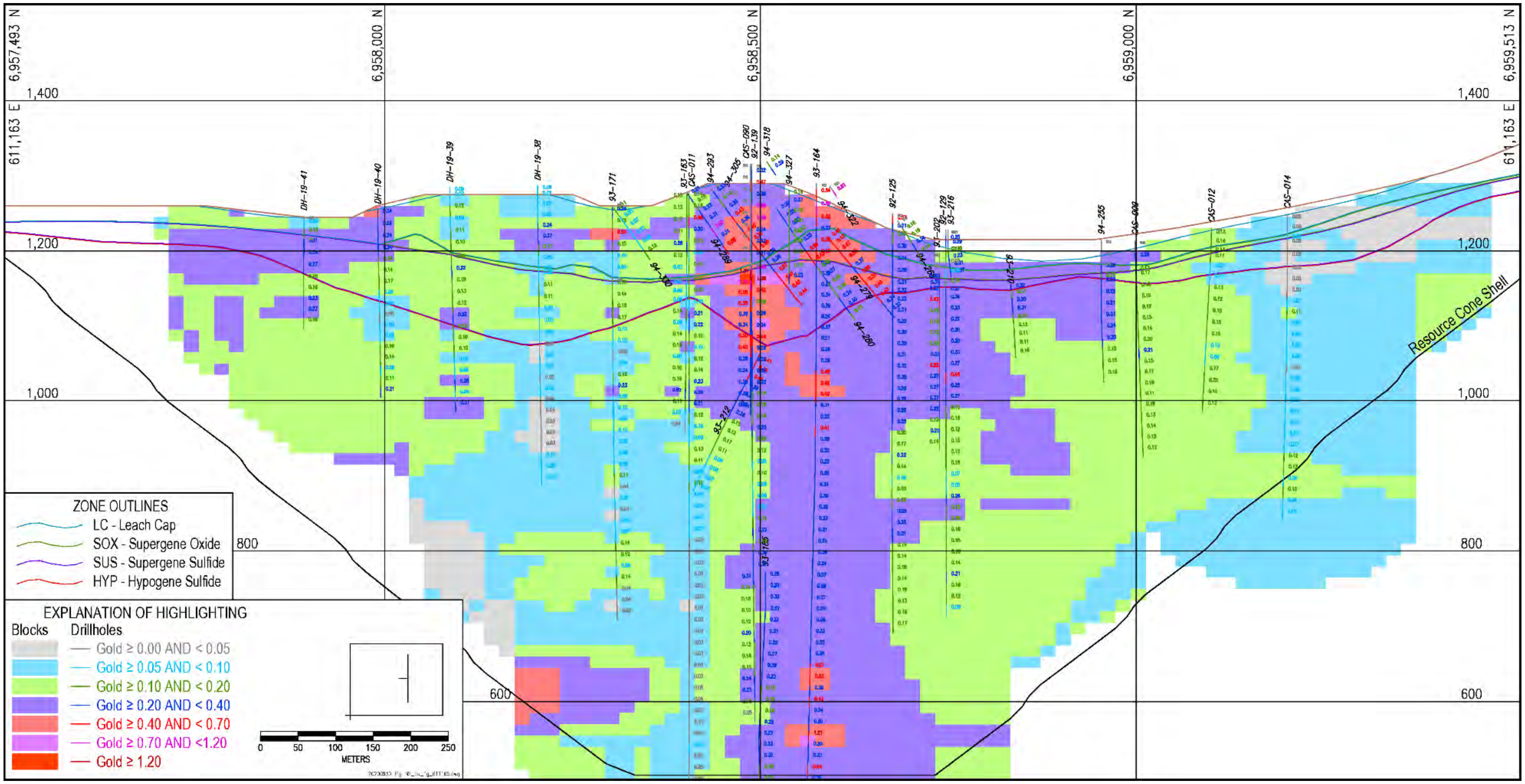
(Source: IMC, 2020)

Figure 14-15: Total Copper Grades on North-South Cross Section 611,165E



(Source: IMC, 2020)

Figure 14-16: Gold Grades on East-West Cross Section 6,958,600N



(Source: IMC, 2020)

Figure 14-17: Gold Grades on North-South Cross Section 611,165E

14.5.8 Bulk Density

Over 13,000 specific gravity measurements on core samples were included with the Casino assay database. IMC excluded four measurements that exceeded 4.0 and 6 measurements less than 1.25 and tabulated the remaining values by oxidation type as shown in Table 14-13.

Table 14-13: Statistics of Specific Gravity Measurement by Oxidation Zone

Oxidation Zone	Zone Code	No. of Samples	Mean S.G.	Std. Dev. S.G.
OVB	1	92	2.496	0.170
LC	2	2,199	2.518	0.128
SOX	3	937	2.580	0.132
SUS	4	2,532	2.624	0.137
HYP	5	7,198	2.651	0.114
WST	6	104	2.684	0.095
TOTAL		13,062	2.617	0.132

It can be seen the mean values increase from OVB to LC to SOX to SUS to HYP to WST, i.e. the higher the level of oxidation the lower the specific gravity.

IMC also examined the specific gravity measurements by rock type, but other than the overburden, the averages by rock type are very similar to each other, ranging from 2.612 to 2.637; it is more meaningful to group the data by oxidation type.

The average specific gravity values on the table were also assumed to represent bulk density measurements, in tonnes per cubic metre, without any adjustments, and assigned to the block model based on oxidation type.

There are sufficient measurements that IMC also investigated estimating values in a similar manner as the grade estimates. The means shown on the table were used as background values for blocks without sufficient close data to estimate them. The averages of blocks done by estimation tended to exceed the table values by a percent or so. Because of this IMC assigned values as the average zone values rather than estimation of the individual blocks.

14.5.9 Resource Classifications

For the purpose of classifying measured and indicated versus inferred mineral resources, an additional block estimate was done. This was based on the same search orientations and search radii as the grade estimates. The estimate was based on a maximum of three composites, a minimum of three, and a maximum of one composite per hole. This estimate provides the average distance to the nearest three holes to each block and was put into the block model. Note the grades from this estimate were not used.

Blocks with an average distance to the nearest three holes less than 100 m were assigned as indicated mineral resource. Blocks with an average distance to three holes greater than 100 m were assigned to inferred mineral resource. Generally (not specific to Casino) an average distance to the nearest three holes of 100 m corresponds to an average drill spacing of about 133 m. These estimates are approximate. It is noted that the nominal spacing for much of the Casino drilling is about 100 m.

Figure 14-18 shows the probability plots of the average distances to the nearest 3 holes for the supergene sulphide. Figure 14-19 shows the plot for the hypogene sulphide.

On Figure 14-2 it can be seen that there is an area on the eastern side of the deposit where there is a combination of vertical and angle holes that reduce the average sample spacing in the area to about 70 m or so. A solid was designed in this area to define measured mineral resources. Figure 14-20 and Figure 14-21 show the resource classification on and east-west and north-south cross section respectively.

The analytical method of distinguishing between indicated and inferred mineral resources resulted in some small groupings of inferred blocks surrounded by indicated blocks. Some filtering was done to remove many, but not all, of these blocks. The filters identified inferred blocks that contacted two, three, or four indicated blocks and set them to indicated blocks. Several passes of filtering were done.

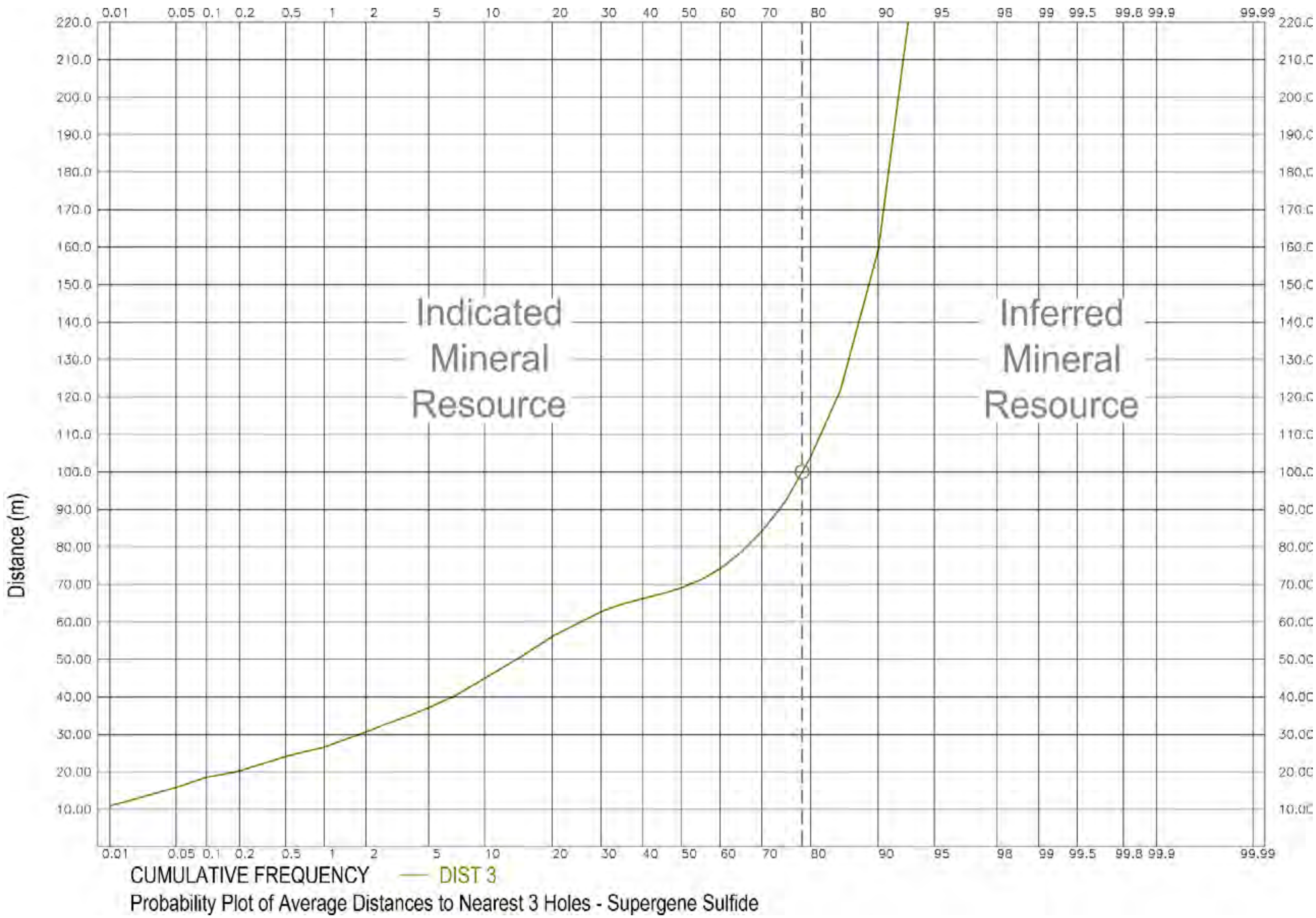


Figure 14-18: Probability Plot of Average Distance to Nearest 3 Holes – Supergene Sulphide (Source: IMC, 2020)

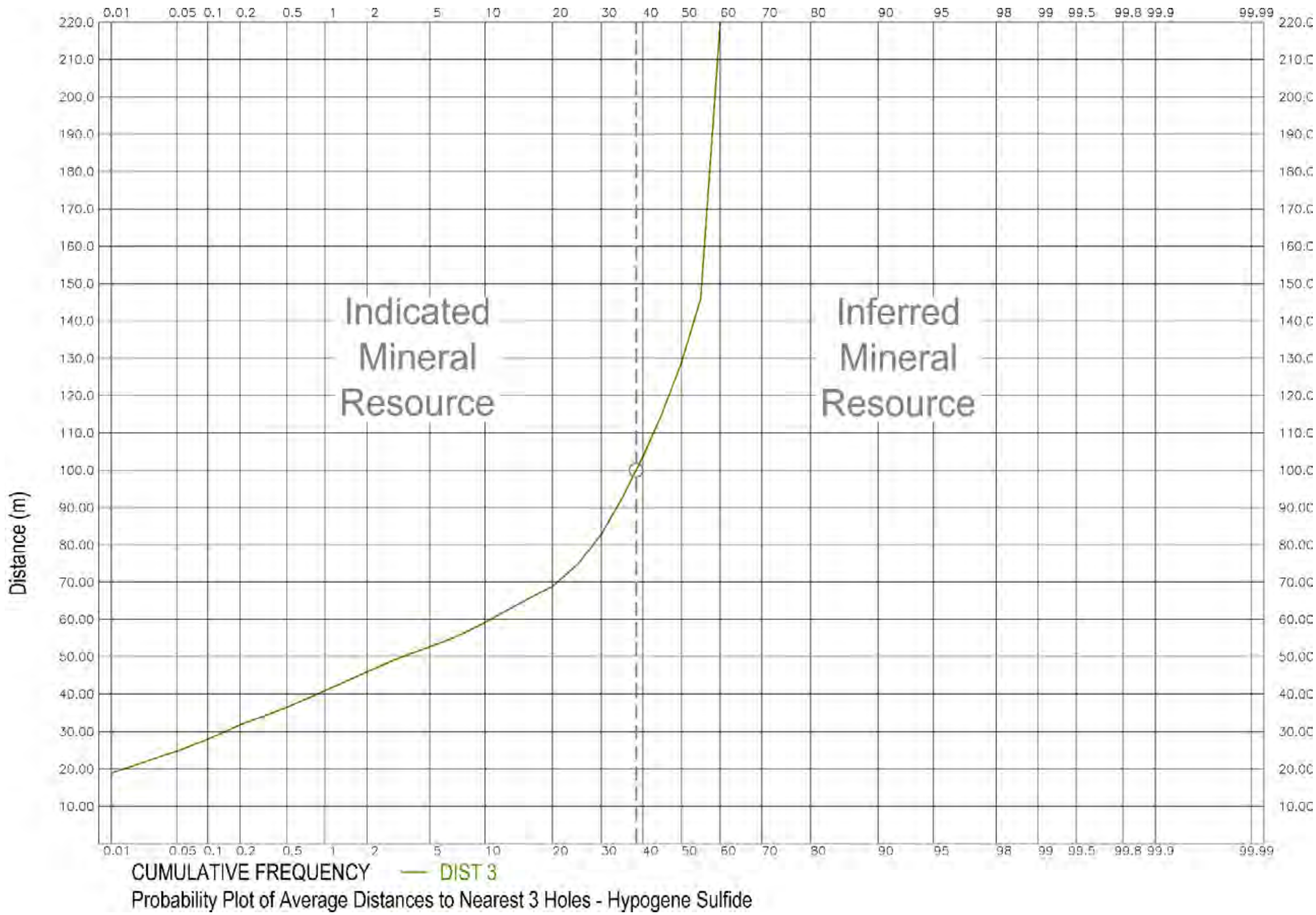


Figure 14-19: Probability Plot of Average Distance to Nearest 3 Holes – Hypogene Sulphide (Source: IMC, 2020)

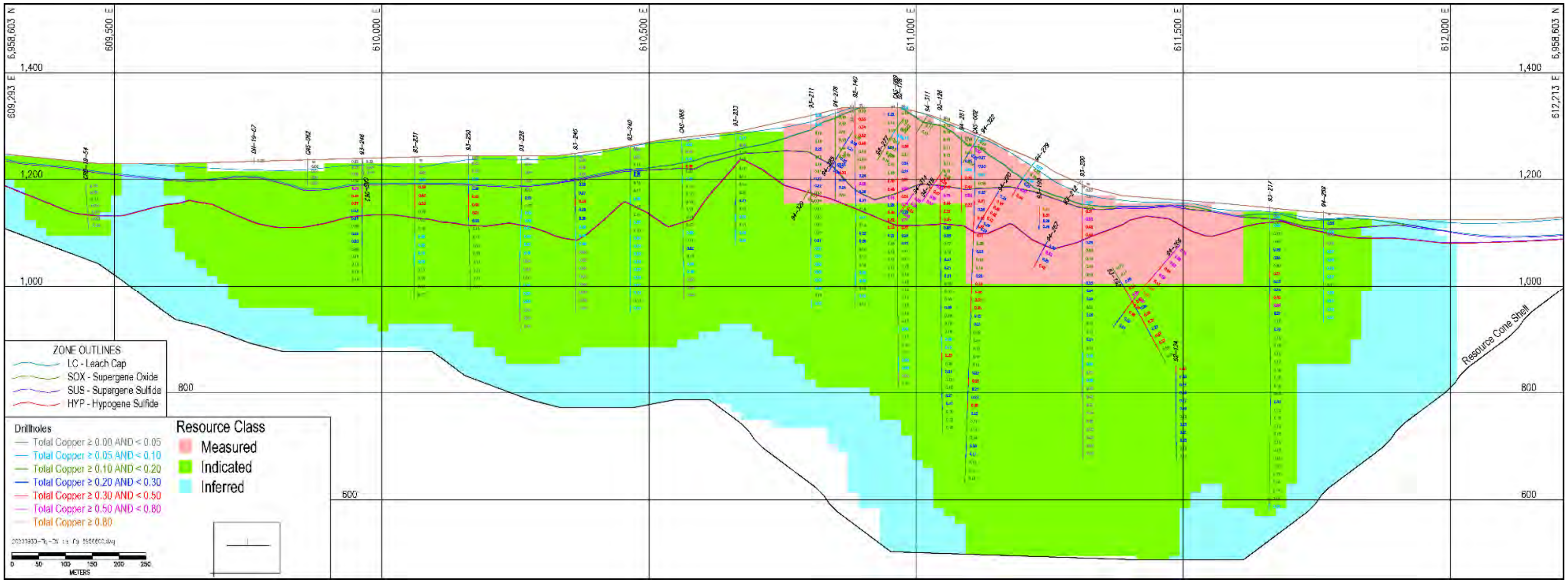


Figure 14-20: Resource Classification on East-West Cross Section 6,958,600N (Source: IMC, 2020)

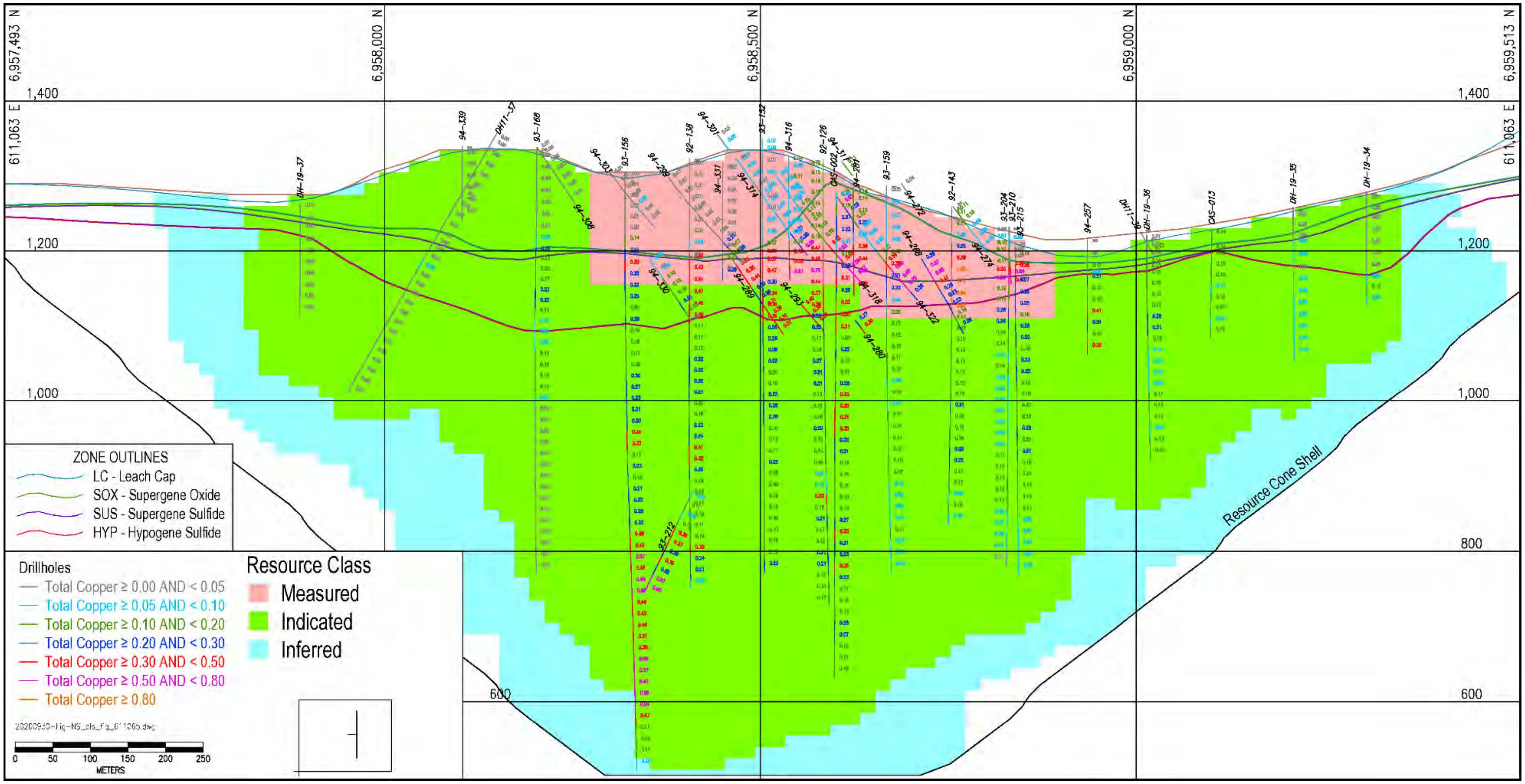


Figure 14-21: Resource Classification on North-South Cross Section 611,065E (Source: IMC, 2020)

14.5.10 Comparison of 2010 and 2020 Mineral Resource

Table 14-4 compares mineral resources amenable to milling. These include the supergene oxide, supergene sulphide and hypogene sulphide materials. For the 2010 mineral resource, measured and indicated mineral resources amounted to 1.06 billion tonnes at 0.20% total copper, 0.23 g/t gold, 0.022% moly and 1.7 g/t silver. This amounted to 4.7 billion pounds of contained copper, 7.9 million ounces of contained gold, 522.1 million pounds of contained moly and 58.0 million ounces of contained silver. Inferred mineral resources was an additional 1.70 billion tonnes at 0.15% total copper, 0.16 g/t gold, 0.019% moly and 1.4 g/t silver.

The 2010 mineral resource estimate was not contained within a potential mining pit shell, it was a tabulation of all the blocks in the resource model. The mineral resource was also tabulated at a 0.25% copper equivalent cutoff grade, where copper equivalent was defined as:

$$\text{Copper Equivalent (\%)} = \text{Total Copper (\%)} + 0.638 \times \text{Gold (g/t)} + 5.625 \times \text{Moly (\%)} + 0.0082 \times \text{Silver (g/t)}$$

An economic justification of the cutoff grade was not included in the available documentation. The factors for the copper equivalency calculation were based on relative commodity prices only and did not consider potential recoveries or treatment charges. The commodity prices used were US$ 2.00 per pound copper, US$ 875 per ounce gold, US$ 11.25 per pound moly, and US$ 11.25 per ounce silver.

For the 2020 mineral resource, measured and indicated mineral resources amounted to 2.17 billion tonnes at 0.16% total copper, 0.18 g/t gold, 0.017% moly, and 1.4 g/t silver. This amounted to 7.4 billion pounds of contained copper, 12.7 million ounces of contained gold, 811.6 million pounds of contained moly and 100.2 million ounces of contained silver. Inferred mineral resources was an additional 1.43 billion tonnes at 0.10% total copper, 0.14 g/t gold, 0.010% moly, and 1.2 g/t silver.

Compared to the 2010 mineral resource, the current measured and indicated mineral resource has 105.7% more tonnes, at a 22.9% lower copper grade, a 21.1% lower gold grade, a 24.4% lower moly grade, and a 16.0% lower silver grade. This amounts to 58.7% more contained copper, 62.0% more contained gold, 55.4% more contained moly, and 72.8% more contained silver.

The 2020 mineral resource estimate is contained within a floating cone pit shell and is tabulated at an NSR cutoff grade of C$5.70 per tonne. Due to higher commodity prices than were prevalent during 2010 the effective cutoff grade for the 2020 mineral resource is lower, resulting in the significant increase in tonnage at lower grades. There is also a significant increase in indicated mineral resources due to conversion of inferred mineral resources. This is partly due to new drilling. It is also the opinion of IMC that the resource classification of indicated mineral resource for 2010 was overly conservative for a copper porphyry system.

Table 14-15 compares mineral resources for material amenable to leaching. This includes the leach cap material. For the 2010 mineral resource, measured and indicated mineral resource amounts to 83.8 million tonnes at 0.40 g/t gold, 0.04% total copper, and 2.6 g/t silver. This amounts to 1.1 million ounces of contained gold, 68.9 million pounds of contained copper and 6.9 million ounces of contained silver. Moly will not be recovered in the leaching process. The 2010 mineral resource estimate was based on a gold cutoff grade of 0.25 g/t gold.

For the 2020 mineral resource estimate, measured and indicated mineral resource amounts to 217.4 million tonnes at 0.25 g/t gold, 0.03% total copper, and 1.9 g/t silver. This amounts to 1.8 million ounces of contained gold, 166.5 million pounds of contained copper and 13.3 million ounces of contained silver. This estimate is based on an NSR cutoff grade of C$5.46 per tonne. As with the mill material, a portion of the increased resource is due to higher commodity prices which reduce the effective cutoff. The new drilling is also a significant factor in the increased mineral resource.

Table 14-14: Comparison of 2010 and 2020 Mineral Resource – Mill Material

Mineral Resource Estimate	Ktonnes	Cu Eq (%)	Copper (%)	Gold (g/t)	Moly (%)	Silver (g/t)	Copper (mlbs)	Gold (moz)	Moly (mlbs)	Silver (moz)
2010 Mineral Resource										
Measured Mineral Resource	93.7	0.78	0.34	0.42	0.027	2.2	695	1.3	56.1	6.7
Indicated Mineral Resource	963.0	0.46	0.19	0.21	0.022	1.7	3,991	6.6	466.0	51.3
Meas/Ind Mineral Resource	**1,056.7**	**0.49**	**0.20**	**0.23**	**0.022**	**1.7**	**4,686**	**7.9**	**522.1**	**58.0**
Inferred Mineral Resource	1,696.4	0.37	0.15	0.16	0.019	1.4	5,440	8.8	719.7	74.7
2020 Mineral Resource										
Measured Mineral Resource	145.3	0.74	0.31	0.40	0.025	2.1	986	1.9	80.6	9.8
Indicated Mineral Resource	2,028.0	0.33	0.14	0.17	0.016	1.4	6,448	10.9	731.0	90.4
Meas/Ind Mineral Resource	**2,173.3**	**0.36**	**0.16**	**0.18**	**0.017**	**1.4**	**7,434**	**12.7**	**811.6**	**100.2**
Inferred Mineral Resource	1,430.2	0.24	0.10	0.14	0.010	1.2	3,240	6.4	322.8	53.5
Percent Difference										
Measured Mineral Resource	55.1%	-5.4%	-8.5%	-5.8%	-7.4%	-5.4%	41.8%	46.1%	43.6%	46.7%
Indicated Mineral Resource	110.6%	-28.8%	-23.3%	-21.5%	-25.5%	-16.3%	61.6%	65.3%	56.9%	76.2%
Meas/Ind Mineral Resource	**105.7%**	**-27.4%**	**-22.9%**	**-21.1%**	**-24.4%**	**-16.0%**	**58.7%**	**62.0%**	**55.4%**	**72.8%**
Inferred Mineral Resource	-15.7%	-34.7%	-29.3%	-14.2%	-46.8%	-15.0%	-40.4%	-27.7%	-55.1%	-28.3%

Table 14-15: Comparison of 2010 and 2020 Mineral Resource – Leach Material

Mineral Resource Estimate	Tonnes Mt	Gold (g/t)	Copper (%)	Silver (g/t)	Gold (moz)	Copper (mlbs)	Silver (moz)
2010 Mineral Resource							
Measured Mineral Resource	30.6	0.52	0.05	2.9	0.5	33.7	2.9
Indicated Mineral Resource	53.2	0.33	0.03	2.4	0.6	35.2	4.0
Meas/Ind Mineral Resource	**83.8**	**0.40**	**0.04**	**2.6**	**1.1**	**68.9**	**6.9**
Inferred Mineral Resource	17.1	0.31	0.01	1.9	0.2	3.8	1.1
2020 Mineral Resource							
Measured Mineral Resource	37.2	0.45	0.05	2.8	0.5	39.3	3.3
Indicated Mineral Resource	180.2	0.21	0.03	1.7	1.2	127.2	10.0
Meas/Ind Mineral Resource	**217.4**	**0.25**	**0.03**	**1.9**	**1.8**	**166.5**	**13.3**
Inferred Mineral Resource	31.1	0.17	0.03	1.7	0.2	17.2	1.7
Percent Difference							
Measured Mineral Resource	21.4%	-14.0%	-4.0%	-6.5%	4.4%	16.6%	13.6%
Indicated Mineral Resource	238.8%	-35.8%	6.7%	-26.7%	117.7%	261.4%	148.4%
Meas/Ind Mineral Resource	**159.4%**	**-36.9%**	**-6.9%**	**-26.0%**	**63.8%**	**141.6%**	**92.1%**
Inferred Mineral Resource	82.1%	-46.1%	150.0%	-11.9%	-1.9%	355.2%	60.4%

15 MINERAL RESERVE ESTIMATES

There are no current mineral reserves for the Casino Project.

16 MINING METHODS

16.1 OPERATING PARAMETERS AND CRITERIA

This PEA is based on a conventional open pit mine plan. Mine operations will consist of drilling large diameter blast holes (31 cm), blasting with a bulk emulsion, and loading into large off-road trucks with cable shovels and a hydraulic shovel. Resource amenable to processing will be delivered to the primary crusher or various resource stockpiles. Waste rock will be placed inside the limits of the tailings management facility (TMF). There will be a fleet of track dozers, rubber-tired dozers, motor graders and water trucks to maintain the working areas of the pit, stockpiles, and haul roads.

The following general parameters guided the development of the mining plan:

- Mill material is limited to about 1.1 billion tonnes, due to the storage capacity limitation of the selected TMF site and embankment design,
- Total mine waste to be co-disposed with tailings is limited to about 500 million tonnes,
- Mill capacity is a nominal 120,000 tonnes per day (t/d), but actual plant throughput for the schedule is based on hardness of the various material types, and usually exceeds 120,000 t/d.

This PEA is based on using only measured and indicated mineral resources for potential plant feed. Inferred mineral resources are considered waste for this study. The geotechnical parameters relevant to the mine plan are discussed in Section 16.3 and are adequate for this Technical Report.

16.2 SLOPE ANGLES

Slope angles recommendations were developed by Knight Piésold Ltd. (KP) and documented in the report "Open Pit Geotechnical Design" (Knight Piésold, 2012). Table 16-1 shows the recommended angles by design sector and Figure 16-1 shows the design sectors. Note the wall geology descriptions on Table 16-1 are described in the Figure 16-1 legend.

Forty-five-degree inter-ramp angles were recommended for most of the slope sectors. The north sectors of the main pit and west pit were recommended to be designed at 42-degree inter-ramp angles. For the small amount of overburden on the north wall, the recommended angle was 27 degrees. The slope angle recommendations also specified that there be no more than 200 m of vertical wall at the inter-ramp angle without an extra wide catch bench (16 m instead of 8 m).

Figure 16-2 shows the final pit design for this study.

Table 16-1: Recommended Slope Angles (Knight-Piésold, 2012)

Design Sector	Max. Slope Height	Wall Geology	Bench Face Angle	Bench Height	Bench Width	Inter-ramp Angle	Max. Inter-ramp Slope Height	Max. Overall Slope Angle	Comments
	m		°	m	m	°	m	°	
M-North	630	Overburden	**40**	**5**	**4**	**27**	**200**	**39**	Reduction of inter-ramp angle to 42° will reduce the risk of multi-bench wedge failure in bedrock.
		DRB	**60**	**15**	**8**	**42**			
M-Northeast	630	Overburden	**40**	**5**	**4**	**27**	**100 (in Weathered Zone) 200 (in Fresh Bedrock)**	**40**	Weathered Zone down to 250 m deep, additional stepouts/ramps should be incorporated into the Weathered Zone slopes.
		DRB	**65**	**15**	**8**	**45**			
M-South	540	DRB, PMS	**65**	**15**	**8**	**45**	**200**	**42**	Potential planar and wedge failures are kinematically possible, but at the limit of the defect friction angle.
Central	210	PMS	**65**	**15**	**8**	**45**	**200**	**N/A**	Central lower pit walls with various orientations. The south facing maybe subject to potential wedge failure.
W-North	285	DRB	**60**	**15**	**8**	**42**	**200**	**39**	Reduction of the inter-ramp slope to 42° will reduce the risk of multi-bench planar and wedge failures.
W-South	480	DRB	**65**	**15**	**8**	**45**	**200**	**42**	Potential bench scale toppling failure is expected.
W-Southwest	345	PMS	**65**	**15**	**8**	**45**	**200**	**42**	
W-West	225	DRB	**65**	**15**	**8**	**45**	**200**	**42**	Potential bench scale planar failure is expected.

X:\p19808\report\KPSlopeAngles\[Table 7.1 - Recommended Pit Slope Angles.xlsm]Table 7.1

NOTES:

1. MAXIMUM SLOPE HEIGHT REPRESENTS THE HIGHEST WALL IN EACH DESIGN SECTOR.
2. RECOMMENDED SLOPE ANGLES DETERMINED BY THE KINEMATIC AND ROCK MASS STABILITY ANALYSES.
3. OVERBURDEN IS NEGLIGIBLE IN THE WESTERN AND SOUTHERN PIT WALLS.
4. THE OVERALL SLOPE ANGLES INCLUDED 1 TO 3 STEPOUTS OR HAUL RAMPS IN THE FINAL PIT WALLS.

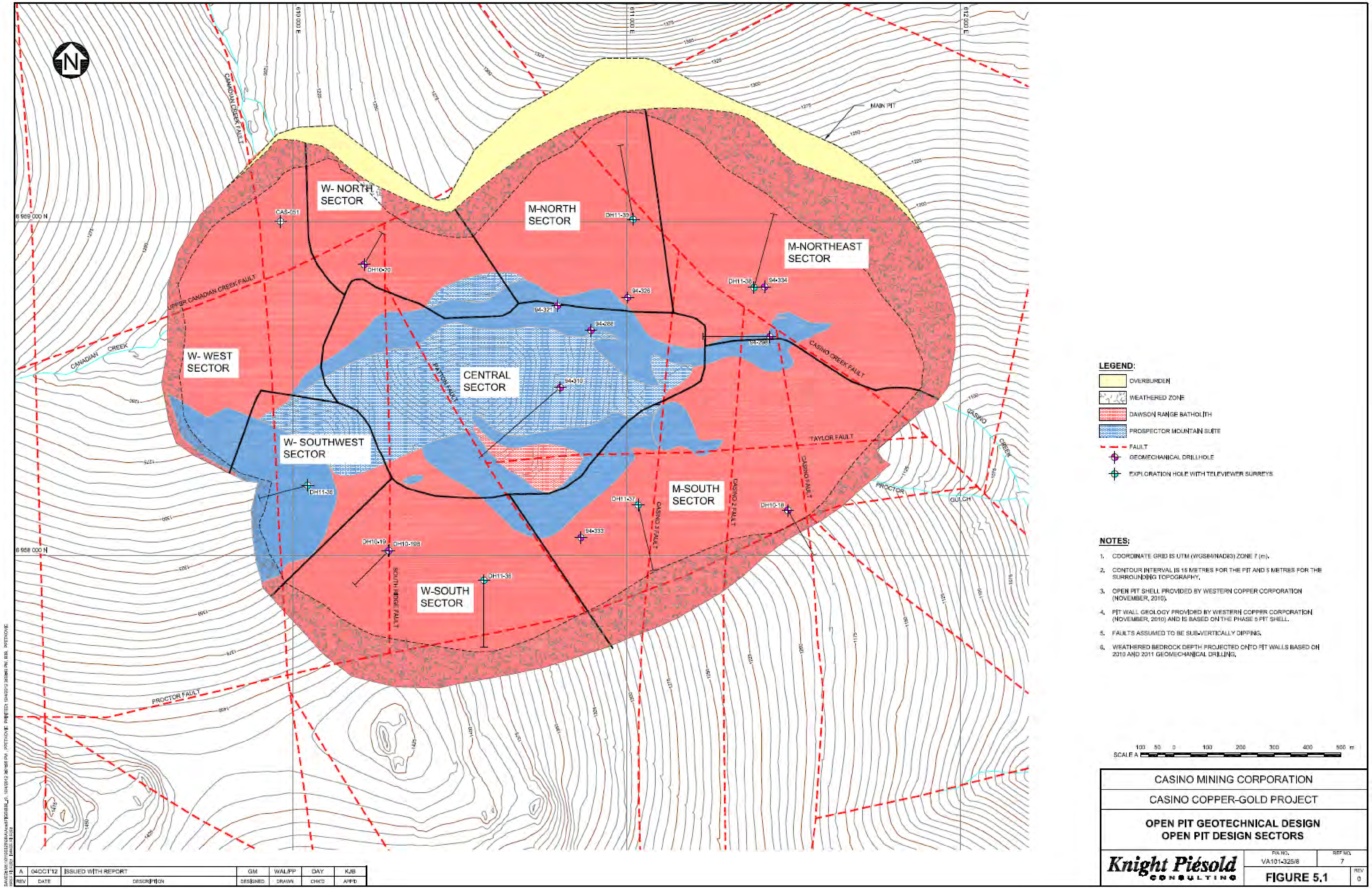


Figure 16-1: Open Pit Design Sectors (Knight-Piésold, 2012)

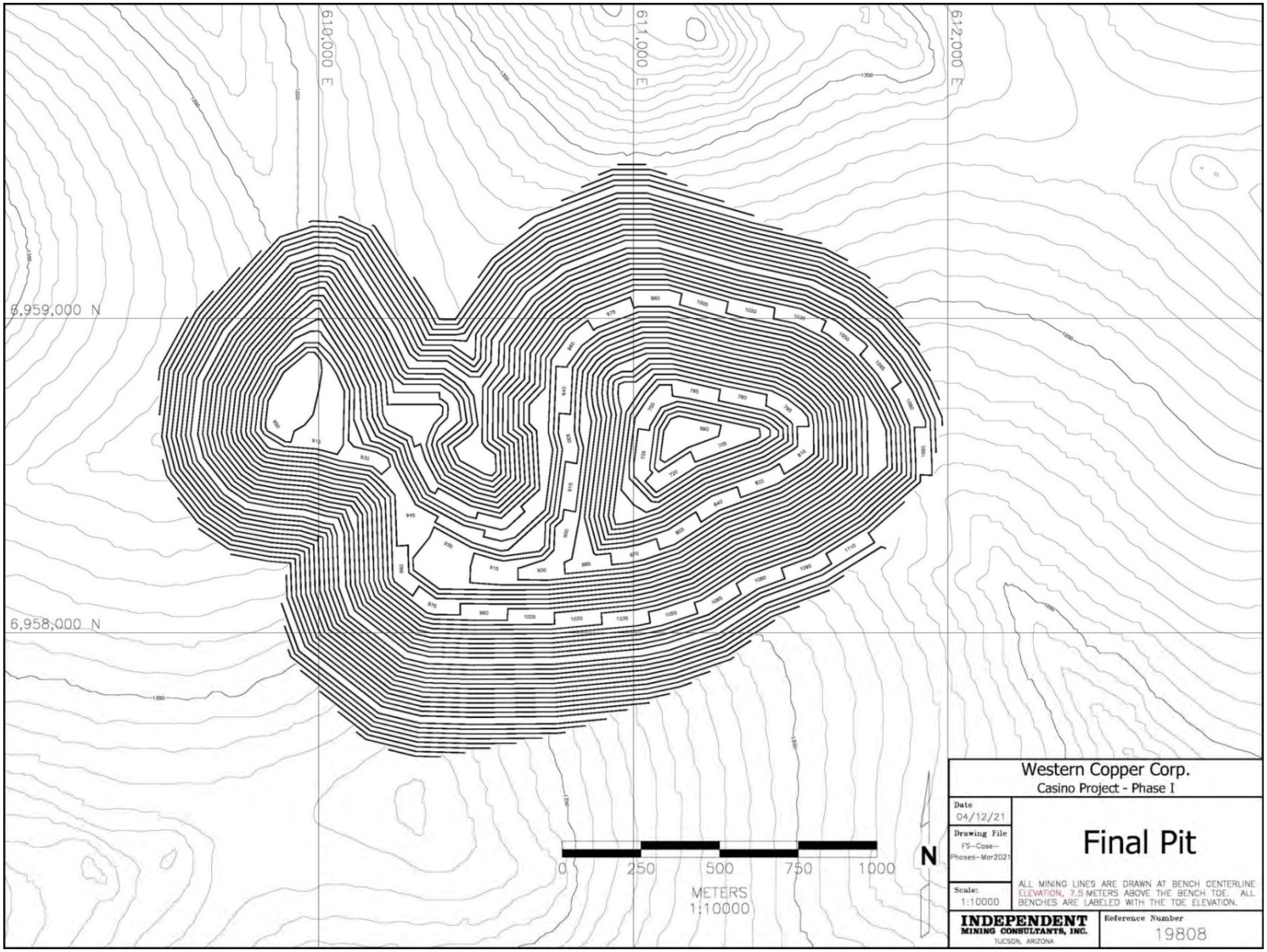


Figure 16-2: Final Pit Design (IMC, 2021)

16.3 ECONOMIC PARAMETERS FOR MINE DESIGN

16.3.1 Metal Prices

Table 16-2 shows the economic and recovery parameters used for mine design and production scheduling. In US dollar (US$) terms, commodity prices are US$3.00 per pound copper, US$1500 per ounce gold, US$24 per ounce silver, and US$9.00 per pound moly. A conversion of US$0.78 = C$1.00 was used to convert the prices to C$. IMC believes these prices to be reasonable based on: 1) historical 3-year trailing averages, 2) prices used by other companies for comparable projects, and 3) long range consensus price forecasts prepared by various bank economists.

16.3.2 Cost and Recovery Estimates

16.3.2.1 Mining Cost

The base mining cost of C$1.75 per total tonne was estimated by IMC. This estimate was based on likely production rates and equipment requirements and considered typical prices for fuel, blasting agents, equipment parts, and labor, etc.

16.3.2.2 Processing of Mill Material

Mill material refers to the supergene oxide, supergene sulphide, and hypogene sulphide zones of the mineral deposit. The processing will be in a conventional sulphide flotation plant that will produce copper and moly concentrates that will be sold to commercial copper smelters and moly roasting plants. The base unit costs for processing and G&A are estimated at C$5.33 and C$0.37 per tonne, respectively, provided by M3. The estimated plant recoveries for gold, moly, and silver in the supergene and hypogene zones are shown on Table 16-2. Copper recovery is estimated at 92.2% for hypogene sulphide material. The plant recovery for supergene material is estimated as follows:

Copper recovery = 94%(Cut% - Cuw%) / Cut%

Where,

Cut% = Total copper grade
Cuw% = Weak acid soluble copper grade

The copper, gold, and silver payable percentages shown on Table 16-2 are typical terms for copper concentrates, assuming a clean concentrate with a copper concentrate grade of 28% copper or greater. The off-site cost per pound of copper is estimated at C$0.583. This is based on payment for 96.5% of the copper in concentrate, smelting cost at US$85 per tonne, refining at US$0.085 per pound, and concentrate freight of US$135 per tonne. The moisture content was estimated at 8.0% and 0.5% concentrate loss during shipping. Gold and silver refining is estimated at C$8.00 and C$0.667, respectively.

Note that the off-site cost for moly is assumed to be accounted in the 85% payable percentage for molybdenum in concentrate, i.e. this is assumed to be the net payable after treatment and transportation charges. This is applicable to a clean moly concentrate with a moly grade of about 50% or greater.

16.3.2.3 Processing of Leach Material

Leach material refers to the leach capping of the mineral deposit. Processing is by crushing and heap leaching with cyanide. Gold and silver from the heap leach will report to a typical doré which will be sent to a refinery. The SART process will be used to extract copper from the cyanide solution and produce a copper concentrate that can be sold to conventional copper smelters. Heap leach mineralized material processing is estimated at C$5.09 per tonne. The G&A cost of C$0.37 per tonne is also applied to leach material.

Heap leach recoveries are estimated at 18% for copper, 66% for gold, and 26% for silver. Typical terms for refining costs are shown and are C$1.733 per ounce for gold and C$0.667 for silver. The payable percentage is estimated at 98% for gold and silver.

It is also assumed that the SART process will produce a copper concentrate with a grade of about 60% copper. Smelting and refining terms are assumed the same as for the flotation concentrate. This results in a smelting, refining, and freight charge of about C$0.346 per pound.

16.3.3 NSR Calculations

Due to multiple mineral products and also the variable recovery for copper in the supergene zones, Net Smelter Return (NSR) values, in Canadian Dollars, were calculated for each model block to use to classify blocks into potential resource and waste. For the leach material:

NSR_au = ($1923.08 – $1.733) x 0.66 x 0.98 x gold(g/t) / 31.103 = C$39.96 x gold (g/t)

NSR_cu = ($3.85 - $0.346) x 0.18 x 0.965 x 0.995 x copper(%) x 22.046
= C$13.34 x copper (%)

NSR_ag = ($30.77 - $0.667) x 0.26 x 0.98 x silver (g/t) / 31.103 = C$0.247 x silver (g/t)

NSR = NSR_au + NSR_cu + NSR_ag

The internal NSR cutoff for leach material is the processing + G&A cost of C$5.46 per tonne since all the recoveries and refining costs are accounted for in the NSR calculation. Internal cutoff grade applies to blocks that have to be removed from the pit so the mining cost is a sunk cost. Internal cutoff is also generally the minimum cutoff that would be evaluated for mine scheduling. The breakeven NSR cutoff grade for leach material is C$7.21 per tonne (mining plus processing and G&A).

For processing of hypogene sulphide material, the NSR values are calculated as:

NSR_cu = ($3.85 – $0.583) x 0.922 x 0.965 x 0.995 x copper (%) x 22.046
= C$63.69 x copper (%)

NSR_au = ($1923.08 – $8.00) x 0.66 x 0.975 x 0.995 x gold (g/t) / 31.103
= C$39.42 x gold (g/t)

NSR_mo = $11.54 x 0.786 x 0.85 x 0.995 x moly (%) x 22.046 = C$169.10 x moly (%)

NSR_ag = ($30.77 - $0.667) x 0.50 x 0.95 x 0.995 x silver (g/t) / 31.103
= C$0.457 x silver (g/t)

NSR = NSR_cu + NSR_au + NSR_mo + NSR_ag

For processing of supergene material, the NSR values are calculated as:

NSR_cu = ($3.85 – $0.583) x 0.965 x 0.995 x rec_cu (%) x 22.046
= C$69.07 x rec_cu (%)

NSR_au = ($1923.08 – $8.00) x 0.69 x 0.975 x 0.995 x gold(g/t) / 31.103
= C$41.22 x gold (g/t)

NSR_mo = $11.54 x 0.523 x 0.85 x 0.995 x moly (%) x 22.046 = C$112.41 x moly (%)

NSR_ag = ($30.77 - $0.667) x 0.60 x 0.95 x 0.995 x silver (g/t) / 31.103
= C$0.549 x silver (g/t)

NSR = NSR_cu + NSR_au + NSR_mo + NSR_ag

where,

rec_cu = 0.94 x (Cut% – Cuw%)

The internal NSR cutoff for flotation is the processing plus G&A cost of C$5.70. Breakeven NSR cutoff is C$7.45.

Table 16-2: Economic Parameters for Mine Design

Parameter	Units	Mill Material SOX/SUS	Mill Material HYP	Heap Leach
Commodity Prices and Exchange Rate:				
Copper Price Per Pound (US$)	(US$)	3.00	3.00	3.00
Gold Price Per Ounce (US$)	(US$)	1500.00	1500.00	1500.00
Silver Price Per Ounce (US$)	(US$)	24.00	24.00	24.00
Molybdenum Price Per Pound (US$)	(US$)	9.00	9.00	9.00
Exchange Rate (CAD to $US)	(none)	0.78	0.78	0.78
Copper Price Per Pound (C$)	(C$)	3.85	3.85	3.85
Gold Price Per Ounce (C$)	(C$)	1923.08	1923.08	1923.08
Silver Price Per Ounce (C$)	(C$)	30.77	30.77	30.77
Molybdenum Price Per Pound (C$)	(C$)	11.54	11.54	11.54
Mining Cost Per Total Tonne:				
Base Mining Cost	(C$)	1.750	1.750	1.750
Sustaining Capital Allowance	(C$)	0.000	0.000	0.000
Total Mining Cost	(C$)	1.750	1.750	1.750
Processing and G&A Per Tonne Processed				
Processing	(C$)	5.330	5.330	5.090
G&A	(C$)	0.370	0.370	0.370
Total Processing and G&A	(C$)	5.700	5.700	5.460
Average Plant Recoveries:				
Copper Recovery	(%)	Note 1	92.2%	18.0%
Gold Recovery	(%)	69.0%	66.0%	66.0%
Silver Recovery	(%)	60.0%	50.0%	26.0%
Moly Recovery	(%)	52.3%	78.6%	N.A.
Refinery Payables:				
Copper Payable	(%)	96.5%	96.5%	96.5%
Gold Payable	(%)	97.5%	97.5%	98.0%
Silver Payable	(%)	95.0%	95.0%	98.0%
Molybdenum Payable	(%)	85.0%	85.0%	N.A.
Payable Concentrate (0.5% Conc Loss)	(%)	99.5%	99.5%	Cu Only
Offsite Costs:				
Copper SRF Cost Per Pound	(C$)	0.583	0.583	0.346
Gold Refining Per Ounce	(C$)	8.000	8.000	1.733
Silver Refining Per Ounce	(C$)	0.667	0.667	0.667
Molybdenum Freight/Treatment Per Pound	(C$)	Note 2	Note 2	N.A.
NSR Factors:				
Copper Factor (Note 4)	(C$/t)	69.07	63.69	13.34
Gold Factor (Note 3)	(C$/t)	41.22	39.42	39.96
Silver Factor (Note 3)	(C$/t)	0.549	0.457	0.247
Moly Factor (Note 3)	(C$/t)	112.41	169.10	N.A.
NSR Cutoff Grades:				
Breakeven Cutoff (C$/t)	(C$/t)	7.45	7.45	7.21
Internal Cutoff (C$/t)	(C$/t)	5.70	5.70	5.46

Note 1: Recovery = 94% x (Cutotal – CuWAS)/(Cutotal)
Note 2: Moly offsite costs are accounted in payable percentage
Note 3: NSR factors are applied to model grades
Note 4: For copper, SOX/SUS factor is applied to recovered copper grade, for HYP and leach the factor is applied to total copper grade.

16.4 MINE PRODUCTION SCHEDULE

The mine production schedule is based on five mining phases. The designs utilized 35 m wide roads at a maximum grade of 10%. The road width will accommodate trucks up to the 330-tonne class such as the Caterpillar 797. A suite of floating cones at various commodity prices were run to evaluate the final pit design (Phase 5). The final pit design is based on the floating cone shell at a copper price of US$1.50 to US$1.62 per pound copper and US$750 to US$813 per ounce gold. These are low prices compared to current market prices, but the design is constrained by TMF capacity as discussed in Section 16.1.

The Bond Work Index and mill throughput rate have been assigned to model blocks based on rock type, oxidation zone, and alteration based on the recommendations in the FLSmidth 2012 report (FLSmidth, 2012). Generally, argillic altered rocks have the lowest work index, potassic altered rocks the highest work index, and rock with phyllic alteration tend to be in the middle. The production schedule has been developed based on plant hours, so throughput varies by year. It was reported to IMC that all necessary efficiency factors were incorporated in throughput rates; therefore, IMC has based the schedule on 8,760 plant hours per year.

The top section of Table 16-3 shows the proposed plant production schedule. Total plant material is 1.13 billion tonnes at 0.197% copper, 0.226 g/t gold, 0.0219% moly, and 1.70 g/t silver. The average NSR value of this is C$25.01 per tonne. For Years 2 through 25, full production years, plant throughput varies from a low of 44.4 million tonnes in Year 18 to a high of 46.9 million tonnes in Years 2 and 13. The table also shows the average Bond Work Index (14.4) and throughput rate (0.19126 hours per kt). The throughput units are somewhat unconventional, but a parameter that could be weight averaged by tonnes was required. Copper recovery was also incorporated into the model on a block-by-block basis, based on total and soluble copper grades. The average recovered copper grade is 0.170%, indicating an average copper recovery of 86.6%.

Table 16-3 also shows the various components of the plant material. Direct feed material is material that is scheduled to be processed the same year it is mined. This amounts to 828.1 million tonnes at 0.218% total copper, 0.242 g/t gold, 0.0249% moly and 1.83 g/t silver. This is about 73% of total plant material. The average NSR value of this material is C$27.68 per tonne. Note that Year 1 plant production is 34.5 million tonnes, about 75% of nominal capacity and is made up of material mined during preproduction and Year 1.

The Supergene Oxide (SOX) material in the mining phase 1 starter pit is stockpiled and processed during Years 4 through 13 at the rate of 3.6 Mt/y. This is done to maintain the ratio of weak soluble copper to total copper at relatively low levels by year. This material amounts to 35.8 million tonnes at 0.251% total copper, 0.086% weak soluble copper, 0.491 g/t gold, 0.0252% moly, and 2.36 g/t silver. The average NSR value is C$35.08 per tonne.

The operating schedule also results in a significant amount of low grade material that is stockpiled and processed at the end of the mine life during Years 20 through 25. This amounts to 263.0 million tonnes at 0.123% total copper, 0.140 g/t gold, 0.0121 moly, and 1.20 g/t silver. The low grade represents material between an NSR cutoff of C$12/t and the operating cutoff for direct plant feed for the year. Significantly lower grade material is available if the low grade NSR cutoff is reduced. The C$12/t cutoff results in the total maximum plant material of 1.1 billion.

The reclaim schedules for both the SOX and low grade are on a last-in-first-out (LIFO) basis, consistent with stockpiles build up in lifts and reclaimed in reverse order.

Based on the schedule, the commercial life of the project is 25 years after an approximate 3-year preproduction period.

Table 16-4 shows the mine production schedule. The upper section of the table shows the direct feed material. 4.7 million tonnes of this is mined during preproduction and stockpiled near the crusher to be part of the Year 1 plant feed for quarters 1 and 2.

As previously discussed, an NSR value was calculated for each block to classify blocks into potential plant feed and waste. For the mine production schedule, the direct feed material varies by year to balance mine and plant capacities. At Year 1 Q4 and after, the schedule starts at relatively high cutoff grades in early years and declines to the internal cutoff grade of C$5.70 per tonne for the last couple years of the mine life.

The second section of Table 16-4 shows the SOX material from Phase I that is stockpiled and processed during Year 4 through 13. This is at an NSR cutoff grade of C$20 per tonne to limit the total amount to about 36 million tonnes. Lower grade SOX goes to the low grade stockpile. The third section of the table shows low grade material produced by year. This is material with an NSR cutoff between C$12 per tonne and the operating cutoff for the year. As previously discussed, this cutoff grade limited the material to what fits in the TMF.

The bottom of Table 16-4 also shows the schedule of resource mined from the leach cap zone by year. It is assumed that this is processed by crushing, and heap leaching. Leach resource is defined as leach cap material with an NSR above C$5.46/t with leach economics and total copper less than 0.1%. This amounts to 203.8 million tonnes at 0.259 g/t gold, 1.95 g/t silver, and 0.034% total copper.

The bottom of Table 16-4 summarizes tonnages. It can be seen that life of mine total material from the pit is 1.83 billion tonnes. Preproduction is 75.0 million tonnes staged over three years. Year 1 total material is scheduled at about 95 million tonnes after which the peak material movement of 100 Mt/y is maintained over much of the mine life. Total waste is 500 million tonnes, so the waste ratio is about 0.4 if mill resource (including SOX), low grade, and leach resource are all counted as resource.

The upper section of Table 16-5 shows a proposed stacking schedule for the leach resource. This is based on the ability to crush and stack 9,125 kt/y (25,000 t/d for 365 days/year or 36,500 t/d for 250 days per year).

The second section of Table 16-5 shows mine production of leach resource. The third and fourth sections show up to 9,125 ktonnes of mined material as direct crusher feed and the excess going to a stockpile. Both are shown at average grades for the year. The bottom of the table shows the stockpile reclaim on a last-in-first-out basis (LIFO). The stockpile gets to a maximum size of 74.1 million tonnes with this scenario.

Only measured and indicated mineral resource is included in the mine production schedule. The amount of potential plant resource in the pit that is inferred mineral resource is only about 22 million tonnes, an inconsequential amount.

The mine production schedule includes allowances for mining dilution and mineralized material loss. IMC believes that reasonable amounts of dilution and loss were incorporated into the block model used for this Technical Report. Compositing assays into composites and estimating blocks with multiple composites introduces some smoothing of model grades that are analogous to dilution and mineralized material loss effects.

Table 16-3: Proposed Plant Production Schedule

	(Units)	Y1 Q1	Y1 Q2	Y1 Q3	Y1 Q4	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	Year 17	Year 18	Year 19	Year 20	Year 21	Year 22	Year 23	Year 24	Year 25	TOTAL
Proposed Mill Schedule:																														
Ktonnes	(kt)	5,698	8,757	9,711	10,334	46,882	46,472	45,609	45,477	45,506	45,625	46,369	46,510	45,047	45,482	46,337	46,887	46,199	45,105	45,993	45,039	44,374	45,213	45,310	45,814	45,744	45,945	45,232	40,295	1,126,966
NSR Value	(C$/t)	36.91	41.39	40.05	43.11	39.30	38.20	32.46	27.63	27.40	28.65	28.57	27.76	28.61	28.69	27.65	26.25	22.57	21.64	20.14	21.24	23.67	23.30	16.07	14.32	14.76	15.54	15.94	16.19	25.01
Total Copper	(%)	0.281	0.340	0.365	0.398	0.338	0.286	0.280	0.234	0.209	0.207	0.203	0.188	0.254	0.232	0.210	0.195	0.196	0.183	0.154	0.153	0.169	0.153	0.116	0.123	0.111	0.107	0.138	0.153	0.197
Weak Soluble Copper	(%)	0.050	0.055	0.063	0.058	0.028	0.006	0.033	0.024	0.009	0.008	0.006	0.007	0.031	0.019	0.013	0.011	0.016	0.011	0.000	0.000	0.000	0.000	0.000	0.010	0.009	0.001	0.014	0.031	0.013
% Cuw/Cut	(%)	17.8%	16.2%	17.1%	14.6%	8.4%	2.1%	11.8%	10.2%	4.2%	3.8%	2.8%	3.8%	12.3%	8.1%	6.0%	5.8%	8.2%	5.8%	0.1%	0.0%	0.0%	0.0%	0.0%	8.4%	7.7%	1.3%	9.8%	20.0%	6.7%
Gold	(g/t)	0.454	0.469	0.399	0.404	0.357	0.358	0.316	0.273	0.269	0.265	0.257	0.247	0.240	0.242	0.243	0.231	0.167	0.190	0.188	0.181	0.193	0.186	0.135	0.126	0.148	0.145	0.142	0.156	0.226
Molybdenum	(%)	0.0159	0.0177	0.0237	0.0288	0.0297	0.0323	0.0193	0.0156	0.0194	0.0274	0.0304	0.0340	0.0273	0.0295	0.0289	0.0278	0.0219	0.0148	0.0127	0.0199	0.0269	0.0321	0.0165	0.0096	0.0116	0.0148	0.0112	0.0105	0.0219
Silver	(g/t)	2.70	2.90	2.37	2.03	1.90	1.95	1.95	1.93	1.80	2.17	1.72	1.73	1.97	2.01	1.75	1.61	1.75	1.58	1.66	2.21	1.61	1.73	1.25	1.15	1.16	1.22	1.22	1.19	1.70
Recovered Copper	(%)	0.216	0.267	0.284	0.319	0.288	0.259	0.231	0.195	0.184	0.183	0.182	0.168	0.208	0.199	0.183	0.170	0.167	0.161	0.142	0.141	0.156	0.141	0.107	0.105	0.095	0.097	0.116	0.114	0.170
Bond Work Index	(Kwh/t)	14.2	14.0	13.8	13.8	14.1	14.2	14.4	14.5	14.5	14.4	14.3	14.1	14.6	14.5	14.2	14.1	14.3	14.6	14.3	14.6	14.9	14.6	14.5	14.4	14.4	14.4	14.6	14.4	14.4
Hours Per Ktonne	(hr/kt)	0.18871	0.18611	0.18259	0.18364	0.18685	0.18849	0.19207	0.19262	0.19250	0.19201	0.18891	0.18835	0.19446	0.19260	0.18905	0.18683	0.18961	0.19421	0.19046	0.19449	0.19741	0.19375	0.19335	0.19121	0.19150	0.19066	0.19367	0.19081	0.19126
Mill Hours	(hours)	1,075	1,630	1,773	1,898	8,760	8,760	8,760	8,760	8,760	8,761	8,760	8,760	8,760	8,760	8,760	8,760	8,760	8,760	8,760	8,760	8,760	8,760	8,761	8,760	8,760	8,760	8,760	7,689	215,544
Copper Recovery	(%)	77.0%	78.7%	77.8%	80.2%	85.4%	90.6%	82.3%	83.4%	88.4%	88.7%	89.7%	89.2%	82.1%	85.7%	87.3%	87.5%	85.4%	87.9%	92.2%	92.2%	92.3%	92.2%	92.0%	85.1%	85.7%	90.9%	83.8%	75.0%	86.6%
Gold Recovery	(%)	68.7%	68.7%	68.7%	68.6%	67.6%	66.4%	67.7%	67.2%	66.4%	66.4%	66.5%	66.8%	68.6%	67.9%	67.3%	66.8%	66.9%	67.4%	66.0%	66.0%	66.0%	66.0%	66.0%	67.0%	67.2%	66.5%	67.1%	68.4%	67.1%
Moly Recovery	(%)	55.1%	54.6%	54.4%	55.2%	63.0%	74.3%	66.1%	67.6%	75.1%	76.3%	76.3%	75.1%	56.7%	64.1%	70.3%	75.2%	74.6%	64.4%	78.4%	78.6%	78.6%	78.6%	78.6%	71.0%	67.6%	77.7%	68.1%	57.0%	71.3%
Silver Recovery	(%)	59.0%	59.0%	59.1%	58.6%	54.9%	51.5%	56.0%	53.5%	51.0%	50.7%	51.1%	52.2%	58.8%	55.9%	54.0%	52.1%	52.8%	54.8%	50.0%	50.0%	50.0%	50.0%	50.0%	53.1%	54.1%	51.1%	53.0%	58.0%	53.1%
Direct Feed Material:																														
Ktonnes	(kt)	5,698	8,757	9,711	10,334	46,882	46,472	42,009	41,877	41,906	42,025	42,769	42,910	41,447	41,882	42,737	43,446	46,199	45,105	45,993	45,039	44,374	45,213	5,360						828,145
NSR Value	(C$/t)	36.91	41.39	40.05	43.11	39.30	38.20	32.13	26.65	26.81	28.25	28.25	27.35	27.98	28.08	26.95	25.54	22.57	21.64	20.14	21.24	23.67	23.30	25.53						27.68
Total Copper	(%)	0.281	0.340	0.365	0.398	0.338	0.286	0.269	0.216	0.202	0.203	0.204	0.189	0.260	0.237	0.212	0.195	0.196	0.183	0.154	0.153	0.169	0.153	0.193						0.218
Weak Soluble Copper	(%)	0.050	0.055	0.063	0.058	0.028	0.006	0.025	0.013	0.000	0.000	0.000	0.003	0.028	0.015	0.008	0.007	0.016	0.011	0.000	0.000	0.000	0.000	0.000						0.011
Gold	(g/t)	0.454	0.469	0.399	0.404	0.357	0.358	0.313	0.265	0.254	0.248	0.239	0.226	0.210	0.213	0.215	0.205	0.167	0.190	0.188	0.181	0.193	0.186	0.201						0.242
Molybdenum	(%)	0.0159	0.0177	0.0237	0.0288	0.0297	0.0323	0.0190	0.0144	0.0185	0.0271	0.0306	0.0346	0.0279	0.0303	0.0295	0.0284	0.0219	0.0148	0.0127	0.0199	0.0269	0.0321	0.0280						0.0249
Silver	(g/t)	2.70	2.90	2.37	2.03	1.90	1.95	1.98	1.95	1.78	2.18	1.68	1.67	1.89	1.93	1.65	1.52	1.75	1.58	1.66	2.21	1.61	1.73	1.25						1.83
Recovered Copper	(%)	0.216	0.267	0.284	0.319	0.288	0.259	0.227	0.188	0.186	0.187	0.188	0.173	0.216	0.206	0.189	0.174	0.167	0.161	0.142	0.141	0.156	0.141	0.178						0.192
Bond Work Index	(Kwh/t)	14.2	14.0	13.8	13.8	14.1	14.2	14.5	14.6	14.6	14.5	14.3	14.2	14.7	14.6	14.2	14.1	14.3	14.6	14.3	14.6	14.9	14.6	14.2						14.4
Hours Per Ktonne	(hr/kt)	0.18871	0.18611	0.18259	0.18364	0.18685	0.18849	0.19285	0.19354	0.19340	0.19287	0.18949	0.18887	0.19550	0.19347	0.18960	0.18718	0.18961	0.19421	0.19046	0.19449	0.19741	0.19375	0.18829						0.19143
Mill Hours	(hours)	1,075	1,630	1,773	1,898	8,760	8,760	8,102	8,105	8,105	8,105	8,104	8,104	8,103	8,103	8,103	8,132	8,760	8,760	8,760	8,760	8,760	8,760	1,009						158,530
SOX Material (LIFO):																														
Ktonnes	(kt)							3,600	3,600	3,600	3,600	3,600	3,600	3,600	3,600	3,600	3,441													35,841
NSR Value	(C$/t)							36.31	39.07	34.32	33.36	32.28	32.68	35.86	35.86	35.86	35.16													35.08
Total Copper	(%)							0.410	0.445	0.283	0.250	0.195	0.178	0.185	0.185	0.185	0.191													0.251
Weak Soluble Copper	(%)							0.123	0.146	0.108	0.100	0.070	0.061	0.063	0.063	0.063	0.061													0.086
Gold	(g/t)							0.344	0.373	0.449	0.464	0.477	0.501	0.581	0.581	0.581	0.559													0.491
Molybdenum	(%)							0.0225	0.0294	0.0301	0.0303	0.0290	0.0271	0.0212	0.0212	0.0212	0.0202													0.0252
Silver	(g/t)							1.68	1.74	1.97	2.02	2.29	2.49	2.93	2.93	2.93	2.64													2.36
Recovered Copper	(%)							0.270	0.281	0.165	0.141	0.117	0.110	0.115	0.115	0.115	0.122													0.155
Bond Work Index	(Kwh/t)							13.8	13.7	13.7	13.7	13.7	13.7	13.7	13.7	13.7	13.7													13.7
Hours Per Ktonne	(hr/kt)							0.18287	0.18198	0.18202	0.18203	0.18212	0.18222	0.18252	0.18252	0.18252	0.18234													0.18231
Mill Hours	(hours)							658	655	655	655	656	656	657	657	657	627													6,534
Low Grade Mill Stock (LIFO):																														
Ktonnes	(kt)																							39,950	45,814	45,744	45,945	45,232	40,295	262,980
NSR Value	(C$/t)																							14.81	14.32	14.76	15.54	15.94	16.19	15.25
Total Copper	(%)																							0.106	0.123	0.111	0.107	0.138	0.153	0.123
Weak Soluble Copper	(%)																							0.000	0.010	0.009	0.001	0.014	0.031	0.011
Gold	(g/t)																							0.126	0.126	0.148	0.145	0.142	0.156	0.140
Molybdenum	(%)																							0.0149	0.0096	0.0116	0.0148	0.0112	0.0105	0.0121
Silver	(g/t)																							1.25	1.15	1.16	1.22	1.22	1.19	1.20
Recovered Copper	(%)																							0.097	0.105	0.095	0.097	0.116	0.114	0.104
Bond Work Index	(Kwh/t)																							14.6	14.4	14.4	14.4	14.6	14.4	14.4
Hours Per Ktonne	(hr/kt)																							0.19402	0.19121	0.19150	0.19066	0.19367	0.19081	0.19195
Mill Hours	(hours)																							7,751	8,760	8,760	8,760	8,760	7,689	50,480

Table 16-4: Mine Production Schedule

	(Units)	-3	-2	-1	Y1 Q1	Y1 Q2	Y1 Q3	Y1 Q4	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	Year 17	Year 18	Year 19	Year 20	Year 21	Year 22	Year 23	Year 24	Year 25	TOTAL
Direct Feed Material:																																	
NSR Cutoff Grade	(C$)	5.70	5.70	5.70	5.70	5.70	5.70	25.00	18.25	22.00	19.75	19.25	20.00	19.00	18.00	17.75	17.50	18.75	17.50	17.50	16.25	16.00	16.50	16.50	17.25	5.70	5.70						
Ktonnes	(kt)		220	4,448	4,030	5,757	9,711	10,334	46,882	46,472	42,009	41,877	41,906	42,025	42,769	42,910	41,447	41,882	42,737	43,446	46,199	45,105	45,993	45,039	44,374	45,213	5,360						828,145
NSR Value	(C$/t)		30.72	34.75	37.72	45.07	40.05	43.11	39.30	38.20	32.13	26.65	26.81	28.25	28.25	27.35	27.98	28.08	26.95	25.54	22.57	21.64	20.14	21.24	23.67	23.30	25.53						27.68
Total Copper	(%)		0.242	0.247	0.295	0.389	0.365	0.398	0.338	0.286	0.269	0.216	0.202	0.203	0.204	0.189	0.260	0.237	0.212	0.195	0.196	0.183	0.154	0.153	0.169	0.153	0.193						0.218
Weak Soluble Copper	(%)		0.026	0.032	0.058	0.067	0.063	0.058	0.028	0.006	0.025	0.013	0.000	0.000	0.000	0.003	0.028	0.015	0.008	0.007	0.016	0.011	0.000	0.000	0.000	0.000	0.000						0.011
Gold	(g/t)		0.332	0.437	0.460	0.491	0.399	0.404	0.357	0.358	0.313	0.265	0.254	0.248	0.239	0.226	0.210	0.213	0.215	0.205	0.167	0.190	0.188	0.181	0.193	0.186	0.201						0.242
Molybdenum	(%)		0.0177	0.0135	0.0169	0.0198	0.0237	0.0288	0.0297	0.0323	0.0190	0.0144	0.0185	0.0271	0.0306	0.0346	0.0279	0.0303	0.0295	0.0284	0.0219	0.0148	0.0127	0.0199	0.0269	0.0321	0.0280						0.0249
Silver	(g/t)		1.92	2.30	2.87	3.22	2.37	2.03	1.90	1.95	1.98	1.95	1.78	2.18	1.68	1.67	1.89	1.93	1.65	1.52	1.75	1.58	1.66	2.21	1.61	1.73	1.25						1.83
Recovered Copper	(%)		0.203	0.202	0.222	0.302	0.284	0.319	0.288	0.259	0.227	0.188	0.186	0.187	0.188	0.173	0.216	0.206	0.189	0.174	0.167	0.161	0.142	0.141	0.156	0.141	0.178						0.192
Bond Work Index	(Kwh/t)		13.7	14.4	14.1	13.8	13.8	13.8	14.1	14.2	14.5	14.6	14.6	14.5	14.3	14.2	14.7	14.6	14.2	14.1	14.3	14.6	14.3	14.6	14.9	14.6	14.2						14.4
Hours Per Ktonne	(hr/kt)		0.18207	0.19152	0.18741	0.18375	0.18259	0.18364	0.18685	0.18849	0.19285	0.19354	0.19340	0.19287	0.18949	0.18887	0.19550	0.19347	0.18960	0.18718	0.18961	0.19421	0.19046	0.19449	0.19741	0.19375	0.18829						0.19143
Mill Hours	(hours)		40	852	755	1,058	1,773	1,898	8,760	8,760	8,102	8,105	8,105	8,105	8,104	8,104	8,103	8,103	8,103	8,132	8,760	8,760	8,760	8,760	8,760	8,760	1,009						158,530
SOX Stockpile:																																	
NSR Cutoff Grade	(C$)		20.00	20.00	20.00	20.00	20.00	20.00	20.00																								
Ktonnes	(kt)		1,894	13,042	5,595	7,504	5,527	1,501	778																								35,841
NSR Value	(C$/t)		34.58	35.86	31.92	33.36	39.07	38.39	27.60																								35.08
Total Copper	(%)		0.195	0.185	0.176	0.250	0.445	0.467	0.242																								0.251
Weak Soluble Copper	(%)		0.060	0.063	0.060	0.100	0.146	0.136	0.058																								0.086
Gold	(g/t)		0.541	0.581	0.482	0.464	0.373	0.330	0.322																								0.491
Molybdenum	(%)		0.0193	0.0212	0.0285	0.0303	0.0294	0.0221	0.0116																								0.0252
Silver	(g/t)		2.4000	2.93	2.38	2.02	1.74	1.52	1.90																								2.36
Recovered Copper	(%)		0.127	0.115	0.109	0.141	0.281	0.311	0.173																								0.155
Bond Work Index	(Kwh/t)		13.7	13.7	13.7	13.7	13.7	13.8	13.9																								13.7
Hours Per Ktonne	(hr/kt)		0.18219	0.18252	0.18215	0.18203	0.18198	0.18277	0.18459																								0.18231
Mill Hours	(hours)		345	2,380	1,019	1,366	1,006	274	144																								6,534
Low Grade:																																	
NSR Cutoff Grade	(C$)			12.00	12.00	12.00	12.00	12.00	12.00	12.00	12.00	12.00	12.00	12.00	12.00	12.00	12.00	12.00	12.00	12.00	12.00	12.00	12.00	12.00	12.00								
Ktonnes	(kt)			159	409	1,135	498	2,639	5,487	7,259	20,570	36,748	24,138	16,326	9,937	12,922	11,222	16,035	10,252	9,137	11,489	19,954	28,327	11,600	6,737								262,980
NSR Value	(C$/t)			17.60	17.52	17.00	16.22	19.57	15.00	16.53	15.90	15.88	16.13	15.69	15.02	14.72	14.72	14.92	14.62	14.57	14.34	14.08	14.70	14.81	15.14								15.25
Total Copper	(%)			0.156	0.122	0.132	0.188	0.188	0.158	0.124	0.159	0.143	0.121	0.107	0.096	0.094	0.115	0.114	0.111	0.131	0.137	0.118	0.109	0.102	0.102								0.123
Weak Soluble Copper	(%)			0.047	0.061	0.054	0.080	0.029	0.032	0.013	0.035	0.017	0.002	0.000	0.001	0.004	0.015	0.008	0.004	0.016	0.021	0.006	0.000	0.000	0.000								0.011
Gold	(g/t)			0.214	0.257	0.208	0.159	0.178	0.143	0.189	0.141	0.142	0.145	0.135	0.154	0.157	0.157	0.145	0.137	0.129	0.122	0.125	0.130	0.124	0.117								0.140
Molybdenum	(%)			0.0067	0.0195	0.0232	0.0180	0.0107	0.0040	0.0080	0.0121	0.0106	0.0131	0.0175	0.0135	0.0134	0.0102	0.0121	0.0113	0.0101	0.0088	0.0089	0.0122	0.0164	0.0211								0.0121
Silver	(g/t)			1.72	1.42	1.40	1.18	1.38	0.89	1.19	1.22	1.21	1.24	1.32	1.11	1.06	1.21	1.18	1.16	1.13	1.28	1.06	1.24	1.43	0.97								1.20
Recovered Copper	(%)			0.103	0.057	0.073	0.101	0.148	0.118	0.103	0.117	0.118	0.110	0.098	0.087	0.083	0.094	0.098	0.100	0.106	0.109	0.104	0.100	0.094	0.094								0.104
Bond Work Index	(Kwh/t)			13.7	13.7	13.7	13.8	14.3	14.5	14.4	14.4	14.6	14.5	14.3	14.3	14.3	14.6	14.5	14.1	14.1	14.3	14.4	14.5	14.7	14.7								14.4
Hours Per Ktonne	(hr/kt)			0.18207	0.18207	0.18207	0.18322	0.19036	0.19325	0.19134	0.19061	0.19391	0.19290	0.18974	0.18947	0.19009	0.19463	0.19237	0.18816	0.18806	0.19087	0.19203	0.19294	0.19538	0.19518								0.19195
Mill Hours	(hours)			29	74	207	91	502	1,060	1,389	3,921	7,126	4,656	3,098	1,883	2,456	2,184	3,085	1,929	1,718	2,193	3,832	5,465	2,266	1,315								50,480
Leach Material:																																	
NSR Cutoff Grade	(C$)	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46											
Ktonnes	(kt)	5,487	13,831	23,056	6,638	4,940	3,079	4,726	34,372	17,355	13,720	818	3,154	10,002	13,669	14,338	19,939	1,240	6,341	4,531	2,523	31											203,790
NSR Value	(C$/t)	11.49	18.68	17.52	15.70	11.88	10.37	11.90	10.88	9.37	7.52	7.01	8.69	9.93	10.41	8.90	8.98	7.30	6.89	7.12	7.49	7.44											11.29
Gold	(g/t)	0.273	0.436	0.406	0.366	0.274	0.232	0.279	0.256	0.211	0.165	0.149	0.199	0.230	0.240	0.203	0.200	0.156	0.146	0.152	0.152	0.156											0.259
Silver	(g/t)	1.30	2.68	2.85	2.54	1.93	1.63	1.37	1.480	2.130	1.340	0.840	2.610	2.370	2.200	1.740	1.570	0.990	1.630	1.690	2.160	1.310											1.95
Total Copper	(%)	0.021	0.044	0.044	0.035	0.036	0.052	0.031	0.021	0.030	0.044	0.065	0.006	0.013	0.022	0.026	0.045	0.061	0.049	0.047	0.066	0.067											0.034
Weak Soluble Copper	(%)	0.004	0.009	0.010	0.011	0.011	0.012	0.008	0.005	0.008	0.015	0.016	0.002	0.004	0.011	0.006	0.009	0.012	0.013	0.012	0.011	0.012											0.009
Molybdenum	(%)	0.0063	0.0115	0.0160	0.0227	0.0230	0.0210	0.0041	0.0061	0.0075	0.0069	0.0026	0.0099	0.0246	0.0227	0.0256	0.0262	0.0012	0.0077	0.0081	0.0052	0.0036											0.0144
Tonnage Summary:																																	
Direct Feed Material:	(kt)	0	220	4,448	4,030	5,757	9,711	10,334	46,882	46,472	42,009	41,877	41,906	42,025	42,769	42,910	41,447	41,882	42,737	43,446	46,199	45,105	45,993	45,039	44,374	45,213	5,360						828,145
SOX Stockpile	(kt)	0	1,894	13,042	5,595	7,504	5,527	1,501	778	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0						35,841
Low Grade	(kt)	0	0	159	409	1,135	498	2,639	5,487	7,259	20,570	36,748	24,138	16,326	9,937	12,922	11,222	16,035	10,252	9,137	11,489	19,954	28,327	11,600	6,737	0	0						262,980
Leach Material:	(kt)	5,487	13,831	23,056	6,638	4,940	3,079	4,726	34,372	17,355	13,720	818	3,154	10,002	13,669	14,338	19,939	1,240	6,341	4,531	2,523	31	0	0	0	0	0						203,790
Total Material	(kt)	7,424	19,758	47,818	20,000	25,000	25,000	25,000	99,896	100,000	99,453	99,485	100,000	100,000	100,000	100,000	100,000	100,000	100,000	97,250	93,901	97,797	97,158	68,314	56,251	45,994	5,372						1,830,871
Waste Material	(kt)	1,937	3,813	7,113	3,328	5,664	6,185	5,800	12,377	28,914	23,154	20,042	30,802	31,647	33,625	29,830	27,392	40,843	40,670	40,136	33,690	32,707	22,838	11,675	5,140	781	12						500,115
Stockpile Rehandle	(kt)				1,668	3,000					3,600	3,600	3,600	3,600	3,600	3,600	3,600	3,600	3,600	3,441							39,950	45,814	45,744	45,945	45,232	40,295	303,489

Table 16-5: Production Schedule for Leach Material

	(Units)	-3	-2	-1	Y1 Q1	Y1 Q2	Y1 Q3	Y1 Q4	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	Year 17	Year 18	Year 19	Year 20	TOTAL
Leach Pad Stacking Schedule:																												
NSR Cutoff Grade	(C$)	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	
Ktonnes	(kt)	5,487	9,125	9,125	2,281	2,281	2,281	2,282	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	6,678	203,790
NSR Value	(C$/t)	11.49	18.68	17.52	15.70	11.88	10.37	11.90	10.88	9.37	7.52	8.23	9.38	9.93	10.41	8.90	8.98	8.75	7.53	8.02	9.45	10.87	10.88	11.19	15.46	17.52	18.34	11.29
Gold	(g/t)	0.273	0.436	0.406	0.366	0.274	0.232	0.279	0.256	0.211	0.165	0.182	0.214	0.230	0.240	0.203	0.200	0.194	0.162	0.178	0.212	0.256	0.256	0.261	0.359	0.406	0.427	0.259
Silver	(g/t)	1.30	2.68	2.85	2.54	1.93	1.63	1.37	1.48	2.13	1.34	1.62	2.19	2.37	2.20	1.74	1.57	1.49	1.61	1.71	2.13	1.48	1.48	1.50	2.51	2.85	2.73	1.95
Total Copper	(%)	0.021	0.044	0.044	0.035	0.036	0.052	0.031	0.021	0.030	0.044	0.040	0.020	0.013	0.022	0.026	0.045	0.047	0.048	0.037	0.034	0.021	0.021	0.028	0.038	0.044	0.044	0.034
Weak Soluble Copper	(%)	0.004	0.009	0.010	0.011	0.011	0.012	0.008	0.005	0.008	0.015	0.012	0.005	0.004	0.011	0.006	0.009	0.009	0.012	0.009	0.010	0.005	0.005	0.007	0.011	0.010	0.009	0.009
Molybdenum	(%)	0.0063	0.0115	0.0160	0.0227	0.0230	0.0210	0.0041	0.0061	0.0075	0.0069	0.0068	0.0081	0.0246	0.0227	0.0256	0.0262	0.0228	0.0133	0.0169	0.0175	0.0061	0.0061	0.0082	0.0207	0.0160	0.0128	0.0144
As Produced From Mine:																												
NSR Cutoff Grade	(C$)	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46						
Ktonnes	(kt)	5,487	13,831	23,056	6,638	4,940	3,079	4,726	34,372	17,355	13,720	818	3,154	10,002	13,669	14,338	19,939	1,240	6,341	4,531	2,523	31						203,790
NSR Value	(C$/t)	11.49	18.68	17.52	15.70	11.88	10.37	11.90	10.88	9.37	7.52	7.01	8.69	9.93	10.41	8.90	8.98	7.30	6.89	7.12	7.49	7.44						11.29
Gold	(g/t)	0.273	0.436	0.406	0.366	0.274	0.232	0.279	0.256	0.211	0.165	0.149	0.199	0.230	0.240	0.203	0.200	0.156	0.146	0.152	0.152	0.156						0.259
Silver	(g/t)	1.30	2.68	2.85	2.54	1.93	1.63	1.37	1.48	2.13	1.34	0.84	2.61	2.37	2.20	1.74	1.57	0.99	1.63	1.69	2.16	1.31						1.95
Total Copper	(%)	0.021	0.044	0.044	0.035	0.036	0.052	0.031	0.021	0.030	0.044	0.065	0.006	0.013	0.022	0.026	0.045	0.061	0.049	0.047	0.066	0.067						0.034
Weak Soluble Copper	(%)	0.004	0.009	0.010	0.011	0.011	0.012	0.008	0.005	0.008	0.015	0.016	0.002	0.004	0.011	0.006	0.009	0.012	0.013	0.012	0.011	0.012						0.009
Molybdenum	(%)	0.0063	0.0115	0.0160	0.0227	0.0230	0.0210	0.0041	0.0061	0.0075	0.0069	0.0026	0.0099	0.0246	0.0227	0.0256	0.0262	0.0012	0.0077	0.0081	0.0052	0.0036						0.0144
Direct to Crusher:		100.00%	65.97%	39.58%	34.36%	46.17%	74.08%	48.29%	26.55%	52.58%	66.51%	100.00%	100.00%	91.23%	66.76%	63.64%	45.76%	100.00%	100.00%	100.00%	100.00%	100.00%						
Ktonnes	(kt)	5,487	9,125	9,125	2,281	2,281	2,281	2,282	9,125	9,125	9,125	818	3,154	9,125	9,125	9,125	9,125	1,240	6,341	4,531	2,523	31						115,375
NSR Value	(C$/t)	11.49	18.68	17.52	15.70	11.88	10.37	11.90	10.88	9.37	7.52	7.01	8.69	9.93	10.41	8.90	8.98	7.30	6.89	7.12	7.49	7.44						10.80
Gold	(g/t)	0.273	0.436	0.406	0.366	0.274	0.232	0.279	0.256	0.211	0.165	0.149	0.199	0.230	0.240	0.203	0.200	0.156	0.146	0.152	0.152	0.156						0.247
Silver	(g/t)	1.30	2.68	2.85	2.54	1.93	1.63	1.37	1.48	2.13	1.34	0.84	2.61	2.37	2.20	1.74	1.57	0.99	1.63	1.69	2.16	1.31						1.95
Total Copper	(%)	0.021	0.044	0.044	0.035	0.036	0.052	0.031	0.021	0.030	0.044	0.065	0.006	0.013	0.022	0.026	0.045	0.061	0.049	0.047	0.066	0.067						0.034
Weak Soluble Copper	(%)	0.004	0.009	0.010	0.011	0.011	0.012	0.008	0.005	0.008	0.015	0.016	0.002	0.004	0.011	0.006	0.009	0.012	0.013	0.012	0.011	0.012						0.009
Molybdenum	(%)	0.0063	0.0115	0.0160	0.0227	0.0230	0.0210	0.0041	0.0061	0.0075	0.0069	0.0026	0.0099	0.0246	0.0227	0.0256	0.0262	0.0012	0.0077	0.0081	0.0052	0.0036						0.0145
To Leach Stockpile:		0.00%	34.03%	60.42%	65.64%	53.83%	25.92%	51.71%	73.45%	47.42%	33.49%	0.00%	0.00%	8.77%	33.24%	36.36%	54.24%	0.00%	0.00%	0.00%	0.00%	0.00%						
Ktonnes	(kt)	0	4,706	13,931	4,357	2,659	798	2,444	25,247	8,230	4,595	0	0	877	4,544	5,213	10,814	0	0	0	0	0						88,415
NSR Value	(C$/t)	11.49	18.68	17.52	15.70	11.88	10.37	11.90	10.88	9.37	7.52	7.01	8.69	9.93	10.41	8.90	8.98	7.30	6.89	7.12	7.49	7.44						11.93
Gold	(g/t)	0.273	0.436	0.406	0.366	0.274	0.232	0.279	0.256	0.211	0.165	0.149	0.199	0.230	0.240	0.203	0.200	0.156	0.146	0.152	0.152	0.156						0.276
Silver	(g/t)	1.30	2.68	2.85	2.54	1.93	1.63	1.37	1.48	2.13	1.34	0.84	2.61	2.37	2.20	1.74	1.57	0.99	1.63	1.69	2.16	1.31						1.95
Total Copper	(%)	0.021	0.044	0.044	0.035	0.036	0.052	0.031	0.021	0.030	0.044	0.065	0.006	0.013	0.022	0.026	0.045	0.061	0.049	0.047	0.066	0.067						0.033
Weak Soluble Copper	(%)	0.004	0.009	0.010	0.011	0.011	0.012	0.008	0.005	0.008	0.015	0.016	0.002	0.004	0.011	0.006	0.009	0.012	0.013	0.012	0.011	0.012						0.008
Molybdenum	(%)	0.0063	0.0115	0.0160	0.0227	0.0230	0.0210	0.0041	0.0061	0.0075	0.0069	0.0026	0.0099	0.0246	0.0227	0.0256	0.0262	0.0012	0.0077	0.0081	0.0052	0.0036						0.0142
Stockpile Reclaim:																												
Ktonnes	(kt)											8,307	5,971					7,885	2,784	4,594	6,602	9,094	9,125	9,125	9,125	9,125	6,678	88,415
NSR Value	(C$/t)											8.35	9.74					8.98	8.98	8.90	10.20	10.88	10.88	11.19	15.46	17.52	18.34	11.93
Gold	(g/t)											0.186	0.222					0.200	0.200	0.203	0.235	0.256	0.256	0.261	0.359	0.406	0.427	0.276
Silver	(g/t)											1.69	1.97					1.57	1.57	1.73	2.12	1.48	1.48	1.50	2.51	2.85	2.73	1.95
Total Copper	(%)											0.038	0.028					0.045	0.045	0.027	0.021	0.021	0.021	0.028	0.038	0.044	0.044	0.033
Weak Soluble Copper	(%)											0.012	0.007					0.009	0.009	0.006	0.009	0.005	0.005	0.007	0.011	0.010	0.009	0.008
Molybdenum	(%)											0.0072	0.0072					0.0262	0.0262	0.0256	0.0222	0.0061	0.0061	0.0082	0.0207	0.0160	0.0128	0.0142

16.5 WASTE MANAGEMENT

Total waste in the IMC mine plan amounts to 500.1 million tonnes. This material is disposed in the tailing management facility. Figure 16-3 shows three facilities for mine waste: 1) North Waste which contains 200.6 million tonnes, 2) South 1 Waste which contains 154.8 million tonnes, and 3) South 2 Waste which contains 144.7 million tonnes. The material will be placed by trucks and dozers, the rising water level in the TMF facility will cover the material relatively quickly, usually one to two years. The waste material by material type is as follows:

- 55.7 million tonnes of overburden.
- 115.9 million tonnes of leach cap material.
- 24.2 million tonnes of supergene oxide material.
- 97.5 million tonnes of supergene sulphide material.
- 206.8 million tonnes of hypogene material.

Additional rock storage facilities during the life of the project include:

- The heap leach pad which at the end of the project will contain 203.8 million tonnes of spent, non-reactive material, assuming all the potential leach material is processed.
- A low-grade stockpile southeast of the pit that has the capacity for 178.3 million tonnes, and a low-grade stockpile east of the pit that contains 84.7 million tonnes, both which will be processed at the end of the mine life.
- There will also be supergene oxide (SOX) stockpile south of the pit to store Phase I SOX. It will be reclaimed during mining Years 4 through 13. The maximum size of this facility is estimated at 35.8 million tonnes. The SOX stockpile and the leach pad overlap by a small amount, but the SOX stockpile will be reclaimed before the leach pad gets to its final limits.
- There will be a stockpile for leach resource east of the pit. This is expected to reach a maximum size of 74.1 million tonnes during Year 10 and will be reclaimed by the end of Year 20.

The stockpiles are all constructed in lifts from the bottom up. The low-grade stockpile, leach stockpile, and SOX stockpile are designed with 30 m lifts at angle of repose with a 20 m setback between lifts to make the overall slope angle about 2H:1V. This is assumed to be adequate since these are not permanent facilities.

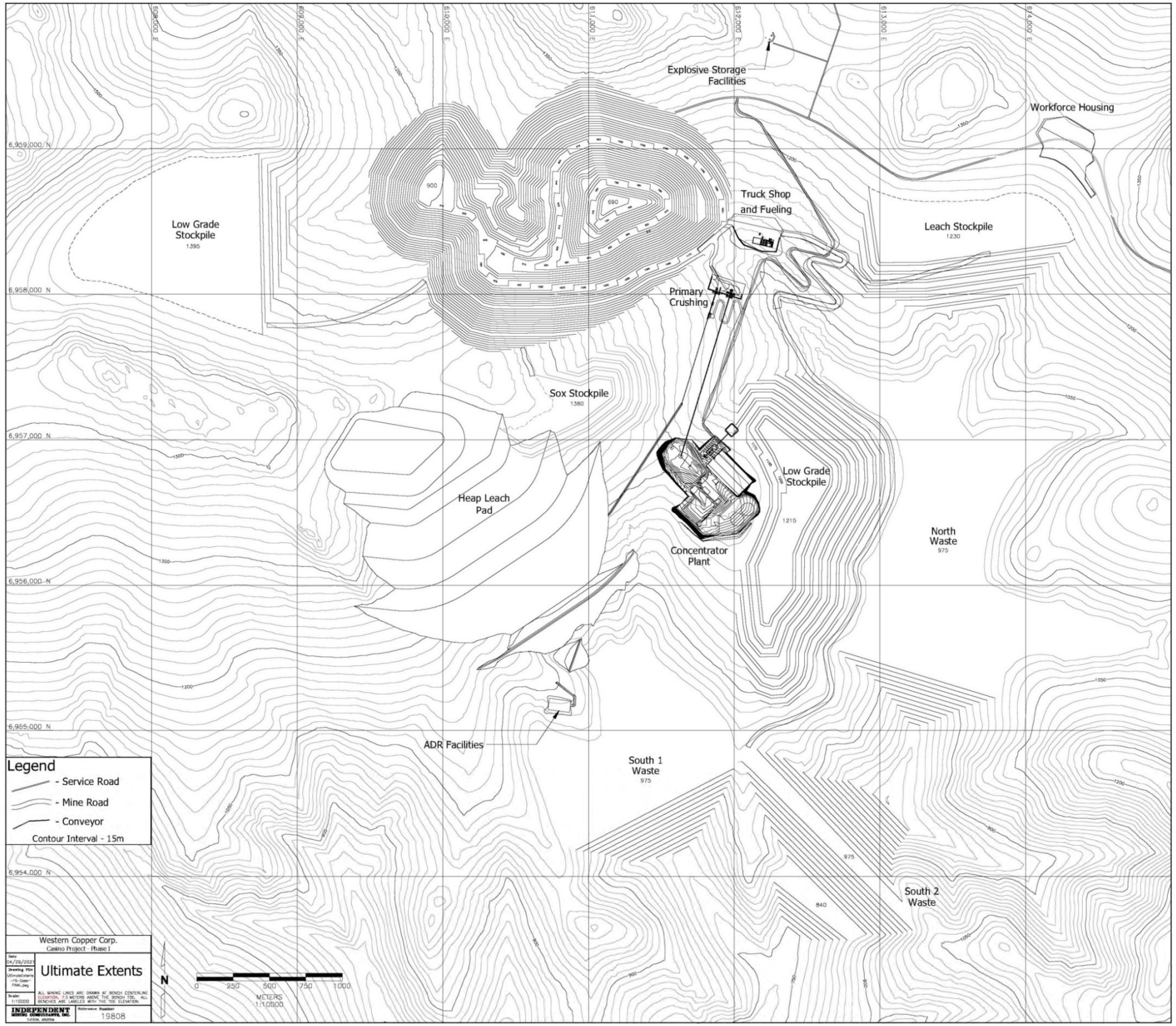


Figure 16-3: Maximum Extent of Waste Storage Areas and Stockpiles (IMC, 2021)

16.6 MINING EQUIPMENT

Mine equipment requirements were sized and estimated on a first principles basis, based on the mine production schedule, the mine work schedule, and estimated equipment shift productivity rates. The size and type of mining equipment is consistent with the size of the project, i.e. peak material movements of 100 Mt/y. The mine equipment estimate is based on owner operation and assumes a well-managed mining operation with a well-trained labour pool, and that all the equipment is new at the start of the operation.

Table 16-6 summarizes the major equipment requirements. The first column shows initial equipment for the first year of mine development (Year -3), the second column shows equipment required for the beginning of commercial production and the third column shows the peak fleet requirements. This represents the equipment required to perform the following duties:

- Develop access roads from the mine to the crusher, various stockpiles, and the waste storage area.
- Mine and transport mill material and leach material to the crushers.
- Mine and transport resource to various stockpiles as required.
- Reclaim stockpiled material and transport it to the crushers.
- Mine and transport waste to the waste storage areas in the TMF.
- Maintain the haul roads and stockpiles and various truck dumping sites.

Table 16-6: Mining Equipment Requirements

Equipment Type	Capacity/ Power	Year -3	Year 1	Peak
P&H 320XPC Drill	(314 mm)	1	4	5
P&H 4100XPC Cable Shovel	(67.6 cu m)	0	2	2
Komatsu PC8000-6 Hyd Shovel	(42 cu m)	0	1	1
Komatsu WA1200-6 Wheel Loa	(20 cu m)	1	1	1
Komatsu 980E Truck	(370 mt)	0	16	21
Komatsu HD1500 Truck	(144 mt)	7	6	7
Komatsu D475A Track Dozer	(664 kw)	0	3	3
Komatsu D375A Track Dozer	(455 kw)	2	3	3
Komatsu WD900 Wheel Dozer	(637 kw)	0	3	3
Komatsu GD825A Motor Grade	(209 kw)	1	3	3
Water Truck - 30,000 gal	(113,550 l)	1	3	3
Komatsu PC360LC-11 Excavat	(1.96 cu m)	1	2	2
Epiroc SmartROC D65	(178 mm)	1	2	2
TOTAL		15	49	56

17 RECOVERY METHODS

17.1 PROCESS DESCRIPTION

The Casino process plant will consist of two processing facilities, one for sulphide mineralized material and one for oxide mineralized material.

The sulphide mineralized material processing facility will produce mineral concentrates of copper and molybdenum using conventional flotation technology. The copper concentrate will be dewatered and transported as a filtered cake by highway trucks. The molybdenum concentrate will be dewatered and packaged in super sacks for transport. Gold and silver contained in the sulphide mineralized material will be recovered as a fraction of the copper concentrate.

The oxide mineralized material processing facility will produce gold and silver Doré bars via heap leach and carbon adsorption technology. Copper contained in the oxide mineralized material will be recovered as a copper sulphide precipitate using SART technology.

17.2 SULPHIDE MINERALIZED MATERIAL PROCESS PLANT DESCRIPTION

17.2.1 Process Design Criteria and Major Equipment

Process design criteria were developed for the Sulphide facility based on a 120,000 t/d (43,800,000 t/y) plant design. The crushing circuit was designed to operate with an overall availability of 80%. The remainder of the sulfide facilities were designed to operate with an overall availability of 93%. The equipment was sized using these criteria. Table 17-1 is a summary of the main components of the sulphide process design criteria used for the study. Table 17-2 is a summary of the major process equipment selected for the study.

Table 17-1: Sulphide Process Design Criteria

Description	Unit	Value
General		
Type of Deposit	-	Supergene Copper Oxide Zone Supergene Sulphide Zone Hypogene Zone
Mill Feed Characteristics		
Specific Gravity	-	2.7
Moisture Content	%	3
Abrasion Index (Average)	-	0.265
Bond Rod Mill Work Index	kWh/t	9.9
Bond Ball Mill Work Index	kWh/t	14.5
Operating Schedule		
Shift/Day	-	2
Hours/Shift	H	12
Hours/Day	H	24
Days/Year	D	365
Plant Availability/Utilization		
Overall Plant Feed	t/y	43,800,000
Overall Plant Feed	t/d	120,000
Crusher Plant Availability	%	80
Grinding and Flotation Plant Availability	%	93
Crushing Rate	t/h	6,667
Grinding Rate	t/h	5,376
Flotation Rate	t/h	5,376
Design Factor	-	TBD

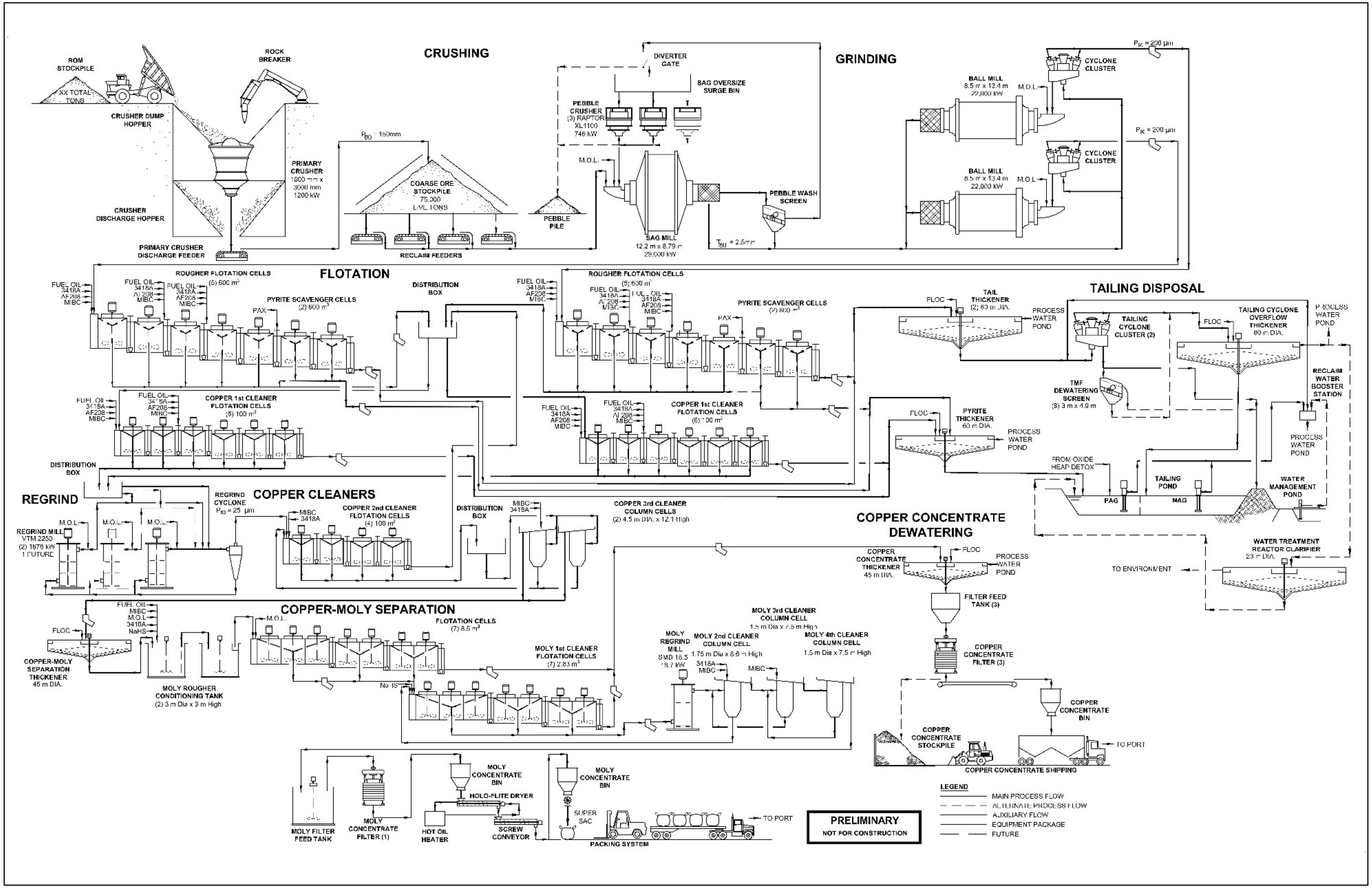
Description	Unit	Value
Head Grades (Design)		
Copper	%	0.293
Molybdenum	%	0.0236
Gold	g/t	0.357
Silver	g/t	2.050
Copper Recovery Supergene Sulfide and Mixed Supergene Mineralized Materials		
Copper recovery to copper concentrate	%	94
Gold recovery to copper concentrate	%	69
Silver recovery to copper concentrate	g/t	60
Molybdenum recovery to molybdenum concentrate	g/t	52.3
Copper Concentrate Grade		
Copper	%	TBD
Molybdenum	%	50
Copper Recovery Hypogene Mineralized Materials		
Copper recovery to copper concentrate	%	92
Gold recovery to copper concentrate	%	66
Silver recovery to copper concentrate	g/t	50
Molybdenum recovery to molybdenum concentrate	g/t	78.6
Copper Concentrate Grade		
Copper	%	TBD
Molybdenum	%	TBD
Copper Concentrate Mass Recovery (Design)	%	
Copper Concentrate Production (Design)	t/y	

Table 17-2: Major Process Equipment

Item	Number	Description	Key Criteria (ea)
Primary Crusher	1	1600 x 3000 TSU Top Service Gyratory Crusher	1,200 kW
SAG Mill	1	12.19m x 8.79m (40ft x 29ft) (EGL 7.99m)	29,000 kW
Pebble Crusher	3	Raptor XL 1300	970 kW
Ball Mill	2	8.53m x 13.72m (28ft x 45ft) (inside shell diameter x EGL)	22,000 kW
Flotation			
Copper Rougher Scavenger	10	FLS nextStep, RT, 600 m3	375 kW
Pyrite Scavenger	2	FLS nextStep, RT, 600 m3	375 kW
Pyrite Scavenger	2	FLS nextStep, RT, 600 m3	375 kW
Copper 1st Cleaner	12	FLS nextStep, RT, 100 m3	132 kW
Copper 2nd Cleaner	4	FLS nextStep, RT, 100 m3	132 kW
Copper 3rd Cleaner	2	Column Cell, 4.5 m diameter x 12.1 m high	75 kW
Copper-Moly Separation	7	Wemco 1+1, 8.5 m3	30 kW
Moly 1st Cleaner	7	Wemco 1+1, 2.8 m3	15 kW
Moly 2nd Cleaner	1	Column Cell, 1.75 m diameter x 8.6 m high	40 kW
Moly 3rd Cleaner	1	Column Cell, 1.5 m diameter x 7.5 m high	40 kW
Moly 4th Cleaner	1	Column Cell, 1.5 m diameter x 7.5 m high	40 kW
Cu. Regrind	2	Vertical Grinding, VTM 2250-WB	1,679 kW
Moly Regrind	1	SMD 18.5	19 kW
Dewatering and Filtration			
Cu-Moly Separation Thickener	1	High-rate thickener	45 m diameter
Cu Concentrate Thickener	1	High-rate thickener	45 m diameter
Tailings Thickener	2	High-rate thickener	80 m diameter

Item	Number	Description	Key Criteria (ea)
Tailing Cyclone Overflow Thickener	1	High-rate thickener	80 m diameter
Pyrite Thickener	1	High-rate thickener	60 m diameter
Copper Concentrate Filter	3	Tower Type - Pressure Filter, FLS Pneumapress M30-12	31.1 m^2 area
Moly Concentrate filter	1	Tower Type - Pressure Filter, FLS Pneumapress M19-2	3.3 m^2 area

Figure 17-1 is a simplified schematic of the overall process for the sulphide mineralized material. This provides the basis for the process description that follows.



(Source: M3)

Figure 17-1: Simplified Sulfide Process Flowsheet

The following items summarise the process operations required to extract copper and molybdenum from the sulphide mineralized material:

- Size reduction of the run-of-mine (ROM) material (900 mm max) to minus 200 mm.
- Stockpiling primary crushed mineralized material and then reclaiming with feeders and a belt conveyor.
- Size reduction of the mineralized material in a semi-autogenous (SAG) mill - ball mill grinding circuit with pebble crushing.
- Concentration and separation of the copper and molybdenum sulphide minerals by froth flotation to produce a bulk (copper/molybdenum) concentrate.
- Separation of the bulk concentrate into separate copper and molybdenum concentrates.
- Final copper concentrate will be thickened, filtered, and loaded in highway haul trucks for shipment.
- Final molybdenum concentrate will be filtered, dried, and packaged in bags for shipment.
- Concentration of the bulk flotation tailings in a pyrite flotation circuit. Pyrite flotation circuit tailings will have a low sulphide sulfur concentration.
- Subaqueous deposition of the pyrite flotation concentrate (PAG tailings) in the TMF.
- Flotation tailings will be thickened and transported by a gravity pipeline to a tailings impoundment area. The tailing will be cycloned with the underflow recovered as sand for tailing dam construction and overflow reporting to the tailing disposal impoundment site.
- Storing, preparing, and distributing reagents used in the sulphide mineralized material process.
- Water from tailings and concentrate dewatering will be recycled for reuse in the process. Plant water stream types include process water, fresh water, potable water, and fire water

17.2.2 Crushing and Coarse Mineralized Material Stockpile

Run-of-Mine (ROM) sulphide mineralized material will be trucked from the mine to the primary crusher and fed to the crusher via a dump pocket. The primary crusher will be a 1,600 mm x 3,000 mm (63”x118”), or equivalent, gyratory crusher, with an open side setting of 200 mm. The crushed mineralized material will drop into a discharge bin equipped with an apron feeder. The apron feeder will discharge onto a belt conveyor that will discharge the primary crushed mineralized material to a covered, conical mineralized material stockpile.

Primary crushed mineralized material will be stockpiled on the ground in a covered, conical mineralized material stockpile. A reclaim tunnel will be installed beneath the stockpile. The stockpile will contain approximately 75,000 tonnes of “live” mineralized material storage. Mineralized material will be moved from the “dead” storage area to the “live” storage area by front-end loader or bulldozer.

Mineralized material for the single grinding line will be withdrawn from the coarse mineralized material reclaim stockpile by variable speed, apron feeders. The feeders will discharge to a conveyor belt which will provide new feed to the SAG mill in the primary grinding circuit.

17.2.2.1 Grinding and Classification

Mineralized material will be ground to rougher flotation feed size in two stages: first, a SAG mill circuit with a single SAG Mill and, second, a ball mill circuit with two ball mills operating in parallel. The SAG mill will operate in closed circuit with a trommel, a pebble wash screen, and pebble crushers. The two ball mills will operate in closed circuit with hydrocyclones.

The SAG mill, 12.19 m diameter x 8.84 m (11.38 m ID x 7.99 m Effective Grinding Length (EGL)), will be equipped with a 29 MW gearless wrap-around drive. SAG mill product (T80 = 2 to 2.5 mm) will discharge through a trommel screen. Trommel undersize will flow by gravity to the primary cyclone feed sump where it will combine with the discharge of the ball mills. Trommel oversize will discharge to a pebble wash screen. Oversize from the pebble wash screen will be transported by belt conveyors to the pebble crushing circuit.

The pebble crushing circuit will consist of a surge bin, belt feeders and three shorthead, cone type crushers, each equipped with 970 kW, or equivalent, drives. The cone crushers will discharge onto the SAG feed conveyor. Pebbles (P80=12mm) may bypass the pebble crushing circuit via a diverter gate, ahead of the pebble crusher surge bin, to the SAG feed conveyor.

Secondary grinding will be performed in two ball mills operated in parallel. Each ball mill, 8.53 m diameter x 13.4 m (8.34 m ID x 13.26 m EGL), will be equipped with a 22 MW gearless wrap-around drive. Each ball mill will operate in closed circuit with a single cluster of hydrocyclones. Discharge from both ball mills will be combined with the undersize from the SAG mill trommel and pebble wash screen in the primary cyclone feed sump and will be pumped to the hydrocyclone clusters via variable speed horizontal centrifugal slurry pumps. Hydrocyclone underflow will return by gravity to the ball mills. Hydrocyclone overflow (final grinding circuit product), with a target particle size distribution of 80 percent finer than 200 microns, will flow by gravity to the flotation circuit.

17.2.3 Flotation

17.2.3.1 Bulk (Copper/Molybdenum) Flotation

Hydrocyclone overflow will flow by gravity to the bulk (copper-moly) flotation circuit. The copper-moly flotation circuit will consist of two rows of mechanical rougher flotation cells, two rows of mechanical first cleaner flotation cells, two concentrate regrind mills operated in closed circuit with hydrocyclones, one row of mechanical second cleaner flotation cells, and two copper-moly third cleaner flotation column cells.

Rougher flotation concentrate will flow by gravity to a sump and will be pumped by variable speed, horizontal centrifugal pumps to the first cleaner flotation circuit. Tailing from the rougher flotation cells will flow by gravity to the pyrite scavenger flotation circuit.

Pyrite concentrate will join the first cleaner tailing at the pyrite thickener. Tailing from the pyrite flotation circuit (final tailing) will flow by gravity to the tailing thickeners.

First cleaner flotation concentrate will flow by gravity to the regrind cyclone feed sump. Tailing from the first cleaner flotation cells will be combined with the concentrate from the pyrite scavenger flotation section in the pyrite thickener.

Copper-moly concentrate regrinding will be performed in two vertical grinding mills operated in parallel. The vertical mills will operate in closed circuit with hydrocyclones. Vertical mill discharge will be combined with copper-moly first cleaner flotation concentrate in the regrind cyclone feed sump and will be pumped by variable speed, horizontal centrifugal slurry pump to a dedicated hydrocyclone cluster for each regrind mill. Hydrocyclone underflow will report back to the respective regrind mill. Hydrocyclone overflow (final regrind circuit product), with a target particle size distribution of 80 percent finer than 25 microns, will flow by gravity to the copper second cleaner flotation circuit.

Second cleaner concentrate will flow by gravity to the third cleaner feed sump and will be pumped by horizontal centrifugal pumps to the third cleaner columns for upgrading. Tailing from the second cleaner flotation cells will return to the first cleaner flotation circuit.

Third cleaner concentrate will flow by gravity to the molybdenum separation circuit. Tailing from the third cleaner flotation columns will return to the second cleaner flotation circuit.

The quantity and size of the flotation cells that will be installed in the bulk flotation circuit are shown in Table 17-3.

Table 17-3: Bulk Flotation Cells

Stage	Quantity of Cells	Size of Cells (m^3)
Copper Rougher	10	600
Pyrite Scavenger	4	600
Copper 1st Cleaner	12	100
Copper 2nd Cleaner	4	100
Copper 3rd Cleaner	2	4.5 m dia. column

Flotation reagents will be added at several points in the bulk flotation circuit as required.

17.2.3.2 Molybdenite Flotation

Concentrate from the final cleaner of the bulk flotation circuit will report to a copper-moly separation thickener. Thickened copper-moly concentrate will be pumped by variable speed, horizontal centrifugal slurry pumps to the molybdenite (moly) flotation circuit.

The moly flotation circuit will consist of two agitated rougher conditioning tanks, one row of separation (rougher) flotation cells, one row of first cleaner flotation cells, a concentrate regrind circuit, one second cleaner flotation column, one third cleaner flotation column, and one fourth cleaner flotation column.

Concentrate from the moly rougher cells will be pumped to the moly first cleaner flotation cells. Tailing from the moly rougher cells, which will be the final copper concentrate flotation product, will flow by gravity to the copper concentrate thickener.

Concentrate from the moly first cleaner cells will flow by gravity to the moly concentrate regrind circuit. Tailing from the moly first cleaner flotation cells will flow by gravity to the copper concentrate thickener.

In order to reduce consumption of NaHS and nitrogen, the rougher and first cleaner cells will be covered, and the flotation gas will be recycled.

Moly concentrate regrinding will be performed in a vertical mill, operated in open circuit. First cleaner moly concentrate will flow by gravity to a regrind sump and be pumped by a variable speed, horizontal centrifugal slurry pump to the mill. Reground moly first cleaner concentrate will be pumped to the moly second cleaner flotation column.

Concentrate from the moly second cleaner column will flow by gravity to the moly third cleaner flotation column. Tailing from the moly second cleaner flotation column will be pumped to the moly first cleaner flotation circuit.

Concentrate from the moly third cleaner column will flow by gravity to the moly fourth cleaner flotation column. Tailing from the moly third cleaner flotation column will be recycled to the moly second cleaner flotation column.

Concentrate from the moly fourth cleaner column, which will be the final moly concentrate flotation product, will flow by gravity to the moly concentrate dewatering circuit. Tailing from the moly fourth cleaner column will be recycled to the moly third cleaner flotation column.

The quantity and size of the flotation cells that will be installed in the moly flotation circuit are shown in Table 17-4.

Table 17-4: Moly Flotation Cells

Stage	Quantity of Cells	Size of Cells (m^3)
Moly Rougher	7	8.50
Moly 1st Cleaner	7	2.83
Moly 2nd Cleaner	1	2.0 m dia. column
Moly 3rd Cleaner	1	1.25 m dia. column
Moly 4th Cleaner	1	1.0 m dia. column

Flotation reagents will be added at several points in the moly flotation circuit as required.

17.2.4 Concentrate Dewatering and Storage

17.2.4.1 Copper Concentrate Dewatering

Moly rougher flotation tailing (copper concentrate) and moly first cleaner flotation tailing (copper concentrate) will flow by gravity to a copper concentrate thickener. Thickened copper concentrate will be filtered in three tower type copper concentrate pressure filters. Filter cake will discharge to a conveyor belt that will discharge to a covered copper concentrate stockpile.

Copper concentrate will be reclaimed by front-end loader onto highway haulage trucks. The loaded haul trucks will proceed to a wash station and be cleaned before exiting the concentrate load out area. This procedure will ensure against tracking of concentrate from the facility.

17.2.4.2 Molybdenite Concentrate Dewatering

Moly concentrate from the final moly cleaner flotation circuit will flow by gravity to an agitated filter feed tank. Moly concentrate slurry will be filtered in one tower type moly concentrate pressure filter. Filter cake will discharge to a Holo-Flite type dryer. The dryer will discharge to the moly concentrate storage bin.

Moly concentrate will be withdrawn from the dried moly concentrate storage bin by a packaging system and will be bagged in super-sacks for shipment by trucks to market.

17.2.5 Pyrite Concentrate Deposition

Pyrite scavenger concentrate and tailing from the copper first cleaner flotation circuit will collect in a pyrite thickener. Thickened pyrite concentrate will flow by gravity to the TMF for subaqueous deposition.

17.2.6 Tailing Dewatering

Tailing from the pyrite scavenger flotation cells will flow by gravity to two tailing thickeners operated in parallel. Overflow solution from the tailing thickeners will be pumped by horizontal centrifugal pumps to the process water pond for reuse in the mill. Thickened tailing will flow by gravity to the tailing cyclone station either to produce sand via the cyclone plant or for direct deposition into the TMF. Tailings discharge into the TMF via valved off-takes along a pipeline on the embankment crest. Discharge into the TMF will not be continuous from any one location in the pipeline. It will be rotated between off-takes as appropriate for tailings distribution within the TMF. This will ensure adequate beach development adjacent to the embankments and maximize the ability of the reclaim system to recover clean water.

17.2.7 Reagents and Consumables

Reagent storage, mixing, and distribution will be provided for all reagents used in the sulphide processing circuits. Table 17-5 below is a summary of reagents used in the process plant.

Table 17-5: Process Consumables

Reagent & Consumables	Units	Consumption Rate
Copper Flotation Reagents		
Lime (hypogene)	kg/t	1.075
Lime (supergene sulfide)	kg/t	2.688
Fuel Oil	kg/t	0.007
3418A (hypogene)	kg/t	0.004
3418A (supergene sulfide)	kg/t	0.008
A208 (hypogene)	kg/t	0.008
A208 (supergene & sulfide)	kg/t	0.017
MIBC	kg/t	0.010
PAX	kg/t	0.040
Flocculant	kg/t	0.022
Moly Flotation Reagents		
NaHS	kg/t	0.053
Fuel Oil	kg/t	0.0004
Flocculant	kg/t	0.0034
	kg/t	
Primary Crusher - Liners	kg/t	0.040
SAG Mill - Liners	kg/t	0.040
Ball Mill - Liners	kg/t	0.048
SAG Mill - Balls	kg/t	0.400
Ball Mill - Balls	kg/t	0.400
Regrind Mill – Balls	Kg/t	0.041

17.2.8 Water System

17.2.8.1 Fresh Water

The water source for Casino is the Yukon River, 17 km away.

Fresh water will be pumped from the fresh-water intake Ranney well through a series of booster stations, pumps, and pipelines to the plant site area. Fresh water will collect in a fresh/fire water pond at the process site. The design capacity of the fresh water collection and transfer system will be approximately 3,200 m^3/h.

Fresh water and fire water will be supplied from the fresh/fire water pond to the facility by gravity. Fresh water will be distributed to:

- A chlorinator package and subsequent potable water tank, for use in offices, laboratory, dry, and rest rooms
- A mine water tank for mine road dust control
- A gland seal water tank to supply seal water for mechanical equipment
- The ADR Fresh/Fire water tank
- The process water pond
- The fire water distribution system in the mill site area

17.2.8.2 Process Water

Water reclaimed from the Tailing Management Facility (TMF), designated as process water, will be pumped from water barges at the TMF pond through a series of booster tanks, booster pumps, and pipelines to the plant site area as described above. Process water will collect in a process water pond at the process site. Overflow solution from the tailing, bulk concentrate, and copper concentrate thickeners will also be pumped to the process water pond. Process water will be distributed from the process water pond by gravity flow through a pipeline to mill process water usage points.

17.2.9 Power

See Section 21.3.1.2 for the estimated power requirements and the costs used in the operating cost estimate and financial model.

17.2.10 Air Service

Separate air supply systems will supply air to the following areas:

- Crushing
- Flotation – air will be provided by two compressors coupled with an air receiver.
- Concentrate Filtration
- Plant Air – high pressure air will be provided by two compressors coupled with an air receiver.
- Instrumentation – high pressure air will be provided from a dedicated compressor, receiver and air dryer.

17.2.11 Metallurgical Performance Projection

According to the metallurgical performance projections developed from the metallurgical test results and the proposed mine plan, life of mine concentrate production is projected and shown in Table 17-6.

Table 17-6: Metallurgical Performance Estimate

Mill Feed			Copper Concentrate			Moly Concentrate		
Tonnage (kt)	Cu %	Mo %	Recovery Cu %	Grade Cu %	Production (kt)	Recovery Mo %	Grade Mo %	Production (kt)
1,126,966	0.197	0.022	86.6	28%	6,857	65.85	56	291

17.3 OXIDE MINERALIZED MATERIAL PROCESS PLANT DESCRIPTION

17.3.1 Process Design Criteria

Process design criteria were developed for the Project based on a 25,000 t/d (9,125,000 t/y) plant design. The crushing circuit was designed to operate with an overall availability of 75%. The remainder of the processing facilities were designed to operate with an overall availability of 92%. The equipment was sized using these criteria. Table 17-7 is a summary of the main components of the oxide process design criteria used for the study. Table 17-8 is a summary of the major process equipment selected for the study.

Table 17-7: Oxide Process Design Criteria

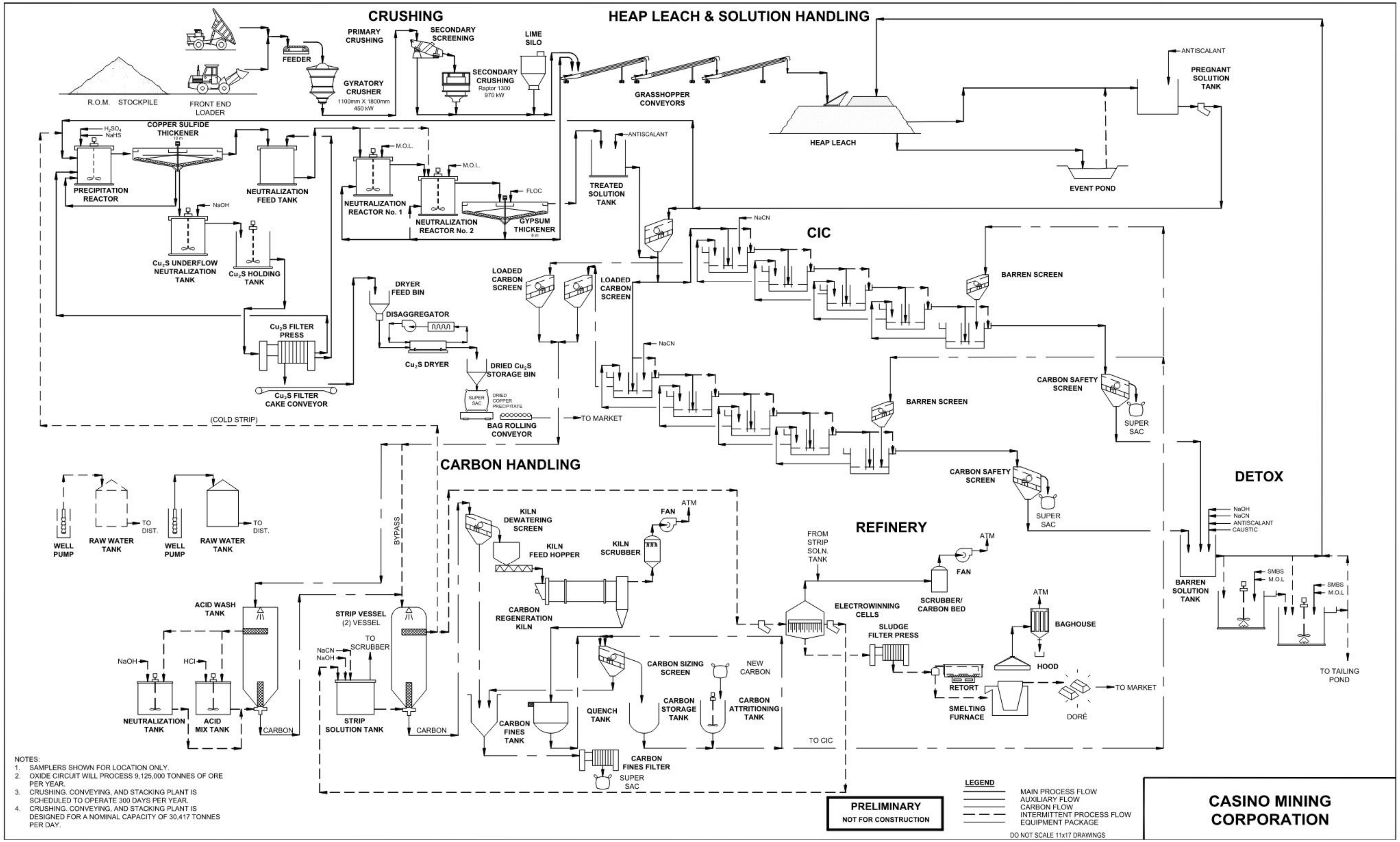
Description	Unit	Value
General		
Type of Deposit	-	Oxide Cap Material
Pad Feed Characteristics		
Specific Gravity	-	2.65
Moisture Content	%	2.0
Abrasion Index (Average)	-	
Bond Crushing Work Index	kWhr/t	N/A
Operating Schedule Crushing Conveying, Stacking		
Shift/Day	-	2

Description	Unit	Value
Hours/Shift	h	12
Hours/Day	h	24
Days/Year	d	300
Availability	%	75
Feed rate	DTPY	9,125,000
Operating Schedule Heap Leaching		
Shift/Day	-	2
Hours/Shift	h	12
Hours/Day	h	24
Days/Year	d	365
Availability	%	98
Feed rate	DTPY	9,125,000
Heap Leach Pile		
Stack leach pile section	d	-
Cure leach pile	d	-
Leach solution application, primary	d	60
Leach solution application, secondary	d	Through subsequent lifts
Drain solution from leach pile	d	-
Total cycle	d	-
Pad mineralized material storage capacity	Tonnes	203,700,000
Pile slope	-	2.5 : 1
Maximum mineralized material thickness	m	8.0
Mineralized material pile setback	m	9
Leach solution application method		Distribution network with emitters
Application rate		12
Precipitation	mm/a	500
Pan Evaporation	mm/a	308
Retained solution, leach pile moisture (wet)	%	10
Operating solution, leach pile, moisture (wet)	%	12
Barren solution flow rate (average)	m^3/h	1,312
Pregnant solution flow rate (average)	m^3/h	1,223

Table 17-8: Major Process Equipment

Item	Number	Description	Key Criteria (ea)
Primary Crusher	1	1100 x 1800 TSU Top Service Gyratory Crusher	450 kW
Secondary Screen	1	3m x 6.7m Inclined; double deck; banana	19 kW
Secondary Crusher	1	Raptor XL 1300	970 kW
Heap Leach Stacking System	1	Conveyors, Stacker	2,200 kW
Barren Solution Pump	2	Vertical Turbine; 703 m^3/hr @ 156 m TDH	448 kW
Pregnant Solution Pump	3	Submersible Vertical Pump; 754 m^3/hr, 125 m head	336 kW
Carbon Handling Plant	1 lot	ADR and Refinery	3 ton
SART plant	1 lot	Design basis: 400 mg/L Cu in the PLS	150 m^3/hr

Figure 17-2 is a simplified schematic of the overall process for the oxide mineralized material. This provides the basis for the process description that follows.



(Source: M3)

Figure 17-2: Simplified Oxide Process Flowsheet

The following items summarise the process operations required to extract gold from the oxide gold mineralized material.

- Size reduction of the run-of-mine (ROM) mineralized material to minus 200 mm using a primary gyratory crusher.
- Size reduction of the primary crushed mineralized material to minus 38 mm through screening and a secondary cone crusher.
- Stacking crushed mineralized material by overland conveyors and a stacker onto a heap leach pad, and subsequently, leaching the mineralized material with cyanide solution.
- Recovering gold and silver from the pregnant leach solution on activated carbon in carbon-in-column tanks (CIC).
- Recovering copper from the pregnant leach solution by the Sulphidization, Acidification, Recycling and Thickening (SART) process.
- Treating gold and silver loaded carbon recovered from the CIC circuit by acid washing, cold stripping with cyanide solution to remove copper, hot stripping with caustic solution to remove gold, and thermal reactivation of the carbon.
- Recovering gold from the pregnant carbon stripping solution as cathode sludge on stainless steel mesh cathodes in an electrowinning cell.
- Melting the cathode sludge with fluxes to produce a gold/silver Doré bar, the final product of the mineralized material processing facility.
- Storing, preparing, and distributing reagents to be used in the process.

17.3.2 Crushing, Conveying, and Stacking

The Oxide mineralized material will have a primary crusher and conveyor system separate from the Sulphide mineralized material.

Run of mine (ROM) oxide mineralized material will be trucked from the mine to the primary crusher apron feeder. Alternatively, ROM may be stockpiled in the ROM Stockpile if the primary crusher is down for maintenance. A front-end loader will reclaim mineralized material from the stockpile and dump onto the primary crusher apron feeder. The apron feeder will provide the feed to the primary crusher. The primary crusher will be a 1,372 mm x 1,905 mm (54"x75"), or equivalent, gyratory crusher. The primary crusher will discharge onto a secondary screen feed belt conveyor.

Secondary screen feed conveyor will discharge onto the secondary screen. Screen undersize will discharge onto the fine mineralized material transfer conveyor. Screen oversize will discharge onto secondary screen discharge belt conveyor, which discharges into secondary crusher feed bin.

Secondary crusher belt feeder will draw mineralized material from the bin and provide feed to the secondary cone crusher, which will be equipped with a 750 kW, or equivalent, drive. Cone crusher discharge will combine with undersize from the secondary screen on the fine mineralized material transfer conveyor. Lime will be added to this conveyor, which discharges onto the intermediate transfer conveyor.

The intermediate transfer conveyor discharges onto a series of overland transfer conveyors, with the last overland conveyor discharging onto the telescoping stacker feed conveyor. The telescoping stacker feed conveyor discharges onto the leach pile stacking conveyor, which places crushed mineralized material onto the heap leach pile.

17.3.3 Heap Leaching

The heap leach pad will consist of liners and a low-permeability soil liner. A perforated pipe drainage system will enhance drainage of leach solutions away from the liner, reduce hydrostatic head, and facilitate pregnant solution recovery. A 600 mm thick layer of sized river gravel or crushed and screened rock will be placed on the liner. This layer is necessary to protect the liner from crushed mineralized material stacking and to allow unimpeded leach solution flow to the drainage pipe system.

Barren process solution will be applied to mineralized material lots. Solution will be applied with drip emitters to minimize evaporation losses. When a mineralized material lot has completed the primary leach cycle, solution application will be stopped and another mineralized material lift (or layer) will be placed on top of the previous lift. Leach solution application will resume. The process of layering and leaching the mineralized material will repeat for a maximum of eight mineralized material lifts or layers on the leach pad. When the last process leach cycle is completed on the last lift, the mineralized material heaps will be rinsed with fresh water to recover the remaining gold and rinse the residue.

Pregnant solution discharging from the mineralized material heaps will be collected in a network of pipe placed throughout the overliner material that will direct the solution to the in-heap collection area.

Pregnant solution will be pumped from the in-heap collection area using horizontal, centrifugal pregnant solution pumps. The pump discharge pipes will be combined in a single pipeline to the carbon-in-column (CIC) / SART circuit for recovery of gold and copper.

An events pond will be installed to handle any overflow that might occur during a large precipitation event. Water that accumulates in the events pond will be periodically pumped by a submersible pump to the barren solution system feeding leach solution to the mineralized material heaps.

The pond system has been sized to contain normal operating solutions and stormwater. Ponds will be fenced to reduce the risk of danger to wildlife.

The design of the heap leach facility is such that the facility will be operated as a zero-discharge facility. A cyanide detox system will be installed as a contingency measure. If a surplus of solution exists, the excess solution will be routed to cyanide destruction prior to transfer to the tailing management facility, which is part of the sulfide concentrator.

17.3.4 Carbon ADR Plant/SART

Gold and silver will be recovered from heap leach pregnant solution by adsorption of their ions on activated carbon followed by desorption and electrowinning of a gold and silver solid product. Copper will be recovered from the pregnant solution by the SART process where the copper will be precipitated by the addition of sulphide to produce a copper sulphide product.

The process steps required to recover gold and silver by the carbon adsorption method include:

- loading gold and silver on activated carbon in a CIC circuit,
- acid washing of the carbon to remove water scale and acid soluble copper,
- cold stripping of carbon (elution) to remove copper,
- stripping gold and silver from the carbon using a hot caustic solution,
- electrowinning gold and silver from the stripping solution in a precious metal sludge using an electrolytic cell,
- reactivating stripped carbon by thermal regeneration, and
- melting the precious metal sludge in a crucible furnace to produce Doré bars.

The process steps required to recover copper by the SART method include:

- bleeding a portion of the pregnant solution to the SART process,
- adding sodium hydrosulphide to the solution,
- decreasing the pH of the solution with acid, thereby precipitating copper,
- removing the copper precipitate from the solution by thickening, filtration, and drying,
- increasing the pH of the solution with lime, thereby precipitating gypsum,
- removing the gypsum from the solution by thickening, and
- shipping the filtered copper sulphide product to a smelter for refining.

17.3.4.1 Carbon-in-Column (CIC) Circuit

Gold will be recovered from pregnant leach solution in a two-train, five-stage carbon-in-column (CIC) circuit. Pregnant leach solution will pass through a stationary screen to remove trash prior to being introduced into the CIC tank line. Most of the pregnant solution will pass directly to the CIC circuit, and the remainder will be processed for copper recovery by the SART process and then be returned to the CIC circuit.

The CIC circuit will consist of two parallel trains of five CIC tanks operated in series. Each CIC tank will be a flat bottom tank with an internal distribution plate between the bottom and the carbon charge. It will be possible to bypass any tank by using a manually operated dart-valve.

Solution (barren solution) will exit the CIC lines, be combined, and pass through a safety, single deck, vibrating screen to capture carbon unintentionally washed from the carbon columns. Screen oversize (carbon) will be transferred by gravity to a carbon quench tank.

Carbon will be advanced through the circuit by a series of recessed impeller, horizontal centrifugal pumps. Loaded carbon advanced from the first CIC vessel will be pumped by the first carbon transfer pump from each train to a carbon distributor tank. Carbon will be advanced through from tank to tank by additional, dedicated carbon transfer pumps – one pump for each CIC vessel.

Regenerated and new carbon will be sized by screening on single deck vibrating screens. Screen undersize will flow by gravity to a carbon fines tank. Screen oversize will flow by gravity to the last carbon column on either of the CIC lines when carbon is advanced.

Barren solution will discharge from the CIC tank line and flow into a barren solution tank. Vertical turbine pumps will pump barren solution from the barren solution tank to the heap leach barren solution distribution system. The pump discharge lines will be combined to a single pipeline to the heap leach area.

Cyanide solution will be added to the barren solution tank and/or the first CIC columns.

17.3.4.2 SART Circuit

A portion of the heap leach pregnant solution, approximately 150 m^3/h, will be pumped to the SART process. Sodium hydrosulphide will be added to the solution through an in-line mixer. Downstream of this in-line mixer, sulfuric acid will be added to the solution prior to mixing in the precipitation reactor.

The discharge from the precipitation reactor will flow by gravity to a covered copper sulphide thickener. Most of the underflow from this thickener will be recirculated to the precipitation reactor, while the balance of the thickener underflow will be advanced to a copper sulphide neutralization tank. Overflow solution from the copper sulphide thickener will collect in a neutralization feed tank.

Sodium hydroxide will be added to the copper thickener underflow neutralization tank. Neutralized copper sulphide, thickened slurry will be fed to the copper sulphide filter press. The filter cake will discharge onto the copper sulphide filter cake belt conveyor, which discharges into a bin. The filtered cake will pass through a dryer. Dried copper sulphide will collect in a bin, be loaded into supersacks, and transported by flatbeds trucks to market.

Copper sulphide thickener overflow will collect in a neutralization feed tank. Neutralization feed pump will transfer the overflow solution to the first neutralization reactor. Lime and recirculated gypsum thickener underflow will be added to this reactor, which overflows into the second neutralization reactor.

Neutralized solution will flow by gravity to a covered, gypsum thickener. A portion of the thickener underflow will be recirculated to the first neutralization reactor. The balance of the underflow will be pumped to a gypsum holding tank.

Gypsum thickener overflow will collect in a treated solution tank, where an antiscalant will be added. This treated solution will be returned to the CIC circuit.

A scrubber will collect fumes from the precipitation reactor, the copper sulphide thickener, the copper sulphide neutralization tank, the neutralization feed tank, the neutralization reactors, the gypsum thickener, and the sodium hydro-sulphide storage tank. The pH of the scrubber will be adjusted by NaOH. Some scrubber discharge will be recirculated, but net scrubber discharge will report to the first neutralization reactor.

17.3.4.3 Carbon Acid Wash

Loaded carbon will be acid washed in a column. A dilute hydrochloric acid solution will be pumped into the bottom of the acid wash tank, flow up through the vessel, and overflow to the acid tank. This pump will either be operated to circulate solution through the carbon, or the carbon may be left to soak. After completion of the acid wash solution cycle, the batch of spent acid solution will be pumped to the acidifying reactor tank in the SART circuit.

A caustic (basic) solution will be pumped into the bottom of the acid wash tank at the end of its cycle, flow up through the vessel, and overflow to the acid tank. The caustic solution pump will either be operated to circulate solution through the carbon or to fill the acid wash tank and allow the carbon to soak. Upon completion of the acid wash cycle, the caustic solution will be used to neutralise the acid solution to pH 8 to 10. Neutralized solution will be pumped to the barren tank.

17.3.4.4 (Copper) Cold Stripping

The acid washed loaded carbon will be cold stripped to remove copper before hot stripping to remove gold.

Water will be added to the acid washed carbon and the carbon and water slurry will be pumped from the acid wash vessel to one of the two strip columns. Water transferred with the carbon (carbon transfer water) will be drained through an internal screen in the bottom of the cold strip tank and will drain to a carbon fines tank.

Cyanide solution will be pumped into the top of the cold strip tank, flow down through the vessel, and discharge through the internal screen. The carbon will then be soaked for the cold strip cycle. Upon completion of the cold strip cycle, the strip solution will be transferred to the SART circuit.

The drained carbon will be rinsed with barren solution to remove the copper strip solution. The rinse solution will also be transferred to the carbon fines tank.

17.3.4.4.1 Carbon Elution Circuit

Elution will be by pressure Zadra techniques. Barren strip solution, containing sodium hydroxide (caustic) will be heated by a solution heating system and circulated through the bottom of the elution vessel by a horizontal centrifugal pump. The solution will flow up through the column, exit the top as pregnant solution, and flow through a heat recovery heat exchanger. The cooled solution will flow to the electrowinning (EW) barren return tank.

After completion of the elution cycle, the strip solution will be drained through an internal screen in the bottom of the elution tank and will be transferred to the carbon fines tank.

17.3.4.5 Electrowinning

Gold and silver will be recovered from pregnant strip solution by the electrowinning process.

Pregnant strip solution will be pumped from the EW barren return tank to an electrowinning cell. Knitted stainless steel mesh cathodes will be used in the cells for deposition of the gold and silver. The stripped solution will discharge from the cell and flow back to the EW barren return tank.

When the electrowinning cycle has been completed, the solution will be pumped from the EW barren return tank to the barren strip solution tank. A high pressure, water wash system will remove the metal sludge from the cathodes and wash it into an electrowinning wash/ sludge tank. Sludge slurry will be pumped by a diaphragm pump to a plate and frame, pressure filter. Filter cake will be cleaned from the filters by hand and placed in filter cake boats for refining.

17.3.4.6 Smelting

Filter cake will be mixed with fluxing materials and charged to an electric, crucible furnace. The melted charge will be poured into conical molds. Doré (gold and silver) will sink to the bottom of the mold, and slag glass containing fused fluxes and impurities, will float to the top of the mold. Doré will be sampled for gold content using vacuum tube samples.

After cooling and solidifying, the molds will be dumped, and the slag will be knocked off the Doré buttons by hand. A vertical drill press will be available as a back-up sample method for Doré buttons.

Buttons will be cleaned under a water stream using a needle gun, weighed, and stamped with an identification number and weight. Doré buttons will be the final product of the operation and will be stored in a safe until shipment.

Slag will be crushed and screened to recover high-grade chips that will be returned to the melting furnace. Remaining slag will be returned to the heap leach.

Fumes from the melting furnace will be collected through ductwork and cleaned in a bag house dust collector system before discharging to atmosphere.

17.3.4.7 Carbon Regeneration

Barren, stripped carbon will be sized on a single deck, vibrating screen. Screen undersize will flow by gravity to the carbon fines tank. Screen oversize will flow by gravity to a kiln feed tank.

Carbon will be withdrawn from the kiln feed tank by a screw conveyor feeder. The feeder will discharge to a horizontal, propane gas fired, carbon regeneration kiln.

The carbon regeneration kiln will discharge into carbon quench tank. Quenching will be done in a conical bottom tank. Regenerated carbon will be transferred from this tank to an activated carbon storage tank. New carbon, after being soaked and attrition agitated in a separate tank, will be added as required. Carbon and water slurry will be pumped from this tank by recessed impeller, horizontal, centrifugal pump to carbon sizing screens for use in the CIC circuit.

17.3.5 Reagents and Consumables

Reagent storage, mixing, and distribution will be provided for all reagents used in the oxide processing circuits. Table 17-9 is a summary of reagents used in the process plant.

Table 17-9: Process Consumables

Reagent & Consumables	Units	Consumption Rate
Heap Leach/SART Reagents		
Sodium Cyanide (NaCN)	kg/t	0.500
Sodium Hydroxide (caustic, NaOH)	kg/t	0.130
Pebble Lime (CaO)	kg/t	3.516
Hydrochloric Acid (HCl)	kg/t	0.010
Sodium Hydrosulphide (NaHS)	kg/t	0.025
Sulfuric Acid H_2SO_4)	kg/t	0.328
Activated Carbon	kg/t	0.001
Antiscalant	kg/t	0.003
Flocculant	kg/t	0.00035
	kg/t	
Primary Crusher - Liners	kg/t	0.040
Secondary Crusher – Liners	kg/t	0.085

17.3.6 Water System

Water will be supplied from the freshwater pond on-site to an ADR fresh/fire water tank. Water from this tank will supply a variety of destinations to meet process, mine, and fire water requirements.

17.3.7 Power

See Section 21.3.1.2 for the estimated power requirements and the costs used in the operating cost estimate and financial model.

17.3.8 Metallurgical Performance Projection

According to the metallurgical performance projections developed from the metallurgical test results average grades and recoveries for the heap leach are shown in Table 17-10.

Table 17-10: Metallurgical Performance Estimate

Heap Leach Feed	Grade			Recovery		
Tonnage (kt)	Cu %	Au g/t	Ag g/t	Cu %	Au %	Ag %
203,790	0.034	0.022	1.951	18	70	26

18 PROJECT INFRASTRUCTURE

The Project site is located about 300 km northwest of Whitehorse, about 200 km northwest of the Village of Carmacks, and about 18 km southwest of the Yukon River, near Patton Hill. The approximate elevation of the mine pit is 1,300 m. The plant site coordinates are 6,956,718.230 N, 611,800.867 E. The elevation of the plant site is approximately 1,190 m.

The mine site is remote from significant population centers, but the site can be accessed by winter road. About 180 km of the roads from Whitehorse to Carmacks are paved. The paved roads include the Alaska Highway and the Klondike Highway that pass through Whitehorse.

Based on the Port Study by Associated Engineering, the Skagway port is the optimal port to utilise in terms of both concentrate exports and mine supply imports. Whitehorse has an international airport with daily flights to Vancouver. The mine site has an airstrip used to access the site by light aircraft.

The Yukon electrical grid is at a significant distance from the mine and at the current time, the utility does not have enough generating capacity to fulfill the requirements for the Casino mine. The cost-effective solution to provide power for the Project is to build and operate an on-site power plant.

18.1 SITE LAYOUT AND ANCILLARY FACILITIES

18.1.1 General

Project facilities will lie east of Patton Hill. The open pit mine is located between the headwaters of Casino Creek and Canadian Creek and will occupy an area of more than 300 ha.

A small valley about 1 km south of the pit, within the Tailings Management Facility (TMF) area, will be filled with oxide mineralized material to form the heap leach pad. An earthen embankment at the eastern end of the pad will provide structural support for the heap leach. A spill and runoff control collection pond termed the “Events Pond” will be built directly downhill from the heap leach pad.

The TMF will be located southeast of the pit and plant site within the valley formed by headwaters of Casino Creek. It will store approximately 712 million tonnes (Mt) of tailings together with 500 Mt of potentially reactive waste rock. The TMF will include a water reclaim system to collect supernatant water and return it to the process.

There are two primary crushers: one for the oxide mineralized material and one for the sulphide mineralized material. The primary gyratory crusher pad where each primary crusher will be located will be constructed east of the pit. An overland conveyor belt will carry the sulphide mineralized material 1.2 km to the southwest from the crusher to the coarse mineralized material stockpile, which is located immediately northwest of the processing plant. The crushed oxide mineralized material will be carried to the heap leach by a separate overland conveyor system.

The concentrator consisting of the grinding mills, flotation circuits, reagent mixing, storage and distribution systems, concentrate filter facility and thickeners will be located on an area of relatively flat terrain about 2 km south of the pit and crusher.

Low-grade mineralized material will be placed in several temporary stockpiles adjacent to the plant site and crusher for processing in later years. Figure 18-1 shows the overall project site layout.

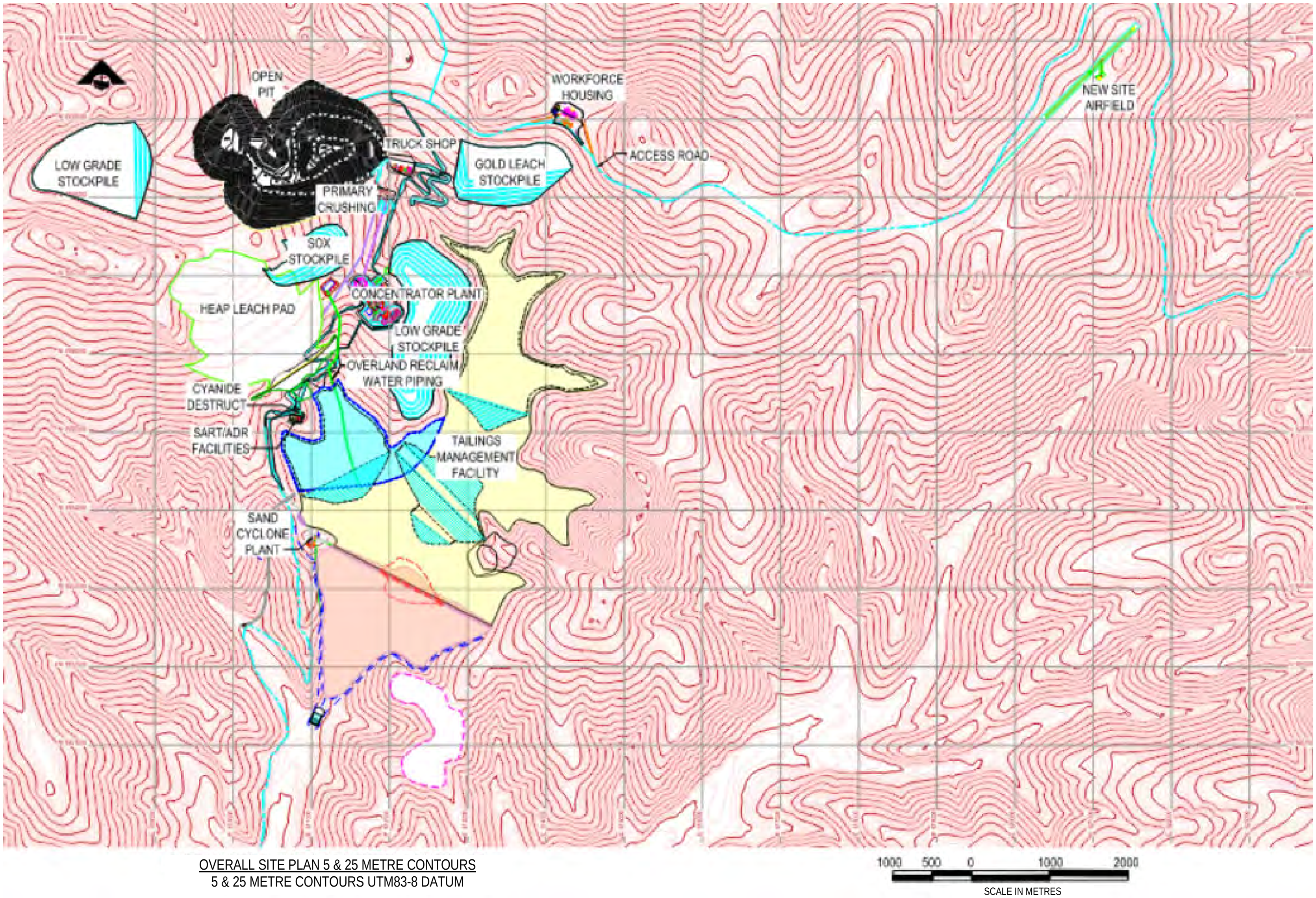


Figure 18-1: Overall Site Plan

18.1.2 Truck Shop

A truck shop and associated facilities will be constructed on a 4.5 ha pad adjacent to the open pit mine's eastern exit.

18.1.3 Residence Camp

A nominal 1,400-man capacity construction and permanent camp will be installed. There will be additional construction camps for off-site construction provided by the contractor(s). The permanent camp will have approximately 880 beds.

18.1.4 Operational Support Facilities

The majority of ancillary buildings are proposed to be located at the processing plant site. The Administration building will provide office space for both the construction effort and operations. The change house (mine dry) and laboratory buildings will be located near the mill and flotation buildings. A warehouse and laydown area will be provided for receiving and storage of parts and supplies, and for maintenance of plant mechanical and electrical equipment. A light vehicle maintenance building is proposed at the plant site apart from the truck shop. All of these buildings will be pre-engineered steel structures.

18.1.5 Guard Shed/Scale House

A Guard Shed/Scale House at the facility entrance will provide 24/7 site security and truck scale service. The guard shed will be of modular construction.

18.1.6 Airstrip

The Casino Mine site is remote and the required workforce is expected to be approximately 1,400 with some operational personnel during construction and 600 to 700 during operations. Access to the site for this number of personnel will be best served by aircraft. The Project plan includes a new 1,600 m airstrip and pre-engineered air terminal building. The airstrip is planned to be located off the Freegold access road into the site. It will be located approximately six kilometers east of the Permanent Camp Site.

Figure 18-2 shows the existing Casino airstrip and the future mill plant site.

Figure 18-2: Current Casino Airstrip

Western intends to replace the existing small airstrip with a larger facility that permits all-season operations. Various aircraft have been considered to provide air service transport to the site. Aircraft considered suitable for this service includes the ATR 42 (capacity 42 passengers) and the Bombardier Dash 8- 200 series turbo-prop aircraft which can be configured with 37 seats. Both aircraft are high-wing turboprop and well suited for the service conditions.

Flights to the site will originate from Whitehorse connecting with scheduled flights from Vancouver or other major centers.

The new airstrip site poses few aeronautical challenges and provides safe aircraft operations during all visible weather conditions. The airstrip design criteria have been developed to conform to the Transport Canada Aerodrome Standards and Recommended Practices (TP 312). The airstrip consists of a Code 3C non-instrument runway generally oriented northeast to southwest. It is 1,600 m long with 60 m overrun beyond the thresholds at each end for a total length of 1,720 m. The required runway width is 30 m and the total graded width is 80 m. The airstrip embankment is raised a minimum of 1.5 m above the existing ground to provide a stable base and to prevent any ground ice from melting. There is a taxiway at the northeast end connecting to an apron for aircraft unloading and a parking area at the start of the airstrip access road. The new airfield is located on the right side of Figure 18-1.

18.2 ACCESS ROADS

18.2.1 Service Roads

Service roads will connect the various mine and process facilities together with the ancillaries. The roads will be constructed with a minimum 4 m wide all-weather gravel surface. In general, the maximum grade for the road will be 10%.

An existing road leads northward from the mine facility to the Yukon River along Britannia Creek. The new freshwater pipeline will generally follow the road alignment. The roadway will be graded and improved as a service road to facilitate construction, access, inspection, and maintenance of the pipeline.

18.2.2 Freegold Access Road

A winter access road from Kluane Lake near the Alaska Highway has been used in the past to service the Casino Mine. Historically, access to the area has been available, form the east, via the Casino Trail, and Western has used this access in recent years.

Access to the site is also available from a barge landing at the junction of Britannia Creek and the Yukon River that was prepared in 2010. The lower 10 km of the 23 km road to the site was realigned to avoid all but one creek crossing at Canadian Creek.

Associated Engineering examined various route options for a year-round access road to Casino. The selected option which appears to have the least environmental impact and the greatest stakeholder support is a new road from the end of the existing Freegold Road approximately 70 km northwest of the village of Carmacks. The Casino Mine access road is a 132 km, two-lane, gravel resource road designed to accommodate B-Train Double (BTD) and Tridem trucks. The road design criteria satisfies the guidelines in the BC Ministry of Forests and Range Forest Road Engineering Guidebook (2nd Edition, 2002) for a 70 km/h design speed with some 50 km/h sections where road geometry is limited by the terrain.

In order to maximise the design speed and avoid unstable terrain, the route is located as much as possible in valley bottoms. The road surface elevation is designed to be 2.0 m above existing ground in these areas, with the fill material placed over undisturbed soils. This embankment height stabilises the road against washouts and protects against permafrost degradation under the road.

Where the road climbs out of the valley bottoms, the road construction method includes both cut and fill. Permafrost rich areas may require buttressing of cut slopes with a layer of angular rock fill on top of filter fabric. This will prevent permafrost degradation and act as a retaining structure to improve slope stability.

Most of the fill required for road construction will be developed from borrow pits located along the road alignment and then hauled to where it is needed. The section of road from the Selwyn River to the mine is located in soil that is mainly suitable for road embankment construction and can be utilized for fill. Further soil testing may reveal other locations with borrow suitable for road construction which will result in shorter haul distances and reduced road construction costs.

There are 16 major bridge crossings located along the main access road. These include crossings of Bow Creek, Big Creek, Hayes Creek, and Selwyn River along with several tributaries and side channels. Associated Engineering completed Hydro-technical analysis and topographic site surveys to provide the necessary information to then complete conceptual bridge designs for each crossing. There are also 71 major culvert crossings that have been identified for the main access road with estimated diameters ranging from 1,500 mm to 2,400 mm. Detailed hydro-technical analysis has not been completed for the culvert crossings. Culverts in fish bearing streams will be embedded to provide a simulated natural stream bottom.

Smaller 500 mm and 600 mm culverts will be used to control road drainage with the culvert spacing determined by road gradient and natural depressions in the topography. The cost of upgrading this road will be shared between Industry (in this case the Casino Mine), the Yukon Government and the Canadian Government. The proportions will be 70% to Industry, and 30% to the Yukon and Canadian Governments. Figure 18-3 shows the proposed mine access road route.

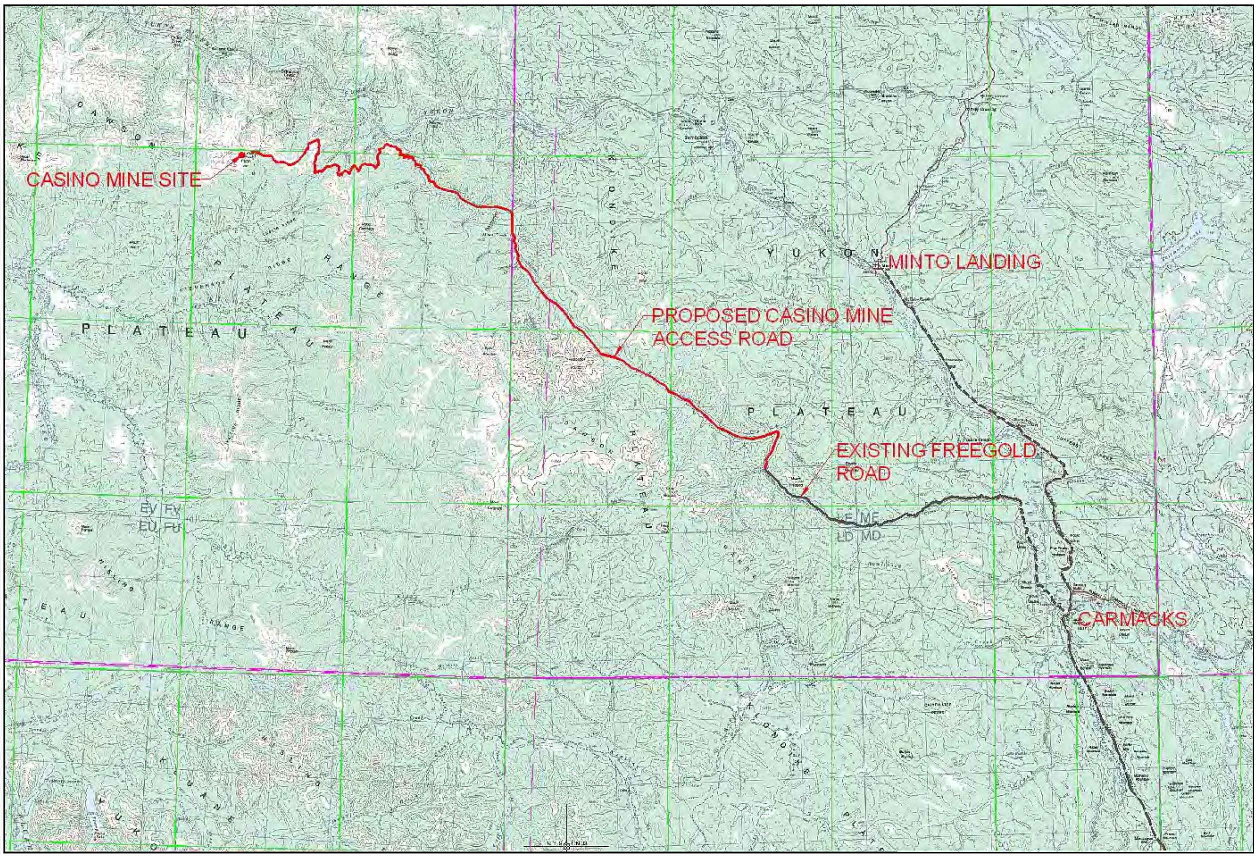


Figure 18-3: Proposed New Access Road

18.3 PORT FACILITIES

The Port of Skagway is located 560 km from the Casino site and has been selected as the port of export for the Project. The port has historically exported over 600,000 tonnes annually of lead and zinc concentrates and currently exports the copper concentrates from the Minto operation. An engineering evaluation has determined that the existing concentrate storage and handling facilities at Skagway can be economically upgraded to serve the Casino export requirements. The Port of Skagway and AIDEA have expressed interest in providing concentrate storage and load-out services to Casino in their to be expanded facilities, consistent with the conceptual design prepared by WC&G. The PEA study operating cost reflects exporting of copper concentrate through the Port of Skagway facilities. Figure 18-4 shows a conceptual rendering of the proposed port facilities expansion at Skagway.

Figure 18-4: Conceptual Rendering of Skagway Port Facilities

18.4 PROCESS BUILDINGS

18.4.1 Crushing Plant

There will be two primary crushers: One crusher will do the initial crushing of the sulphide mineralized material, and the other will do the initial crushing of the oxide mineralized material. Each primary crusher will be housed within approximately 30 m high concrete structures. They will be placed only a few metres below existing grade in order to minimise blasting of bedrock. These structures will be surrounded by “U”-shaped Mechanically Stabilized Earth (Hilfiker or equivalent) retaining walls to form the truck maneuvering area around the dump hopper, roughly at the elevation of the exit from the pit.

18.4.2 Gold Recovery & SART Building

The oxide crushing facility and overland conveyor will feed the heap leach pad, and the pregnant solution from there will feed the ADR/SART Gold Recovery facility. The ADR (Adsorption, Desorption and Recovery) and SART (Sulphidization, Acidification, Recycling, and Thickening) facilities will be located in a single structure.

18.4.3 Grinding

The Grinding Building (Mill) will house the 12.2 m (40 ft.) diameter SAG mill and two 8.5 m (28 ft.) diameter ball mills. The SAG mill and ball mill areas will each have a 90-tonne overhead bridge crane for maintenance.

18.4.4 Flotation/Reagent Storage & Mixing

The flotation circuits will be located within a structurally independent building adjacent and connected to the grinding building. Reagent storage and mixing facilities will be fully enclosed and placed adjacent to the flotation building. The 8,000-tonne lime silo will be located apart from the Mill/Flotation building.

18.5 TAILINGS MANAGEMENT FACILITY

18.5.1 Design Basis

The principal objectives for the design of the Tailings Management Facility (TMF) are safe and economic storage for tailings and waste rock, protection of the regional groundwater and surface waters both during operations and in the long term (post-closure), and to achieve effective reclamation at mine closure. The design of the TMF addresses the following requirements:

- Permanent, secure and total confinement of all tailings and waste rock materials within an engineered facility.
- Control, collection, and removal of free draining liquids from the tailings during operations for recycling as process water to the maximum practical extent.
- The use of cyclones to generate sand for embankment construction from the bulk Non-Acid Generating (NAG) tailings.
- The inclusion of monitoring features for all aspects of the facility to confirm the quantitative performance objectives (QPOs) are achieved and the design intent is met.
- Staged development of the facility over the life of the Project.
- An end land use that meets the objectives of the project stakeholders and Communities of Interest.

Mining of the open pit will yield approximately 1.1 billion tonnes (Bt) of mineralized material. The mill will operate at a nominal throughput of approximately 125,000 tonnes per day (tpd), producing two primary tailings materials:

- 20% of the mill tailings are predicted to be Potentially-Acid Generating (PAG) that will be maintained in a sub-aqueous state within the TMF impoundment.
- 80% of the mill tailings are predicted to be NAG and will be used to produce sand for construction of the TMF Main Dam. The NAG tailings are relatively coarse with an average P80 of approximately 200 µm (the underflow has a P80 of 240 µm) which makes them amenable to efficient sand recovery with cyclones and dewatering screens.

It is assumed that 97% of the NAG tailings will be processed through a cyclone station for approximately nine months of each year (approximately 73% annually) to produce suitable cyclone underflow sand for embankment construction. The remaining 3% of NAG tailings are assumed to be discharged directly to the TMF impoundment during periods of downtime and/or maintenance of the cyclone station. Hydraulic placement of cyclone sand is not anticipated to be possible for a portion of the year due to freezing temperatures in the winter. The coarse cyclone underflow will be processed using dewatering screens during the winter months to produce sand that can be transported by conveyor to the toe of the dam for future use in construction of the Main Dam.

It is estimated that approximately 50% of the NAG tailings can be recovered as cyclone underflow sand when the cyclone station is operating. The remaining 50% of NAG tailings (cyclone overflow) will be thickened prior to being discharged into the TMF impoundment. It is estimated that 85% of the underflow sand will be recovered by the dewatering screens; the remaining material lost through the screens will be thickened with the cyclone overflow and directed to the TMF for storage. A schematic flow diagram showing the tailings mass balance and distribution is shown on Figure 18-5.

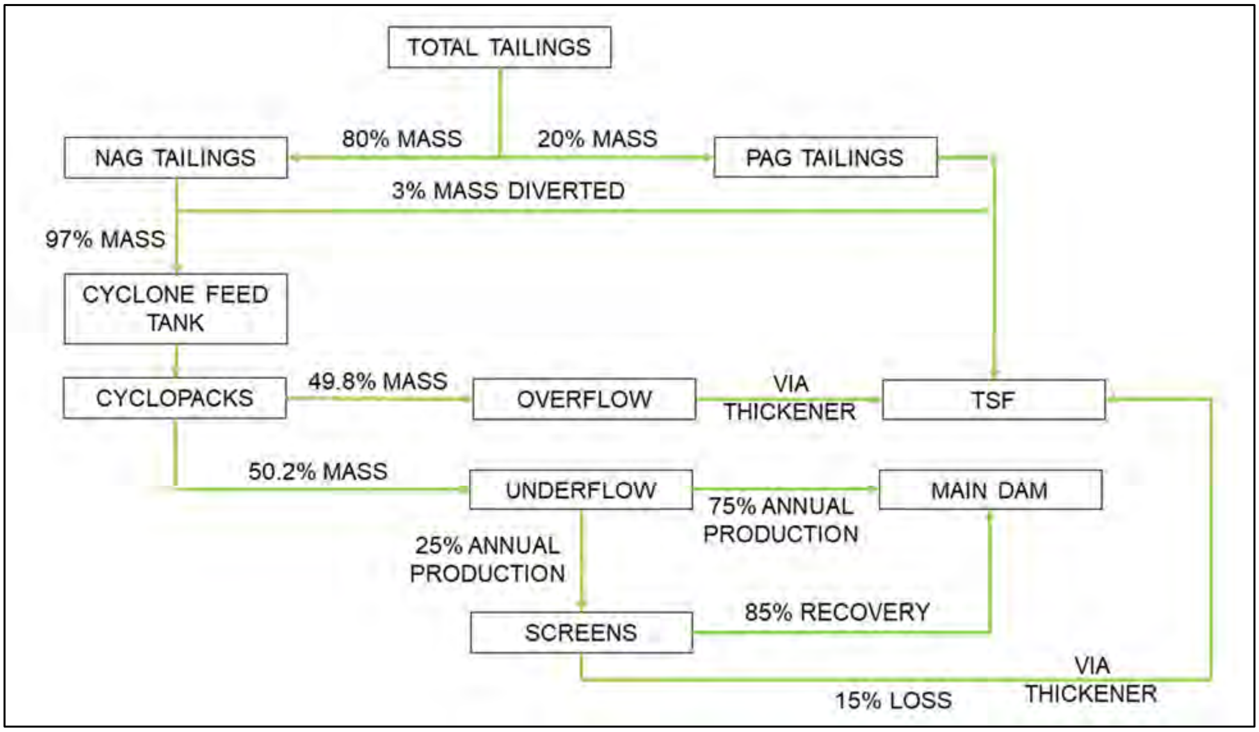


Figure 18-5: Tailings Conceptual Mass Balance

Approximately 500 Mt of waste rock and overburden will be mined from the open pit. The waste characterization studies indicate that the waste rock is potentially reactive and will need to be stored in the TMF.

The TMF is sized to provide sufficient capacity to sub-aqueously store approximately 712 Mt of tailings and 500 Mt of potentially reactive waste rock and overburden materials. The remaining 415 Mt of NAG tailings will be used for dam construction. The annual TMF storage requirement is shown in Table 18-1.

18.5.2 Hazard Classification

A preliminary consequence classification of HIGH has been assigned to the TMF based on criteria provided by the Canadian Dam Association (CDA) Guidelines (CDA, 2019). There is no permanent population at risk downstream of the TMF. The potential for loss of life from a dam failure is likely minor but cannot be discounted, particularly during operations when there will be work activities in the waste storage area and intermittently in areas downstream of the TMF. The economic consequences (including clean-up, repair and remedial works) would also be high. The environmental impact on downstream watercourses has the potential to be significant if a failure resulted in the release of tailings and/or process water into Casino Creek. An uncontrolled release into Casino Creek may flow into Dip Creek and potentially to the Yukon River by way of the Klotassin, Donjek and White Rivers. Fish species present in Casino Creek include Arctic Grayling, Burbot, and Slimy Sculpin. In addition to these species, Round Whitefish have been observed, and Chinook Salmon are likely present in Dip Creek. The fish species present are all currently listed as 'Unthreatened' under the Species at Risk Act and are widespread in the region, as is rearing habitat.

Table 18-1: Tailings Management Facility Storage Requirement

Mine Year	PAG Tailings to TSF	NAG Tailings to TSF	Total Tailings to TSF	Total Tailings to TSF	Waste Rock to TSF	Waste Rock to TSF	Total TSF Storage	Total TSF Storage
	Annual	Annual	Annual	Annual	Annual	Annual	Annual	Cumulative
	(Mt)	(Mt)	(Mt)	(Mm^3)	(Mt)	(Mm^3)	(Mm^3)	(Mm^3)
-1	-	-	-	-	13	6.4	6	6.4
1	6.9	15	22	16	21	10	26	33
2	9.4	21	30	21	12	6.2	28	60
3	9.3	20	30	21	29	14	36	96
4	9.1	20	29	21	23	12	32	128
5	9.1	20	29	21	20	10	31	159
6	9.1	20	29	21	31	15	36	195
7	9.1	20	29	21	32	16	36	232
8	9.3	20	29	21	34	17	38	269
9	9.3	20	29	21	30	15	36	305
10	9.0	20	29	20	27	14	34	340
11	9.1	19	29	20	41	20	41	380
12	9.3	20	29	21	41	20	41	421
13	9.4	20	29	21	40	20	41	463
14	9.2	20	29	21	34	17	38	500
15	9.0	19	28	20	33	16	37	537
16	9.2	20	29	21	23	11	32	569
17	9.0	19	28	20	12	5.8	26	595
18	8.9	19	28	20	5.1	2.6	22	617
19	9.0	19	28	20	0.8	0.4	21	638
20	9.1	19	28	20	0.0	0.0	20	658
21	9.2	20	29	21	-	-	21	679
22	9.1	20	29	21	-	-	21	699
23	9.2	20	29	21	-	-	21	720
24	9.0	19	28	20	-	-	20	740
25	8.1	17	25	18	-	-	18	758

Notes:

1. Annual tailings and waste rock tonnages initially provided by Western and factored for cycloning and sand screen operations.
2. Assumed tailings dry density of 1.4 t/m^3.
3. Assumed waste rock dry density of 2.2 t/m^3.

18.5.3 Main Dam Construction

The TMF Main Dam foundations will require clearing and stripping in preparation for fill placement. An average stripping depth of 1 metre has been assumed within the embankment footprint. Areas of unsuitable material, including colluvial apron or other ice-rich overburden, within the embankment foundation will be excavated to competent foundation, absent of frost susceptible soil. The average thickness of the colluvial apron and other ice-rich overburden is expected to be approximately 10 metres based on the findings of site investigations; the material is localized within the valley bottom and the northwest facing abutment.

The Starter Dam will comprise an approximately 120 m high embankment (crest to downstream toe) constructed to an elevation of 830 m. A cofferdam will be required to dewater and prepare the foundation prior to construction, beginning in pre-production Year -3. The embankment shell zones for the Starter Dam will require approximately 10.8 million cubic metres (Mm^3) of fill material for construction, sourced from a local borrow developed within the upstream TMF basin. Zone A is a 150 mm minus material processed from local borrow or from the plant site excavation, and Zone B is general rockfill from local borrow. The Starter Dam embankment will be constructed with 3H:1V upstream and 2H:1V downstream slopes. A grout curtain will be installed along the upstream toe of the dam, along with construction of a concrete plinth to facilitate tie-in of a linear low-density polyethylene (LLDPE) geomembrane installed on the upstream face of the dam. A 0.3 m liner bedding material and 16 oz non-woven geotextile will be placed on the upstream face of the dam prior to installation of the geomembrane. The grout curtain and geomembrane will limit seepage from the facility during initial operations when water storage may be required. A conceptual section of the Starter Dam is shown on Figure 18-6.

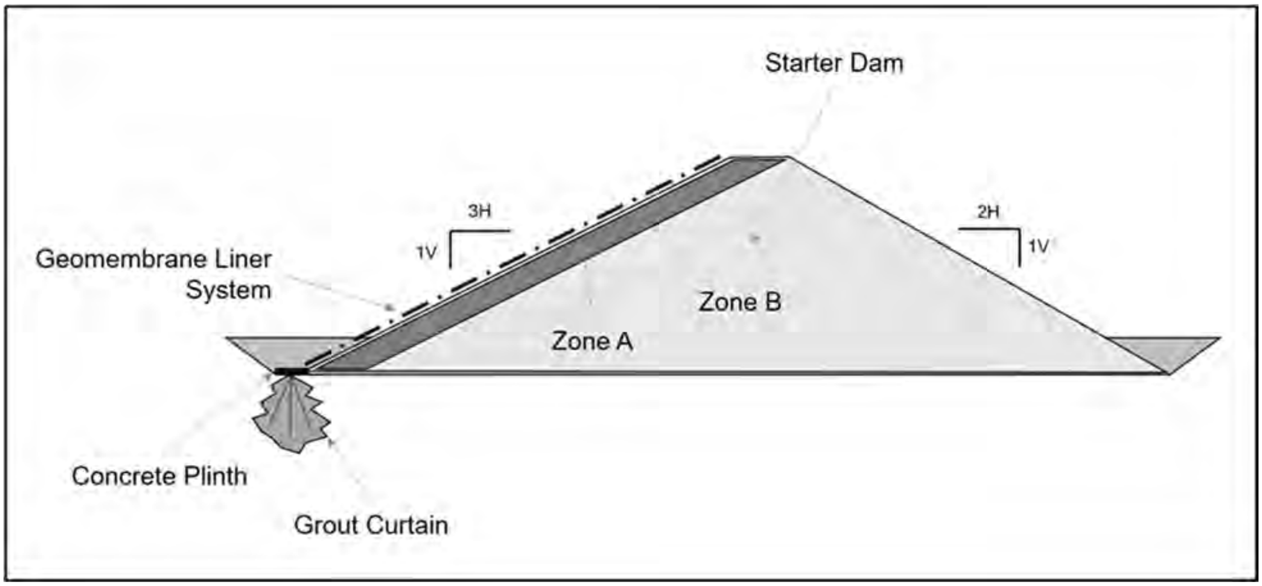


Figure 18-6: Tailings Management Facility Starter Dam – Conceptual Section (KP, 2021)

Cyclone sand will be used to construct centerline raises of the dam from Year 1 through Year 25 to an elevation of 981 m with maximum dam height of approximately 280 m (crest to toe). The cyclone sand will be well-drained to develop a zone of drained tailings solids within the facility along the dam. The Main Dam will be constructed with a primary target downstream slope of 3.5H:1V. Additional surplus sand produced through cycloning and screening operations will be utilized for construction to a final downstream slope of approximately 6H:1V. A total of approximately 230 Mm^3 of cyclone sand will be used in construction of the dam. An average in situ dry density of 1.65 t/m^3 is assumed for compacted fill to estimate the cyclone sand fill requirements for embankment construction. Granular soils, such as cyclone sand, can be potentially liquefiable when saturated if they are contractive and demonstrate strain softening behavior in response to undrained shearing. Flattening of the Main Dam final slope with additional cyclone sand will further improve stability and reduce the risks associated with potential saturation of cyclone sand at the base of the facility. A drainage zone will be constructed at the maximum dam section, with connection to the dam abutments through regularly spaced toe drains to improve drainage of cyclone sand near the base of the dam section.

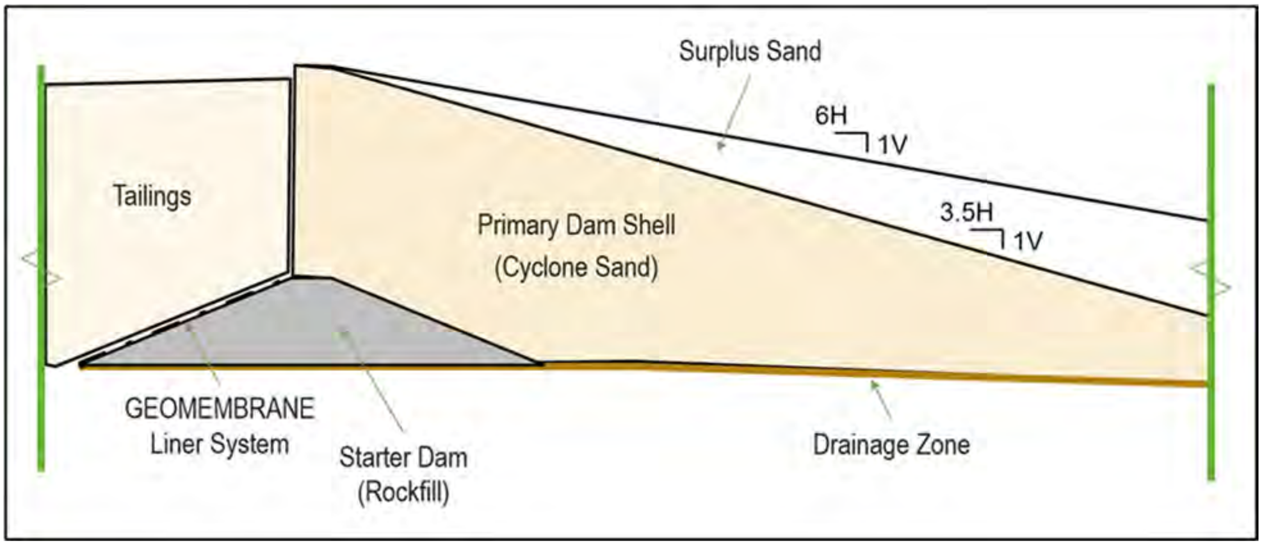


Figure 18-7: Tailings Management Facility Main Dam – Conceptual Section (KP, 2021)

18.5.4 Water Management

The Starter Dam will be constructed starting in pre-production Year -3 to impound water for mill commissioning and start-up. Mill process water for ongoing operations and water for cyclone plant operations will be reclaimed from the TMF supernatant pond and the Yukon River fresh water supply system. The fresh water supply system will be able to offset any predicted water deficit conditions during mine operations and is sized to supply all of the process water requirements, if needed.

Seepage water from the TMF will be collected by a Water Management System, which consists of underdrains and surface ditches that discharge into a water management pond located downstream of the Main Embankment. Seepage and runoff collected in the pond will be pumped back to the process plant using the reclaim water system during operations. The Water Management System will accommodate seepage through the TMF embankments, water recovered from cyclone sand embankment construction, and surface runoff.

A spillway will be constructed at closure to safely route and discharge excess water accumulation within the TMF and to provide safe passage of stormwater volumes from the TMF.

18.6 HEAP LEACH FACILITY

A Heap Leach Facility (HLF) will be constructed on a southeast facing hillslope, approximately 1 km south of the open pit and upgradient of the TMF. HLF operations will commence during pre-production stripping of the open pit. The HLF was designed to accommodate 204 Mt of mineralized material. The heap leach pad will be stacked with mineralized material and leached from Year -3 through Year 20 of mine operations. The mineralized material will be stacked at a nominal rate of approximately 9.1 Mt per year.

The following is a summary of the design features, operational requirements and construction methods which will be required for the proposed facility:

- Excavation within the pad foundation to remove unsuitable materials including ice-rich overburden within the permafrost layer, and a surficial layer of topsoil and organic material. The estimated depth of unsuitable material is approximately 2.5 metres.

- An events pond will be constructed at the foot of the HLF confining embankment for temporary storage of storm runoff and pregnant solution overflow during shutdowns. The events pond will be constructed in the pre-production phase.
- A composite liner system comprising a textured LLDPE liner, compacted soil liner, and leak detection and recovery system to maximise leachate collection and minimise seepage losses will be constructed over the upper portion of the leach pad.
- A double composite liner system comprising two LLDPE liners and a compacted soil liner will be constructed within the lower portion of the leach pad (the in-heap water management pond area) and in the downstream events pond. The double composite liner system will include a leak detection and recovery system for intercepting and collecting potential leakage through the upper liner.
- An overliner drainage layer will be installed on top of the liner system comprising of crushed mineralized material. The overliner material will be carefully placed and spread to avoid damaging the liner and solution collection pipes, and to avoid impacting the integrity of the liner system.
- Construction materials for the confining embankment and events pond may be sourced from mine waste rock. Soil liner material may be sourced from suitable overburden (residual and colluvial soils) and weathered bedrock along the well-drained, non-frozen, south-facing slopes east or west of the HLF.
- The heap leach pad will be developed in stages by loading in successive lifts upslope from the platform developed within the in-heap pond area behind the HLF confining embankment. This will provide initial stability and minimise initial capital costs. The pad will be developed in eight-metre lifts constructed at repose bench face angles of approximately 1.4H:1V. Bench widths approximately nine metres wide will be left at the toe of each lift to establish a final overall slope of 2.5H:1V. Intermittent wider benches will be constructed to limit the vertical height of the HLF along its profile to a maximum of 120 m.
- Operations will involve the irrigation of a weak cyanide solution over the mineralized material lift and the recovery of pregnant solution by means of solution collection pipes and pumps. The solution will be pumped to the gold extraction plant for metal extraction and recycled for re-use in the leaching process. The irrigation lines will be buried to prevent freezing during winter conditions.

The final heap leach pad required for a mineralized material tonnage of 204 Mt will have a surface area of approximately 2.5 million square metres (Mm^2). An events pond will provide storage for excess leachate and storm water runoff. The HLF confining embankment and events pond embankment will require approximately 2.8 Mm^3 of embankment fill materials for construction. The total quantity of LLDPE liner required over the operating life of the HLF is approximately 3 Mm^2. The total quantity of liner required for the events pond is approximately 50,000 m^2.

The HLF will provide approximately 245,000 m^3 of in-heap storage (i.e., storage within the voids of the stacked mineralized material) behind the confining embankment for storage of leachate solution and contact water, with appropriate freeboard for a design storm event (the 1 in 100-year 24-hr storm event). A spillway will be established in the HLF confining embankment to allow for the peak flow from larger storm events to be conveyed safely from the HLF, while maintaining embankment crest freeboard.

The Events Pond is designed to provide additional storage for flood events exceeding the design storm event of the HLF in-heap pond, as well as functioning as a seepage collection pond for leachate solution that may leak past the double composite liner system in the HLF in-heap pond area. The Events Pond will be operated as a dry pond during normal operations to have the maximum capacity available for storage of excess HLF surface runoff from storm events. Water collected in the Events Pond will be used to supplement the HLF water supply for irrigation. Water volumes exceeding the Events Pond storage capacity will be conveyed to the TMF via the Events Pond Spillway.

Preliminary water balance modelling indicates that the HLF will operate in an annual water deficit. During the initial years of HLF operation (i.e., Year -3 to Year -1), a temporary fresh water supply pond will be established through construction of a cofferdam in the TMF. The TMF embankment will be constructed and commissioned in Year 1 of mine operations and HLF water will be sourced from the TMF supernatant pond for the remainder of operations. Makeup water, if required, will be sourced from the Yukon River, which is the primary source of process water makeup for the Project.

The HLF is shown in plan view on Figure 18-8, and in section view on Figure 18-9.

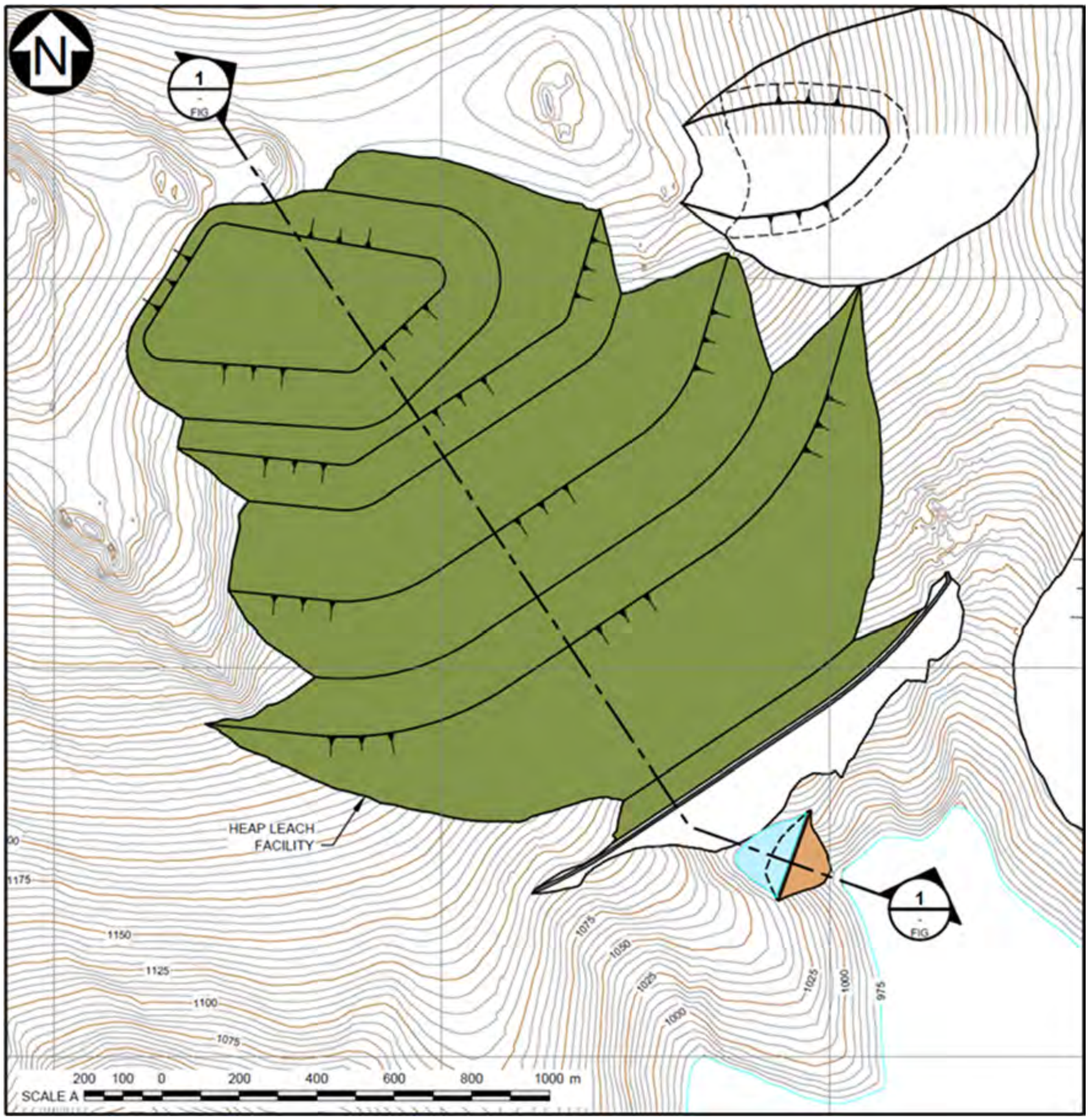


Figure 18-8: Heap Leach Facility General Arrangement (KP, 2021)

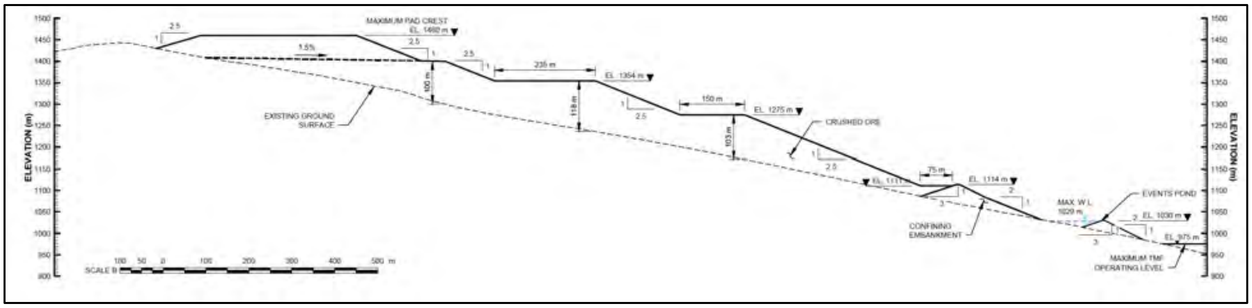


Figure 18-9: Heap Leach Facility Cross-Section (KP, 2021)

18.7 POWER GENERATION AND DISTRIBUTION

18.7.1 Power Generation

Electrical power generation for the Project will be developed in two phases. An initial power plant designated the Supplementary Power Plant will be constructed in the vicinity of the main workforce housing complex to provide power to the camp, for construction activities, and to oxide crushing, conveying and heap leach facilities that go into operation before the main power plant is operational.

The Supplementary Power Plant will consist of three 2250 kW diesel internal combustion engines (ICE). Two of the generators will remain at the Workforce Housing complex and the third will be relocated to the Sand Cyclone (Area 640) facility to provide standby/emergency power to this area after the concentrator start-up.

A Main Power Plant will be constructed at the Casino main mill and concentrator complex to supply the electrical energy required for operations throughout the mine site. The primary electrical power generation will be provided by three Gas Turbine driven generators (two Single Fuel Gas Turbines, one Dual Fuel Gas Turbine) and a steam generator, operating in combined cycle mode (CCGT) with a total installed capacity of approximately 200 MW. The nominal running load to the mine and concentrator complex is about 130 MW. Three diesel internal combustion engine (ICE) driven generators will provide another 6.75 MW of power for black start capability, emergency power, and to complement the gas turbine generation when required. The gas turbines will be fuelled by natural gas (supplied as liquefied natural gas, or LNG). One of the three will have Dual Fuel capabilities - LNG and Diesel.

18.7.2 LNG Receiving, Storage and Distribution Facilities

LNG will be transported to the site from Fort Nelson, British Columbia via tanker trucks and stored on-site in a large 10,000 m^3 site-fabricated storage tank to provide fuel for the power plant. An LNG receiving station is provided to unload the LNG tankers and transfer the LNG into the storage facility. An LNG vaporization facility is provided to convert the LNG into gas at a suitable supply pressure to operate the power generation equipment. In addition to providing fuel for the power plants, the LNG facility will provide fuel for the mine haulage fleet, and fuel for over-the-highway tractors hauling concentrates, lime, grinding media, and LNG from Fort Nelson. Distribution to the on-site vehicles will be by two portable fueling stations, and two mobile refuelers.

18.7.3 Power Distribution

The power system for the Casino Project consists of two generating stations and the distribution system.

The main generating station will consist of a combined cycle plant with three 50.5 megawatt (MW) gas turbines (GT) and approximately 40 MW steam turbine (ST). There will also be three 2.25 MW diesel powered reciprocating engine generators at the main power plant. The GT and the ST units will all generate power at 13.8 kilovolts (kV) which will be stepped up to 34.5 kV through four (4) 13.8 kV to 34.5 kV transformers for the distribution system. The three ICE generators will be stepped up from 600V to 34.5 kV through three transformers.

The second generating station will be located at the main/construction camp site and will consist of three 2.25 MW diesel powered reciprocating engine generators. These units will generate power at 600V and will be stepped up through a 1,500 kVA, 33kV Delta to 400/231 V Wye transformer. This station will be the first installed and will provide power for the Project construction.

The 34.5 kV distribution systems will radiate from a 34.5 kV switchgear line-up with feeders to the SAG mill, Ball Mill No. #1, Ball Mill No. #2, and feeders to the mill and flotation areas in cable tray using insulated copper conductors. Overhead line feeder circuits with aluminum conductor steel reinforced (ACSR) will be provided for the tailings reclaim water, fresh water from the Yukon River, crushing/conveying and SART/ADR, camp site and two feeders to the pit loop.

Electric power utilization voltages will be 4,160 volts for motors 300 horsepower (hp) and above, 575 volts for three-phase motors 250 hp and below. For lighting, small loads and building services 600/347 or 208/120 volts will be the utilization voltage.

18.8 WATER SUPPLY AND DISTRIBUTION

18.8.1 Oxide Mineralized Material Processing

Initial water requirements for the gold plant and oxide mineralized material heap leach operations will be met by pumping water retained behind a temporary cofferdam located in the heap leach event pond. After start-up of the concentrator, process water with be provided from the TMF reclaim water system.

18.8.2 Sulphide Mineralized Material Processing

The main fresh water supply will be supplied from the Yukon River. The water will be collected in a riverbank caisson and radial well system (Ranney Well) and pumped through an above ground insulated 0.9 m (36 in) diameter by 17.4 km long pipeline with four booster stations. The design flow rate of the system is 3,400 m^3/h.

The fire water requirement is 341 m^3/h for two hours. This demand is satisfied by designing the storage pond with a fire reserve capacity of 682 m^3 in the lower portion of the pond that will be unavailable for other uses.

Potable water will be produced by filtering and chlorinating fresh water and will be stored and distributed separately.

Process water reclaimed from the tailings pond by way of two relay stations and one reclaim booster station. The reclaimed water will come from the Sand Cyclone area thickener and from the process plant. One of the relay stations will be taken off-line as the level of the TMF rises. Water will be collected in a 63,700 m^3 process water pond.

18.9 WASTEWATER DISPOSAL

Packaged sewage treatment plant systems will accept and treat all sanitary wastewater.

18.10 COMMUNICATIONS

No communications infrastructure exists in the area of the site. The Project will develop communications infrastructure to meet construction and operations requirements.

19 MARKET STUDIES AND CONTRACTS

No market studies were performed and no sales contracts are in place. The commodities involved in this project are commonly traded on the open market.

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 INTRODUCTION

This section provides further details on environmental studies conducted to date, the environmental assessment process, territorial and federal regulatory approvals required to bring this property into production, and the status of First Nations' consultation and agreements.

20.2 ENVIRONMENTAL STUDIES

Numerous environmental studies have been completed on the Casino property to support previous submissions to the Yukon Environmental and Socio-economic Assessment Board (YESAB), under the Yukon Environmental and Socioeconomic Assessment Act (YESAA). The majority of these studies were completed from 2012 through 2014, with bi-annual surface and groundwater monitoring and climate monitoring programs on-going. Environmental studies were conducted on terrain and terrain hazards, water quality and hydrology, geochemistry, hydrogeology, air quality, noise, fish and aquatic resources, rare plants and vegetation and wildlife. The details of, and reports on, these studies can be found on both the YESAB registry (https://yesabregistry.ca/projects/815c7843-b66d-469f-b9d3-5b62e2c276d4/) and on the CMC website (https://casinomining.com/project/yesab-proposal/).

20.3 PERMITTING

Mining projects in the Yukon require several permits and licences issued either by the Yukon Government or by the various departments of the Government of Canada. The primary regulatory approvals are a Water Licence, issued under the Waters Act, and a Quartz Mining Licence, issued under the Quartz Mining Act. Federal authorizations are required under the Fisheries Act and Navigable Waters Act, amongst others. In advance of licence applications, mining projects require a screening report issued by the YESAB. The environmental assessment and permitting requirements are further detailed below.

20.3.1 Existing Assessments and Permits

Exploration activities at mining projects in the Yukon are undertaken under a Mining Land Use approval, issued by the Yukon Government, Department of Energy, Mines & Resources. Current exploration at the Casino property is approved under Class 4 Quartz Mining Land Use Approval LQ00510, and Class 3 Quartz Mining Land Use Approval LQ00320c. CMC recently underwent assessment through YESAB to combine these two approvals (YESAB project 2020-0083), and a decision document approving this assessment was issued in September 2020. Other existing permits include Waste Management Permit 81-079.

The Yukon Government currently holds $672 in security for the Casino property.

20.3.2 Environmental and Socio-Economic Assessment Process

Larger quartz mining projects (i.e., those that begin mining as opposed to just exploration activities) are typically categorized as assessable activities under the Assessable Activities, Exceptions and Executive Committee Projects Regulations (SOR/2005-379), and require an Executive Committee screening. There is no fee for an assessment to be conducted. Upon completion of the assessment, YESAB issues a screening report and a recommendation, which is sent to federal, territorial and/or First Nation governments who act as Decision Bodies. The recommendation will include one of four options. YESAB will recommend that the project:

- Be allowed to proceed;
- Be allowed to proceed with terms and conditions;

- Not proceed; or
- The Executive Committee can recommend that the project be required to undergo a review by a Panel of the Board.

The Decision Body for the assessment will be a regulating body or authority. Decision Bodies can be federal, territorial or First Nation governments and agencies that regulate and permit the proposed activity. The Decision Body will issue, in writing, a Decision Document that accepts, varies or rejects the recommendation. Once the Decision Document has been issued, an agency can issue authorizations or permits in accordance with their process.

A Project Proposal for the Casino Mine was submitted to the executive committee of the YESAB in January 2014 and underwent several rounds of adequacy review information requests from 2014 through 2016. On February 18, 2016, the Executive Committee determined that the Casino Mine Project requires a Panel Review, the highest level of environmental and socio-economic assessment under YESAA. A Panel Review is an assessment process by which a Panel of the Board (comprised of one YESAB board member nominated by the Council of Yukon First Nations, and two YESAB board members nominated by the territorial or federal governments) conducts technical analysis of an Environmental and Socio-economic Effects Statement submitted by CMC, followed by public hearings. The Panel of the Board then issues their recommendations (similar to the other levels of assessment under YESAA) to the relevant Decision Body(s), which can be federal, territorial and/or First Nation governments. The Decision Body(s) will then decide whether to accept, reject or vary the recommendation of YESAB and issue a Decision Document. The Decision body has 60 days to issue a decision document, or 45 days in which to refer the recommendations back to the Panel of the Board for reconsideration. Regulatory permitting, discussed in the following section, would follow on the heels of a positive decision document being issued. All documents issued and submitted during the environmental assessment are placed on the YESAB Online Registry and are available to the public (https://yesabregistry.ca/).

Guidelines for the Environmental and Socio-economic Effects Statement were issued by YESAB on June 20, 2016. The next steps would be for CMC to prepare and submit the Environmental and Socio-economic Effects Statement in accordance with those guidelines.

20.3.3 Licensing

The mining project will be regulated under the legislation of federal and territorial boundaries thus requiring many permits and approvals. A Quartz Mining Licence will be required and must adhere to the regulations of the Quartz Mining Act particularly as per section 135, issued and administered by the Yukon Government. Additionally, CMC will be required to obtain a Type A Water Licence under the Waters Act for mine operations with use of water and deposit of waste, as well as considerations of tailings creation and storage according to the project design. The Yukon Water Board would administer this licence.

The following federal legislation will also be considered, including:

- Section 35(2) Authorization under the Fisheries Act (harmful alteration, disruption or destruction (HADD) of fish habitat).
- Section 36(4) Regulation or Order in Council under the Fisheries Act (deposit of deleterious substances).
- Section 5(2) Approval under the Navigable Waters Protection Act.
- Blasting Permit under the Explosives Act and Regulations and the Occupational Health and Safety Act.

Other applicable territorial legislation includes:

- Energy and Operating Certificates under the Public Utilities Act.

- Work in Highway Right-of-Way Permit, Access Permit, and Highways Hauling Permit under the Highways Act.
- Land Use, Quarry, and Timber Permits under the Lands Act.
- Air Emissions, Special Waste, and Storage Tank Systems Permits under the Environment Act.
- Burning Permit under the Forest Protection Act.
- Archaeological Sites Permit under the Historic Resources Act.
- Sewage Disposal System Permit under the Public Health and Safety Act.
- Certificate for Transport of Dangerous Goods under the Dangerous Goods Transportation Act.
- Building and Plumbing Permits under the Building Standards Act / Electrical Protection Act.
- Gas Installation Permit under the Gas Burning Devices Act.
- Pressure Vessel Boiler Permit under the Boiler and Pressure Vessel Act.
- Compliance with the Public Health and Safety Act.

20.3.4 Environmental and Mine Operation Plans

Environmental management plans will be assembled under an Environmental Management Plan, which provides overarching direction for environmental and development management at the Casino Project. It is supported by a suite of project-specific mitigation, monitoring and/or management plans that set out the Project's standards and requirements under the Quartz Mining Licence and/or Water Licence for particular areas of environmental management.

20.3.5 Reclamation and Closure

A reclamation and closure plan must be prepared by the mine owner and submitted for review and approval by the government prior to receiving a Quartz Mining License. The reclamation and closure plan must be updated periodically throughout the operating mine life (minimum every five years). A conceptual plan will be expected to support the environmental assessment process, while a detailed plan is expected to be required as a condition for the Quartz Mine License. Provisions for changes and updates as mining progresses are also expected. The Yukon Government will require the company to post security for this project. Both the Yukon Waters Act and the Yukon Quartz Mining Act have provisions for security to be held by government.

A reclamation and closure plan (RCP) will be submitted with the Environmental and Socio-economic Effects Statement. The RCP will include a liability estimate for reclaiming and closing the mine. The RCP will demonstrate how CMC has considered and addressed the expectations and concerns throughout the mine planning process. Over the life of the mine, successive iterations of the plan may be expected every two years, each iteration providing more detail and greater certainty regarding the sequence of events to occur during reclamation and closure.

20.4 FIRST NATIONS AND COMMUNITY ENGAGEMENT

CMC is committed to developing and operating the Casino Project (the "Project") in a safe, ethical and socially responsible manner. CMC has been sharing information and consulting with First Nations, local communities, Yukon government and federal agencies, non-government organizations (NGOs), and individuals since 2008.

The property has components that are located within the Traditional Territories of three of the Yukon First Nations that entered into Final Agreements under the 1993 Yukon Land Claims Umbrella Final Agreement among the Governments of Yukon and Canada and the Yukon First Nations (UFA): the Selkirk First Nation, the Little Salmon/Carmacks First

Nation, and the Tr'ondëk Hwëch'in First Nation. The term 'Traditional Territory' refers to those lands that were historically used by the First Nation for traditional pursuits and were recognized and accepted as such by the Governments of Canada and Yukon in the UFA.

The main deposit and camp infrastructure, and most of the proposed Freegold Road Extension falls within the Traditional Territory of the Selkirk First Nation. The existing Freegold Road and some of the proposed Freegold Extension falls within the Traditional Territory of the Little Salmon/Carmacks First Nation. The barge landing on the Yukon River falls within the Traditional Territory of the Tr'ondëk Hwëch'in First Nation.

In addition to the Selkirk First Nation, the Little Salmon/Carmacks First Nation, and the Tr'ondëk Hwëch'in First Nation, the Kluane First Nation and White River First Nation have also been identified by YESAB as being potentially affected due to potential downstream effects, and require individual consideration within the Environmental and Socio-economic Effects Statement. Kluane First Nation are also a signatory to the UFA, whereas the White River First Nation is an Indian Act band who have not entered into a land claim or a self-government agreement with the Crown.

CMC has signed cooperation agreements with the Selkirk First Nation, the Little Salmon/Carmacks First Nation, and the Tr'ondëk Hwëch'in First Nation, which provided funding for participation in the Executive Committee review of the Project Proposal. Agreements were also reached with Selkirk First Nation, Tr'ondëk Hwëch'in First Nation and White River First Nation for the funding of nation specific Traditional Land Use studies, which were completed in 2018 and 2019. Subsequent updated agreements will be required to facilitate participation in the Panel Review process or in any other processes to be conducted under the proposed mining project. CMC is in regular communication with all five First Nations and meets with leadership regularly.

21 CAPITAL AND OPERATING COSTS

21.1 CAPITAL COST

21.1.1 Initial Capital Cost

Table 21-1 summarizes the initial capital costs.

Table 21-1: Initial Capital Cost Summary

Cost Item	Total (C$M)
Process Plant and Infrastructure	
Project Directs including freight	1,777
Project Indirects	390
Contingency	412
Subtotal	2,579
Mine	
Mine Equipment	409
Mine Preproduction	211
Subtotal	620
Owner's Costs	52
Total	3,251

21.1.2 Basis of Capital Cost Estimate

In general, M3 based this capital cost estimate on its knowledge and experience of similar types of facilities and work in similar locations. To assist in the estimating, M3 used quantity estimates, and in some cases, costs supplied by specialist sub-consultants, such as the following:

- Associated Engineering (AE): Main access from Carmacks to site.
- Knight Piésold (KP): Geotechnical quantities associated with the Heap Leach Facility, Waste Rock Storage Area and the Tailing Management Facility.
- Independent Mining Consultants (IMC): Mine capital and operating costs.

"Initial Capital" is defined as all capital costs through to the end of construction or the end of Year 1 of the mine life. Capital costs predicted for later years are carried as sustaining capital in the financial model.

All costs are from end of 1st quarter 2021 Canadian dollars except as noted otherwise. Canadian to US exchange rate used is C$1.25 = US$1.00.

21.1.2.1 EPCM Execution

The capital costs are based on this project being executed by experienced EPCM contractor(s) in the hard rock mining industry with a recent record of bringing projects on budget or under budget. It is assumed that at least two sufficiently sized self-performing local contractors are in place for all trades, such as civil, concrete, steel, architectural, mechanical, electrical, instrumentation and controls, and process piping. Certain contractors will have multiple trade capabilities.

21.1.2.2 Exclusions and Qualifications

This capital cost estimate excludes the following items:

1. Future escalation.
2. The cost of all prior and future studies.
3. Start-up and initial operating expenses subsequent to Owner's acceptance of the plant ready to accept feed are excluded.
4. Future foreign currency exchange variation.
5. The cost to provide insurance coverage for the duration of the project.
6. Environmental and ecological considerations.

This capital cost estimate depends on the following qualifications:

1. Environmental permits and licenses required to operate the facilities are obtained in a timely manner.
2. Unfettered access to the project site is assured for the duration of the project development and operation.

21.1.2.3 Contingency

Contingency is a cost that statistically will occur based on historical data. The term is not used to cover changes in scope, errors, or lack of sufficient information to meet a desired accuracy range. Contingency is used to cover items of cost which fall within the scope of work but are not known or sufficiently detailed at the time that the estimate is developed. A 20% contingency was applied to the majority of the process plant and infrastructure direct and indirect costs.

21.1.2.4 Documents and MTOs

Documents developed for this PEA included the process design criteria, flowsheet, equipment list, general arrangements, civil earthwork drawings, overall electrical one line diagram and duty specifications for the major process equipment.

Civil, concrete and structural steel material takeoffs (MTOs) were developed from general arrangement drawings. Mechanical, electrical and I&C material and labour is included as allowances based on experience with similar installations.

21.1.2.5 Construction Labour

Burdened construction labour rates used in the process plant and infrastructure capital cost estimate are shown in Table 21-2. The rates were provided in Q2 2021 by a BC industrial contractor experienced in the scope and scale of the Project.

Table 21-2: Burdened Labour Rates

Trade	Labour Rate (C$/ hour)
Civil Work	$119.47
Concrete	$120.67
Architectural	$131.68
Structural Steel	$131.68
Equipment Installation	$131.68
Piping	$131.68
Electrical	$126.32
Instrumentation	$126.32

21.1.3 Direct Costs

21.1.3.1 Equipment

Budgetary quotes were received by M3 for the following major equipment & services:

- ADR Plant
- Apron Feeders
- Barge Pumps
- Belt Feeders
- BOG Compressor
- Column Cells
- Cone Crushers
- Conveyors Sand
- Conveyors – Coarse Mineralized Material
- Cyclone Clusters
- Cyclone Feed Pumps
- Flotation Cells
- Fresh Water Supply (Well) pumps
- Gearless Mill Drives
- LNG Plant Ancillary Pumps
- LNG Plant Glycol Water Heat Exchanger
- Waste Treatment Plant
- Air Transport to/from Whitehorse to Site
- Cyanide Destruction System
- All equipment and installation bulk material for Power Plant
- Diesel Generators
- LNG Pressure Vessels
- LNG Pumps
- Moly Concentrate Dryer
- Press Filter
- Plant Conveyors
- Primary Crushers
- Pumps – Process
- Pumps – Process Water
- Regrind Mills & Moly Regrind Mill
- SAG & Ball Mills
- Screens
- Thickeners
- Water Treatment Plant
- Reagents

Western Copper and Gold supplied the following quotes:

- Camp & Catering
- Coarse Mineralized Material Conveyor Drive & Cyclone Feed Pump Drives
- LNG Tanks
- Diesel generators
- Power Plant, equipment & bulk installation materials
- Siemens Electrical Equipment & E-houses
- Mine Equipment
- Waste Treatment Plant
- Personnel air transport to/from Whitehorse & site

21.1.3.2 Civil Sitework

Civil earthwork quantities and unit rates for the major civil works including the Heap Leach Facility, and Tailings Management Facility were provided by KP.

21.1.3.3 Structural Steel and Concrete

Structural steel and concrete costs for the process plant were estimated using parametric factors collated from constructed projects of a similar size and nature.

21.1.3.4 Architectural

Architectural costs are based on M3 database of similar sized projects for the major buildings.

21.1.3.5 Mechanical, Piping, Electrical and Instrumentation

Cost allowances for mechanical, piping, electrical and instrumentation were estimated as a percentage of the mechanical equipment costs using parametric factors collated from constructed projects of a similar size and nature.

21.1.3.6 Power Generation

The Capital cost estimate was developed based on a quotation for all the power generation equipment and installation bulk materials from the potential Gas Turbine Supplier. The building cost to house the combined cycle gas turbine generation equipment, heat recovery boilers, and steam driven generator and all concrete requirements were estimated by M3. The power generation design criteria used in this study were provided by Western.

21.1.3.7 Main Access Road

AE selected the main access route and designed the 132 km gravel road which is an extension of the Freegold Road from approximately Big Creek to the site. AE's estimate of $128.5 million includes all construction costs and the road construction crew support camp. It is assumed that 30% of this cost will be borne by the Yukon government under the Yukon Resource Gateway Program.

21.1.3.8 Freight

Freight is 8% of equipment and materials.

21.1.4 Indirect Costs

Indirect costs include:

- Camp construction and operating costs
- Construction power
- Engineering, Procurement and Construction Management
- Vendor support
- Commissioning
- Capital and commissioning spares

21.1.5 Mine Capital Costs

The estimated mine capital cost includes the following items:

- Mine major equipment
- Mine support equipment
- Mine preproduction development expense

The estimated cost of the following mining facilities was developed by others and is included in the infrastructure capital budget:

- The mine shop and warehouse
- Fuel and lubricant storage facilities
- Explosive storage facilities
- Office facilities

Table 21-3 summarizes the mine capital cost by category for initial and sustaining capital. The initial capital period is considered to be the four-year period from Years -3 through Year 1, as these are the years of significant capital build-up.

Table 21-3: Mining Capital – Mine Equipment and Mine Development (C$ x 1000)

Category		Initial Capital by Time Period				Initial Capital	Sustaining Capital	Total Capital
		Yr -3	Yr -2	Yr -1	Year 1			
Major Equipment		45,103	63,635	92,723	134,574	336,035	147,108	483,143
Support Equipment @	15.00%	6,765	9,545	13,908	20,186	50,405	22,066	72,471
Initial Spare Parts @	0.00%	0	0	0	0	0	0	0
Shop Tools @	0.00%	0	0	0	0	0	0	0
Equipment Subtotal		51,869	73,180	106,631	154,760	386,440	169,175	555,614
Equipment Contingency @	10.0%	5,187	7,318	10,663	0	23,168	0	23,168
Mine Development		33,361	65,109	112,153	0	210,624	0	210,624
TOTAL MINE CAPITAL		90,416	145,608	229,448	154,760	620,232	169,175	789,406
Exclusions: Mine shop and warehouse, fuel and lubricant storage, explosives storage, and offices.								

Mine preproduction development of C$210.6 million is based on the estimated mine operating costs during the preproduction period. The cost estimate is based on owner operating costs with large equipment plus a contingency to provide additional allowance for additional road construction or other site preparation and subcontracting portions of the mining, etc. Table 21-4 shows the components of the cost during mine development by year. Total preproduction development is estimated as 75.0 million tonnes, corresponding to a unit rate of C$2.81 per tonne.

Table 21-4: Mine Development Direct Costs Plus Contingency (C$ x 1000)

Item	Year -3	Year -2	Year -1	Total
Owner Operating Cost – Large Equipment	26,925	56,765	102,160	185,850
Mine Development Contingency	6,436	8,344	9,994	24,774
Total Mine Development Cost	33,361	65,109	112,154	210,624
% Contingency	23.9%	14.7%	9.8%	13.3%

21.1.6 Owner's Costs

Costs Included in Estimate

Owner's costs prior to the start of project engineering and construction are deemed sunk and not included in this estimate. Owner's costs incurred during the project development include but are not limited to:

- First fills and consumables

- Owner's On-Site team
- Consultants other than EPCM
- Shop tools & furnishings, special tools
- Office equipment, furniture and hardware
- General supplies and safety equipment
- Temporary Offices

21.2 SUSTAINING CAPITAL COSTS

Project sustaining capital costs are capital costs incurred during operations and are included in LOM totals shown in Table 21-5 below. Mining capital includes equipment overhauls and replacements. The Tailings Management Facility capital includes earthworks to raise the embankment per annum. The Process Plant capital includes mobile equipment replacements every 7 years and heap leach facility expansions at Years 3, 7 and 11.

Table 21-5: LOM Sustaining Capital Costs

	Total (C$M)
Mine	169
Tailings Management Facility	293
Process Plant (includes Heap Leach)	257
Total	719

21.3 OPERATING COSTS

This section addresses the following costs:

- Process Plant Operating & Maintenance Cost
- General and Administrative Costs

The operating and maintenance costs for the Casino operations are summarised by areas of the plant and shown in Table 21-6. Cost centers include mine operations, process plant operations, and the General and Administration area. Process operating costs were determined on year by year of operations shown below is (Year 2) for illustration, based on an annual mill mineralized material tonnage of 47.0 million tonnes that will produce approximately 483,000 tonnes of copper concentrates and 16,000 tonnes of molybdenum concentrates. The heap leach plant costs are based on gold mineralized material processed of 9.13 million tonnes and production of approximately 53,000 ounces of gold, 113,000 ounces of silver, and 575 tonnes of copper precipitates. These costs were adjusted on a year-by-year basis and incorporated into the financial model.

Life of mine average operating cost is $9.84 per tonne for sulphide mineralized material, which includes mining, concentrator plant and general and administrative costs. The life of mine average operating cost is $6.03 per tonne for processing oxide mineralized material; mining cost for this material is included in the mine operating cost.

Table 21-6: Operating Cost – Mine Site Cost Summary

Concentrator Processing Units Base Rate (tonnes/year mineralized material)	46,882,000	
Heap Leach Processing Units Base Rate (tonnes/year mineralized material)	9,125,000	
Total Tons Mined	99,897,000	
	Year 2	
	Cost	$/Mill Mineralized material
Mining Operations		
Drilling	$18,103,077	$0.39
Blasting	$23,739,416	$0.51
Loading	$21,181,126	$0.45
Hauling	$75,229,789	$1.60
Roads and Dumps	$23,954,224	$0.51
Mine Services	$10,988,823	$0.23
Mine Administration	$7,438,687	$0.16
Subtotal Mining	$180,635,142	$3.85
Processing Operations		
Concentrator		$/Mill Mineralized Material
Primary Crushing & Stockpile Feed	$15,782,528	$0.34
Grinding, Classification & Pebble Crushing	$144,221,470	$3.08
Flotation & Regrind	$58,950,135	$1.26
Concentrate Thickening/Filtration	$6,810,029	$0.15
Tailings Dewatering & Disposal	$37,908,203	$0.81
Fresh Water/Plant Water	$3,592,351	$0.08
Flotation Reagents	$1,471,567	$0.03
Ancillary Services	$4,601,923	$0.10
Subtotal Concentrator	$273,338,206	$5.83
Supporting Facilities		$/Mill Mineralized Material
Laboratory	$1,636,910	$0.04
General and Administrative	$19,321,105	$0.41
Subtotal Supporting Facilities	$20,958,015	$0.45
Total Mill Mineralized Material Cost	$474,931,363	$10.13
Heap Leach		$/Heap Leach Mineralized Material
Heap Leach - Gold Mineralized Material	$11,961,496	$1.31
ADR/SART - Gold Mineralized Material	$42,912,808	$4.70
Subtotal Heap Leach	$54,874,304	$6.01

21.3.1 Process Plant Operating & Maintenance Costs

The process plant (concentrator and heap leach) operating costs are summarised by cost elements of labour, power, reagents, maintenance parts and supplies and services. The process plant operating costs are shown by cost element in Table 21-7 and Table 21-8.

21.3.1.1 Process Labour & Fringes

Process labour costs were derived from a staffing plan and based on prevailing daily or annual labour rates referenced from an industry survey for Canadian wages and benefits. Labour rates and fringe benefits for employees include all applicable social security benefits as well as all applicable payroll taxes.

21.3.1.2 Power

Power costs were based on obtaining power from an LNG fuelled power plant at a rate of C$0.095 per kWh. Power consumption was based on the equipment list connected kW, discounted for operating time per day and anticipated operating load level. A summary of the power cost and consumption are shown at the end of this section.

21.3.1.3 Reagents

Reagents for the process plants are shown below:

- Copper Flotation Reagents
  - Lime
  - Fuel Oil
  - 3418A
  - A208
  - MIBC
  - Flocculant
  - PAX
- Moly Flotation Reagents
  - NaSH
  - Flocculant
  - Fuel Oil
- Heap Leach Reagents
  - NaSH
  - Sulfuric Acid
  - Hydrochloric Acid
  - Lime
  - Sodium Hydroxide
  - Sodium Cyanide (NaCN)
  - Carbon
  - Anti-scalant
  - Flocculant

Consumption rates were determined from the metallurgical test data or industry practice. Budget quotations were received for reagents supplied to Skagway, AK, or from local sources where available with allowance for freight to site.

21.3.1.4 Maintenance Wear Parts and Consumables

Grinding media and part consumption are based on industry practice for the crusher and grinding operations. An allowance was made to cover the cost of maintenance of all items not specifically identified and the cost of maintenance of the facilities. The allowance made was 5.0% of the direct capital cost of equipment, which totalled approximately $15.8 million annually for repair parts and $1.6 million annually for outside repairs the concentrator and $2.4 million annually for repair parts and $0.2 million annually for outside repairs for the heap leach plants.

21.3.1.5 Process Supplies & Services

Allowances were provided in concentrator, heap leach process plants for outside consultants, outside contractors, vehicle maintenance, and miscellaneous supplies. The allowances were estimated using historical information from other operations and projects. Operating costs for the process plants are summarised in Table 21-7 and Table 21-8.

21.3.2 General Administration

General and administration costs include labour and fringe benefits for the administrative personnel, human resources, and accounting. Also included are office supplies, communications, insurance, employee transportation and camp, and other expenses in the administrative area. Labour costs for G&A are based on a staff of 40. Labour rates are based on a daily rate and include benefits. All other G&A costs were developed as allowances based on historical information from other operations and other projects. Laboratory costs estimates are based on labour and fringe benefits, power, reagents, assay consumables, and supplies and services. All other laboratory costs were developed as allowances based on historical information from other operations and other projects. Note that laboratory services are likely to be contracted out. This estimate retains the costs under the assumption that contract and in-house laboratory services costs are equal.

Table 21-7: Operating Cost – Concentrator Cost Summary – Typical Year of Operation

Processing Units Base Rate (tonnes/year mineralized material)	46,882,000	
	Year 2 Annual Cost	
Primary Crushing & Stockpile Feed		
Labor and Fringes	$2,237,255	$0.05
Power	$5,030,942	$0.11
Liners	$6,085,317	$0.13
Maintenance	$2,110,536	$0.05
Supplies & Services	$318,476	$0.01
Subtotal Primary Crushing & Stockpile Feed	$15,782,528	$0.34
Grinding, Classification & Pebble Crushing		
Labour and Fringes	$2,196,316	$0.05
Power	$64,267,813	$1.37
Liners	$14,318,522	$0.31
Grinding Media	$52,689,506	$1.12
Maintenance	$10,190,700	$0.22
Supplies and Services	$558,613	$0.01
Subtotal Grinding, Classification & Pebble Crushing	$144,221,470	$3.08
Flotation & Regrind		
Labour and Fringes	$2,544,254	$0.05
Power	$9,107,869	$0.19
Reagents	$43,846,055	$0.94
Maintenance	$3,325,664	$0.07
Supplies and Services	$126,292	$0.00
Subtotal Flotation & Regrind	$58,950,135	$1.26
Concentrate Thickening/Filtration		
Labour and Fringes	$3,291,300	$0.07
Power	$1,240,222	$0.03
Maintenance	$2,152,215	$0.05
Supplies and Services	$126,292	$0.00
Subtotal Concentrate Thickening/Filtration	$6,810,029	$0.15
Tailings Dewatering & Disposal		$0.00
Labour and Fringes	$1,208,250	$0.03
Power	$9,020,243	$0.19
Maintenance	$4,920,919	$0.10
Supplies and Services	$22,758,792	$0.49
Subtotal Tailings Dewatering & Disposal	$37,908,203	$0.81
Fresh Water/Plant Water		
Labour and Fringes	$581,025	$0.01
Power	$1,998,100	$0.04
Maintenance	$919,879	$0.02
Supplies and Services	$93,347	$0.00
Subtotal Fresh Water/Plant Water	$3,592,351	$0.08
Flotation Reagents		
Labour and Fringes	$581,025	$0.01
Power	$186,909	$0.00
Maintenance	$621,268	$0.01
Supplies and Services	$82,365	$0.00
Subtotal Flotation Reagents	$1,471,567	$0.03
Ancillary Services		
Labour and Fringes	$1,159,142	$0.02
Power	$712,987	$0.02
Maintenance	$2,647,430	$0.06
Supplies and Services	$82,365	$0.00
Subtotal Ancillary Services	$4,601,923	$0.10
Total Process Plant	$273,338,206	$5.83

Table 21-8: Operating Cost – Heap Leach Cost Summary – Typical Year of Operation

Processing Units Base Rate (tonnes/year mineralized material) Processing Units Base Rate (doré oz/year) - Gold Mineralized material	9,125,000 89,538	
Heap Leach		
Labour and Fringes	$1,153,053	$0.13
Power	$3,113,040	$0.34
Liners	$3,706,324	$0.41
Maintenance	$2,617,069	$0.29
Supplies & Services	$1,374,840	$0.15
Subtotal Heap Leach	$11,964,327	$1.31
ADR/SART (Gold Mineralized Material)		
Labour and Fringes	$2,447,614	$0.27
Power	$1,596,159	$0.17
Reagents	$35,971,325	$3.94
Maintenance	$2,318,020	$0.25
Supplies and Services	$579,690	$0.06
Subtotal ADR/SART (Gold Mineralized Material)	$42,912,808	$4.70
Total Process Plant	$54,874,304	$6.01

21.3.3 Mine Operating Costs

Table 21-9 summarizes the mine operating costs. Total cost, the cost per total tonne, and cost per mill tonne are shown by various time periods. During commercial production, the unit costs for mining are C$1.902 per total tonne and C$3.476 per mill tonne. The leach tonnes are not in the divisor for the cost per mill tonne calculations. Years 21 to 25 are low grade stockpile re-handle.

Table 21-9: Summary of Total and Unit Mining Costs

Category	Total Material (kt)	Mill Material (kt)	Total Cost (C$x1000)	Cost Per Total Tonne (C$/t)	Cost Per Mill Tonne (C$/t)
Mine Development (PP)	74,999	0	210,624	2.808	0.000
Commercial Production (Years 1 to 25)	2,059,367	1,126,966	3,917,116	1.902	3.476
All Time Periods	2,134,366	1,126,966	4,127,740	1.934	3.663
Commercial Production Years 1 - 5	505,706	218,940	945,555	1.870	4.319
Commercial Production Years 6 - 10	518,002	229,057	1,033,484	1.995	4.512
Commercial Production Years 11 - 15	499,590	230,010	1,006,178	2.014	4.374
Commercial Production Years 16 - 20	313,039	225,929	660,691	2.111	2.924
Commercial Production Yr 21 - 25 (LG)	223,030	223,030	271,209	1.216	1.216

The estimate is based on assumed prices for commodities such as fuel, explosives, parts, etc. that are subject to wide variations depending on market conditions. The current estimate is based on the following estimated prices for key commodities:

- Diesel fuel delivered to the site for C$0.98 per liter.
- Electrical power at C$0.095 per kWh.
- Bulk emulsion at C$0.85 per kg delivered to the site.
- Tires at approximately 75% of US list prices.
- Exchange rate of C$1.26 = US$1.00.

22 ECONOMIC ANALYSIS

The financial evaluation presents the determination of the Net Present Value (NPV), payback period (time in years to recapture the initial capital investment), and the Internal Rate of Return (IRR) for the project. Annual cash flow projections were estimated over the life of the mine based on the estimates of capital expenditures and production cost and sales revenue. The sales revenue is based on the production of copper concentrate with gold and silver credits, molybdenum concentrate, and gold & silver doré. The estimates of capital expenditures and site production costs have been developed specifically for this project and have been presented in earlier sections of this report.

This economic analysis is based on measured and indicated mineral resources only. Inferred mineral resources are considered waste for this analysis.

All amounts are in Canadian dollars with an exchange rate of 1.25:1 with the US dollar except as noted otherwise.

22.1 MINE PRODUCTION STATISTICS

Mine production is reported as direct feed mill mineralized material, SOX stockpile mineralized material, low grade mineralized material, leach mineralized material and waste material from the mining operation. The annual production figures were obtained from the mine plan as reported earlier in this report. The financial model reflects the stockpiling of low-grade mineralized material and its subsequent processing at the end of mine life.

The life of mine mineralized material and waste quantities and mineralized material grade are presented in Table 22-1.

Table 22-1: Life of Mine Mineralized Material, Waste Quantities and Mineralized Material Grade

	Tonnes (000')	Copper %	Moly %	Gold g/t	Silver g/t
Direct Mill Feed Mineralized Material	828,145	0.22%	0.02%	0.242	1.834
SOX Stockpile Mineralized Material	35,841	0.25%	0.03%	0.491	2.361
Low Grade Mineralized Material	262,980	0.12%	0.01%	0.140	1.196
Leach Mineralized Material	203,790	0.03%		0.259	1.951
Waste	500,121				
Total Material Mined	1,830,877				

22.2 PLANT PRODUCTION STATISTICS

In the current plan, all the mill mineralized material and leach mineralized material is processed directly, with the low-grade mill mineralized material being stockpiled and processed at the end of mine life. The leach mineralized material will begin to be processed in Year -3, that is, three years before mill mineralized material processing begins.

The gold mineralized material processing by heap leaching will produce two products, a gold and silver doré and a copper precipitate. The estimated production over the life of the heap leach is 1,190,000 ounces of gold, 3,324,000 ounces of silver, and 27.2 million pounds of copper.

Production from the flotation plant will produce a copper-gold-silver concentrate and a molybdenum concentrate. The estimated copper concentrate production for the life of the flotation plant is 6.86 million tonnes containing 4.2 billion pounds of copper and 5.5 million ounces of gold and 32.8 million ounces of silver. The estimated molybdenum concentrate production for the life of the flotation plant is 315,000 tonnes containing 388.7 million pounds of molybdenum.

22.3 CAPITAL EXPENDITURE

22.3.1 Initial Capital

The base case financial indicators have been determined based on 100% equity financing of the initial capital. The base case assumes LNG is used to fuel the power plant from initial start-up of the concentrator and the haulage units throughout the life of the mine. Any acquisition cost or expenditures prior to the full project production decision have been treated as “sunk” cost and have not been included in the analysis.

The total capital carried in the financial model for new construction is $3.251 billion in Canadian dollars and is expended over a five-year period (4 years of construction plus one additional year for finalizing invoices and other miscellaneous items). The cash flow for the new construction is shown being expended in the years before production and ending in the first year of production. The initial capital includes Owner’s costs and contingency and the capital for the power plant.

22.3.2 Sustaining Capital

A schedule of capital cost expenditures during the production period was estimated and included in the financial analysis under the category of sustaining capital. The total life of mine sustaining capital is estimated to be $718.7 million. This capital will be expended during a 25-year period.

22.3.3 Working Capital

Accounts receivable for sale of the metals vary by year depending on sales revenue. Operating working capital is allowed at two months of sales revenue to provide cash to meet operating expenses prior to receipt of sales revenue. In addition, working capital for accounts payable is being allowed for 30 days, also an allowance for plant consumable inventory is estimated in Year -3 for the Heap Leach plant and -1 for the process plant. All the working capital is recaptured at the end of the mine life and the final value of the account is $0.

22.3.4 Salvage Value

A $31.5 million allowance for salvage value has been included in the cash flow analysis.

22.4 REVENUE

Annual revenue is determined by applying selected metal prices to the annual payable metal contained in the concentrates and doré estimated for each operating year. Sales prices have been applied to all life of mine production without escalation or hedging. The base case financial evaluation uses long term prices that were based on analyst consensus of US$3.35 per pound of copper, US$1,600 per oz of gold, US$24.00 per oz of silver and US$12.00 per pound of molybdenum with an exchange rate of US$0.80 per C$. Prices used are as shown in Table 22-2.

Table 22-2: Metal Prices Used in Financial Model

	Base Case Long Term Price
Copper (C$/lb)	4.19
Molybdenum (C$/lb)	15.00
Gold (C$/oz)	2,000.00
Silver (C$/oz)	30.00

The revenue is the gross value of payable metals sold before transportation and smelting charges.

22.5 TOTAL CASH OPERATING COST

The sulphide average cash operating costs are $11.67 per tonne of mill mineralized material processed. Included in these costs are the mining operations, concentrator, general administration cost and smelter and transportation cost. Heap Leach process operating cost is $6.03 per tonne of leach mineralized material. Included in these costs are the heap leach operations, ADR/SART operations and transportation and refining.

Specifics of operating costs are discussed in Section 21.

22.6 TOTAL CASH PRODUCTION COST

Total Cash Production Cost is the Total Cash Operating Cost plus royalties, carbon tax, reclamation & closure, property tax and salvage income. The sulphide average cash production cost including these items totals $14.11 per tonne of mill mineralized material processed. The Heap Leach costs remain the same at $6.03 per tonne of leach mineralized material as the sulphide process bears these additional costs alone.

22.6.1 Royalty

A NSR royalty will be paid using a rate of 2.75% totalling $883.1 million over the life of the mine.

It is estimated that $2.131 billion will be paid in Yukon mining royalties. Yukon mining royalties are based on a sliding scale of 12% of revenues less operating expenses, depreciation and pre-production expenses.

22.6.2 Taxes

22.6.2.1 Corporate Tax

Corporate income taxes paid is estimated to be $3.940 billion for the life of the mine based on a 30% combined federal and territorial corporate income tax rate of taxable income. A deduction of depreciation for class 41A assets is being taken which results in no income tax being paid until initial capital is fully depreciated. These deductions against income are applied each year but cannot create a loss.

22.6.2.2 Property Tax

An allowance of $100,000 per year was included in the cash flow to account for property tax.

22.6.2.3 Carbon Tax

Canadian Federal Carbon Tax pricing without any rebate was included in the cash flow to account for the carbon tax.

22.6.3 Reclamation & Closure

$200.0 million spread over four years (one year of production and three years of post-production) was allowed for post-closure reclamation & final closeout.

22.7 TOTAL PRODUCTION COST

Total Production Cost is the Total Cash Cost plus depreciation. Depreciation is calculated by the 25% Declining Balance method starting with the first year of production. The last year of production is the catch-up year if the assets are not fully depreciated by that time. An additional deduction for the initial capital is being taken in the early years until the initial capital is fully depreciated.

22.8 PROJECT FINANCING

It is assumed the project will be all equity financed.

22.9 NET INCOME AFTER TAX

Net Income after tax amounts to $9.1 billion for the life of the mine.

22.10 NPV AND IRR

The base case economic analysis (Table 22-3) indicates that the project has an Internal Rate of Return (IRR) of 19.5% after taxes with a payback period of 3.0 years.

Table 22-3 compares the base case project financial indicators with the financial indicators for other cases when the sales price, the amount of capital expenditure, operating cost, and copper recovery are varied from the base case values. By comparing the results of this sensitivity study it can be seen that the project IRR's sensitivity to variation in sales price has the most impact, while variation of operating cost, variation of mill recovery, and variation of capital cost are approximately equal. Table 22-4 further demonstrates the project's sensitivity to metals prices.

Table 22-3: Sensitivity Analysis (After tax)

	NPV @ 0%	NPV @ 5%	NPV @ 8%	NPV @ 10%	IRR	Payback Years
Base Case (LTP)	$9,073	$3,896	$2,332	$1,623	19.5%	3.0
Base-Case Sensitivities						
Metals Price +10%	$11,245	$4,995	$3,111	$2,256	22.6%	2.7
Metals Price -10%	$6,900	$2,795	$1,552	$989	16.1%	3.5
Capex +10%	$8,872	$3,703	$2,146	$1,442	17.8%	3.2
Capex -10%	$9,274	$4,089	$2,517	$1,803	21.4%	2.8
Opex +10%	$8,310	$3,516	$2,064	$1,406	18.3%	3.2
Opex -10%	$9,835	$4,275	$2,599	$1,839	20.6%	2.9
Mill Recovery +5%	$10,023	$4,372	$2,667	$1,893	20.8%	2.8
Mill Recovery -5%	$8,123	$3,420	$1,997	$1,352	18.1%	3.2
$ in millions						

Base Case Commodity Prices	
Copper	$4.19
Molybdenum	$15.00
Gold	$2,000.00
Silver	$30.00

Table 22-4: Metal Price Sensitivity

Exchange Rate - .80$US			Copper Price ($US)						
			$2.50	$3.00	$3.35	$4.00	$4.50	$5.00	$5.50
Gold Price ($US)	$1,200.00	NPV 8% (millions)	$654	$1,209	$1,597	$2,309	$2,857	$3,405	$3,952
		IRR%	11.6%	14.4%	16.1%	19.1%	21.2%	23.3%	25.2%
		Payback (years)	4.8	3.9	3.5	3.0	2.8	2.6	2.4
	$1,400.00	NPV 8% (millions)	$1,028	$1,581	$1,965	$2,677	$3,224	$3,772	$4,318
		IRR%	13.6%	16.2%	17.8%	20.7%	22.8%	24.8%	26.6%
		Payback (years)	4.1	3.5	3.2	2.9	2.6	2.4	2.3
	$1,600.00	NPV 8% (millions)	$1,399	$1,948	$2,332	$3,044	$3,591	$4,138	4,682
		IRR%	15.4%	17.9%	19.5%	22.3%	24.3%	26.2%	28.1%
		Payback (years)	3.7	3.2	3.0	2.7	2.5	2.3	2.2
	$1,800.00	NPV 8% (millions)	$1,767	$2,315	$2,698	$3,410	$3,957	$4,501	$5,045
		IRR%	17.2%	19.5%	21.1%	23.8%	25.8%	27.7%	29.5%
		Payback (years)	3.3	3.0	2.8	2.5	2.3	2.2	2.1
	$2,000.00	NPV 8% (millions)	$2,133	$2,681	$3,064	$3,776	$4,320	$4,864	$5,408
		IRR%	18.9%	21.2%	22.7%	25.4%	27.3%	29.1%	30.8%
		Payback (years)	3.0	2.8	2.6	2.4	2.2	2.1	2.0
	$2,200.00	NPV 8% (millions)	$2,498	$3,046	$3,429	$4,138	$4,682	$5,226	$5,770
		IRR%	20.6%	22.8%	24.3%	26.8%	28.7%	30.5%	32.2%
		Payback (years)	2.8	2.6	2.5	2.2	2.1	2.0	1.9

Exchange Rate - .80$US	Molybdenum Price ($US)						
	$8.00	$10.00	$12.00	$14.00	$16.00	$18.00	$20.00
NPV 8% (millions)	$2,006	$2,169	$2,332	$2,495	$2,657	$2,820	$2,983
IRR%	18.2%	18.9%	19.5%	20.0%	20.6%	21.2%	21.7%
Payback (years)	3.1	3.0	3.0	2.9	2.9	2.8	2.8

	Exchange Rate						
	0.70	0.75	0.80	0.85	0.90	0.95	1.00
NPV 8% (millions)	$3,445	$2,851	$2,332	$1,873	$1,464	$1,096	$764
IRR%	23.9%	21.6%	19.5%	17.5%	15.7%	13.9%	12.3%
Payback (years)	2.5	2.8	3.0	3.3	3.6	3.9	4.5

Mineralized material produced during the first four years is substantially higher in copper, gold, silver, and molybdenum than the life-of-mine average. The results are more robust cash flow during those years allowing payback in 3.0 years under the base case. Table 22-5 illustrates the difference during these early years.

Table 22-5: Project Cash Flow

	Years 1-4	Life of Mine
Average Annual Pre-Tax Cashflow ($ thousand)	$1,081,070	$644,751
Average Annual After-Tax Cashflow ($ thousand)	$965,163	$487,156
Average NSR (sulphide mineralized material)	$41.92	$28.14
Average Annual Mill Feed Grade		
Copper (%)	0.312%	0.197%
Gold (g/t)	0.360	0.226
Silver (g/t)	2.039	1.702
Molybdenum (%)	0.026%	0.022%
Average Concentrate Production		
Copper (dry kt)	408	274
Molybdenum (dry kt)	13	13
Average Annual Metal Production		
Copper & Molybdenum Concentrate		
Copper (Mlbs)	252	169
Gold (koz)	339	220
Silver (koz)	1,572	1,310
Molybdenum (Mlbs)	17	16
Gold/Silver Doré		
Gold (koz)	47	42
Silver (koz)	130	119
Copper Precipitate		
Copper (Mlbs)	1.2	1.0

The Financial Model is shown on the following page in Table 22-6.

Table 22-6: Financial Model

		Total	-4	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25
Mining Operations																															
Direct Feed Mill Mineralized Material	Beginning Inventory (kt)	828,145		828,145	828,145	827,925	823,477	793,645	746,763	700,291	658,282	616,405	574,499	532,474	489,705	446,795	405,348	363,466	320,729	277,283	231,084	185,979	139,986	94,947	50,573	5,360	-	-	-	-	-
	Mined (kt)	828,145		-	220	4,448	29,832	46,882	46,472	42,009	41,877	41,906	42,025	42,769	42,910	41,447	41,882	42,737	43,446	46,199	45,105	45,993	45,039	44,374	45,213	5,360	-	-	-	-	-
	Ending Inventory (kt)	-		828,145	827,925	823,477	793,645	746,763	700,291	658,282	616,405	574,499	532,474	489,705	446,795	405,348	363,466	320,729	277,283	231,084	185,979	139,986	94,947	50,573	5,360	-	-	-	-	-	-
	Copper Grade (%)	0.218%		0.000%	0.242%	0.247%	0.372%	0.338%	0.286%	0.269%	0.216%	0.202%	0.203%	0.204%	0.189%	0.260%	0.237%	0.212%	0.195%	0.196%	0.183%	0.154%	0.153%	0.169%	0.153%	0.193%	0.000%	0.000%	0.000%	0.000%	0.000%
	Molybdenum Grade (%)	0.025%		0.0000%	0.0177%	0.0135%	0.0238%	0.0297%	0.0323%	0.0190%	0.0144%	0.0185%	0.0271%	0.0306%	0.0346%	0.0279%	0.0303%	0.0295%	0.0284%	0.0219%	0.0148%	0.0127%	0.0199%	0.0269%	0.0321%	0.0280%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%
	Gold Grade (g/t)	0.242		-	0.332	0.437	0.427	0.357	0.358	0.313	0.265	0.254	0.248	0.239	0.226	0.210	0.213	0.215	0.205	0.167	0.190	0.188	0.181	0.193	0.186	0.201	-	-	-	-	-
	Silver Grade (g/t)	1.834		-	1.920	2.302	2.485	1.901	1.951	1.977	1.948	1.784	2.180	1.677	1.669	1.890	1.929	1.649	1.524	1.747	1.584	1.659	2.210	1.610	1.730	1.250	-	-	-	-	-
	Contained Copper (klbs)	3,978,640		-	1,174	24,238	244,405	349,015	292,897	249,155	199,315	186,999	188,078	192,231	178,791	237,130	218,408	199,505	186,991	199,409	181,811	156,527	151,920	165,329	152,507	22,806	-	-	-	-	-
	Contained Molybdenum (klbs)	455,165		-	86	1,326	15,661	30,646	33,069	17,606	13,267	17,098	25,108	28,810	32,705	25,472	27,941	27,794	27,241	22,280	14,765	12,910	19,759	26,316	31,996	3,309	-	-	-	-	-
	Contained Gold (kozs)	6,436		-	2	62	409	538	534	423	357	342	335	328	312	280	287	295	287	248	276	278	262	275	270	35	-	-	-	-	-
	Contained Silver (kozs)	48,843		-	14	329	2,384	2,866	2,915	2,670	2,623	2,404	2,945	2,307	2,302	2,519	2,597	2,266	2,129	2,595	2,297	2,454	3,200	2,297	2,515	215	-	-	-	-	-
SOX Stockpile Mineralized Material	Beginning Inventory (kt)	35,841		35,841	35,841	33,947	20,905	778	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	Mined (kt)	35,841		-	1,894	13,042	20,127	778	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	Ending Inventory (kt)	-		35,841	33,947	20,905	778	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	Copper Grade (%)	0.398%		0.000%	0.195%	0.185%	0.561%	0.242%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%
	Molybdenum Grade (%)	0.031%		0.0000%	0.0193%	0.0212%	0.0399%	0.0116%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%
	Gold Grade (g/t)	0.572		-	0.541	0.581	0.579	0.322	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	Silver Grade (g/t)	2.736		-	2.400	2.930	2.674	1.900	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	Contained Copper (klbs)	314,517		-	8,142	53,192	249,032	4,151	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	Contained Molybdenum (klbs)	24,807		-	806	6,096	17,707	199	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	Contained Gold (kozs)	659		-	33	244	374	8	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	Contained Silver (kozs)	3,153		-	146	1,229	1,731	48	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Low Grade Mineralized Material	Beginning Inventory(kt)	262,980		262,980	262,980	262,980	262,821	258,140	252,653	245,394	224,824	188,076	163,938	147,612	137,675	124,753	113,531	97,496	87,244	78,107	66,618	46,664	18,337	6,737	-	-	-	-	-	-	-
	Mined (kt)	262,980		-	-	159	4,681	5,487	7,259	20,570	36,748	24,138	16,326	9,937	12,922	11,222	16,035	10,252	9,137	11,489	19,954	28,327	11,600	6,737	-	-	-	-	-	-	-
	Ending Inventory (kt)	-		262,980	262,980	262,821	258,140	252,653	245,394	224,824	188,076	163,938	147,612	137,675	124,753	113,531	97,496	87,244	78,107	66,618	46,664	18,337	6,737	-	-	-	-	-	-	-	-
	Copper Grade (%)	0.123%		0.000%	0.000%	0.156%	0.169%	0.158%	0.124%	0.159%	0.143%	0.121%	0.107%	0.096%	0.094%	0.115%	0.114%	0.111%	0.131%	0.137%	0.118%	0.109%	0.102%	0.102%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%
	Molybdenum Grade (%)	0.012%		0.0000%	0.0000%	0.0067%	0.0153%	0.0040%	0.0080%	0.0121%	0.0106%	0.0131%	0.0175%	0.0135%	0.0134%	0.0102%	0.0121%	0.0113%	0.0101%	0.0088%	0.0089%	0.0122%	0.0164%	0.0211%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%
	Gold Grade (g/t)	0.140		-	-	0.214	0.190	0.143	0.189	0.141	0.142	0.145	0.135	0.154	0.157	0.157	0.145	0.137	0.129	0.122	0.125	0.130	0.124	0.117	-	-	-	-	-	-	-
	Silver Grade (g/t)	1.196		-	-	1.720	1.367	0.887	1.193	1.216	1.207	1.244	1.316	1.114	1.060	1.210	1.176	1.165	1.127	1.280	1.060	1.238	1.430	0.970	-	-	-	-	-	-	-
	Contained Copper (klbs)	712,041		-	-	547	17,390	19,055	19,901	71,902	116,175	64,455	38,541	21,106	26,740	28,567	40,354	25,181	26,300	34,642	51,856	68,095	26,085	15,150	-	-	-	-	-	-	-
	Contained Molybdenum (klbs)	69,962		-	-	23	1,574	488	1,281	5,478	8,558	6,996	6,289	2,954	3,823	2,529	4,271	2,547	2,031	2,242	3,931	7,618	4,194	3,134	-	-	-	-	-	-	-
	Contained Gold (kozs)	1,188		-	-	1	29	25	44	93	167	113	71	49	65	57	75	45	38	45	81	118	46	25	-	-	-	-	-	-	-
	Contained Silver (kozs)	10,114		-	-	9	206	156	278	804	1,426	966	691	356	440	436	606	384	331	473	680	1,128	533	210	-	-	-	-	-	-	-
Total Mill Mineralized Material	Beginning Inventory(kt)	1,126,966	-	1,126,966	1,126,966	1,124,852	1,107,203	1,052,563	999,416	945,685	883,106	804,481	738,437	680,086	627,380	571,548	518,879	460,962	407,973	355,390	297,702	232,643	158,323	101,684	50,573	5,360	-	-	-	-	-
	Mined (kt)	1,126,966	-	-	2,114	17,649	54,640	53,147	53,731	62,579	78,625	66,044	58,351	52,706	55,832	52,669	57,917	52,989	52,583	57,688	65,059	74,320	56,639	51,111	45,213	5,360	-	-	-	-	-
	Ending Inventory (kt)	-		1,126,966	1,124,852	1,107,203	1,052,563	999,416	945,685	883,106	804,481	738,437	680,086	627,380	571,548	518,879	460,962	407,973	355,390	297,702	232,643	158,323	101,684	50,573	5,360	-	-	-	-	-	-
	Copper Grade (%)	0.201%	0.000%	0.000%	0.200%	0.200%	0.424%	0.318%	0.264%	0.233%	0.182%	0.173%	0.176%	0.184%	0.167%	0.229%	0.203%	0.192%	0.184%	0.184%	0.163%	0.137%	0.143%	0.160%	0.153%	0.193%	0.000%	0.000%	0.000%	0.000%	0.000%
	Molybdenum Grade (%)	0.022%	0.000%	0.000%	0.019%	0.019%	0.029%	0.027%	0.029%	0.017%	0.013%	0.017%	0.024%	0.027%	0.030%	0.024%	0.025%	0.026%	0.025%	0.019%	0.013%	0.013%	0.019%	0.026%	0.032%	0.028%	0.000%	0.000%	0.000%	0.000%	0.000%
	Gold Grade (g/t)	0.229	-	-	0.519	0.541	0.462	0.335	0.335	0.257	0.207	0.214	0.216	0.223	0.210	0.199	0.194	0.200	0.192	0.158	0.171	0.166	0.169	0.183	0.186	0.201	-	-	-	-	-
	Silver Grade (g/t)	1.714	-	-	2.350	2.761	2.459	1.797	1.849	1.727	1.602	1.587	1.938	1.571	1.528	1.745	1.720	1.556	1.455	1.654	1.423	1.499	2.050	1.526	1.730	1.250	-	-	-	-	-
	Contained Copper (klbs)	5,005,198	-	-	9,316	77,977	510,827	372,220	312,798	321,057	315,490	251,454	226,619	213,337	205,531	265,697	258,763	224,685	213,292	234,051	233,667	224,621	178,005	180,479	152,507	22,806	-	-	-	-	-
	Contained Molybdenum (klbs)	549,935	-	-	892	7,445	34,943	31,334	34,350	23,084	21,825	24,095	31,397	31,764	36,527	28,001	32,212	30,341	29,272	24,521	18,696	20,528	23,954	29,450	31,996	3,309	-	-	-	-	-
	Contained Gold (kozs)	8,283	-	-	35	307	812	572	578	516	524	454	406	377	377	336	362	340	324	293	357	396	308	301	270	35	-	-	-	-	-
	Contained Silver (kozs)	62,109	-	-	160	1,567	4,320	3,070	3,193	3,474	4,049	3,369	3,636	2,662	2,743	2,955	3,203	2,650	2,460	3,068	2,977	3,582	3,733	2,507	2,515	215	-	-	-	-	-
Leach Mineralized Material	Beginning Inventory(kt)	203,790		203,790	198,303	184,472	161,416	142,033	107,661	90,306	76,586	75,768	72,614	62,612	48,943	34,605	14,666	13,426	7,085	2,554	31	-	-	-	-	-	-	-	-	-	-
	Mined (kt)	203,790		5,487	13,831	23,056	19,383	34,372	17,355	13,720	818	3,154	10,002	13,669	14,338	19,939	1,240	6,341	4,531	2,523	31	-	-	-	-	-	-	-	-	-	-
	Ending Inventory (kt)	-		198,303	184,472	161,416	142,033	107,661	90,306	76,586	75,768	72,614	62,612	48,943	34,605	14,666	13,426	7,085	2,554	31	-	-	-	-	-	-	-	-	-	-	-
	Copper Grade (%)	0.034%		0.0210%	0.0440%	0.0440%	0.0370%	0.0210%	0.0300%	0.0440%	0.0650%	0.0060%	0.0130%	0.0220%	0.0260%	0.0450%	0.0610%	0.0490%	0.0470%	0.0660%	0.0670%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%
	Gold Grade (g/t)	0.259		0.273	0.436	0.406	0.300	0.256	0.211	0.165	0.149	0.199	0.230	0.240	0.203	0.200	0.156	0.146	0.152	0.152	0.156	-	-	-	-	-	-	-	-	-	-
	Silver Grade (g/t)	1.951		1.300	2.680	2.850	1.955	1.480	2.130	1.340	0.840	2.610	2.370	2.200	1.740	1.570	0.990	1.630	1.690	2.160	1.310	-	-	-	-	-	-	-	-	-	-
	Contained Copper (klbs)	150,839		2,540	13,417	22,365	15,802	15,913	11,478	13,309	1,172	417	2,867	6,630	8,219	19,781	1,668	6,850	4,695	3,671	46	-	-	-	-	-	-	-	-	-	-
	Contained Gold (kozs)	1,699		48	194	301	187	283	118	73	4	20	74	105	94	128	6	30	22	12	0	-	-	-	-	-	-	-	-	-	-
	Contained Silver (kozs)	12,786		229	1,192	2,113	1,218	1,636	1,188	591	22	265	762	967	802	1,006	39	332	246	175	1	-	-	-	-	-	-	-	-	-	-
Waste	Beginning Inventory(kt)	500,121		500,121	498,185	494,372	487,259	466,280	453,902	424,989	401,835	381,791	350,988	319,341	285,716	255,886	228,493	187,650	146,980	106,844	73,153	40,446	17,608	5,933	793	12	-	-	-	-	-
	Mined (kt)	500,121		1,936	3,813	7,113	20,979	12,378	28,913	23,154	20,044	30,803	31,647	33,625	29,830	27,393	40,843	40,670	40,136	33,691	32,707	22,838	11,675	5,140	781	12	-	-	-	-	-
	Ending Inventory (kt)	-		498,185	494,372	487,259	466,280	453,902	424,989	401,835	381,791	350,988	319,341	285,716	255,886	228,493	187,650	146,980	106,844	73,153	40,446	17,608	5,933	793	12	-	-	-	-	-	-
	Total Material Mined (kt)	1,830,877		7,423	19,758	47,818	95,002	99,897	99,999	99,453	99,487	100,001	100,000	100,000	100,000	100,001	100,000	100,000	97,250	93,902	97,797	97,158	68,314	56,251	45,994	5,372	-	-	-	-	-
	Waste to Mineralized Material Ratio	0.38		0.35	0.24	0.17	0.28	0.14	0.41	0.30	0.25	0.45	0.46	0.51	0.43	0.38	0.69	0.69	0.70	0.56	0.50	0.31	0.21	0.10	0.02	0.00	-	-	-	-	
	Mine Re-handle (kt)	303,489		-	-	-	4,668	-	-	3,600	3,600	3,600	3,600	3,600	3,600	3,600	3,600	3,600	3,441	-	-	-	-	-	-	39,950	45,814	45,744	45,945	45,232	40,295
Process Plant Operations - Concentrator																															
Supergene Oxide Mineralized Material	Beginning Mineralized Material Inventory (kt)	-		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	Mined Mineralized Material to Concentrator (kt)	76,231	-	-	-	-	-	250	478	7,366	5,035	3,600	3,600	3,957	5,281	5,243	3,646	3,608	4,592	2,405	18	-	-	-	-	-	2,800	2,271	1,549	4,836	15,696
	Mined Mineralized Material - Processed (kt)	76,231	-	-	-	-	-	250	478	7,366	5,035	3,600	3,600	3,957	5,281	5,243	3,646	3,608	4,592	2,405	18	-	-	-	-	-	2,800	2,271	1,549	4,836	15,696
	Ending Mineralized Material Inventory		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	Copper Grade (%)	0.225%		0.000%	0.000%	0.000%	0.000%	0.178%	0.208%	0.345%	0.397%	0.283%	0.250%	0.189%	0.174%	0.185%	0.184%	0.185%	0.241%	0.319%	0.207%	0.000%	0.000%	0.000%	0.000%	0.000%	0.198%	0.138%	0.102%	0.199%	0.172%
	Molybdenum Grade (%)	0.019%		0.0000%	0.0000%	0.0000%	0.0000%	0.0110%	0.0108%	0.0181%	0.0237%	0.0301%	0.0303%	0.0306%	0.0286%	0.0223%	0.0209%	0.0212%	0.0177%	0.0094%	0.0080%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0070%	0.0123%	0.0064%	0.0125%	0.0139%
	Gold Grade (g/t)	0.319		-	-	-	-	0.253	0.360	0.278	0.342	0.449	0.464	0.457	0.410	0.472	0.579	0.580	0.462	0.166	0.160	-	-	-	-	-	0.128	0.165	0.181	0.107	0.149
	Silver Grade (g/t)	1.758		-	-	-	-	1.490	2.500	1.753	1.578	1.973	2.020	2.233	2.280	2.645	2.902	2.944	2.288	1.060	1.050	-	-	-	-	-	1.043	1.144	1.262	0.668	1.116
	Contained Copper (klbs)	377,836		-	-	-	-	981	2,192	55,978	44,081	22,447	19,842	16,468	20,258	21,420	14,809	14,706	24,374	16,914	82	-	-	-	-	-	12,207	6,911	3,496	21,259	59,413
	Contained Molybdenum (klbs)	32,162		-	-	-	-	61	114	2,941	2,631	2,393	2,405	2,669	3,325	2,574	1,683	1,683	1,790	498	3	-	-	-	-	-	431	617	217	1,333	4,795
	Contained Gold (kozs)	782		-	-	-	-	2	6	66	55	52	54	58	70	80	68	67	68	13	0	-	-	-	-	-	12	12	9	17	75
	Contained Silver (kozs)	4,309		-	-	-	-	12	38	415	255	228	234	284	387	446	340	341	338	82	1	-	-	-	-	-	94	84	63	104	563
	Recovery Copper (%)	62.55%		0.00%	0.00%	0.00%	0.00%	71.35%	68.27%	64.85%	63.73%	58.19%	56.40%	60.63%	66.57%	68.31%	62.17%	62.20%	56.28%	59.25%	69.57%	0.00%	0.00%	0.00%	0.00%	0.00%	57.83%	67.79%	75.03%	60.69%	63.89%
	Recovery Molybdenum (%)	52.30%		0.00%	0.00%	0.00%	0.00%	52.30%	52.30%	52.30%	52.30%	52.30%	52.30%	52.30%	52.30%	52.30%	52.30%	52.30%	52.30%	52.30%	52.30%	0.00%	0.00%	0.00%	0.00%	0.00%	52.30%	52.30%	52.30%	52.30%	52.30%
	Recovery Gold (%)	69.00%		0.00%	0.00%	0.00%	0.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	0.00%	0.00%	0.00%	0.00%	0.00%	69.00%	69.00%	69.00%	69.00%	69.00%
	Recovery Silver (%)	60.00%		0.00%	0.00%	0.00%	0.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	0.00%	0.00%	0.00%	0.00%	0.00%	60.00%	60.00%	60.00%	60.00%	60.00%
	Copper Concentrate (kt)	383		-	-	-	-	1	2	59	46	21	18	16	22	24	15	15	22	16	0	-	-	-	-	-	11	8	4	21	61
	Copper Concentrate Grade (%)	28.00%		0.00%	0.00%	0.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%
	Recovered Copper (klbs)	236,326		-	-	-	-	700	1,496	36,304	28,091	13,061	11,191	9,986	13,486	14,633	9,206	9,147	13,716	10,021	57	-	-	-	-	-	7,059	4,685	2,623	12,901	37,961
	Recovered Gold (kozs)	540		-	-	-	-	1	4	45	38	36	37	40	48	55	47	46	47	9	0	-	-	-	-	-	8	8	6	12	52
	Recovered Silver (kozs)	2,586		-	-	-	-	7	23	249	153	137	140	170	232	267	204	205	203	49	0	-	-	-	-	-	56	50	38	62	338
	Molybdenum Concentrate (kt)	14		-	-	-	-	0	0	1	1	1	1	1	1	1	1	1	1	0	0	-	-	-	-	-	0	0	0	1	2
	Molybdenum Concentrate Grade (%)	56.00%		0.00%	0.00%	0.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%
	Recovered Molybdenum (klbs)	16,821		-	-	-	-	32	60	1,538	1,376	1,251	1,258	1,396	1,739	1,346	880	880	936	261	2	-	-	-	-	-	226	323	113	697	2,508

		Total	-4	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25
Supergene Sulfide Mineralized Material	Beginning Mineralized Material Inventory (kt)	-		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	Mined Mineralized Material to Concentrator (kt)	270,493	-	-	-	-	31,099	24,445	7,077	21,782	12,388	389	-	135	2,574	33,367	22,476	13,777	3,650	13,236	20,709	181	-	-	-	104	13,002	16,498	3,667	12,848	17,090
	Mined Mineralized Material - Processed (kt)	270,493	-	-	-	-	31,099	24,445	7,077	21,782	12,388	389	-	135	2,574	33,367	22,476	13,777	3,650	13,236	20,709	181	-	-	-	104	13,002	16,498	3,667	12,848	17,090
	Ending Mineralized Material Inventory		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	Copper Grade (%)	0.255%		0.000%	0.000%	0.000%	0.364%	0.408%	0.318%	0.260%	0.238%	0.246%	0.000%	0.049%	0.234%	0.275%	0.276%	0.268%	0.210%	0.226%	0.225%	0.248%	0.000%	0.000%	0.000%	0.117%	0.137%	0.130%	0.071%	0.145%	0.143%
	Molybdenum Grade (%)	0.020%		0.0000%	0.0000%	0.0000%	0.0225%	0.0337%	0.0336%	0.0131%	0.0142%	0.0193%	0.0000%	0.0040%	0.0225%	0.0273%	0.0296%	0.0251%	0.0240%	0.0100%	0.0175%	0.0208%	0.0000%	0.0000%	0.0000%	0.0120%	0.0083%	0.0118%	0.0040%	0.0110%	0.0077%
	Gold Grade (g/t)	0.240		-	-	-	0.422	0.361	0.322	0.291	0.253	0.216	-	0.398	0.276	0.206	0.213	0.196	0.175	0.154	0.197	0.157	-	-	-	0.149	0.124	0.140	0.220	0.142	0.161
	Silver Grade (g/t)	1.712		-	-	-	2.427	1.780	1.810	1.880	1.830	2.220	-	0.900	2.290	1.930	1.930	1.560	1.550	1.520	1.660	1.530	-	-	-	0.910	1.049	1.167	1.173	1.044	1.198
	Contained Copper (klbs)	1,518,604		-	-	-	249,568	219,879	49,615	124,855	65,000	2,110	-	146	13,279	202,294	136,761	81,400	16,898	65,948	102,725	990	-	-	-	268	39,161	47,152	5,732	41,127	53,699
	Contained Molybdenum (klbs)	118,780		-	-	-	15,437	18,162	5,242	6,291	3,878	166	-	12	1,277	20,082	14,667	7,624	1,931	2,918	7,990	83	-	-	-	27	2,383	4,277	324	3,121	2,888
	Contained Gold (kozs)	2,091		-	-	-	422	284	73	204	101	3	-	2	23	221	154	87	21	66	131	1	-	-	-	0	52	74	26	59	89
	Contained Silver (kozs)	14,891		-	-	-	2,426	1,399	412	1,317	729	28	-	4	190	2,070	1,395	691	182	647	1,105	9	-	-	-	3	439	619	138	431	658
	Recovery Copper (%)	81.50%		0.00%	0.00%	0.00%	77.50%	81.62%	83.96%	82.31%	79.83%	83.33%	0.00%	83.67%	84.19%	82.18%	84.42%	85.07%	84.29%	79.65%	84.44%	83.87%	0.00%	0.00%	0.00%	81.20%	79.81%	80.55%	82.70%	80.23%	79.96%
	Recovery Molybdenum (%)	52.30%		0.00%	0.00%	0.00%	52.30%	52.30%	52.30%	52.30%	52.30%	52.30%	52.30%	52.30%	52.30%	52.30%	52.30%	52.30%	52.30%	52.30%	52.30%	52.30%	0.00%	0.00%	0.00%	52.30%	52.30%	52.30%	52.30%	52.30%	52.30%
	Recovery Gold (%)	69.00%		0.00%	0.00%	0.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	0.00%	0.00%	0.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%
	Recovery Silver (%)	60.00%		0.00%	0.00%	0.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	0.00%	0.00%	0.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%
	Copper Concentrate (kt)	2,005		-	-	-	313	291	67	166	84	3	-	0	18	269	187	112	23	85	141	1	-	-	-	0	51	62	8	53	70
	Copper Concentrate Grade (%)	28.00%		0.00%	0.00%	0.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%
	Recovered Copper (klbs)	1,237,663		-	-	-	193,406	179,460	41,658	102,765	51,891	1,758	-	122	11,179	166,249	115,454	69,251	14,243	52,525	86,745	830	-	-	-	217	31,256	37,979	4,741	32,998	42,938
	Recovered Gold (kozs)	1,443		-	-	-	291	196	51	141	70	2	-	1	16	152	106	60	14	45	91	1	-	-	-	0	36	51	18	40	61
	Recovered Silver (kozs)	8,935		-	-	-	1,456	839	247	790	437	17	-	2	114	1,242	837	415	109	388	663	5	-	-	-	2	263	372	83	259	395
	Molybdenum Concentrate (kt)	50		-	-	-	7	8	2	3	2	0	-	0	1	9	6	3	1	1	3	0	-	-	-	0	1	2	0	1	1
	Molybdenum Concentrate Grade (%)	56.00%		0.00%	0.00%	0.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%
	Recovered Molybdenum (klbs)	62,122		-	-	-	8,074	9,499	2,742	3,290	2,028	87	-	6	668	10,503	7,671	3,987	1,010	1,526	4,179	43	-	-	-	14	1,246	2,237	169	1,632	1,511
Hypogene Mineralized Material	Beginning Mineralized Material Inventory (kt)	-		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	Mined Mineralized Material to Concentrator (kt)	780,242	-	-	-	-	3,401	22,187	38,917	16,461	28,054	41,517	42,025	42,277	38,655	6,437	19,360	28,952	38,645	30,558	24,378	45,812	45,039	44,374	45,213	45,207	30,012	26,978	40,729	27,554	7,502
	Mined Mineralized Material - Processed (kt)	780,242	-	-	-	-	3,401	22,187	38,917	16,461	28,054	41,517	42,025	42,277	38,655	6,437	19,360	28,952	38,645	30,558	24,378	45,812	45,039	44,374	45,213	45,207	30,012	26,978	40,729	27,554	7,502
	Ending Mineralized Material Inventory		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	Copper Grade (%)	0.174%		0.000%	0.000%	0.000%	0.270%	0.262%	0.281%	0.278%	0.203%	0.202%	0.203%	0.205%	0.187%	0.198%	0.191%	0.185%	0.188%	0.173%	0.147%	0.154%	0.153%	0.169%	0.153%	0.116%	0.111%	0.098%	0.111%	0.124%	0.136%
	Molybdenum Grade (%)	0.023%		0.0000%	0.0000%	0.0000%	0.0218%	0.0254%	0.0323%	0.0280%	0.0147%	0.0185%	0.0271%	0.0305%	0.0355%	0.0317%	0.0311%	0.0316%	0.0294%	0.0280%	0.0126%	0.0127%	0.0199%	0.0269%	0.0321%	0.0165%	0.0104%	0.0114%	0.0161%	0.0110%	0.0100%
	Gold Grade (g/t)	0.212		-	-	-	0.473	0.354	0.364	0.365	0.270	0.254	0.248	0.238	0.223	0.224	0.213	0.224	0.209	0.173	0.185	0.188	0.181	0.193	0.186	0.135	0.127	0.151	0.137	0.149	0.159
	Silver Grade (g/t)	1.693		-	-	-	2.743	2.040	1.970	2.140	2.040	1.780	2.180	1.680	1.620	1.650	1.930	1.690	1.530	1.900	1.520	1.660	2.210	1.610	1.730	1.250	1.207	1.162	1.219	1.392	1.305
	Contained Copper (klbs)	2,992,470		-	-	-	20,249	128,154	241,090	100,887	125,552	184,889	188,078	191,070	159,361	28,098	81,522	118,082	160,171	116,548	79,004	155,537	151,920	165,329	152,507	115,730	73,309	58,285	99,268	75,411	22,421
	Contained Molybdenum (klbs)	394,127		-	-	-	1,636	12,424	27,712	10,161	9,092	16,933	25,108	28,427	30,253	4,499	13,274	20,170	25,048	18,863	6,772	12,827	19,759	26,316	31,996	16,422	6,867	6,780	14,443	6,686	1,658
	Contained Gold (kozs)	5,316		-	-	-	52	253	455	193	244	339	335	323	277	46	133	209	260	170	145	277	262	275	270	196	122	131	179	132	38
	Contained Silver (kozs)	42,476		-	-	-	300	1,455	2,465	1,133	1,840	2,376	2,945	2,284	2,013	341	1,201	1,573	1,901	1,867	1,191	2,445	3,200	2,297	2,515	1,816	1,165	1,008	1,596	1,233	315
	Recovery Copper (%)	92.20%		0.00%	0.00%	0.00%	92.20%	92.20%	92.20%	92.20%	92.20%	92.20%	92.20%	92.20%	92.20%	92.20%	92.20%	92.20%	92.20%	92.20%	92.20%	92.20%	92.20%	92.20%	92.20%	92.20%	92.20%	92.20%	92.20%	92.20%	92.20%
	Recovery Molybdenum (%)	78.60%		0.00%	0.00%	0.00%	78.60%	78.60%	78.60%	78.60%	78.60%	78.60%	78.60%	78.60%	78.60%	78.60%	78.60%	78.60%	78.60%	78.60%	78.60%	78.60%	78.60%	78.60%	78.60%	78.60%	78.60%	78.60%	78.60%	78.60%	78.60%
	Recovery Gold (%)	66.00%		0.00%	0.00%	0.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%
	Recovery Silver (%)	50.00%		0.00%	0.00%	0.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%
	Copper Concentrate (kt)	4,470		-	-	-	30	191	360	151	188	276	281	285	238	42	122	176	239	174	118	232	227	247	228	173	109	87	148	113	33
	Copper Concentrate Grade (%)	28.00%		0.00%	0.00%	0.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%
	Recovered Copper (klbs)	2,759,058		-	-	-	18,670	118,158	222,285	93,018	115,759	170,468	173,408	176,166	146,930	25,907	75,163	108,872	147,678	107,457	72,842	143,405	140,070	152,433	140,611	106,703	67,591	53,738	91,525	69,529	20,672
	Recovered Gold (kozs)	3,509		-	-	-	34	167	301	127	161	224	221	214	183	31	88	138	171	112	96	183	173	182	178	129	81	86	118	87	25
	Recovered Silver (kozs)	21,238		-	-	-	150	728	1,232	566	920	1,188	1,473	1,142	1,007	171	601	787	950	933	596	1,222	1,600	1,148	1,257	908	583	504	798	617	157
	Molybdenum Concentrate (kt)	251		-	-	-	1	8	18	6	6	11	16	18	19	3	8	13	16	12	4	8	13	17	20	10	4	4	9	4	1
	Molybdenum Concentrate Grade (%)	56.00%		0.00%	0.00%	0.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%
	Recovered Molybdenum (klbs)	309,784		-	-	-	1,286	9,765	21,782	7,987	7,146	13,309	19,735	22,344	23,779	3,536	10,433	15,853	19,688	14,827	5,323	10,082	15,531	20,684	25,149	12,908	5,397	5,329	11,353	5,255	1,303
Total Mill Mineralized Material	Beginning Mineralized Material Inventory (kt)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	Mined Mineralized Material to Concentrator (kt)	1,126,966	-	-	-	-	34,500	46,882	46,472	45,609	45,477	45,506	45,625	46,369	46,510	45,047	45,482	46,337	46,887	46,199	45,105	45,993	45,039	44,374	45,213	45,311	45,813	45,747	45,944	45,237	40,287
	Mined Mineralized Material - Processed (kt)	1,126,966	-	-	-	-	34,500	46,882	46,472	45,609	45,477	45,506	45,625	46,369	46,510	45,047	45,482	46,337	46,887	46,199	45,105	45,993	45,039	44,374	45,213	45,311	45,813	45,747	45,944	45,237	40,287
	Ending Mineralized Material Inventory		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	Copper Grade (%)	0.197%	0.000%	0.000%	0.000%	0.000%	0.355%	0.338%	0.286%	0.280%	0.234%	0.209%	0.207%	0.203%	0.188%	0.254%	0.232%	0.210%	0.195%	0.196%	0.183%	0.154%	0.153%	0.169%	0.153%	0.116%	0.123%	0.111%	0.107%	0.138%	0.153%
	Molybdenum Grade (%)	0.022%	0.000%	0.000%	0.000%	0.000%	0.022%	0.030%	0.032%	0.019%	0.016%	0.019%	0.027%	0.030%	0.034%	0.027%	0.030%	0.029%	0.028%	0.022%	0.015%	0.013%	0.020%	0.027%	0.032%	0.016%	0.010%	0.012%	0.015%	0.011%	0.011%
	Gold Grade (g/t)	0.226	-	-	-	-	0.427	0.357	0.358	0.316	0.273	0.269	0.265	0.257	0.247	0.240	0.242	0.243	0.231	0.167	0.190	0.188	0.181	0.193	0.186	0.135	0.126	0.148	0.145	0.142	0.156
	Silver Grade (g/t)	1.702	-	-	-	-	2.458	1.901	1.951	1.953	1.932	1.799	2.167	1.725	1.732	1.973	2.008	1.749	1.606	1.747	1.584	1.659	2.210	1.610	1.730	1.249	1.152	1.163	1.217	1.216	1.186
	Contained Copper (klbs)	4,888,911	-	-	-	-	269,817	349,015	292,897	281,719	234,633	209,445	207,919	207,684	192,898	251,813	233,091	214,187	201,443	199,409	181,811	156,527	151,920	165,329	152,507	115,997	124,676	112,347	108,497	137,797	135,533
	Contained Molybdenum (klbs)	545,070	-	-	-	-	17,073	30,646	33,069	19,393	15,601	19,491	27,513	31,108	34,855	27,155	29,624	29,477	28,770	22,280	14,765	12,910	19,759	26,316	31,996	16,449	9,681	11,674	14,984	11,140	9,342
	Contained Gold (kozs)	8,189	-	-	-	-	474	538	534	463	400	394	389	383	370	347	354	363	348	248	276	278	262	275	270	197	185	217	214	207	202
	Contained Silver (kozs)	61,677	-	-	-	-	2,726	2,866	2,915	2,864	2,824	2,632	3,179	2,572	2,590	2,858	2,936	2,606	2,421	2,595	2,297	2,454	3,200	2,297	2,515	1,819	1,698	1,711	1,797	1,768	1,536
	Recovery Copper (%)	86.58%	0.00%	0.00%	0.00%	0.00%	78.60%	85.47%	90.63%	82.38%	83.42%	88.47%	88.78%	89.69%	88.96%	82.12%	85.73%	87.43%	87.19%	85.25%	87.81%	92.15%	92.20%	92.20%	92.20%	92.17%	84.94%	85.81%	91.14%	83.77%	74.94%
	Recovery Molybdenum (%)	71.32%	0.00%	0.00%	0.00%	0.00%	54.82%	62.96%	74.34%	66.08%	67.63%	75.15%	76.30%	76.33%	75.13%	56.66%	64.08%	70.30%	75.20%	74.57%	64.36%	78.43%	78.60%	78.60%	78.60%	78.56%	70.95%	67.57%	77.65%	68.08%	56.97%
	Recovery Gold (%)	67.05%	0.00%	0.00%	0.00%	0.00%	68.67%	67.59%	66.44%	67.75%	67.17%	66.42%	66.41%	66.47%	66.75%	68.60%	67.88%	67.28%	66.76%	66.95%	67.43%	66.01%	66.00%	66.00%	66.00%	66.01%	67.02%	67.19%	66.49%	67.09%	68.43%
	Recovery Silver (%)	53.11%	0.00%	0.00%	0.00%	0.00%	58.90%	54.92%	51.54%	56.05%	53.48%	50.97%	50.74%	51.12%	52.23%	58.81%	55.91%	53.96%	52.15%	52.81%	54.81%	50.04%	50.00%	50.00%	50.00%	50.02%	53.14%	54.11%	51.12%	53.03%	57.95%
	Copper Concentrate (kt)	6,857	-	-	-	-	344	483	430	376	317	300	299	302	278	335	324	303	285	275	259	234	227	247	228	173	172	156	160	187	165
	Copper Concentrate Grade (%)	28.00%	0.00%	0.00%	0.00%	0.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%
	Recovered Copper (klbs)	4,233,047	-	-	-	-	212,076	298,318	265,439	232,087	195,741	185,287	184,598	186,274	171,596	206,789	199,823	187,269	175,637	170,003	159,644	144,235	140,070	152,433	140,611	106,920	105,906	96,402	98,889	115,428	101,571
	Recovered Gold (kozs)	5,491	-	-	-	-	325	364	355	314	268	261	258	255	247	238	241	244	233	166	186	183	173	182	178	130	124	146	143	139	138
	Recovered Silver (kozs)	32,758	-	-	-	-	1,606	1,574	1,503	1,605	1,511	1,342	1,613	1,315	1,353	1,681	1,642	1,406	1,262	1,371	1,259	1,228	1,600	1,148	1,257	910	902	926	919	938	890
	Molybdenum Concentrate (kt)	315	-	-	-	-	8	16	20	10	9	12	17	19	21	12	15	17	18	13	8	8	13	17	20	10	6	6	9	6	4
	Molybdenum Concentrate Grade (%)	56.00%	0.00%	0.00%	0.00%	0.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%
	Recovered Molybdenum (klbs)	388,727	-	-	-	-	9,360	19,296	24,583	12,815	10,550	14,647	20,993	23,746	26,186	15,385	18,984	20,721	21,634	16,613	9,503	10,125	15,531	20,684	25,149	12,922	6,869	7,889	11,635	7,585	5,322
Heap Leach	Beginning Mineralized Material Inventory (kt)	-		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	Mined Mineralized Material to Heap Leach (kt)	203,790		5,487	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	6,678	-	-	-	-	-
	Mined Mineralized Material - Processed (kt)	203,790		5,487	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	6,678	-	-	-	-	-
	Ending Mineralized Material Inventory (kt)	-		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	Copper Grade (%)	0.034%		0.021%	0.044%	0.044%	0.038%	0.021%	0.030%	0.044%	0.040%	0.020%	0.013%	0.022%	0.026%	0.045%	0.047%	0.048%	0.037%	0.034%	0.021%	0.021%	0.028%	0.038%	0.044%	0.044%	0.000%	0.000%	0.000%	0.000%	0.000%
	Gold Grade (g/t)	0.259		0.273	0.436	0.406	0.288	0.256	0.211	0.165	0.182	0.214	0.230	0.240	0.203	0.200	0.194	0.162	0.178	0.212	0.256	0.256	0.261	0.359	0.406	0.427	-	-	-	-	-
	Silver Grade (g/t)	1.951		1.300	2.680	2.850	1.867	1.480	2.130	1.340	1.617	2.192	2.370	2.200	1.740	1.570	1.491	1.612	1.712	2.134	1.479	1.480	1.499	2.507	2.850	2.730	-	-	-	-	-
	Contained Copper (klbs)	150,839		2,540	8,852	8,852	7,745	4,225	6,035	8,852	8,085	4,078	2,615	4,426	5,230	9,053	9,490	9,612	7,389	6,757	4,256	4,225	5,549	7,646	8,852	6,478	-	-	-	-	-
	Contained Gold (kozs)	1,699		48	128	119	84	75	62	48	53	63	67	70	60	59	57	48	52	62	75	75	77	105	119	92	-	-	-	-	-
	Contained Silver (kozs)	12,786		229	786	836	548	434	625	393	474	643	695	645	510	461	437	473	502	626	434	434	440	735	836	586	-	-	-	-	-
	Recovery Copper (%)	18%		18%	18%	18%	18%	18%	18%	18%	18%	18%	18%	18%	18%	18%	18%	18%	18%	18%	18%	18%	18%	18%	18%	18%	18%	18%	18%	18%	18%
	Recovery Gold (%)	70%		70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%
	Recovery Silver (%)	26%		26%	26%	26%	26%	26%	26%	26%	26%	26%	26%	26%	26%	26%	26%	26%	26%	26%	26%	26%	26%	26%	26%	26%	26%	26%	26%	26%	26%
	Copper Precipitate (kt)	21		0	1	1	1	1	1	1	1	1	0	1	1	1	1	1	1	1	1	1	1	1	1	1	-	-	-	-	-

		Total	-4	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25
	Copper Precipitate Grade	60%		60%	60%	60%	60%	60%	60%	60%	60%	60%	60%	60%	60%	60%	60%	60%	60%	60%	60%	60%	60%	60%	60%	60%	60%	60%	60%	60%	60%
	Recovered Copper (klbs)	27,151		457	1,593	1,593	1,394	760	1,086	1,593	1,455	734	471	797	941	1,629	1,708	1,730	1,330	1,216	766	760	999	1,376	1,593	1,166	-	-	-	-	-
	Recovered Gold Dore - Heap Leach (kozs)	1,190		34	90	83	59	53	43	34	37	44	47	49	42	41	40	33	36	44	53	53	54	74	83	64	-	-	-	-	-
	Total Recovered Gold Dore (kozs)	1,190		34	90	83	59	53	43	34	37	44	47	49	42	41	40	33	36	44	53	53	54	74	83	64	-	-	-	-	-
	Recovered Silver Dore (kozs)	3,324		60	204	217	142	113	162	102	123	167	181	168	133	120	114	123	131	163	113	113	114	191	217	152	-	-	-	-	-
Payable Metals	Copper Concentrate																														
	Payable Copper (klbs)	4,084,890	-	-	-	-	204,653	287,877	256,149	223,964	188,890	178,802	178,137	179,754	165,590	199,551	192,829	180,715	169,490	164,053	154,057	139,187	135,167	147,098	135,690	103,178	102,199	93,028	95,428	111,388	98,016
	Payable Gold (kozs)	5,354	-	-	-	-	317	355	346	306	262	255	252	248	241	232	235	238	227	162	182	179	169	177	174	127	121	142	139	136	135
	Payable Silver (kozs)	31,121	-	-	-	-	1,526	1,495	1,427	1,525	1,435	1,275	1,532	1,249	1,285	1,596	1,559	1,336	1,199	1,302	1,196	1,166	1,520	1,091	1,195	864	857	879	873	891	846
	Molybdenum Concentrates																														
	Payable Molybdenum (klbs)	330,418	-	-	-	-	7,956	16,401	20,896	10,893	8,968	12,450	17,844	20,184	22,258	13,077	16,137	17,613	18,389	14,121	8,077	8,606	13,201	17,582	21,377	10,984	5,839	6,706	9,890	6,447	4,524
	Gold/Silver Dore																														
	Payable Metal Gold (kozs)	1,166		33	88	82	58	52	42	33	37	43	46	48	41	40	39	33	36	43	51	52	53	72	82	63	-	-	-	-	-
	Payable Metal Silver (kozs)	3,258		58	200	213	140	111	159	100	121	164	177	164	130	117	111	120	128	160	111	111	112	187	213	149	-	-	-	-	-
	Copper Precipitate																														
	Payable Metal (klbs)	26,201		441	1,538	1,538	1,345	734	1,048	1,538	1,404	708	454	769	909	1,572	1,648	1,670	1,283	1,174	739	734	964	1,328	1,538	1,125	-	-	-	-	-
Income Statement (C$000)																															
	Copper (C$/lb.)	$4.19		$4.19	$4.19	$4.19	$4.19	$4.19	$4.19	$4.19	$4.19	$4.19	$4.19	$4.19	$4.19	$4.19	$4.19	$4.19	$4.19	$4.19	$4.19	$4.19	$4.19	$4.19	$4.19	$4.19	$4.19	$4.19	$4.19	$4.19	$4.19
	Molybdenum (C$/lb)	$15.00		$15.00	$15.00	$15.00	$15.00	$15.00	$15.00	$15.00	$15.00	$15.00	$15.00	$15.00	$15.00	$15.00	$15.00	$15.00	$15.00	$15.00	$15.00	$15.00	$15.00	$15.00	$15.00	$15.00	$15.00	$15.00	$15.00	$15.00	$15.00
	Gold (C$/oz)	$2,000.00		$2,000.00	$2,000.00	$2,000.00	$2,000.00	$2,000.00	$2,000.00	$2,000.00	$2,000.00	$2,000.00	$2,000.00	$2,000.00	$2,000.00	$2,000.00	$2,000.00	$2,000.00	$2,000.00	$2,000.00	$2,000.00	$2,000.00	$2,000.00	$2,000.00	$2,000.00	$2,000.00	$2,000.00	$2,000.00	$2,000.00	$2,000.00	$2,000.00
	Silver (C$/oz)	$30.00		$30.00	$30.00	$30.00	$30.00	$30.00	$30.00	$30.00	$30.00	$30.00	$30.00	$30.00	$30.00	$30.00	$30.00	$30.00	$30.00	$30.00	$30.00	$30.00	$30.00	$30.00	$30.00	$30.00	$30.00	$30.00	$30.00	$30.00	$30.00
	Revenues																														
	Copper Concentrates - Cu	$17,105,478		$0	$0	$0	$856,985	$1,205,486	$1,072,623	$937,848	$790,978	$748,732	$745,950	$752,721	$693,409	$835,620	$807,472	$756,744	$709,740	$686,971	$645,112	$582,845	$566,014	$615,974	$568,200	$432,057	$427,958	$389,553	$399,604	$466,436	$410,444
	Copper Concentrates - Au	$10,707,708		$0	$0	$0	$634,682	$709,471	$692,174	$611,414	$523,539	$509,853	$503,509	$496,820	$481,014	$464,047	$469,072	$475,742	$453,555	$324,193	$363,218	$357,604	$337,316	$354,369	$347,973	$253,085	$242,259	$284,304	$277,966	$271,059	$269,469
	Copper Concentrates - Ag	$933,615		$0	$0	$0	$45,766	$44,863	$42,824	$45,751	$43,050	$38,237	$45,971	$37,465	$38,552	$47,894	$46,784	$40,072	$35,975	$39,063	$35,887	$34,993	$45,602	$32,731	$35,836	$25,935	$25,707	$26,381	$26,185	$26,721	$25,370
	Molybdenum Concentrates	$4,956,269		$0	$0	$0	$119,337	$246,019	$313,437	$163,388	$134,516	$186,752	$267,655	$302,761	$333,865	$196,158	$242,051	$264,191	$275,835	$211,820	$121,162	$129,097	$198,020	$263,723	$320,653	$164,756	$87,580	$100,584	$148,350	$96,706	$67,854
	Dore' - Gold	$2,331,483		$66,076	$175,495	$163,420	$115,822	$103,043	$84,930	$66,414	$73,369	$86,145	$92,578	$96,603	$81,710	$80,502	$78,096	$65,398	$71,497	$85,470	$102,906	$103,043	$105,253	$144,471	$163,420	$125,824	$0	$0	$0	$0	$0
	Dore' - Silver	$97,734		$1,753	$6,010	$6,391	$4,188	$3,319	$4,777	$3,005	$3,625	$4,917	$5,315	$4,934	$3,902	$3,521	$3,344	$3,614	$3,840	$4,785	$3,318	$3,319	$3,363	$5,622	$6,391	$4,481	$0	$0	$0	$0	$0
	Copper Precipitate	$109,716		$1,848	$6,438	$6,438	$5,633	$3,073	$4,390	$6,438	$5,880	$2,966	$1,902	$3,219	$3,804	$6,585	$6,903	$6,991	$5,374	$4,915	$3,096	$3,073	$4,036	$5,561	$6,438	$4,712	$0	$0	$0	$0	$0
	Total Revenues	$36,242,001		$69,677	$187,943	$176,249	$1,782,413	$2,315,274	$2,215,154	$1,834,259	$1,574,958	$1,577,602	$1,662,880	$1,694,522	$1,636,256	$1,634,327	$1,653,722	$1,612,753	$1,555,817	$1,357,216	$1,274,699	$1,213,974	$1,259,603	$1,422,450	$1,448,911	$1,010,850	$783,504	$800,822	$852,105	$860,923	$773,137
Operating Cost	Mining	$4,127,740		$33,361	$65,109	$112,153	$178,366	$180,635	$194,113	$192,440	$200,002	$200,781	$205,214	$210,670	$212,092	$204,728	$198,630	$204,744	$212,264	$202,969	$187,570	$199,481	$148,024	$128,085	$118,332	$66,768	$49,515	$47,447	$47,139	$67,466	$59,642
	Concentrator	$6,450,929		$0	$0	$0	$210,558	$273,338	$266,569	$268,444	$264,529	$259,584	$260,046	$263,750	$265,477	$267,062	$264,328	$265,959	$266,023	$264,814	$261,023	$259,510	$254,914	$251,746	$255,743	$256,237	$263,021	$253,010	$251,010	$251,276	$232,959
	Heap Leach - Gold Mineralized Material	$1,218,002		$33,024	$54,880	$54,878	$54,875	$54,874	$54,874	$54,875	$54,878	$54,878	$54,873	$54,874	$54,878	$54,880	$54,877	$54,873	$54,876	$54,873	$54,870	$52,791	$52,791	$52,793	$52,793	$40,925	$0	$0	$0	$0	$0
	General Administration	$509,923		$8,327	$8,637	$10,129	$19,899	$20,958	$21,130	$21,281	$21,550	$21,330	$21,472	$21,677	$21,813	$21,637	$21,440	$21,611	$21,761	$21,520	$21,167	$21,514	$19,854	$19,062	$18,767	$17,408	$16,387	$16,362	$10,849	$11,295	$11,087
	Treatment & Refining Charges																														
	Copper Concentrates																														
	Treatment Charges	$514,307		$0	$0	$0	$25,767	$36,245	$32,250	$28,198	$23,782	$22,512	$22,428	$22,632	$20,849	$25,124	$24,278	$22,753	$21,340	$20,655	$19,396	$17,524	$17,018	$18,520	$17,084	$12,991	$12,867	$11,713	$12,015	$14,024	$12,341
	Copper Refining Charges	$317,479		$0	$0	$0	$15,906	$22,374	$19,908	$17,406	$14,681	$13,897	$13,845	$13,971	$12,870	$15,509	$14,987	$14,045	$13,173	$12,750	$11,973	$10,818	$10,505	$11,433	$10,546	$8,019	$7,943	$7,230	$7,417	$8,657	$7,618
	Gold Refining Charges	$35,198		$0	$0	$0	$2,086	$2,332	$2,275	$2,010	$1,721	$1,676	$1,655	$1,633	$1,581	$1,525	$1,542	$1,564	$1,491	$1,066	$1,194	$1,176	$1,109	$1,165	$1,144	$832	$796	$935	$914	$891	$886
	Silver Refining Charges	$41,931		$0	$0	$0	$2,055	$2,015	$1,923	$2,055	$1,933	$1,717	$2,065	$1,683	$1,731	$2,151	$2,101	$1,800	$1,616	$1,754	$1,612	$1,572	$2,048	$1,470	$1,609	$1,165	$1,155	$1,185	$1,176	$1,200	$1,139
	Transportation	$1,076,610		$0	$0	$0	$53,938	$75,873	$67,510	$59,028	$49,784	$47,125	$46,950	$47,376	$43,643	$52,594	$50,822	$47,629	$44,671	$43,238	$40,603	$36,684	$35,625	$38,769	$35,762	$27,193	$26,935	$24,518	$25,151	$29,357	$25,833
	Road Maintenance	$78,481		$0	$0	$0	$3,139	$3,139	$3,139	$3,139	$3,139	$3,139	$3,139	$3,139	$3,139	$3,139	$3,139	$3,139	$3,139	$3,139	$3,139	$3,139	$3,139	$3,139	$3,139	$3,139	$3,139	$3,139	$3,139	$3,139	$3,139
	Molybdenum Concentrate																														
	Treatment Charges	$0		$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
	Transportation	$0		$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
	Gold Dore																														
	Gold Refining Charges	$1,546		$44	$116	$108	$77	$68	$56	$44	$49	$57	$61	$64	$54	$53	$52	$43	$47	$57	$68	$68	$70	$96	$108	$83	$0	$0	$0	$0	$0
	Silver Refining Charges	$997		$18	$61	$65	$43	$34	$49	$31	$37	$50	$54	$50	$40	$36	$34	$37	$39	$49	$34	$34	$34	$57	$65	$46	$0	$0	$0	$0	$0
	Transportation	$1,404		$29	$91	$94	$63	$51	$64	$42	$50	$66	$71	$68	$54	$50	$48	$49	$52	$64	$51	$51	$52	$82	$94	$67	$0	$0	$0	$0	$0
	Copper Precipitate																														
	Treatment Charges	$1,539		$26	$90	$90	$79	$43	$62	$90	$83	$42	$27	$45	$53	$92	$97	$98	$75	$69	$43	$43	$57	$78	$90	$66	$0	$0	$0	$0	$0
	Refining Charges	$2,036		$34	$119	$119	$105	$57	$81	$119	$109	$55	$35	$60	$71	$122	$128	$130	$100	$91	$57	$57	$75	$103	$119	$87	$0	$0	$0	$0	$0
	Transportation	$3,524		$59	$207	$207	$181	$99	$141	$207	$189	$95	$61	$103	$122	$211	$222	$225	$173	$158	$99	$99	$130	$179	$207	$151	$0	$0	$0	$0	$0
	Total Cash Operating Cost	$14,381,646		$74,922	$129,312	$177,844	$567,135	$672,136	$664,146	$649,409	$636,515	$627,004	$631,996	$641,795	$638,468	$648,915	$636,724	$638,698	$640,840	$627,265	$602,902	$604,561	$545,445	$526,778	$515,603	$435,178	$381,759	$365,539	$358,809	$387,306	$354,643
	Royalty	$883,147		$1,910	$4,995	$4,828	$43,890	$56,082	$54,061	$45,213	$39,388	$39,564	$41,134	$40,863	$39,614	$39,155	$39,689	$38,854	$37,642	$32,857	$30,920	$29,639	$30,568	$34,283	$35,019	$24,598	$18,947	$19,425	$20,604	$20,702	$18,702
	Property Tax	$2,800		$100	$100	$100	$100	$100	$100	$100	$100	$100	$100	$100	$100	$100	$100	$100	$100	$100	$100	$100	$100	$100	$100	$100	$100	$100	$100	$100	$100
	Carbon Tax	$1,692,210		$1,261	$3,911	$8,119	$13,728	$20,737	$29,273	$39,101	$50,329	$62,959	$76,989	$76,989	$76,989	$76,989	$76,989	$76,989	$76,989	$76,989	$76,989	$76,989	$76,989	$76,989	$76,989	$76,989	$76,989	$76,989	$76,989	$76,989	$76,989
	Salvage Value	-$31,491		$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	-$31,491
	Reclamation & Closure	$200,000		$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$50,000
	Total Cash Production Cost	$17,128,312		$78,193	$138,318	$190,891	$624,853	$749,055	$747,580	$733,823	$726,332	$729,627	$750,219	$759,747	$755,171	$765,159	$753,503	$754,641	$755,571	$737,211	$710,910	$711,289	$653,102	$638,150	$627,711	$536,865	$477,795	$462,053	$456,502	$485,097	$468,944
	Operating Income	$19,113,689	$0	-$8,517	$49,625	-$14,642	$1,157,560	$1,566,219	$1,467,574	$1,100,436	$848,626	$847,975	$912,662	$934,776	$881,086	$869,169	$900,219	$858,112	$800,246	$620,005	$563,788	$502,685	$606,501	$784,300	$821,200	$473,985	$305,709	$338,769	$395,603	$375,826	$304,193
	Yukon Mining Royalty	$2,130,991		$0	$5,604	$0	$86,108	$136,744	$124,970	$80,907	$50,254	$51,623	$79,860	$120,922	$114,683	$113,053	$116,170	$111,497	$104,229	$82,456	$75,213	$68,066	$81,314	$103,840	$108,647	$65,688	$44,842	$48,880	$56,208	$53,979	$45,233
	Net Income before Depreciation	$16,982,698	$0	-$8,517	$44,021	-$14,642	$1,071,453	$1,429,475	$1,342,604	$1,019,530	$798,371	$796,352	$832,801	$813,853	$766,403	$756,116	$784,049	$746,615	$696,017	$537,548	$488,576	$434,618	$525,187	$680,460	$712,552	$408,297	$260,867	$289,889	$339,395	$321,847	$258,960
	Capital Cost Depreciation	$1,557,068		-$8,517	$44,021	-$14,642	$807,547	$541,301	$187,356	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
	Additional Deduction (Class 41A Assets)	$1,693,985		$0	$0	$0	$257,439	$874,477	$562,069	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
	Sustaining Capital Depreciation	$718,733				$0	$6,466	$13,696	$20,285	$16,998	$27,794	$22,095	$32,536	$28,640	$26,762	$29,883	$34,713	$34,975	$29,391	$33,115	$25,426	$21,393	$20,185	$18,892	$10,627	$18,070	$17,329	$17,873	$29,324	$34,383	$147,884
	Total Depreciation	$3,969,786		-$8,517	$44,021	-$14,642	$1,071,453	$1,429,475	$769,710	$16,998	$27,794	$22,095	$32,536	$28,640	$26,762	$29,883	$34,713	$34,975	$29,391	$33,115	$25,426	$21,393	$20,185	$18,892	$10,627	$18,070	$17,329	$17,873	$29,324	$34,383	$147,884
	Net Income After Depreciation	$13,012,912		$0	$0	$0	$0	$0	$572,895	$1,002,532	$770,578	$774,257	$800,266	$785,214	$739,641	$726,233	$749,336	$711,640	$666,626	$504,434	$463,150	$413,226	$505,002	$661,568	$701,925	$390,226	$243,538	$272,016	$310,071	$287,464	$111,076
	Tax Loss Carry Forward Applied	-$30,000		$0	$0	$0	$0	$0	-$30,000	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
	Net Income After Tax Loss Carry Forward	$12,982,912		$0	$0	$0	$0	$0	$542,895	$1,002,532	$770,578	$774,257	$800,266	$785,214	$739,641	$726,233	$749,336	$711,640	$666,626	$504,434	$463,150	$413,226	$505,002	$661,568	$701,925	$390,226	$243,538	$272,016	$310,071	$287,464	$111,076
	Taxable Income	$12,982,912		$0	$0	$0	$0	$0	$542,895	$1,002,532	$770,578	$774,257	$800,266	$785,214	$739,641	$726,233	$749,336	$711,640	$666,626	$504,434	$463,150	$413,226	$505,002	$661,568	$701,925	$390,226	$243,538	$272,016	$310,071	$287,464	$111,076
	Taxes at 30%	$3,939,874		$0	$0	$0	$0	$0	$162,868	$300,759	$231,173	$232,277	$240,080	$235,564	$221,892	$217,870	$224,801	$213,492	$199,988	$151,330	$138,945	$123,968	$151,500	$198,470	$210,578	$117,068	$73,061	$81,605	$93,021	$86,239	$33,323
	Net Income After Taxes	$9,073,039		$0	$0	$0	$0	$0	$410,026	$701,772	$539,404	$541,980	$560,186	$549,650	$517,749	$508,363	$524,535	$498,148	$466,638	$353,103	$324,205	$289,258	$353,501	$463,098	$491,348	$273,158	$170,476	$190,411	$217,050	$201,225	$77,753
	Cash Flow																														
	Net Income before Depreciation plus Tax Loss Carry Forward	$16,982,698		-$8,517	$44,021	-$14,642	$1,071,453	$1,429,475	$1,342,604	$1,019,530	$798,371	$796,352	$832,801	$813,853	$766,403	$756,116	$784,049	$746,615	$696,017	$537,548	$488,576	$434,618	$525,187	$680,460	$712,552	$408,297	$260,867	$289,889	$339,395	$321,847	$258,960
	Working Capital																														
	Account Receivable	$0		-$11,454	-$19,441	$1,922	-$264,027	-$87,594	$16,458	$62,613	$42,625	-$435	-$14,018	-$5,201	$9,578	$317	-$3,188	$6,735	$9,359	$32,647	$13,564	$9,982	-$7,501	-$26,769	-$4,350	$72,010	$37,372	-$2,847	-$8,430	-$1,450	$14,431
	Accounts Payable	$0		$6,158	$4,470	$3,989	$31,997	$8,630	-$657	-$1,211	-$1,060	-$782	$410	$805	-$273	$859	-$1,002	$162	$176	-$1,116	-$2,002	$136	-$4,859	-$1,534	-$919	-$6,610	-$4,391	-$1,333	-$553	$2,342	-$2,685
	Inventory - Parts, Supplies	$0		-$500	$0	-$5,800	$0	$0	$0	$500	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$5,800	$0	$0	$0
	Total Working Capital	$0		-$5,796	-$14,971	$111	-$232,030	-$78,963	$15,801	$61,902	$41,565	-$1,216	-$13,608	-$4,396	$9,304	$1,176	-$4,190	$6,897	$9,535	$31,531	$11,562	$10,119	-$12,360	-$28,304	-$5,268	$65,400	$32,981	$1,620	-$8,983	$893	$11,746
	Capital Expenditures																														
	Initial Capital																														
	Mine	$620,232	$0	$90,416	$145,608	$229,448	$154,760	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
	Concentrator	$2,578,536	$87,405	$758,625	$1,040,491	$652,340	$39,676	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
	Heap Leach	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0

	Total	-4	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12
Owners Cost	$52,285	$2,091	$14,117	$21,960	$11,503	$2,614	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Sustaining Capital																	
Mine	$169,175		$0	$0	$0	$0	$9,521	$11,274	$0	$32,068	$0	$22,548	$1,593	$12,315	$12,848	$17,211	$9,521
Tailings Management Facility	$292,868		$0	$0	$0	$20,865	$20,865	$0	$2,138	$23,112	$0	$0	$0	$0	$19,533	$0	$0
Process Plant (includes Heap Leach)	$256,690		$0	$0	$0	$5,000	$5,000	$28,775	$5,000	$5,000	$5,000	$41,310	$5,000	$5,000	$5,000	$45,171	$5,000
Total Capital Expenditures	$3,969,786	$89,496	$863,158	$1,208,058	$893,290	$222,916	$35,386	$40,049	$7,138	$60,181	$5,000	$63,857	$6,593	$17,315	$37,381	$62,382	$14,521
Cash Flow before Taxes	$13,012,912	-$89,496	-$877,470	-$1,179,007	-$907,821	$616,507	$1,315,125	$1,318,357	$1,074,293	$779,756	$790,136	$755,336	$802,864	$758,392	$719,911	$717,477	$738,991
Cumulative Cash Flow before Taxes		-$89,496	-$966,967	-$2,145,974	-$3,053,795	-$2,437,288	-$1,122,163	$196,194	$1,270,487	$2,050,243	$2,840,379	$3,595,714	$4,398,578	$5,156,971	$5,876,882	$6,594,359	$7,333,350
						$1	$1	$1	$0	$0	$0	$0	$0	$0	$0	$0	$0
Taxes																	
Income Taxes	$3,939,874	$0	$0	$0	$0	$0	$0	$162,868	$300,759	$231,173	$232,277	$240,080	$235,564	$221,892	$217,870	$224,801	$213,492
Cash Flow after Taxes	$9,073,039	-$89,496	-$877,470	-$1,179,007	-$907,821	$616,507	$1,315,125	$1,155,488	$773,534	$548,582	$557,859	$515,256	$567,300	$536,500	$502,041	$492,676	$525,499
Cumulative Cash Flow after Taxes		-$89,496	-$966,967	-$2,145,974	-$3,053,795	-$2,437,288	-$1,122,163	$33,325	$806,859	$1,355,441	$1,913,300	$2,428,556	$2,995,856	$3,532,356	$4,034,398	$4,527,074	$5,052,573
						$1	$1	$1	$0	$0	$0	$0	$0	$0	$0	$0	$0

	13	14	15	16	17	18	19	20	21	22	23	24	25
Owners Cost	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Sustaining Capital													
Mine	$29,735	$0	$0	$0	$3,527	$0	$0	$7,015	$0	$0	$0	$0	$0
Tailings Management Facility	$0	$0	$20,290	$0	$0	$0	$0	$20,290	$0	$12,673	$66,803	$43,149	$43,149
Process Plant (includes Heap Leach)	$5,000	$20,717	$5,000	$5,000	$5,000	$5,000	$5,000	$5,000	$20,717	$5,000	$5,000	$5,000	$5,000
Total Capital Expenditures	$34,735	$20,717	$25,290	$5,000	$8,527	$5,000	$5,000	$32,305	$20,717	$17,673	$71,803	$48,149	$48,149
Cash Flow before Taxes	$670,817	$548,362	$474,848	$439,737	$504,301	$647,156	$702,284	$441,391	$273,131	$273,835	$258,609	$274,591	$222,557
Cumulative Cash Flow before Taxes	$8,004,168	$8,552,530	$9,027,378	$9,467,115	$9,971,415	$10,618,571	$11,320,855	$11,762,247	$12,035,378	$12,309,213	$12,567,822	$12,842,413	$13,064,970
	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Taxes													
Income Taxes	$199,988	$151,330	$138,945	$123,968	$151,500	$198,470	$210,578	$117,068	$73,061	$81,605	$93,021	$86,239	$33,323
Cash Flow after Taxes	$470,830	$397,032	$335,903	$315,769	$352,800	$448,686	$491,707	$324,324	$200,070	$192,231	$165,588	$188,352	$189,235
Cumulative Cash Flow after Taxes	$5,523,403	$5,920,435	$6,256,338	$6,572,107	$6,924,907	$7,373,593	$7,865,299	$8,189,623	$8,389,692	$8,581,923	$8,747,510	$8,935,862	$9,125,097
	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0

Economic Indicators before Taxes		
NPV @ 0%	0%	$13,012,912
NPV @ 5%	5%	$5,790,431
NPV @ 8%	8%	$3,616,543
NPV @ 10%	10%	$2,632,163
IRR		23.3%
Payback		2.9
Economic Indicators after Taxes		
NPV @ 0%	0%	$9,073,039
NPV @ 5%	5%	$3,895,851
NPV @ 8%	8%	$2,331,765
NPV @ 10%	10%	$1,622,605
IRR		19.5%
Payback	Years	3.0
NSR $/Sulphide Mineralized Material		$ 28.14
Sulphide Cash Cost		$ 11.67
Total Cash Production Cost		$ 14.11
Heap Leach Cash Cost		$ 6.03

23 ADJACENT PROPERTIES

Several quartz mineral claim blocks and placer claims registered to other owners are staked adjacent to and in the general vicinity of CMC's claim block. Some of the placer claims on Canadian and Britannia Creeks overlap the Casino claims in the area of the pit. These placer claims along the upper part of Canadian creek are located within the projected pit shell and are worked by their owners on a seasonal basis with small heavy equipment. The northwestern boundary of the Casino property adjoins the Coffee Creek project of Newmont Mining. The property hosts a structurally controlled gold deposit in metamorphic rocks of the Yukon Tanana terrane and granitoids of mid Cretaceous age. The mineralization is associated with quartz- carbonate and illite alteration and is best described as an orogenic deposit. The project is at a pre-feasibility stage of development.

The northeastern boundary of the Casino property abuts the "Betty and Hayes" property held by White Gold Corp. This property abuts the northern boundary of the narrow eastern extension of the Casino property. At this time, the property has undergone fairly early stages of exploration for similar orogenic-style gold mineralization to that within the Coffee Creek property.

Part of the eastern extension is also directly surrounded by the Idaho claim block held by Atac Resources Ltd.

24 OTHER RELEVANT DATA AND INFORMATION

24.1 PROPOSED PHASE II EXPANSION

The information presented in Sections 16 through 22 of this study is based on measured and indicated mineral resources with a mine plan constrained by the capacity of the selected site and design of the Tailings Management Facility (TMF), i.e. Phase I. This section presents the results for a larger pit design which includes inferred mineral resources and an expanded tailings capacity based on building an additional embankment south of the Phase I embankment; i.e. the Phase II plan.

The economic assessment of the proposed Phase II expansion is preliminary in nature and includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the preliminary economic assessment will be realized.

The plant production rate is a nominal 120,000 tonnes per day and the peak mining rate is 100 Mt/y or about 275,000 t/d for this plan, the same as the Phase I plan. As with the Phase I plan, all of the waste rock will be co-disposed in the TMF facility.

The concentrator and associated facilities are as per Phase I and continue to process mill feed at a nominal 120,000 t/d. Table 24-1 shows a summary of the capital costs for both cases.

Table 24-1: Phase I vs. Phase II Capital Costs

Cost Item	Phase I Total ($M)	Phase I + II Total ($M)
Process Plant and Infrastructure		
Project Directs including freight	1,777	1,777
Project Indirects	390	390
Contingency	412	412
Subtotal	2,579	2,579
Mine		
Mine Equipment	409	419
Mine Preproduction	211	206
Subtotal	620	625
Owner's Costs	52	52
Total Initial Capital	3,251	3,256
Sustaining Capital	719	1,808
Total Life of Mine Capital Costs	3,970	5,064

Table 24-2 shows the economics of Phase I and Phase II.

Table 24-2: Phase I vs. Phase I + II Economic Indicators

Economic Indicators before Taxes	Phase I	Phase I + II
NPV @ 0% ($M)	13,012	17,175
NPV @ 5% ($M)	5,790	6,237
NPV @ 8% ($M)	3,617	3,700
NPV @ 10% ($M)	2,632	2,640
IRR	23.3%	23.1%
Payback (years)	2.9	2.8
Economic Indicators after Taxes		
NPV @ 0% ($M)	9,073	11,968
NPV @ 5% ($M)	3,896	4,198
NPV @ 8% ($M)	2,332	2,384
NPV @ 10% ($M)	1,623	1,624
IRR (%)	19.5%	19.3%
Payback (years)	3.0	3.0

24.2 MINING METHODS

24.2.1 Slope Angles and Final Pit

The slope angles used for this expansion are the same as the Phase I design and are based on the Knight Piésold Ltd. (KP) report “Open Pit Geotechnical Design”, dated October 12, 2012. For future work it will be necessary to review these angles with the larger pit design.

Figure 24-1 shows the final pit design for this study. Compared with the Phase I pit design this design has significant expansion mostly to the south and north. The pit design is based on the base case commodity prices of US$3.00 per pound copper, US$1500 per ounce gold, US$24 per ounce silver, and US$9.00 per pound moly. In Phase I, only measured and indicated mineral resources were considered to define the final pit design. The Phase II mine plan includes 321 Mt of inferred mineral resource in the expanded mining phases that is included in the mine production schedule for this plan.

24.2.2 Economic Parameters

The commodity prices, unit costs, metal recoveries and NSR calculations for this mine plan are the same as was reported in Section 16.3 and Table 16-2.

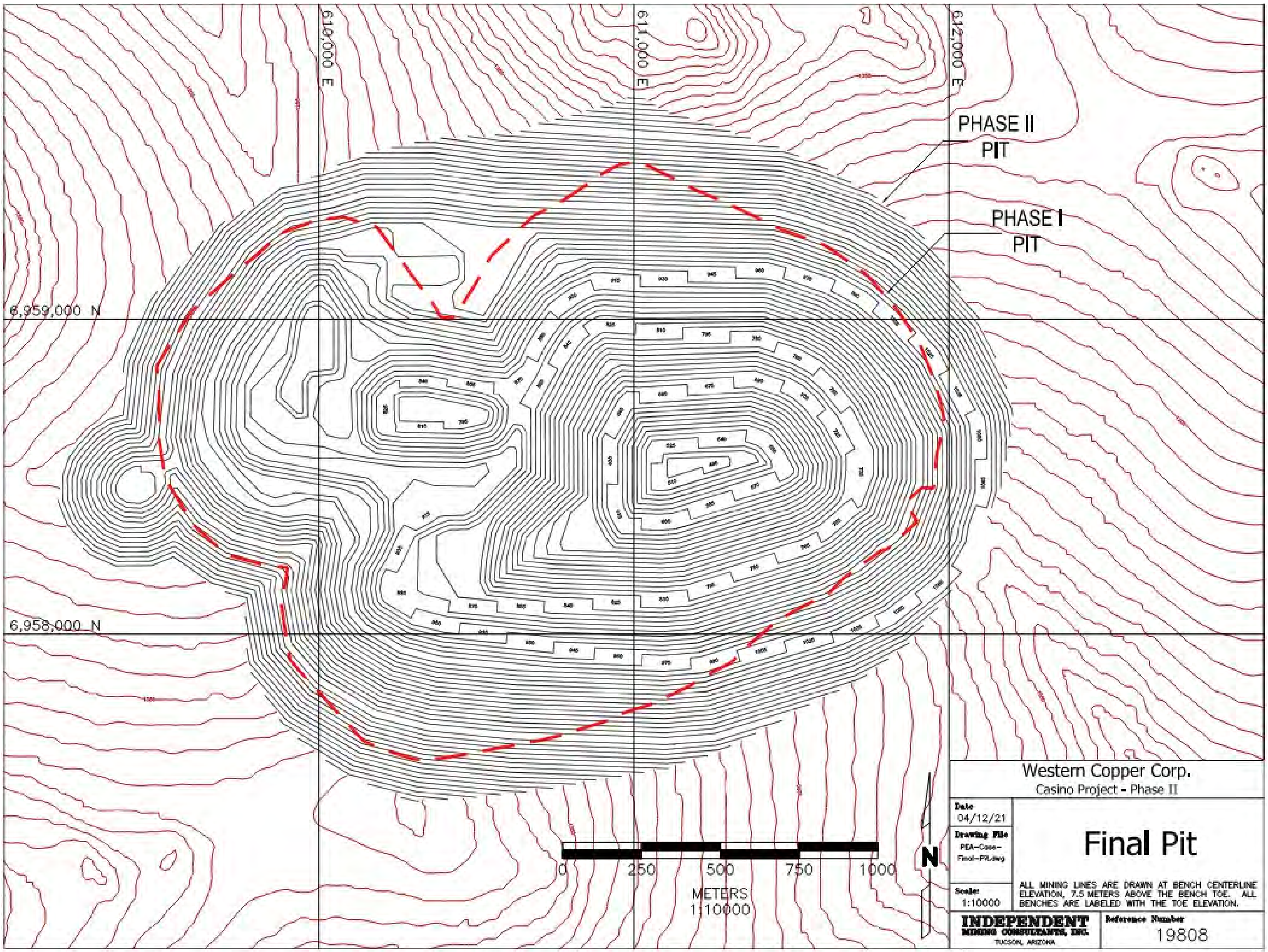


Figure 24-1: Final Phase II Pit Design (IMC, 2021)

24.2.3 Mine Production Schedule

The mine production schedule is based on seven mining phases, versus five phases for the Phase I plan. The designs utilized 35 m wide roads at a maximum grade of 10%. The road width will accommodate trucks up to the 330 tonne class such as the Caterpillar 797.

As with the Phase I production schedule, the Bond Work Index and mill throughput rate has been assigned to model blocks based on rock type, mineralized material type, and alteration. The mill schedule has been developed based on plant hours, so throughput varies by year. It was reported to IMC that all necessary efficiency factors were incorporated in throughput rates so IMC has based the schedule on 8,760 plant hours per year.

The top section of Table 24-3 (spread over two pages to be readable) shows the proposed plant production schedule. Total mill resource is 2.13 billion tonnes at 0.163% copper, 0.192 g/t gold, 0.0182% moly, and 1.51 g/t silver. The average NSR value of this is $21.16 per tonne. For Years 2 through 46, full production years, plant throughput varies from a low of 44.0 million tonnes in Year 35 to a high of 46.9 million tonnes in Years 2 and 13. The table also shows the average Bond Work Index (14.4) and throughput rate (0.19101 h/kt). The throughput units are somewhat unconventional, but a parameter that could be weight averaged by tonnes was required. Copper recovery was also incorporated into the model on a block-by-block basis, based on total and soluble copper grades for supergene materials, and a fixed copper recovery of 92.2% for hypogene. The average recovered copper grade is 0.143%, indicating an average copper recovery of 87.9%.

The table also shows the various components of the mill resource. Direct mill feed is resource that is scheduled to be processed the same year it is mined. This amounts to 1.64 billion tonnes at 0.179% total copper, 0.201 g/t gold, 0.0204% moly and 1.61 g/t silver. This is about 77% of total plant material. The average NSR value of this resource is $23.06 per tonne. Note that Year 1 plant production is 34.5 million tonnes, about 75% of nominal capacity and is made up of resource mined during preproduction and Year 1.

The SOX resource in the mining phase 1 starter pit is stockpiled and processed during Years 4 through 13 at the rate of 3.6 Mt/y. This is done to maintain the ratio of weak soluble copper to total copper at relatively low levels by year. This resource amounts to 35.8 million tonnes at 0.251% total copper, 0.086% weak soluble copper, 0.491 g/t gold, 0.0252% moly, and 2.36 g/t silver. The NSR value for this resource is $35.08/t.

The operating schedule also results in a significant amount of low-grade mineralized material that is stockpiled and processed at the end of the mine life during Years 37 through 47. This amounts to 449.4 million tonnes at 0.098% total copper, 0.135 g/t gold, 0.0092% moly, and 1.07 g/t silver. The low grade represents material between an NSR cutoff of $10 per tonne and the operating cutoff for direct feed mineralized material for the year. This cutoff grade is somewhat arbitrary; it was chosen to produce the approximate 450 million tonnes shown. It is understood that 10 years of low-grade processing is unusually long, but this material is included to demonstrate resource that is available for the plant feed stream if the plant capacity is increased.

The reclaim schedules for the SOX and low grade are on a last-in-first-out (LIFO) basis, consistent with stockpiles build up in lifts and reclaimed in reverse order.

Based on the schedule, the commercial life of the project is 47 years after a 3-year preproduction period.

Table 24-5 and Table 24-7 show the mine production schedule. The upper section of the table shows the direct feed mill resource. 4.67 million tonnes of this is mined during preproduction and stockpiled near the crusher to be part of the Year 1 mill feed for quarters 1 and 2.

As previously discussed, an NSR value was calculated for each block to classify blocks into resource and waste. For the mine production schedule, the direct feed varies by year to balance mine and plant capacities. After the first three quarters of Year 1, Year 1 Q4 and after, the schedule starts at relatively high cutoff grades and declines to the internal cutoff grade of $5.70 per tonne for the last couple years of the mine life.

The second section of Table 24-5 shows the SOX mill resource from mining phase 1 that is stockpiled and processed during Year 4 through 13. This is at an NSR cutoff of $20 per tonne to limit the total amount to about 36 million tonnes. Lower grade SOX goes to the low-grade stockpile. The third section of Table 24-5 and Table 24-6 show low grade resource produced by year. This is material with an NSR cutoff between $10 per tonne and the operating cutoff for the year.

The bottom of Table 24-5 and Table 24-6 also show the schedule of resource mined from the leach cap zone by year. It is assumed that this is processed by crushing, and heap leaching. Leach resource is defined as leach cap material with an NSR above $5.46/t with leach economics and total copper less than 0.1%. This amounts to 239.8 million tonnes at 0.245 g/t gold, 1.87 g/t silver, and 0.034% total copper. Of this amount, 24.9 million tonnes is inferred mineral resource, about 10% of the total.

The bottom of Table 24-5 and Table 24-6 summarise tonnages. It can be seen that life of mine total material from the pit is 3.47 billion tonnes. Preproduction is 75.0 million tonnes staged over three years. Year 1 total material is scheduled at about 95 million tonnes after which the peak material movement of 100 Mt/y is maintained over much of the mine life. Total waste is 1.10 billion tonnes so the waste ratio is about 0.47 if mill resource (including SOX), low grade, and leach resource are all counted as resource for the calculation.

The upper section of Table 24-7 and Table 24-8 show a proposed stacking schedule for the leach resource. This is based on the ability to crush and stack 9,125 kt/y (25,000 t/d for 365 days/year or 36,500 t/d for 250 days per year).

The rest of the table shows the details of the schedule. The second section of Table 24-7 and Table 24-8 show mine production of leach resource. The third and fourth sections show up to 9,125 kt of mined resource as direct crusher feed and the excess going to a stockpile. Both are shown at average grades for the year. The bottom of the tables show the stockpile reclaim on a last-in-first-out basis (LIFO). The stockpile gets to a maximum size of 77.9 million tonnes in Year 11 with this scenario.

As previously mentioned, only measured and indicated mineral resource was used to develop the final pit geometry. However, inferred mineral resource in the pit is included in the production schedule. Of the 2.13 billion tonne mill feed, 321.3 Mt is inferred mineral resource, about 15%. This is compared to only 22 million tonnes of inferred mineral resource in the Phase I mining plan that is counted as waste for that plan.

The economic assessment of the proposed Phase II expansion is preliminary in nature and includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the preliminary economic assessment will be realized.

The mine production schedule includes allowances for mining dilution and loss. IMC believes that reasonable amounts of dilution and loss were incorporated into the block model used for this Technical Report. Compositing assays into composites and estimating blocks with multiple composites introduces some smoothing of model grades that are analogous to dilution and resource loss effects.

Table 24-3: Proposed Plant Production Schedule

	(Units)	Y1 Q1	Y1 Q2	Y1 Q3	Y1 Q4	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	Year 17	Year 18	Year 19	Year 20	Year 21	Year 22	Year 23
Proposed Mill Schedule:																											
Ktonnes	(kt)	5,698	8,757	9,711	10,334	46,882	46,472	45,612	45,478	45,507	45,626	46,361	46,597	44,545	45,785	46,256	46,939	46,084	45,437	45,838	45,881	45,348	45,162	45,410	45,846	46,145	46,579
NSR Value	(C$/t)	36.91	41.39	40.05	43.11	39.30	38.20	32.46	27.63	27.40	28.65	28.60	27.76	27.69	28.94	26.97	26.52	21.49	22.63	19.54	19.21	19.75	20.65	21.08	19.75	17.77	20.96
Total Copper	(%)	0.281	0.340	0.365	0.398	0.338	0.286	0.280	0.234	0.209	0.207	0.203	0.187	0.229	0.253	0.209	0.192	0.189	0.196	0.150	0.145	0.139	0.148	0.146	0.128	0.111	0.149
Weak Soluble Copper	(%)	0.050	0.055	0.063	0.058	0.028	0.006	0.033	0.024	0.009	0.008	0.006	0.006	0.026	0.025	0.013	0.011	0.019	0.011	0.001	0.000	0.000	0.000	0.001	0.001	0.001	0.000
% Cuw/Cut	(%)	17.8%	16.2%	17.1%	14.6%	8.4%	2.1%	11.8%	10.2%	4.2%	3.8%	2.7%	3.5%	11.2%	10.0%	6.4%	5.6%	10.1%	5.5%	1.0%	0.0%	0.0%	0.2%	1.0%	0.4%	0.6%	0.0%
Gold	(g/t)	0.454	0.469	0.399	0.404	0.357	0.358	0.316	0.273	0.269	0.265	0.256	0.247	0.245	0.235	0.236	0.233	0.163	0.185	0.186	0.172	0.169	0.172	0.175	0.169	0.172	0.192
Molybdenum	(%)	0.0159	0.0177	0.0237	0.0288	0.0297	0.0323	0.0193	0.0156	0.0194	0.0274	0.0304	0.0343	0.0282	0.0273	0.0272	0.0301	0.0204	0.0172	0.0123	0.0144	0.0193	0.0224	0.0253	0.0258	0.0198	0.0188
Silver	(g/t)	2.70	2.90	2.37	2.03	1.90	1.95	1.95	1.93	1.80	2.17	1.72	1.70	2.05	1.94	1.74	1.62	1.69	1.60	1.45	1.72	2.12	1.55	1.45	1.32	1.23	1.58
Recovered Copper	(%)	0.216	0.267	0.284	0.319	0.288	0.259	0.231	0.195	0.184	0.183	0.183	0.167	0.190	0.213	0.181	0.168	0.158	0.172	0.137	0.134	0.128	0.136	0.134	0.118	0.102	0.137
Bond Work Index	(Kwh/t)	14.2	14.0	13.8	13.8	14.1	14.2	14.4	14.5	14.5	14.4	14.3	14.1	14.8	14.4	14.2	14.0	14.3	14.5	14.4	14.4	14.5	14.6	14.5	14.4	14.2	14.1
Hours Per Ktonne	(hr/kt)	0.18871	0.18611	0.18259	0.18364	0.18685	0.18849	0.19207	0.19262	0.19250	0.19201	0.18894	0.18800	0.19666	0.19133	0.18938	0.18664	0.19008	0.19280	0.19111	0.19093	0.19317	0.19398	0.19291	0.19108	0.18984	0.18809
Mill Hours	(hours)	1,075	1,630	1,773	1,898	8,760	8,760	8,761	8,760	8,760	8,761	8,759	8,760	8,760	8,760	8,760	8,761	8,760	8,760	8,760	8,760	8,760	8,761	8,760	8,760	8,760	8,761
Copper Recovery	(%)	77.0%	78.7%	77.8%	80.2%	85.4%	90.6%	82.3%	83.4%	88.4%	88.7%	89.8%	89.5%	82.9%	84.1%	87.0%	87.5%	83.5%	87.8%	91.2%	92.4%	92.1%	91.9%	91.6%	92.5%	91.6%	91.9%
Gold Recovery	(%)	68.7%	68.7%	68.7%	68.6%	67.6%	66.4%	67.7%	67.2%	66.4%	66.4%	66.5%	66.7%	68.5%	68.1%	67.4%	66.7%	67.1%	67.4%	66.3%	66.0%	66.0%	66.1%	66.2%	66.1%	66.2%	66.0%
Moly Recovery	(%)	55.1%	54.6%	54.4%	55.2%	63.0%	74.3%	66.1%	67.6%	75.1%	76.3%	76.5%	75.6%	56.6%	62.3%	69.3%	75.5%	73.7%	67.5%	75.4%	78.6%	78.6%	78.5%	78.4%	78.2%	78.5%	78.6%
Silver Recovery	(%)	59.0%	59.0%	59.1%	58.6%	54.9%	51.5%	56.0%	53.5%	51.0%	50.7%	51.1%	51.9%	58.3%	56.9%	54.2%	52.1%	53.2%	54.3%	51.0%	50.0%	50.0%	50.6%	50.6%	50.3%	50.6%	50.1%
Direct Feed:																											
Ktonnes	(kt)	5,698	8,757	9,711	10,334	46,882	46,472	42,012	41,878	41,907	42,026	42,761	42,997	40,945	42,185	42,656	43,498	46,084	45,437	45,838	45,881	45,348	45,162	45,410	45,846	46,145	46,579
NSR Value	(C$/t)	36.91	41.39	40.05	43.11	39.30	38.20	32.13	26.65	26.81	28.25	28.29	27.35	26.97	28.35	26.22	25.84	21.49	22.63	19.54	19.21	19.75	20.65	21.08	19.75	17.77	20.96
Total Copper	(%)	0.281	0.340	0.365	0.398	0.338	0.286	0.269	0.216	0.202	0.203	0.204	0.187	0.233	0.259	0.211	0.192	0.189	0.196	0.150	0.145	0.139	0.148	0.146	0.128	0.111	0.149
Weak Soluble Copper	(%)	0.050	0.055	0.063	0.058	0.028	0.006	0.025	0.013	0.000	0.000	0.000	0.002	0.022	0.022	0.009	0.007	0.019	0.011	0.001	0.000	0.000	0.000	0.001	0.001	0.001	0.000
Gold	(g/t)	0.454	0.469	0.399	0.404	0.357	0.358	0.313	0.265	0.254	0.248	0.238	0.226	0.215	0.205	0.207	0.207	0.163	0.185	0.186	0.172	0.169	0.172	0.175	0.169	0.172	0.192
Molybdenum	(%)	0.0159	0.0177	0.0237	0.0288	0.0297	0.0323	0.0190	0.0144	0.0185	0.0271	0.0305	0.0349	0.0288	0.0279	0.0277	0.0309	0.0204	0.0172	0.0123	0.0144	0.0193	0.0224	0.0253	0.0258	0.0198	0.0188
Silver	(g/t)	2.70	2.90	2.37	2.03	1.90	1.95	1.98	1.95	1.78	2.18	1.68	1.63	1.98	1.86	1.64	1.53	1.69	1.60	1.45	1.72	2.12	1.55	1.45	1.32	1.23	1.58
Recovered Copper	(%)	0.216	0.267	0.284	0.319	0.288	0.259	0.227	0.188	0.186	0.187	0.188	0.172	0.197	0.221	0.187	0.171	0.158	0.172	0.137	0.134	0.128	0.136	0.134	0.118	0.102	0.137
Bond Work Index	(Kwh/t)	14.2	14.0	13.8	13.8	14.1	14.2	14.5	14.6	14.6	14.5	14.3	14.1	14.9	14.5	14.2	14.0	14.3	14.5	14.4	14.4	14.5	14.6	14.5	14.4	14.2	14.1
Hours Per Ktonne	(hr/kt)	0.18871	0.18611	0.18259	0.18364	0.18685	0.18849	0.19285	0.19354	0.19340	0.19287	0.18951	0.18848	0.19790	0.19208	0.18996	0.18698	0.19008	0.19280	0.19111	0.19093	0.19317	0.19398	0.19291	0.19108	0.18984	0.18809
Mill Hours	(hours)	1,075	1,630	1,773	1,898	8,760	8,760	8,102	8,105	8,105	8,106	8,104	8,104	8,103	8,103	8,103	8,133	8,760	8,760	8,760	8,760	8,760	8,761	8,760	8,760	8,760	8,761
SOX Stockpile (LIFO):																											
Ktonnes	(kt)							3,600	3,600	3,600	3,600	3,600	3,600	3,600	3,600	3,600	3,441										
NSR Value	(C$/t)							36.31	39.07	34.32	33.36	32.28	32.68	35.86	35.86	35.86	35.16										
Total Copper	(%)							0.410	0.445	0.283	0.250	0.195	0.178	0.185	0.185	0.185	0.191										
Weak Soluble Copper	(%)							0.123	0.146	0.108	0.100	0.070	0.061	0.063	0.063	0.063	0.061										
Gold	(g/t)							0.344	0.373	0.449	0.464	0.477	0.501	0.581	0.581	0.581	0.559										
Molybdenum	(%)							0.0225	0.0294	0.0301	0.0303	0.0290	0.0271	0.0212	0.0212	0.0212	0.0202										
Silver	(g/t)							1.68	1.74	1.97	2.02	2.29	2.49	2.93	2.93	2.93	2.64										
Recovered Copper	(%)							0.270	0.281	0.165	0.141	0.117	0.110	0.115	0.115	0.115	0.122										
Bond Work Index	(Kwh/t)							13.8	13.7	13.7	13.7	13.7	13.7	13.7	13.7	13.7	13.7										
Hours Per Ktonne	(hr/kt)							0.18287	0.18198	0.18202	0.18203	0.18212	0.18222	0.18252	0.18252	0.18252	0.18234										
Mill Hours	(hours)							658	655	655	655	656	656	657	657	657	627										
Low Grade Stockpile (LIFO):																											
Ktonnes	(kt)																										
NSR Value	(C$/t)																										
Total Copper	(%)																										
Weak Soluble Copper	(%)																										
Gold	(g/t)																										
Molybdenum	(%)																										
Silver	(g/t)																										
Recovered Copper	(%)																										
Bond Work Index	(Kwh/t)																										
Hours Per Ktonne	(hr/kt)																										
Mill Hours	(hours)																										

Table 24-4: Proposed Plant Production Schedule (Continued)

	(Units)	Year 24	Year 25	Year 26	Year 27	Year 28	Year 29	Year 30	Year 31	Year 32	Year 33	Year 34	Year 35	Year 36	Year 37	Year 38	Year 39	Year 40	Year 41	Year 42	Year 43	Year 44	Year 45	Year 46	Year 47	TOTAL
Proposed Mill Schedule:																										
Ktonnes	(kt)	46,628	46,707	46,827	46,725	45,789	46,168	45,983	45,578	45,096	45,034	44,415	43,984	45,092	46,496	45,601	46,362	46,092	45,816	46,170	45,527	46,023	45,434	45,511	30,442	2,127,790
NSR Value	(C$/t)	19.04	21.63	21.93	24.75	19.42	14.92	14.48	14.55	15.66	16.50	16.38	15.38	18.73	26.79	11.04	10.98	11.87	12.46	13.14	13.18	13.97	15.11	15.13	15.46	21.16
Total Copper	(%)	0.131	0.160	0.182	0.224	0.191	0.115	0.107	0.107	0.112	0.116	0.119	0.112	0.133	0.209	0.087	0.062	0.065	0.099	0.111	0.088	0.087	0.120	0.142	0.136	0.163
Weak Soluble Copper	(%)	0.000	0.000	0.000	0.005	0.024	0.002	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.005	0.003	0.001	0.004	0.012	0.008	0.001	0.007	0.022	0.026	0.008
% Cuw/Cut	(%)	0.0%	0.0%	0.0%	2.2%	12.3%	1.5%	0.2%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	5.9%	5.1%	1.1%	3.8%	10.6%	8.9%	1.3%	6.0%	15.8%	19.2%	5.1%
Gold	(g/t)	0.199	0.198	0.175	0.184	0.157	0.133	0.130	0.129	0.148	0.160	0.139	0.117	0.143	0.210	0.096	0.152	0.140	0.110	0.125	0.159	0.148	0.138	0.135	0.163	0.192
Molybdenum	(%)	0.0132	0.0174	0.0167	0.0172	0.0111	0.0110	0.0115	0.0114	0.0115	0.0127	0.0161	0.0190	0.0245	0.0260	0.0097	0.0046	0.0097	0.0093	0.0080	0.0080	0.0122	0.0114	0.0104	0.0085	0.0182
Silver	(g/t)	1.38	1.49	1.35	1.48	1.44	1.41	1.27	1.60	1.72	1.47	1.31	0.89	1.02	1.69	0.94	0.96	1.10	1.02	1.12	1.04	1.13	1.17	1.13	1.13	1.51
Recovered Copper	(%)	0.121	0.148	0.168	0.201	0.156	0.105	0.098	0.098	0.103	0.107	0.110	0.103	0.123	0.193	0.075	0.054	0.060	0.088	0.092	0.075	0.079	0.105	0.112	0.103	0.143
Bond Work Index	(Kwh/t)	14.1	14.1	14.1	14.1	14.4	14.3	14.3	14.5	14.6	14.6	14.8	15.0	14.6	14.1	14.5	14.2	14.3	14.4	14.3	14.5	14.3	14.5	14.5	14.4	14.4
Hours Per Ktonne	(hr/kt)	0.18787	0.18755	0.18708	0.18748	0.19131	0.18974	0.19051	0.19220	0.19426	0.19452	0.19723	0.19917	0.19427	0.18840	0.19209	0.18895	0.19006	0.19120	0.18973	0.19242	0.19034	0.19282	0.19249	0.19185	0.19101
Mill Hours	(hours)	8,760	8,760	8,760	8,760	8,760	8,760	8,760	8,760	8,760	8,760	8,760	8,760	8,760	8,760	8,760	8,760	8,760	8,760	8,760	8,760	8,760	8,760	8,760	5,840	406,422
Copper Recovery	(%)	92.4%	92.5%	92.3%	90.0%	81.5%	90.9%	91.5%	91.6%	92.0%	92.2%	92.4%	92.0%	92.5%	92.2%	87.0%	87.7%	91.5%	89.3%	83.1%	85.1%	91.4%	87.1%	78.8%	75.6%	87.9%
Gold Recovery	(%)	66.0%	66.0%	66.0%	66.1%	67.2%	66.2%	66.1%	66.0%	66.0%	66.0%	66.0%	66.0%	66.0%	66.0%	66.4%	66.4%	66.6%	66.5%	67.2%	67.3%	66.8%	66.7%	67.9%	68.4%	66.8%
Moly Recovery	(%)	78.6%	78.6%	78.6%	78.0%	68.5%	76.4%	78.5%	78.6%	78.6%	78.6%	78.6%	78.6%	78.6%	78.6%	76.6%	74.5%	77.5%	75.0%	69.2%	64.9%	77.4%	74.1%	60.5%	57.8%	73.6%
Silver Recovery	(%)	50.0%	50.0%	50.0%	50.3%	54.0%	50.5%	50.3%	50.0%	50.0%	50.0%	50.0%	50.0%	50.0%	50.0%	51.5%	51.9%	52.3%	51.6%	54.1%	54.5%	52.3%	52.0%	55.8%	57.9%	52.2%
Direct Feed:																										
Ktonnes	(kt)	46,628	46,707	46,827	46,725	45,789	46,168	45,983	45,578	45,096	45,034	44,415	43,984	45,092	40,058											1,642,533
NSR Value	(C$/t)	19.04	21.63	21.93	24.75	19.42	14.92	14.48	14.55	15.66	16.50	16.38	15.38	18.73	29.35											23.06
Total Copper	(%)	0.131	0.160	0.182	0.224	0.191	0.115	0.107	0.107	0.112	0.116	0.119	0.112	0.133	0.230											0.179
Weak Soluble Copper	(%)	0.000	0.000	0.000	0.005	0.024	0.002	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000											0.007
Gold	(g/t)	0.199	0.198	0.175	0.184	0.157	0.133	0.130	0.129	0.148	0.160	0.139	0.117	0.143	0.230											0.201
Molybdenum	(%)	0.0132	0.0174	0.0167	0.0172	0.0111	0.0110	0.0115	0.0114	0.0115	0.0127	0.0161	0.0190	0.0245	0.0282											0.0204
Silver	(g/t)	1.38	1.49	1.35	1.48	1.44	1.41	1.27	1.60	1.72	1.47	1.31	0.89	1.02	1.84											1.61
Recovered Copper	(%)	0.121	0.148	0.168	0.201	0.156	0.105	0.098	0.098	0.103	0.107	0.110	0.103	0.123	0.212											0.160
Bond Work Index	(Kwh/t)	14.1	14.1	14.1	14.1	14.4	14.3	14.3	14.5	14.6	14.6	14.8	15.0	14.6	14.1											14.4
Hours Per Ktonne	(hr/kt)	0.18787	0.18755	0.18708	0.18748	0.19131	0.18974	0.19051	0.19220	0.19426	0.19452	0.19723	0.19917	0.19427	0.18795											0.19115
Mill Hours	(hours)	8,760	8,760	8,760	8,760	8,760	8,760	8,760	8,760	8,760	8,760	8,760	8,760	8,760	7,529											313,975
SOX Stockpile (LIFO):																										
Ktonnes	(kt)																									35,841
NSR Value	(C$/t)																									35.08
Total Copper	(%)																									0.251
Weak Soluble Copper	(%)																									0.086
Gold	(g/t)																									0.491
Molybdenum	(%)																									0.0252
Silver	(g/t)																									2.36
Recovered Copper	(%)																									0.155
Bond Work Index	(Kwh/t)																									13.7
Hours Per Ktonne	(hr/kt)																									0.18231
Mill Hours	(hours)																									6,534
Low Grade Stockpile (LIFO):																										
Ktonnes	(kt)														6,438	45,601	46,362	46,092	45,816	46,170	45,527	46,023	45,434	45,511	30,442	449,416
NSR Value	(C$/t)														10.87	11.04	10.98	11.87	12.46	13.14	13.18	13.97	15.11	15.13	15.46	13.12
Total Copper	(%)														0.081	0.087	0.062	0.065	0.099	0.111	0.088	0.087	0.120	0.142	0.136	0.098
Weak Soluble Copper	(%)														0.000	0.005	0.003	0.001	0.004	0.012	0.008	0.001	0.007	0.022	0.026	0.008
Gold	(g/t)														0.083	0.096	0.152	0.140	0.110	0.125	0.159	0.148	0.138	0.135	0.163	0.135
Molybdenum	(%)														0.0125	0.0097	0.0046	0.0097	0.0093	0.0080	0.0080	0.0122	0.0114	0.0104	0.0085	0.0092
Silver	(g/t)														0.78	0.94	0.96	1.10	1.02	1.12	1.04	1.13	1.17	1.13	1.13	1.07
Recovered Copper	(%)														0.075	0.075	0.054	0.060	0.088	0.092	0.075	0.079	0.105	0.112	0.103	0.084
Bond Work Index	(Kwh/t)														14.4	14.5	14.2	14.3	14.4	14.3	14.5	14.3	14.5	14.5	14.4	14.4
Hours Per Ktonne	(hr/kt)														0.19121	0.19209	0.18895	0.19006	0.19120	0.18973	0.19242	0.19034	0.19282	0.19249	0.19185	0.19117
Mill Hours	(hours)														1,231	8,760	8,760	8,760	8,760	8,760	8,760	8,760	8,760	8,760	5,840	85,913

Table 24-5: Mine Production Schedule

	(Units)	-3	-2	-1	Y1 Q1	Y1 Q2	Y1 Q3	Y1 Q4	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	Year 17	Year 18	Year 19	Year 20	Year 21
Direct Feed:																												
NSR Cutoff Grade	(C$)	5.70	5.70	5.70	5.70	5.70	5.70	25.00	18.25	22.00	19.75	19.25	20.00	19.00	18.00	17.75	16.50	18.00	16.25	17.25	15.75	15.00	14.75	14.75	13.75	14.00	13.00	11.50
Ktonnes	(kt)		220	4,448	4,030	5,757	9,711	10,334	46,882	46,472	42,012	41,878	41,907	42,026	42,761	42,997	40,945	42,185	42,656	43,498	46,084	45,437	45,838	45,881	45,348	45,162	45,410	45,846
NSR Value	(C$/t)		30.72	34.75	37.72	45.07	40.05	43.11	39.30	38.20	32.13	26.65	26.81	28.25	28.29	27.35	26.97	28.35	26.22	25.84	21.49	22.63	19.54	19.21	19.75	20.65	21.08	19.75
Total Copper	(%)		0.242	0.247	0.295	0.389	0.365	0.398	0.338	0.286	0.269	0.216	0.202	0.203	0.204	0.187	0.233	0.259	0.211	0.192	0.189	0.196	0.150	0.145	0.139	0.148	0.146	0.128
Weak Soluble Copper	(%)		0.026	0.032	0.058	0.067	0.063	0.058	0.028	0.006	0.025	0.013	0.000	0.000	0.000	0.002	0.022	0.022	0.009	0.007	0.019	0.011	0.001	0.000	0.000	0.000	0.001	0.001
Gold	(g/t)		0.332	0.437	0.460	0.491	0.399	0.404	0.357	0.358	0.313	0.265	0.254	0.248	0.238	0.226	0.215	0.205	0.207	0.207	0.163	0.185	0.186	0.172	0.169	0.172	0.175	0.169
Molybdenum	(%)		0.0177	0.0135	0.0169	0.0198	0.0237	0.0288	0.0297	0.0323	0.0190	0.0144	0.0185	0.0271	0.0305	0.0349	0.0288	0.0279	0.0277	0.0309	0.0204	0.0172	0.0123	0.0144	0.0193	0.0224	0.0253	0.0258
Silver	(g/t)		1.92	2.30	2.87	3.22	2.37	2.03	1.90	1.95	1.98	1.95	1.78	2.18	1.68	1.63	1.98	1.86	1.64	1.53	1.69	1.60	1.45	1.72	2.12	1.55	1.45	1.32
Recovered Copper	(%)		0.203	0.202	0.222	0.302	0.284	0.319	0.288	0.259	0.227	0.188	0.186	0.187	0.188	0.172	0.197	0.221	0.187	0.171	0.158	0.172	0.137	0.134	0.128	0.136	0.134	0.118
Bond Work Index	(Kwh/t)		13.7	14.4	14.1	13.8	13.8	13.8	14.1	14.2	14.5	14.6	14.6	14.5	14.3	14.1	14.9	14.5	14.2	14.0	14.3	14.5	14.4	14.4	14.5	14.6	14.5	14.4
Hours Per Ktonne	(hr/kt)		0.18207	0.19152	0.18741	0.18375	0.18259	0.18364	0.18685	0.18849	0.19285	0.19354	0.19340	0.19287	0.18951	0.18848	0.19790	0.19208	0.18996	0.18698	0.19008	0.19280	0.19111	0.19093	0.19317	0.19398	0.19291	0.19108
Mill Hours	(hours)		40	852	755	1,058	1,773	1,898	8,760	8,760	8,102	8,105	8,105	8,106	8,104	8,104	8,103	8,103	8,103	8,133	8,760	8,760	8,760	8,760	8,760	8,761	8,760	8,760
SOX Stockpile:																												
NSR Cutoff Grade	(C$)		20.00	20.00	20.00	20.00	20.00	20.00	20.00																			
Ktonnes	(kt)		1,894	13,042	5,595	7,504	5,527	1,501	778																			
NSR Value	(C$/t)		34.58	35.86	31.92	33.36	39.07	38.39	27.60																			
Total Copper	(%)		0.195	0.185	0.176	0.250	0.445	0.467	0.242																			
Weak Soluble Copper	(%)		0.060	0.063	0.060	0.100	0.146	0.136	0.058																			
Gold	(g/t)		0.541	0.581	0.482	0.464	0.373	0.330	0.322																			
Molybdenum	(%)		0.0193	0.0212	0.0285	0.0303	0.0294	0.0221	0.0116																			
Silver	(g/t)		2.4000	2.93	2.38	2.02	1.74	1.52	1.90																			
Recovered Copper	(%)		0.127	0.115	0.109	0.141	0.281	0.311	0.173																			
Bond Work Index	(Kwh/t)		13.7	13.7	13.7	13.7	13.7	13.8	13.9																			
Hours Per Ktonne	(hr/kt)		0.18219	0.18252	0.18215	0.18203	0.18198	0.18277	0.18459																			
Mill Hours	(hours)		345	2,380	1,019	1,366	1,006	274	144																			
Low Grade:																												
NSR Cutoff Grade	(C$)			10.00	10.00	10.00	10.00	10.00	10.00	10.00	10.00	10.00	10.00	10.00	10.00	10.00	10.00	10.00	10.00	10.00	10.00	10.00	10.00	10.00	10.00	10.00	10.00	10.00
Ktonnes	(kt)			159	409	1,153	512	2,692	7,745	9,014	23,504	44,355	29,965	20,967	15,642	20,156	23,301	16,091	12,386	11,920	17,290	20,466	19,888	17,064	13,372	19,292	16,671	8,986
NSR Value	(C$/t)			17.60	17.52	16.90	16.07	19.41	13.86	15.46	15.29	15.05	15.16	14.61	13.48	13.56	12.74	13.69	12.95	13.51	13.02	12.42	12.52	12.43	11.67	11.86	11.19	10.68
Total Copper	(%)			0.156	0.122	0.130	0.184	0.187	0.129	0.113	0.150	0.138	0.114	0.098	0.082	0.075	0.084	0.103	0.092	0.116	0.122	0.111	0.095	0.079	0.063	0.061	0.058	0.053
Weak Soluble Copper	(%)			0.047	0.061	0.053	0.079	0.029	0.025	0.012	0.032	0.018	0.003	0.000	0.001	0.003	0.008	0.010	0.003	0.014	0.017	0.008	0.001	0.000	0.001	0.001	0.001	0.000
Gold	(g/t)			0.214	0.257	0.206	0.158	0.177	0.152	0.184	0.138	0.134	0.140	0.131	0.152	0.172	0.165	0.141	0.134	0.127	0.116	0.108	0.112	0.113	0.144	0.150	0.150	0.155
Molybdenum	(%)			0.0067	0.0195	0.0233	0.0183	0.0105	0.0032	0.0068	0.0114	0.0099	0.0118	0.0151	0.0106	0.0096	0.0060	0.0105	0.0091	0.0090	0.0064	0.0067	0.0100	0.0139	0.0086	0.0091	0.0062	0.0045
Silver	(g/t)			1.72	1.42	1.39	1.17	1.36	0.91	1.15	1.17	1.12	1.18	1.28	1.03	1.01	1.04	1.07	1.08	1.09	1.20	0.94	1.03	1.21	1.05	1.07	1.18	0.92
Recovered Copper	(%)			0.103	0.057	0.073	0.099	0.147	0.097	0.094	0.111	0.112	0.102	0.091	0.075	0.067	0.071	0.087	0.082	0.094	0.099	0.096	0.087	0.073	0.058	0.056	0.053	0.048
Bond Work Index	(Kwh/t)			13.7	13.7	13.7	13.8	14.3	14.7	14.5	14.4	14.5	14.5	14.3	14.3	14.4	14.6	14.3	14.1	14.1	14.5	14.4	14.3	14.3	14.3	14.3	14.2	14.1
Hours Per Ktonne	(hr/kt)			0.18207	0.18207	0.18207	0.18319	0.19043	0.19555	0.19251	0.19077	0.19331	0.19277	0.19007	0.19011	0.19114	0.19415	0.19025	0.18728	0.18792	0.19244	0.19178	0.19101	0.19024	0.19028	0.18988	0.18979	0.18722
Mill Hours	(hours)			29	74	210	94	513	1,515	1,735	4,484	8,574	5,776	3,985	2,974	3,853	4,524	3,061	2,320	2,240	3,327	3,925	3,799	3,246	2,544	3,663	3,164	1,682
Leach:																												
NSR Cutoff Grade	(C$)	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46
Ktonnes	(kt)	5,487	13,831	23,056	6,638	4,940	3,079	5,406	34,620	19,995	14,807	817	4,606	10,501	13,992	14,045	6,431	19,763	5,625	8,124	2,900	82	88	1,104	1,380	1,432	623	1,705
NSR Value	(C$/t)	11.49	18.68	17.52	15.70	11.88	10.37	11.77	10.86	9.29	7.46	7.01	8.04	9.40	10.57	8.96	8.62	8.92	6.80	6.96	7.34	7.76	5.53	6.79	8.22	8.22	9.25	7.11
Gold	(g/t)	0.273	0.436	0.406	0.366	0.274	0.232	0.277	0.256	0.211	0.164	0.149	0.185	0.217	0.244	0.206	0.195	0.199	0.144	0.149	0.149	0.166	0.126	0.161	0.190	0.179	0.200	0.159
Silver	(g/t)	1.30	2.68	2.85	2.54	1.93	1.63	1.34	1.470	2.010	1.310	0.840	2.250	2.430	2.290	1.710	1.320	1.570	1.750	1.720	2.010	1.540	1.680	1.120	1.130	2.060	0.690	0.360
Total Copper	(%)	0.021	0.044	0.044	0.035	0.036	0.052	0.027	0.021	0.027	0.042	0.065	0.008	0.009	0.020	0.021	0.038	0.044	0.047	0.044	0.067	0.055	0.005	0.007	0.027	0.042	0.082	0.049
Weak Soluble Copper	(%)	0.004	0.009	0.010	0.011	0.011	0.012	0.007	0.005	0.007	0.014	0.016	0.001	0.003	0.011	0.006	0.007	0.009	0.013	0.011	0.013	0.009	0.000	0.001	0.004	0.008	0.014	0.008
Molybdenum	(%)	0.0063	0.0115	0.0160	0.0227	0.0230	0.0210	0.0036	0.0060	0.0065	0.0065	0.0026	0.0045	0.0198	0.0228	0.0221	0.0228	0.0256	0.0066	0.0072	0.0049	0.0040	0.0007	0.0018	0.0007	0.0012	0.0006	0.0002
Tonnage Summary:																												
Direct Feed	(kt)	0	220	4,448	4,030	5,757	9,711	10,334	46,882	46,472	42,012	41,878	41,907	42,026	42,761	42,997	40,945	42,185	42,656	43,498	46,084	45,437	45,838	45,881	45,348	45,162	45,410	45,846
SOX Stockpile	(kt)	0	1,894	13,042	5,595	7,504	5,527	1,501	778	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Low Grade	(kt)	0	0	159	409	1,153	512	2,692	7,745	9,014	23,504	44,355	29,965	20,967	15,642	20,156	23,301	16,091	12,386	11,920	17,290	20,466	19,888	17,064	13,372	19,292	16,671	8,986
Leach	(kt)	5,487	13,831	23,056	6,638	4,940	3,079	5,406	34,620	19,995	14,807	817	4,606	10,501	13,992	14,045	6,431	19,763	5,625	8,124	2,900	82	88	1,104	1,380	1,432	623	1,705
Total Material	(kt)	7,424	19,758	47,818	20,000	25,000	25,000	25,000	99,896	100,000	99,461	99,488	100,000	100,000	100,000	100,000	100,000	100,000	100,000	100,000	100,000	100,000	100,000	100,000	99,999	100,000	99,999	100,000
Waste Material	(kt)	1,937	3,813	7,113	3,328	5,646	6,171	5,067	9,871	24,519	19,138	12,438	23,522	26,506	27,605	22,802	29,323	21,961	39,333	36,458	33,726	34,015	34,186	35,951	39,899	34,114	37,295	43,463
Stockpile Rehandle	(kt)				1,668	3,000					3,600	3,600	3,600	3,600	3,600	3,600	3,600	3,600	3,600	3,441								

Table 24-6: Mine Production Schedule (Continued)

	(Units)	Year 22	Year 23	Year 24	Year 25	Year 26	Year 27	Year 28	Year 29	Year 30	Year 31	Year 32	Year 33	Year 34	Year 35	Year 36	Year 37	Year 38	Year 39	Year 40	Year 41	Year 42	Year 43	Year 44	Year 45	Year 46	Year 47	TOTAL
Direct Feed:																												
NSR Cutoff Grade	(C$)	11.25	10.50	9.25	10.50	13.00	13.00	12.50	10.75	11.00	10.25	11.00	12.50	12.00	11.00	5.70	5.70											
Ktonnes	(kt)	46,145	46,579	46,628	46,707	46,827	46,725	45,789	46,168	45,983	45,578	45,096	45,034	44,415	43,984	45,092	40,058											1,642,533
NSR Value	(C$/t)	17.77	20.96	19.04	21.63	21.93	24.75	19.42	14.92	14.48	14.55	15.66	16.50	16.38	15.38	18.73	29.35											23.06
Total Copper	(%)	0.111	0.149	0.131	0.160	0.182	0.224	0.191	0.115	0.107	0.107	0.112	0.116	0.119	0.112	0.133	0.230											0.179
Weak Soluble Copper	(%)	0.001	0.000	0.000	0.000	0.000	0.005	0.024	0.002	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000											0.007
Gold	(g/t)	0.172	0.192	0.199	0.198	0.175	0.184	0.157	0.133	0.130	0.129	0.148	0.160	0.139	0.117	0.143	0.230											0.201
Molybdenum	(%)	0.0198	0.0188	0.0132	0.0174	0.0167	0.0172	0.0111	0.0110	0.0115	0.0114	0.0115	0.0127	0.0161	0.0190	0.0245	0.0282											0.0204
Silver	(g/t)	1.23	1.58	1.38	1.49	1.35	1.48	1.44	1.41	1.27	1.60	1.72	1.47	1.31	0.89	1.02	1.84											1.61
Recovered Copper	(%)	0.102	0.137	0.121	0.148	0.168	0.201	0.156	0.105	0.098	0.098	0.103	0.107	0.110	0.103	0.123	0.212											0.160
Bond Work Index	(Kwh/t)	14.2	14.1	14.1	14.1	14.1	14.1	14.4	14.3	14.3	14.5	14.6	14.6	14.8	15.0	14.6	14.1											14.4
Hours Per Ktonne	(hr/kt)	0.18984	0.18809	0.18787	0.18755	0.18708	0.18748	0.19131	0.18974	0.19051	0.19220	0.19426	0.19452	0.19723	0.19917	0.19427	0.18795											0.19115
Mill Hours	(hours)	8,760	8,761	8,760	8,760	8,760	8,760	8,760	8,760	8,760	8,760	8,760	8,760	8,760	8,760	8,760	7,529											313,975
SOX Stockpile:																												
NSR Cutoff Grade	(C$)																											
Ktonnes	(kt)																											35,841
NSR Value	(C$/t)																											35.08
Total Copper	(%)																											0.251
Weak Soluble Copper	(%)																											0.086
Gold	(g/t)																											0.491
Molybdenum	(%)																											0.0252
Silver	(g/t)																											2.36
Recovered Copper	(%)																											0.155
Bond Work Index	(Kwh/t)																											13.7
Hours Per Ktonne	(hr/kt)																											0.18231
Mill Hours	(hours)																											6,534
Low Grade:																												
NSR Cutoff Grade	(C$)	10.00	10.00	10.00	10.00	10.00	10.00	10.00	10.00	10.00	10.00	10.00	10.00	10.00	10.00													
Ktonnes	(kt)	8,977	2,872	0	929	7,562	6,641	6,540	4,184	6,486	1,336	4,120	12,273	11,699	2,797													449,416
NSR Value	(C$/t)	10.59	10.24	0.00	10.22	11.49	11.56	11.45	10.36	10.47	10.11	10.50	11.48	11.10	10.57													13.12
Total Copper	(%)	0.044	0.048	0.000	0.056	0.080	0.115	0.112	0.084	0.081	0.070	0.079	0.081	0.078	0.085													0.098
Weak Soluble Copper	(%)	0.001	0.000	0.000	0.000	0.001	0.029	0.024	0.001	0.000	0.000	0.000	0.000	0.000	0.000													0.008
Gold	(g/t)	0.179	0.163	0.000	0.147	0.134	0.116	0.107	0.090	0.094	0.096	0.097	0.094	0.088	0.076													0.135
Molybdenum	(%)	0.0026	0.0022	0.0000	0.0027	0.0047	0.0059	0.0059	0.0068	0.0076	0.0086	0.0070	0.0130	0.0135	0.0111													0.0092
Silver	(g/t)	0.65	0.92	0.00	0.88	0.95	1.05	1.25	0.90	0.79	0.91	0.97	0.92	0.83	0.71													1.07
Recovered Copper	(%)	0.040	0.044	0.000	0.052	0.072	0.079	0.083	0.076	0.075	0.064	0.073	0.075	0.072	0.078													0.084
Bond Work Index	(Kwh/t)	14.3	14.1	0.0	14.1	14.1	14.5	14.6	14.5	14.4	14.4	14.5	14.5	14.3	14.5													14.4
Hours Per Ktonne	(hr/kt)	0.18891	0.18746	0.00000	0.18741	0.18792	0.19355	0.19379	0.19171	0.19206	0.19114	0.19244	0.19235	0.19005	0.19273													0.19117
Mill Hours	(hours)	1,696	538	0	174	1,421	1,285	1,267	802	1,246	255	793	2,361	2,223	539													85,913
Leach:																												
NSR Cutoff Grade	(C$)	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46																		
Ktonnes	(kt)	356	0	475	60	2,683	9,944	1,116	11	34																		239,756
NSR Value	(C$/t)	7.53	0.00	5.61	5.62	7.01	7.01	6.73	9.88	9.88																		10.72
Gold	(g/t)	0.182	0.000	0.129	0.129	0.147	0.147	0.134	0.231	0.231																		0.245
Silver	(g/t)	0.960	0.000	0.620	0.570	1.480	1.300	1.600	2.650	2.650																		1.87
Total Copper	(%)	0.001	0.000	0.023	0.023	0.056	0.061	0.075	0.000	0.000																		0.034
Weak Soluble Copper	(%)	0.000	0.000	0.000	0.000	0.019	0.018	0.026	0.000	0.000																		0.009
Molybdenum	(%)	0.0002	0.0000	0.0009	0.0008	0.0046	0.0067	0.0087	0.0002	0.0002																		0.0128
Tonnage Summary:																												
Direct Feed	(kt)	46,145	46,579	46,628	46,707	46,827	46,725	45,789	46,168	45,983	45,578	45,096	45,034	44,415	43,984	45,092	40,058											1,642,533
SOX Stockpile	(kt)	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0											35,841
Low Grade	(kt)	8,977	2,872	0	929	7,562	6,641	6,540	4,184	6,486	1,336	4,120	12,273	11,699	2,797	0	0											449,416
Leach	(kt)	356	0	475	60	2,683	9,944	1,116	11	34	0	0	0	0	0	0	0											239,756
Total Material	(kt)	100,000	100,000	100,000	100,000	100,000	99,053	93,082	92,977	85,928	80,988	72,288	70,123	65,410	57,089	46,444	40,131											3,472,356
Waste Material	(kt)	44,522	50,549	52,897	52,304	42,928	35,743	39,637	42,614	33,425	34,074	23,072	12,816	9,296	10,308	1,352	73											1,104,810
Stockpile Rehandle	(kt)																6,438	45,601	46,362	46,092	45,816	46,170	45,527	46,023	45,434	45,511	30,442	489,925

Table 24-7: Production Schedule for Leach Material

	(Units)	-3	-2	-1	Y1 Q1	Y1 Q2	Y1 Q3	Y1 Q4	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12
Leach Pad Stacking Schedule:																			
NSR Cutoff Grade	(C$)	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46
Ktonnes	(kt)	5,487	9,125	9,125	2,281	2,281	2,281	2,282	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125
NSR Value	(C$/t)	11.49	18.68	17.52	15.70	11.88	10.37	11.77	10.86	9.29	7.46	7.95	8.66	9.40	10.57	8.96	8.72	8.92	7.61
Gold	(g/t)	0.273	0.436	0.406	0.366	0.274	0.232	0.277	0.256	0.211	0.164	0.176	0.198	0.217	0.244	0.206	0.198	0.199	0.165
Silver	(g/t)	1.30	2.68	2.85	2.54	1.93	1.63	1.34	1.470	2.010	1.310	1.469	2.131	2.430	2.290	1.710	1.435	1.570	1.681
Total Copper	(%)	0.021	0.044	0.044	0.035	0.036	0.052	0.027	0.021	0.027	0.042	0.040	0.017	0.009	0.020	0.021	0.033	0.044	0.046
Weak Soluble Copper	(%)	0.004	0.009	0.010	0.011	0.011	0.012	0.007	0.005	0.007	0.014	0.012	0.004	0.003	0.011	0.006	0.007	0.009	0.011
Molybdenum	(%)	0.0063	0.0115	0.0160	0.0227	0.0230	0.0210	0.0036	0.0060	0.0065	0.0065	0.0062	0.0055	0.0198	0.0228	0.0221	0.0226	0.0256	0.0139
As Produced From Mine:																			
NSR Cutoff Grade	(C$)	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46
Ktonnes	(kt)	5,487	13,831	23,056	6,638	4,940	3,079	5,406	34,620	19,995	14,807	817	4,606	10,501	13,992	14,045	6,431	19,763	5,625
NSR Value	(C$/t)	11.49	18.68	17.52	15.70	11.88	10.37	11.77	10.86	9.29	7.46	7.01	8.04	9.40	10.57	8.96	8.62	8.92	6.80
Gold	(g/t)	0.273	0.436	0.406	0.366	0.274	0.232	0.277	0.256	0.211	0.164	0.149	0.185	0.217	0.244	0.206	0.195	0.199	0.144
Silver	(g/t)	1.30	2.68	2.85	2.54	1.93	1.63	1.34	1.470	2.010	1.310	0.840	2.250	2.430	2.290	1.710	1.320	1.570	1.750
Total Copper	(%)	0.021	0.044	0.044	0.035	0.036	0.052	0.027	0.021	0.027	0.042	0.065	0.008	0.009	0.020	0.021	0.038	0.044	0.047
Weak Soluble Copper	(%)	0.004	0.009	0.010	0.011	0.011	0.012	0.007	0.005	0.007	0.014	0.016	0.001	0.003	0.011	0.006	0.007	0.009	0.013
Molybdenum	(%)	0.0063	0.0115	0.0160	0.0227	0.0230	0.0210	0.0036	0.0060	0.0065	0.0065	0.0026	0.0045	0.0198	0.0228	0.0221	0.0228	0.0256	0.0066
Direct to Crusher:		100.00%	65.97%	39.58%	34.36%	46.17%	74.08%	42.21%	26.36%	45.64%	61.63%	100.00%	100.00%	86.90%	65.22%	64.97%	100.00%	46.17%	100.00%
Ktonnes	(kt)	5,487	9,125	9,125	2,281	2,281	2,281	2,282	9,125	9,125	9,125	817	4,606	9,125	9,125	9,125	6,431	9,125	5,625
NSR Value	(C$/t)	11.49	18.68	17.52	15.70	11.88	10.37	11.77	10.86	9.29	7.46	7.01	8.04	9.40	10.57	8.96	8.62	8.92	6.80
Gold	(g/t)	0.273	0.436	0.406	0.366	0.274	0.232	0.277	0.256	0.211	0.164	0.149	0.185	0.217	0.244	0.206	0.195	0.199	0.144
Silver	(g/t)	1.30	2.68	2.85	2.54	1.93	1.63	1.34	1.470	2.010	1.310	0.840	2.250	2.430	2.290	1.710	1.320	1.570	1.750
Total Copper	(%)	0.021	0.044	0.044	0.035	0.036	0.052	0.027	0.021	0.027	0.042	0.065	0.008	0.009	0.020	0.021	0.038	0.044	0.047
Weak Soluble Copper	(%)	0.004	0.009	0.010	0.011	0.011	0.012	0.007	0.005	0.007	0.014	0.016	0.001	0.003	0.011	0.006	0.007	0.009	0.013
Molybdenum	(%)	0.0063	0.0115	0.0160	0.0227	0.0230	0.0210	0.0036	0.0060	0.0065	0.0065	0.0026	0.0045	0.0198	0.0228	0.0221	0.0228	0.0256	0.0066
To Leach Stockpile:		0.00%	34.03%	60.42%	65.64%	53.83%	25.92%	57.79%	73.64%	54.36%	38.37%	0.00%	0.00%	13.10%	34.78%	35.03%	0.00%	53.83%	0.00%
Ktonnes	(kt)	0	4,706	13,931	4,357	2,659	798	3,124	25,495	10,870	5,682	0	0	1,376	4,867	4,920	0	10,638	0
NSR Value	(C$/t)	11.49	18.68	17.52	15.70	11.88	10.37	11.77	10.86	9.29	7.46	7.01	8.04	9.40	10.57	8.96	8.62	8.92	6.80
Gold	(g/t)	0.273	0.436	0.406	0.366	0.274	0.232	0.277	0.256	0.211	0.164	0.149	0.185	0.217	0.244	0.206	0.195	0.199	0.144
Silver	(g/t)	1.30	2.68	2.85	2.54	1.93	1.63	1.34	1.470	2.010	1.310	0.840	2.250	2.430	2.290	1.710	1.320	1.570	1.750
Total Copper	(%)	0.021	0.044	0.044	0.035	0.036	0.052	0.027	0.021	0.027	0.042	0.065	0.008	0.009	0.020	0.021	0.038	0.044	0.047
Weak Soluble Copper	(%)	0.004	0.009	0.010	0.011	0.011	0.012	0.007	0.005	0.007	0.014	0.016	0.001	0.003	0.011	0.006	0.007	0.009	0.013
Molybdenum	(%)	0.0063	0.0115	0.0160	0.0227	0.0230	0.0210	0.0036	0.0060	0.0065	0.0065	0.0026	0.0045	0.0198	0.0228	0.0221	0.0228	0.0256	0.0066
Stockpile Reclaim:																			
Ktonnes	(kt)											8,308	4,519				2,694		3,500
NSR Value	(C$/t)											8.04	9.29				8.96		8.92
Gold	(g/t)											0.179	0.211				0.206		0.199
Silver	(g/t)											1.531	2.010				1.710		1.570
Total Copper	(%)											0.037	0.027				0.021		0.044
Weak Soluble Copper	(%)											0.012	0.007				0.006		0.009
Molybdenum	(%)											0.0065	0.0065				0.0221		0.0256
Stockpile Balance	(kt)	0	4,706	18,637	22,994	25,653	26,451	29,575	55,070	65,940	71,622	63,314	58,795	60,171	65,038	69,958	67,264	77,902	74,402

Table 24-8: Production Schedule for Leach Material (Continued)

	(Units)	Year 13	Year 14	Year 15	Year 16	Year 17	Year 18	Year 19	Year 20	Year 21	Year 22	Year 23	Year 24	Year 25	Year 26	Year 27	Year 28	Year 29	Year 30	TOTAL
Leach Pad Stacking Schedule:																				
NSR Cutoff Grade	(C$)	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	
Ktonnes	(kt)	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	946	475	60	2,683	9,125	1,935	11	34	239,756
NSR Value	(C$/t)	7.18	8.42	9.90	10.28	10.37	10.46	10.72	14.45	15.57	17.61	18.68	5.61	5.62	7.01	7.01	6.85	9.88	9.88	10.72
Gold	(g/t)	0.154	0.183	0.228	0.240	0.245	0.246	0.249	0.335	0.360	0.410	0.436	0.129	0.129	0.147	0.147	0.140	0.231	0.231	0.245
Silver	(g/t)	1.704	1.711	2.148	1.653	1.428	1.419	1.533	2.288	2.385	2.706	2.680	0.620	0.570	1.480	1.300	1.473	2.650	2.650	1.87
Total Copper	(%)	0.044	0.051	0.019	0.023	0.019	0.022	0.029	0.040	0.045	0.042	0.044	0.023	0.023	0.056	0.061	0.069	0.000	0.000	0.034
Weak Soluble Copper	(%)	0.011	0.010	0.008	0.006	0.005	0.005	0.007	0.011	0.010	0.009	0.009	0.000	0.000	0.019	0.018	0.023	0.000	0.000	0.009
Molybdenum	(%)	0.0092	0.0190	0.0208	0.0061	0.0055	0.0052	0.0058	0.0202	0.0130	0.0135	0.0115	0.0009	0.0008	0.0046	0.0067	0.0079	0.0002	0.0002	0.0128
As Produced From Mine:																				
NSR Cutoff Grade	(C$)	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	5.46	
Ktonnes	(kt)	8,124	2,900	82	88	1,104	1,380	1,432	623	1,705	356	0	475	60	2,683	9,944	1,116	11	34	239,756
NSR Value	(C$/t)	6.96	7.34	7.76	5.53	6.79	8.22	8.22	9.25	7.11	7.53	0.00	5.61	5.62	7.01	7.01	6.73	9.88	9.88	10.72
Gold	(g/t)	0.149	0.149	0.166	0.126	0.161	0.190	0.179	0.200	0.159	0.182	0.000	0.129	0.129	0.147	0.147	0.134	0.231	0.231	0.245
Silver	(g/t)	1.720	2.010	1.540	1.680	1.120	1.130	2.060	0.690	0.360	0.960	0.000	0.620	0.570	1.480	1.300	1.600	2.650	2.650	1.87
Total Copper	(%)	0.044	0.067	0.055	0.005	0.007	0.027	0.042	0.082	0.049	0.001	0.000	0.023	0.023	0.056	0.061	0.075	0.000	0.000	0.034
Weak Soluble Copper	(%)	0.011	0.013	0.009	0.000	0.001	0.004	0.008	0.014	0.008	0.000	0.000	0.000	0.000	0.019	0.018	0.026	0.000	0.000	0.009
Molybdenum	(%)	0.0072	0.0049	0.0040	0.0007	0.0018	0.0007	0.0012	0.0006	0.0002	0.0002	0.0000	0.0009	0.0008	0.0046	0.0067	0.0087	0.0002	0.0002	0.0128
Direct to Crusher:		100.00%	100.00%	100.00%	100.00%	100.00%	100.00%	100.00%	100.00%	100.00%	100.00%	0.00%	100.00%	100.00%	100.00%	91.76%	100.00%	100.00%	100.00%	
Ktonnes	(kt)	8,124	2,900	82	88	1,104	1,380	1,432	623	1,705	356	0	475	60	2,683	9,125	1,116	11	34	145,514
NSR Value	(C$/t)	6.96	7.34	7.76	5.53	6.79	8.22	8.22	9.25	7.11	7.53	0.00	5.61	5.62	7.01	7.01	6.73	9.88	9.88	10.06
Gold	(g/t)	0.149	0.149	0.166	0.126	0.161	0.190	0.179	0.200	0.159	0.182	0.000	0.129	0.129	0.147	0.147	0.134	0.231	0.231	0.229
Silver	(g/t)	1.720	2.010	1.540	1.680	1.120	1.130	2.060	0.690	0.360	0.960	0.000	0.620	0.570	1.480	1.300	1.600	2.650	2.650	1.83
Total Copper	(%)	0.044	0.067	0.055	0.005	0.007	0.027	0.042	0.082	0.049	0.001	0.000	0.023	0.023	0.056	0.061	0.075	0.000	0.000	0.035
Weak Soluble Copper	(%)	0.011	0.013	0.009	0.000	0.001	0.004	0.008	0.014	0.008	0.000	0.000	0.000	0.000	0.019	0.018	0.026	0.000	0.000	0.009
Molybdenum	(%)	0.0072	0.0049	0.0040	0.0007	0.0018	0.0007	0.0012	0.0006	0.0002	0.0002	0.0000	0.0009	0.0008	0.0046	0.0067	0.0087	0.0002	0.0002	0.0125
To Leach Stockpile:		0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	8.24%	0.00%	0.00%	0.00%	
Ktonnes	(kt)	0	0	0	0	0	0	0	0	0	0	0	0	0	0	819	0	0	0	94,242
NSR Value	(C$/t)	6.96	7.34	7.76	5.53	6.79	8.22	8.22	9.25	7.11	7.53	0.00	5.61	5.62	7.01	7.01	6.73	9.88	9.88	11.74
Gold	(g/t)	0.149	0.149	0.166	0.126	0.161	0.190	0.179	0.200	0.159	0.182	0.000	0.129	0.129	0.147	0.147	0.134	0.231	0.231	0.271
Silver	(g/t)	1.720	2.010	1.540	1.680	1.120	1.130	2.060	0.690	0.360	0.960	0.000	0.620	0.570	1.480	1.300	1.600	2.650	2.650	1.93
Total Copper	(%)	0.044	0.067	0.055	0.005	0.007	0.027	0.042	0.082	0.049	0.001	0.000	0.023	0.023	0.056	0.061	0.075	0.000	0.000	0.032
Weak Soluble Copper	(%)	0.011	0.013	0.009	0.000	0.001	0.004	0.008	0.014	0.008	0.000	0.000	0.000	0.000	0.019	0.018	0.026	0.000	0.000	0.008
Molybdenum	(%)	0.0072	0.0049	0.0040	0.0007	0.0018	0.0007	0.0012	0.0006	0.0002	0.0002	0.0000	0.0009	0.0008	0.0046	0.0067	0.0087	0.0002	0.0002	0.0133
Stockpile Reclaim:																				
Ktonnes	(kt)	1,001	6,225	9,043	9,037	8,021	7,745	7,693	8,502	7,420	8,769	946					819			94,242
NSR Value	(C$/t)	8.92	8.92	9.92	10.33	10.86	10.86	11.18	14.83	17.52	18.02	18.68					7.01			11.74
Gold	(g/t)	0.199	0.199	0.228	0.241	0.256	0.256	0.262	0.344	0.406	0.419	0.436					0.147			0.271
Silver	(g/t)	1.570	1.572	2.154	1.653	1.470	1.470	1.435	2.405	2.850	2.777	2.680					1.300			1.93
Total Copper	(%)	0.044	0.044	0.019	0.023	0.021	0.021	0.027	0.037	0.044	0.044	0.044					0.061			0.032
Weak Soluble Copper	(%)	0.009	0.009	0.008	0.006	0.005	0.005	0.007	0.011	0.010	0.010	0.009					0.018			0.008
Molybdenum	(%)	0.0256	0.0256	0.0210	0.0062	0.0060	0.0060	0.0066	0.0216	0.0160	0.0141	0.0115					0.0067			0.0133
Stockpile Balance	(kt)	73,401	67,176	58,133	49,096	41,075	33,330	25,637	17,135	9,715	946	0	0	0	0	819	0	0	0	

24.2.4 Waste Management

Total waste in the Phase II mine plan amounts to 1.10 billion tonnes. This material is disposed in the tailing management facility and is stored sub-aqueously. Figure 24-2 shows four facilities for mine waste: 1) North Waste which contains 198.4 Mt, 2) South 1 Waste which contains 154.9 Mt, 3) South 2 Waste which contains 144.7 Mt, and 4) Southwest Waste which contains 606.8 Mt. The material will be placed by trucks and dozers and the pond water level or saturated NAG tailings will eventually cover the material. The waste material by material type is as follows:

- 76.8 Mt of overburden.
- 184.3 Mt of leach cap material.
- 22.6 Mt of supergene oxide material.
- 149.8 Mt of supergene sulphide material.
- 671.3 Mt of hypogene material.

Additional rock storage facilities during the life of the project include:

- The heap leach pad which at the end of the project will contain 239.8 Mt of spent, non-reactive material, assuming all the potential leach material is processed.
- A low-grade stockpile southeast of the pit that has the capacity for 177.7 Mt, and a low-grade stockpile east of the pit that contains 271.7 Mt, both which will be processed at the end of the mine life.
- There will also be supergene oxide (SOX) stockpile south of the pit to store Phase I SOX. It will be reclaimed during mining Years 4 through 13. The maximum size of this facility is estimated at 35.8 Mt.
- There will be a stockpile for leach resource east of the pit. This is expected to reach a maximum size of 77.9 million tonnes during Year 11 and will be reclaimed by the end of Year 28.

The stockpiles are all constructed in lifts from the bottom up. The low-grade stockpile, leach stockpile, and SOX stockpile are designed with 30 m lifts at angle of repose with a 20 m setback between lifts to make the overall slope angle about 2H:1V. This is deemed to be adequate since these are not permanent facilities.

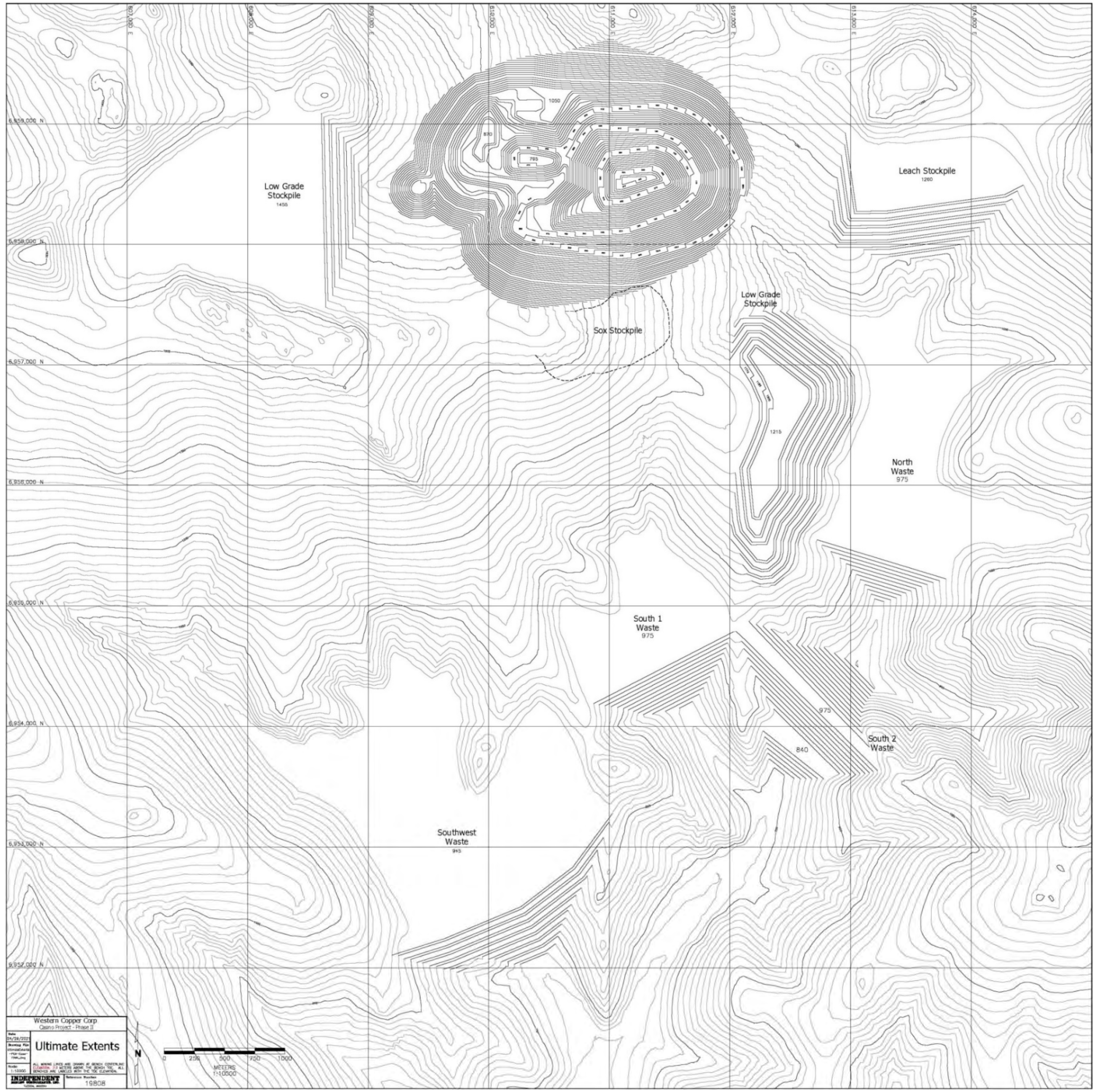


Figure 24-2: Maximum Extent of Phase II Waste Storage Areas and Stockpiles (IMC, 2021)

24.2.5 Mining Equipment

Mine equipment requirements were sized and estimated on a first principles basis, based on the mine production schedule, the mine work schedule, and estimated equipment shift productivity rates. The size and type of mining equipment is consistent with the size of the project, i.e. peak material movements of 100 Mt/y. The mine equipment estimate is based on owner operation and assumes a well-managed mining operation with a well-trained labour pool, and that all the equipment is new at the start of the operation.

Table 24-9 summarizes the major equipment requirements. The first column shows initial equipment for the first year of mine development (Year -3), the second column shows equipment required for the beginning of commercial production, and the third column shows the peak fleet requirements. This represents the equipment required to perform the following duties:

- Develop access roads from the mine to the crusher, various stockpiles, and the waste storage area.
- Mine and transport mill material and leach material to the crushers.
- Mine and transport resource to various stockpiles as required.
- Reclaim stockpiled material and transport it to the crushers.
- Mine and transport waste to the waste storage areas in the TMF.
- Maintain the haul roads and stockpiles and various truck dumping sites.

Table 24-9: Mining Equipment Requirements

Equipment Type	Capacity/ Power	Year -3	Year 1	Peak
P&H 320XPC Drill	(356 mm)	1	4	5
P&H 4100XPC Cable Shovel	(67.6 cu m)	0	2	2
Komatsu PC8000-6 Hyd Shovel	(42 cu m)	0	1	1
Komatsu WA1200-6 Wheel Loa	(20 cu m)	1	1	1
Komatsu 980E Truck	(370 mt)	0	16	31
Komatsu HD1500 Truck	(144 mt)	7	3	7
Komatsu D475A Track Dozer	(664 kw)	0	3	3
Komatsu D375A Track Dozer	(455 kw)	2	3	3
Komatsu WD900 Wheel Dozer	(637 kw)	0	3	3
Komatsu GD825A Motor Grade	(209 kw)	1	3	3
Water Truck - 30,000 gal	(113,550 l)	1	3	3
Komatsu PC360LC-11 Excavat	(1.96 cu m)	1	2	2
Epiroc SmartROC D65	(178 mm)	1	2	2
TOTAL		15	46	66

24.2.6 Phase II Mine Capital Costs

The estimated mine capital cost includes the following items:

- Mine major equipment
- Mine support equipment
- Mine preproduction development expense

The estimated cost of the following mining facilities was developed by others and is included in the infrastructure capital budget:

- The mine shop and warehouse
- Fuel and lubricant storage facilities
- Explosive storage facilities
- Office facilities

Table 24-10 summarizes the Phase II mine capital cost by category for initial and sustaining capital. The initial capital period is considered to be the four-year period from Years -3 through Year 1, as these are the years of significant capital build-up.

Table 24-10: Phase II Mining Capital – Mine Equipment and Mine Development (C$ x 1000)

		Initial Capital by Time Period				Initial	Sustaining	Total
Category		Yr -3	Yr -2	Yr -1	Year 1	Capital	Capital	Capital
Major Equipment		45,103	71,914	84,444	142,853	344,313	588,295	932,608
Support Equipment @	15.00%	6,765	10,787	12,667	21,428	51,647	88,244	139,891
Initial Spare Parts @	0.00%	0	0	0	0	0	0	0
Shop Tools @	0.00%	0	0	0	0	0	0	0
Equipment Subtotal		51,869	82,701	97,111	164,280	395,960	676,539	1,072,500
Equipment Contingency @	10.0%	5,187	8,270	9,711	0	23,168	0	23,168
Mine Development		31,641	63,215	111,057	0	205,913	0	205,913
TOTAL MINE CAPITAL		88,697	154,186	217,879	164,280	625,041	676,539	1,301,581
Exclusions: Mine shop and warehouse, fuel and lubricant storage, explosives storage, and offices.								

Mine preproduction development of $210.6 million is based on the estimated mine operating costs during the preproduction period. The cost estimate is based on owner operating costs with large equipment plus a contingency to provide additional allowance for additional road construction or other site preparation and subcontracting portions of the mining, etc. Table 24-11 shows the components of the cost during mine development by year. Total preproduction development is estimated as 75.0 million tonnes, so the unit rate amounts to $2.81/t.

Table 24-11: Mine Development Direct Costs Plus Contingency (C$ x 1000)

Item	Year -3	Year -2	Year -1	Total
Owner Operating Cost – Large Equip	25,205	54,871	101,064	181,140
Mine Development Contingency	6,436	8,344	9,994	24,774
Total Mine Development Cost	31,641	63,215	111,057	205,913
%Contingency	25.5%	15.2%	9.9%	13.7%

24.2.7 Phase II Mine Operating Costs

Table 24-12 summarizes the Phase II mine operating costs. The total cost, the cost per total tonne, and cost per mill tonne are shown by various time periods. During commercial production the unit costs for mining are $2.077 per total tonne and $3.795 per mill tonne. The leach tonnes are not in the divisor for the cost per mill tonne calculations. Years 38 to 47 are low grade stockpile re-handle.

Table 24-12: Summary of Total and Unit Mining Costs

Category	Total Material (kt)	Mill Material (kt)	Total Cost (C$)	Cost Per Total Tonne (C$)	Cost Per Mill Tonne (C$)
Mine Development (PP)	74,999	0	205,913	2.746	0.000
Commercial Production (Years 1 to 47)	3,887,288	2,127,790	8,075,358	2.077	3.795
All Time Periods	3,962,287	2,127,790	8,281,271	2.090	3.892
Commercial Production Years 1 - 5	505,716	218,944	933,915	1.847	4.266
Commercial Production Years 6 - 10	518,003	228,636	1,024,968	1.979	4.483
Commercial Production Years 11 - 20	1,010,639	458,140	2,225,660	2.202	4.858
Commercial Production Years 21 - 37	1,409,952	779,092	3,277,542	2.325	4.207
Commercial Production Yr 38 - Final (LG)	442,978	442,978	613,274	1.384	1.384

The estimate is based on assumed prices for commodities such as fuel, explosives, parts, etc. that are subject to wide variations depending on market conditions. The current estimate is based on the following estimated prices for key commodities:

- Diesel fuel delivered to the site for C$ 0.98 per liter,
- Electrical power at C$ 0.095 per kWh,
- Bulk emulsion at C$ 0.85 per kg delivered to the site, and
- Tires at approximately 75% of US list prices.
- Exchange rate of C$1.26 = US$1.00.

24.3 PROCESS PLANT AND INFRASTRUCTURE SUSTAINING CAPITAL

24.3.1 Process Plant for the Proposed Phase II Expansion

The Process Plant remains the same for the Proposed Phase II Expansion. There will not be any anticipated increase in the Sustaining Capital; the operating cost includes a higher allowance for maintenance material and service for an aging plant.

24.3.2 Sustaining Capital for Water Management

The Sustaining Capital for Water Management in the proposed Phase II Expansion is limited to the cost of additional Seepage pumps, Seepage “pond,” Relay/Booster Stations, and additional piping to get the reclaimed water up to the Sand Cyclone Thickener Overflow Sump. From there the water is returned to the Process Water Pond. Additionally, there will be cost for the addition of one additional E-house and the extension the overhead electrical power lines to get to the E-house.

24.3.3 Sustaining Capital for Material Movement

The Sustaining Capital for Material movement consist of the conveyors necessary to take the de-watered, Sand Cyclone underflow material (Coarse Sand) to the Phase II Tailings Management Facility Stockpile and to the Phase II Dam Crest. The Sand Cyclone underflow will flow to the Phase II dam Crest. Additionally, the thickened tailings from the Sand Cyclone Thickener will be piped by gravity to the Phase II TMF. See Phase I and Phase II Site Plan in Figure 24-3.

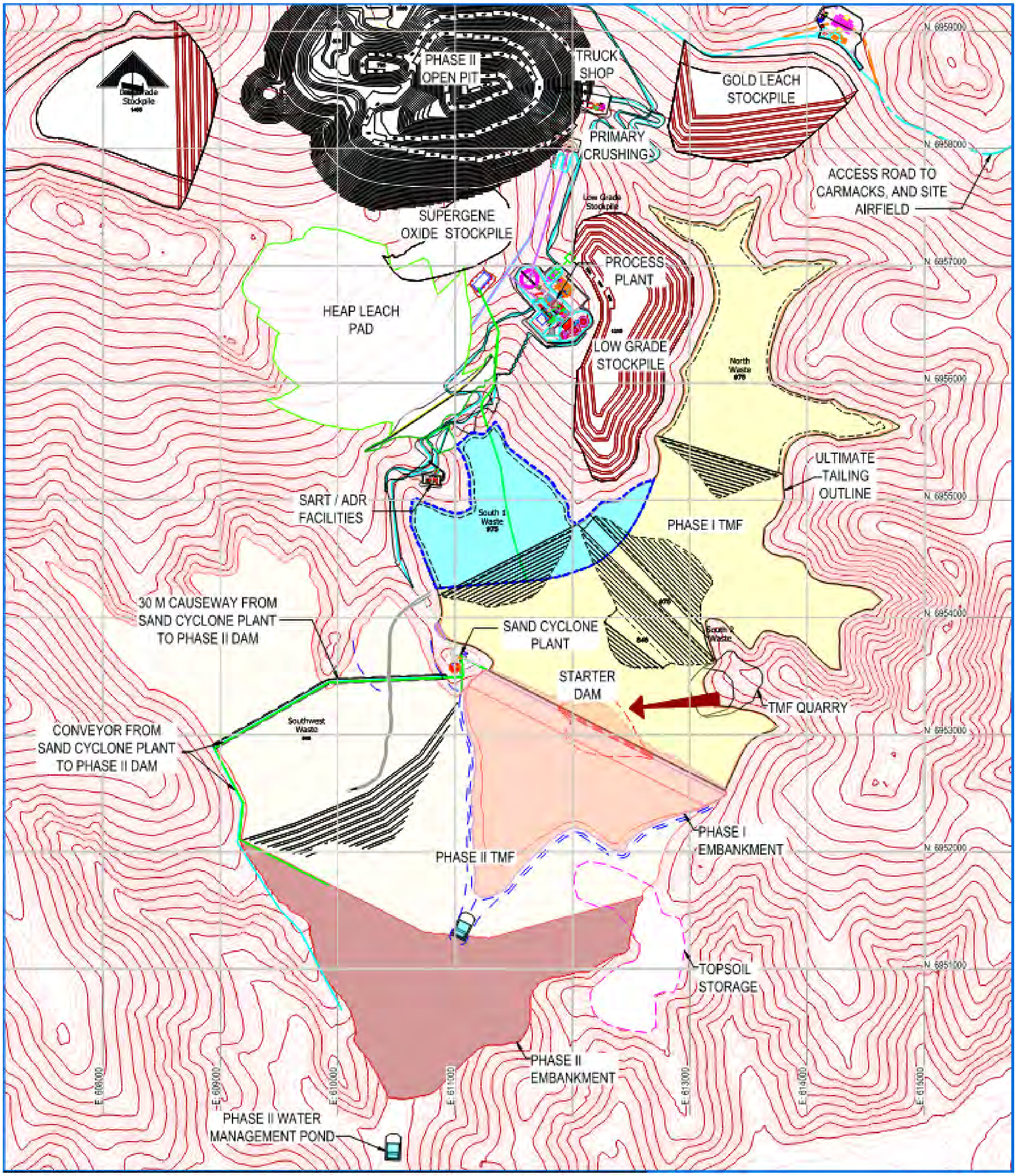


Figure 24-3: Phase I and Phase II Site with TMF (M3, 2021)

This study assumes a 4-year design, procurement, and construction schedule that will begin with a notice to proceed (NTP) by the project owner.

Table 24-13: Phase II Expansion Financial Model

	Total	-4	-3	-2	-1	1	2	3	4	5	6	7	8
Mining Operations													
Direct Feed Mill Mineralized Material													
Beginning Inventory (kt)	1,642,533		1,642,533	1,642,533	1,642,313	1,637,865	1,608,033	1,561,151	1,514,679	1,472,667	1,430,789	1,388,882	1,346,856
Mined (kt)	1,642,533		-	220	4,448	29,832	46,882	46,472	42,012	41,878	41,907	42,026	42,761
Ending Inventory (kt)	-		1,642,533	1,642,313	1,637,865	1,608,033	1,561,151	1,514,679	1,472,667	1,430,789	1,388,882	1,346,856	1,304,095
Copper Grade (%)	0.179%		0.000%	0.242%	0.247%	0.372%	0.338%	0.286%	0.269%	0.216%	0.202%	0.203%	0.204%
Molybdenum Grade (%)	0.020%		0.0000%	0.0177%	0.0135%	0.0238%	0.0297%	0.0323%	0.0190%	0.0144%	0.0185%	0.0271%	0.0305%
Gold Grade (g/t)	0.201		-	0.332	0.437	0.427	0.357	0.358	0.313	0.265	0.254	0.248	0.238
Silver Grade (g/t)	1.609		-	1.920	2.302	2.485	1.901	1.951	1.977	1.948	1.784	2.180	1.677
Contained Copper (klbs)	6,483,386		-	1,174	24,238	244,405	349,015	292,897	249,173	199,318	187,002	188,082	192,460
Contained Molybdenum (klbs)	740,249		-	86	1,326	15,661	30,646	33,069	17,607	13,268	17,099	25,109	28,766
Contained Gold (koz)	10,620		-	2	62	409	538	534	423	357	342	335	327
Contained Silver (koz)	84,944		-	14	329	2,384	2,866	2,915	2,670	2,623	2,404	2,946	2,306
SOX Stockpile Mineralized Material													
Beginning Inventory (kt)	35,841		35,841	35,841	33,947	20,905	778	-	-	-	-	-	-
Mined (kt)	35,841		-	1,894	13,042	20,127	778	-	-	-	-	-	-
Ending Inventory (kt)	-		35,841	33,947	20,905	778	-	-	-	-	-	-	-
Copper Grade (%)	0.251%		0.000%	0.195%	0.185%	0.299%	0.242%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%
Molybdenum Grade (%)	0.025%		0.0000%	0.0193%	0.0212%	0.0289%	0.0116%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%
Gold Grade (g/t)	0.491		-	0.541	0.581	0.434	0.322	-	-	-	-	-	-
Silver Grade (g/t)	2.361		-	2.400	2.930	2.006	1.900	-	-	-	-	-	-
Contained Copper (klbs)	198,230		-	8,142	53,192	132,745	4,151	-	-	-	-	-	-
Contained Molybdenum (klbs)	19,942		-	806	6,096	12,842	199	-	-	-	-	-	-
Contained Gold (koz)	565		-	33	244	281	8	-	-	-	-	-	-
Contained Silver (koz)	2,720		-	146	1,229	1,298	48	-	-	-	-	-	-
Low Grade Mineralized Material													
Beginning Inventory(kt)	449,416		449,416	449,416	449,416	449,257	444,491	436,746	427,732	404,228	359,873	329,908	308,941
Mined (kt)	449,416		-	-	159	4,766	7,745	9,014	23,504	44,355	29,965	20,967	15,642
Ending Inventory (kt)	-		449,416	449,416	449,257	444,491	436,746	427,732	404,228	359,873	329,908	308,941	293,299
Copper Grade (%)	0.098%		0.000%	0.000%	0.156%	0.167%	0.129%	0.113%	0.150%	0.138%	0.114%	0.098%	0.082%
Molybdenum Grade (%)	0.009%		0.0000%	0.0000%	0.6700%	0.0152%	0.0032%	0.0068%	0.0114%	0.0099%	0.0118%	0.0151%	0.0106%
Gold Grade (g/t)	0.135		-	-	0.214	0.189	0.152	0.184	0.138	0.134	0.140	0.131	0.152
Silver Grade (g/t)	1.069		-	-	1.720	1.352	0.908	1.145	1.171	1.117	1.181	1.284	1.028
Contained Copper (klbs)	972,425		-	-	547	17,565	22,039	22,367	77,812	135,076	74,982	45,473	28,115
Contained Molybdenum (klbs)	93,867		-	-	2,349	1,596	544	1,357	5,903	9,642	7,824	6,971	3,666
Contained Gold (koz)	1,951		-	-	1	29	38	53	104	191	135	88	76
Contained Silver (koz)	15,442		-	-	9	207	226	332	885	1,593	1,138	865	517
Total Mill Mineralized Material													
Beginning Inventory(kt)	2,127,790	-	2,127,790	2,127,790	2,125,676	2,108,027	2,053,302	1,997,897	1,942,411	1,876,895	1,790,662	1,718,790	1,655,797
Mined (kt)	2,127,790	-	-	2,114	17,649	54,725	55,405	55,486	65,516	86,233	71,872	62,993	58,403
Ending Inventory (kt)	-		2,127,790	2,125,676	2,108,027	2,053,302	1,997,897	1,942,411	1,876,895	1,790,662	1,718,790	1,655,797	1,597,394
Copper Grade (%)	0.163%	0.000%	0.000%	0.200%	0.200%	0.327%	0.307%	0.258%	0.226%	0.176%	0.165%	0.168%	0.171%
Molybdenum Grade (%)	0.018%	0.000%	0.000%	0.019%	0.025%	0.025%	0.026%	0.028%	0.016%	0.012%	0.016%	0.023%	0.025%
Gold Grade (g/t)	0.192	-	-	0.519	0.541	0.409	0.328	0.329	0.250	0.197	0.206	0.209	0.215
Silver Grade (g/t)	1.507	-	-	2.350	2.761	2.210	1.763	1.820	1.688	1.521	1.533	1.882	1.503
Contained Copper (klbs)	7,654,041	-	-	9,316	77,977	394,714	375,205	315,263	326,984	334,394	261,985	233,555	220,576
Contained Molybdenum (klbs)	854,059	-	-	892	9,770	30,099	31,389	34,426	23,510	22,910	24,923	32,080	32,433
Contained Gold (koz)	13,136	-	-	35	307	719	584	587	527	547	477	423	403
Contained Silver (koz)	103,107	-	-	160	1,567	3,889	3,140	3,247	3,555	4,216	3,542	3,811	2,823
Leach Mineralized Material													
Beginning Inventory(kt)	239,756		239,756	234,269	220,438	197,382	177,319	142,699	122,704	107,897	107,080	102,474	91,973
Mined (kt)	239,756		5,487	13,831	23,056	20,063	34,620	19,995	14,807	817	4,606	10,501	13,992
Ending Inventory (kt)	-		234,269	220,438	197,382	177,319	142,699	122,704	107,897	107,080	102,474	91,973	77,981
Copper Grade (%)	0.034%		0.0210%	0.0440%	0.0440%	0.0357%	0.0210%	0.0270%	0.0420%	0.0650%	0.0080%	0.0090%	0.0200%
Gold Grade (g/t)	0.245		0.273	0.436	0.406	0.299	0.256	0.211	0.164	0.149	0.185	0.217	0.244
Silver Grade (g/t)	1.866		1.300	2.680	2.850	1.927	1.470	2.010	1.310	0.840	2.250	2.430	2.290
Contained Copper (klbs)	179,247		2,540	13,417	22,365	15,790	16,028	11,902	13,710	1,171	812	2,084	6,169
Contained Gold (koz)	1,892		48	194	301	193	285	136	78	4	27	73	110
Contained Silver (koz)	14,383		229	1,192	2,113	1,243	1,636	1,292	624	22	333	820	1,030
Waste													
Beginning Inventory(kt)	1,104,816		1,104,816	1,102,880	1,099,067	1,091,954	1,071,739	1,061,868	1,037,348	1,018,211	1,005,773	982,251	955,744
Mined (kt)	1,104,816		1,936	3,813	7,113	20,215	9,871	24,520	19,137	12,438	23,522	26,507	27,606
Ending Inventory (kt)	-		1,102,880	1,099,067	1,091,954	1,071,739	1,061,868	1,037,348	1,018,211	1,005,773	982,251	955,744	928,138
Total Material Mined (kt)	3,472,362		7,423	19,758	47,818	95,003	99,896	100,001	99,460	99,488	100,000	100,001	100,001
Waste to Mineralized Material Ratio	0.47		0.35	0.24	0.17	0.27	0.11	0.32	0.24	0.14	0.31	0.36	0.38
Mine Re-handle (kt)	489,925		-	-	-	4,668	-	-	3,600	3,600	3,600	3,600	3,600
Process Plant Operations - Concentrator													
Supergene Oxide Mineralized Material													
Beginning Mineralized Material Inventory (kt)	-		-	-	-	-	-	-	-	-	-	-	-
Mined Mineralized Material to Concentrator (kt)	94,273	-	-	-	-	-	250	478	7,366	5,035	3,600	3,600	3,760
Mined Mineralized Material - Processed (kt)	94,273	-	-	-	-	-	250	478	7,366	5,035	3,600	3,600	3,760
Ending Mineralized Material Inventory		-	-	-	-	-	-	-	-	-	-	-	-
Copper Grade (%)	0.210%		0.000%	0.000%	0.000%	0.000%	0.178%	0.208%	0.345%	0.397%	0.283%	0.250%	0.190%
Molybdenum Grade (%)	0.017%		0.0000%	0.0000%	0.0000%	0.0000%	0.0110%	0.0108%	0.0181%	0.0237%	0.0301%	0.0303%	0.0300%
Gold Grade (g/t)	0.284		-	-	-	-	0.253	0.360	0.278	0.342	0.449	0.464	0.467
Silver Grade (g/t)	1.641		-	-	-	-	1.490	2.500	1.753	1.578	1.973	2.020	2.262

	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23
Mining Operations															
Direct Feed Mill Mineralized Material															
Beginning Inventory (kt)	1,304,095	1,261,098	1,220,153	1,177,968	1,135,312	1,091,814	1,045,730	1,000,293	954,455	908,574	863,226	818,064	772,654	726,808	680,663
Mined (kt)	42,997	40,945	42,185	42,656	43,498	46,084	45,437	45,838	45,881	45,348	45,162	45,410	45,846	46,145	46,579
Ending Inventory (kt)	1,261,098	1,220,153	1,177,968	1,135,312	1,091,814	1,045,730	1,000,293	954,455	908,574	863,226	818,064	772,654	726,808	680,663	634,084
Copper Grade (%)	0.187%	0.233%	0.259%	0.211%	0.192%	0.189%	0.196%	0.150%	0.145%	0.139%	0.148%	0.146%	0.128%	0.111%	0.149%
Molybdenum Grade (%)	0.0349%	0.0288%	0.0279%	0.0277%	0.0309%	0.0204%	0.0172%	0.0123%	0.0144%	0.0193%	0.0224%	0.0253%	0.0258%	0.0198%	0.0188%
Gold Grade (g/t)	0.226	0.215	0.205	0.207	0.207	0.163	0.185	0.186	0.172	0.169	0.172	0.175	0.169	0.172	0.192
Silver Grade (g/t)	1.632	1.976	1.855	1.642	1.534	1.687	1.603	1.449	1.722	2.119	1.551	1.446	1.323	1.232	1.583
Contained Copper (klbs)	177,614	210,423	240,585	198,051	184,006	191,658	196,153	152,088	146,468	138,619	147,000	146,460	129,072	113,130	152,650
Contained Molybdenum (klbs)	33,122	26,042	25,912	26,046	29,620	20,703	17,254	12,462	14,542	19,324	22,262	25,334	26,115	20,104	19,343
Contained Gold (koz)	312	284	279	284	290	241	271	274	254	247	249	255	249	256	288
Contained Silver (koz)	2,256	2,601	2,516	2,252	2,146	2,500	2,342	2,135	2,540	3,089	2,252	2,111	1,950	1,828	2,370
SOX Stockpile Mineralized Material															
Beginning Inventory (kt)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Mined (kt)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Ending Inventory (kt)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Copper Grade (%)	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%
Molybdenum Grade (%)	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%
Gold Grade (g/t)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Silver Grade (g/t)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Contained Copper (klbs)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Contained Molybdenum (klbs)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Contained Gold (koz)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Contained Silver (koz)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Low Grade Mineralized Material															
Beginning Inventory(kt)	293,299	273,143	249,842	233,751	221,365	209,445	192,155	171,689	151,801	134,737	121,365	102,073	85,402	76,416	67,439
Mined (kt)	20,156	23,301	16,091	12,386	11,920	17,290	20,466	19,888	17,064	13,372	19,292	16,671	8,986	8,977	2,872
Ending Inventory (kt)	273,143	249,842	233,751	221,365	209,445	192,155	171,689	151,801	134,737	121,365	102,073	85,402	76,416	67,439	64,567
Copper Grade (%)	0.075%	0.084%	0.103%	0.092%	0.116%	0.122%	0.111%	0.095%	0.079%	0.063%	0.061%	0.058%	0.053%	0.044%	0.048%
Molybdenum Grade (%)	0.0096%	0.0060%	0.0105%	0.0091%	0.0090%	0.0064%	0.0067%	0.0100%	0.0139%	0.0086%	0.0091%	0.0062%	0.0045%	0.0026%	0.0022%
Gold Grade (g/t)	0.172	0.165	0.141	0.134	0.127	0.116	0.108	0.112	0.113	0.144	0.150	0.150	0.155	0.179	0.163
Silver Grade (g/t)	1.006	1.041	1.075	1.080	1.095	1.197	0.935	1.033	1.209	1.052	1.073	1.180	0.924	0.647	0.925
Contained Copper (klbs)	33,406	43,081	36,471	25,186	30,544	46,694	50,288	41,527	29,667	18,686	26,150	21,262	10,433	8,769	3,037
Contained Molybdenum (klbs)	4,285	3,058	3,720	2,476	2,355	2,435	3,037	4,368	5,216	2,543	3,856	2,293	899	506	138
Contained Gold (koz)	111	123	73	53	49	65	71	72	62	62	93	81	45	52	15
Contained Silver (koz)	652	780	556	430	420	665	615	661	664	452	666	633	267	187	85
Total Mill Mineralized Material															
Beginning Inventory(kt)	1,597,394	1,534,241	1,469,995	1,411,719	1,356,677	1,301,259	1,237,885	1,171,982	1,106,256	1,043,311	984,591	920,137	858,056	803,224	748,102
Mined (kt)	63,153	64,246	58,276	55,042	55,418	63,374	65,903	65,726	62,945	58,720	64,454	62,081	54,832	55,122	49,451
Ending Inventory (kt)	1,534,241	1,469,995	1,411,719	1,356,677	1,301,259	1,237,885	1,171,982	1,106,256	1,043,311	984,591	920,137	858,056	803,224	748,102	698,651
Copper Grade (%)	0.152%	0.179%	0.216%	0.184%	0.176%	0.171%	0.170%	0.134%	0.127%	0.122%	0.122%	0.123%	0.115%	0.100%	0.143%
Molybdenum Grade (%)	0.027%	0.021%	0.023%	0.024%	0.026%	0.017%	0.014%	0.012%	0.014%	0.017%	0.018%	0.020%	0.022%	0.017%	0.018%
Gold Grade (g/t)	0.208	0.197	0.188	0.191	0.190	0.150	0.161	0.164	0.156	0.164	0.165	0.168	0.167	0.173	0.191
Silver Grade (g/t)	1.432	1.637	1.640	1.516	1.440	1.553	1.396	1.323	1.583	1.876	1.408	1.374	1.258	1.137	1.545
Contained Copper (klbs)	211,020	253,504	277,056	223,237	214,551	238,352	246,441	193,615	176,135	157,305	173,151	167,722	139,505	121,899	155,687
Contained Molybdenum (klbs)	37,407	29,100	29,632	28,523	31,975	23,138	20,291	16,829	19,758	21,867	26,118	27,627	27,014	20,609	19,482
Contained Gold (koz)	423	407	352	338	339	306	342	346	316	309	342	336	294	307	303
Contained Silver (koz)	2,908	3,380	3,072	2,682	2,565	3,165	2,957	2,796	3,203	3,541	2,917	2,743	2,217	2,015	2,456
Leach Mineralized Material															
Beginning Inventory(kt)	77,981	63,936	57,505	37,742	32,117	23,993	21,093	21,011	20,923	19,819	18,439	17,007	16,384	14,679	14,323
Mined (kt)	14,045	6,431	19,763	5,625	8,124	2,900	82	88	1,104	1,380	1,432	623	1,705	356	-
Ending Inventory (kt)	63,936	57,505	37,742	32,117	23,993	21,093	21,011	20,923	19,819	18,439	17,007	16,384	14,679	14,323	14,323
Copper Grade (%)	0.0210%	0.0380%	0.0440%	0.0470%	0.0440%	0.0670%	0.0550%	0.0050%	0.0070%	0.0270%	0.0420%	0.0820%	0.0490%	0.0010%	0.0000%
Gold Grade (g/t)	0.206	0.195	0.199	0.144	0.149	0.149	0.166	0.126	0.161	0.190	0.179	0.200	0.159	0.182	-
Silver Grade (g/t)	1.710	1.320	1.570	1.750	1.720	2.010	1.540	1.680	1.120	1.130	2.060	0.690	0.360	0.960	-
Contained Copper (klbs)	6,502	5,388	19,171	5,828	7,881	4,284	99	10	170	821	1,326	1,126	1,842	8	-
Contained Gold (koz)	93	40	126	26	39	14	0	0	6	8	8	4	9	2	-
Contained Silver (koz)	772	273	998	316	449	187	4	5	40	50	95	14	20	11	-
Waste															
Beginning Inventory(kt)	928,138	905,335	876,012	854,051	814,717	778,259	744,533	710,519	676,333	640,382	600,483	566,369	529,074	485,611	441,089
Mined (kt)	22,803	29,323	21,961	39,334	36,458	33,726	34,014	34,186	35,951	39,899	34,114	37,295	43,463	44,522	50,549
Ending Inventory (kt)	905,335	876,012	854,051	814,717	778,259	744,533	710,519	676,333	640,382	600,483	566,369	529,074	485,611	441,089	390,540
Total Material Mined (kt)	100,001	100,000	100,000	100,001	100,000	100,000	99,999	100,000	100,000	99,999	100,000	99,999	100,000	100,000	100,000
Waste to Mineralized Material Ratio	0.30	0.41	0.28	0.65	0.57	0.51	0.52	0.52	0.56	0.66	0.52	0.59	0.77	0.80	1.02
Mine Re-handle (kt)	3,600	3,600	3,600	3,600	3,441	-	-	-	-	-	-	-	-	-	-
Process Plant Operations - Concentrator															
Supergene Oxide Mineralized Material															
Beginning Mineralized Material Inventory (kt)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Mined Mineralized Material to Concentrator (kt)	4,986	5,112	4,265	3,624	4,489	3,116	93	-	-	-	-	-	87	128	-
Mined Mineralized Material - Processed (kt)	4,986	5,112	4,265	3,624	4,489	3,116	93	-	-	-	-	-	87	128	-
Ending Mineralized Material Inventory	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Copper Grade (%)	0.171%	0.189%	0.183%	0.185%	0.237%	0.318%	0.204%	0.000%	0.000%	0.000%	0.000%	0.000%	0.095%	0.136%	0.000%
Molybdenum Grade (%)	0.0290%	0.0238%	0.0203%	0.0211%	0.0179%	0.0086%	0.0078%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0003%	0.0002%	0.0000%
Gold Grade (g/t)	0.425	0.475	0.526	0.579	0.468	0.158	0.148	-	-	-	-	-	0.216	0.207	-
Silver Grade (g/t)	2.301	2.646	2.749	2.938	2.319	1.070	1.020	-	-	-	-	-	0.400	2.130	-

	Total	-4	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23
Contained Copper (klbs)	437,034		-	-	-	-	981	2,192	55,978	44,081	22,447	19,842	15,788	18,843	21,250	17,175	14,755	23,439	21,845	418	-	-	-	-	-	182	384	-
Contained Molybdenum (klbs)	34,375		-	-	-	-	61	114	2,941	2,631	2,393	2,405	2,488	3,189	2,683	1,910	1,686	1,767	591	16	-	-	-	-	-	1	1	-
Contained Gold (koz)	859		-	-	-	-	2	6	66	55	52	54	56	68	78	72	67	68	16	0	-	-	-	-	-	1	1	-
Contained Silver (koz)	4,973		-	-	-	-	12	38	415	255	228	234	273	369	435	377	342	335	107	3	-	-	-	-	-	1	9	-
Recovery Copper (%)	61.31%		0.00%	0.00%	0.00%	0.00%	71.35%	68.27%	64.85%	63.73%	58.19%	56.40%	60.25%	65.37%	68.84%	63.73%	62.29%	56.21%	57.23%	69.12%	0.00%	0.00%	0.00%	0.00%	0.00%	62.11%	60.29%	0.00%
Recovery Molybdenum (%)	52.30%		0.00%	0.00%	0.00%	0.00%	52.30%	52.30%	52.30%	52.30%	52.30%	52.30%	52.30%	52.30%	52.30%	52.30%	52.30%	52.30%	52.30%	52.30%	52.30%	52.30%	52.30%	52.30%	52.30%	52.30%	52.30%	52.30%
Recovery Gold (%)	69.00%		0.00%	0.00%	0.00%	0.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%
Recovery Silver (%)	60.00%		0.00%	0.00%	0.00%	0.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%
Copper Concentrate (kt)	434		-	-	-	-	1	2	59	46	21	18	15	20	24	18	15	21	20	0	-	-	-	-	-	0	0	-
Copper Concentrate Grade (%)	28.00%		0.00%	0.00%	0.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%
Recovered Copper (klbs)	267,945		-	-	-	-	700	1,496	36,304	28,091	13,061	11,191	9,512	12,318	14,627	10,945	9,192	13,176	12,503	289	-	-	-	-	-	113	231	-
Recovered Gold (koz)	593		-	-	-	-	1	4	45	38	36	37	39	47	54	50	47	47	11	0	-	-	-	-	-	0	1	-
Recovered Silver (koz)	2,984		-	-	-	-	7	23	249	153	137	140	164	221	261	226	205	201	64	2	-	-	-	-	-	1	5	-
Molybdenum Concentrate (kt)	15		-	-	-	-	0	0	1	1	1	1	1	1	1	1	1	1	0	0	-	-	-	-	-	0	0	-
Molybdenum Concentrate Grade (%)	56.00%		0.00%	0.00%	0.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%
Recovered Molybdenum (klbs)	17,978		-	-	-	-	32	60	1,538	1,376	1,251	1,258	1,301	1,668	1,403	999	882	924	309	8	-	-	-	-	-	0	0	-
Supergene Sulfide Mineralized Material																												
Beginning Mineralized Material Inventory (kt)	-		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
Mined Mineralized Material to Concentrator (kt)	349,359	-	-	-	-	31,099	24,445	7,077	21,784	12,386	388	-	103	1,775	31,623	26,627	15,016	3,419	14,502	20,174	3,750	78	194	1,690	2,544	1,009	2,384	157
Mined Mineralized Material - Processed (kt)	349,359	-	-	-	-	31,099	24,445	7,077	21,784	12,386	388	-	103	1,775	31,623	26,627	15,016	3,419	14,502	20,174	3,750	78	194	1,690	2,544	1,009	2,384	157
Ending Mineralized Material Inventory		-				-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
Copper Grade (%)	0.224%		0.000%	0.000%	0.000%	0.364%	0.408%	0.318%	0.260%	0.238%	0.246%	0.000%	0.025%	0.221%	0.244%	0.301%	0.267%	0.210%	0.221%	0.223%	0.201%	0.029%	0.058%	0.087%	0.185%	0.206%	0.077%	0.046%
Molybdenum Grade (%)	0.017%		0.0000%	0.0000%	0.0000%	0.0225%	0.0337%	0.0336%	0.0131%	0.0142%	0.0193%	0.0000%	0.0003%	0.0209%	0.0294%	0.0258%	0.0246%	0.0245%	0.0103%	0.0163%	0.0183%	0.0008%	0.0028%	0.0033%	0.0035%	0.0164%	0.0018%	0.0002%
Gold Grade (g/t)	0.220		-	-	-	0.422	0.361	0.322	0.291	0.253	0.216	-	0.448	0.284	0.211	0.204	0.193	0.177	0.150	0.190	0.199	0.292	0.232	0.215	0.172	0.158	0.232	0.310
Silver Grade (g/t)	1.593		-	-	-	2.427	1.780	1.810	1.880	1.830	2.220	-	0.540	2.210	1.970	1.860	1.550	1.550	1.500	1.560	1.770	2.700	1.870	2.600	1.540	1.530	1.220	2.440
Contained Copper (klbs)	1,722,569		-	-	-	249,568	219,879	49,615	124,866	64,989	2,104	-	57	8,648	170,109	176,694	88,389	15,829	70,657	99,181	16,617	50	248	3,241	10,376	4,582	4,047	159
Contained Molybdenum (klbs)	129,068		-	-	-	15,437	18,162	5,242	6,291	3,878	165	-	1	818	20,497	15,145	8,144	1,847	3,293	7,250	1,513	1	12	123	196	365	95	1
Contained Gold (koz)	2,466		-	-	-	422	284	73	204	101	3	-	1	16	215	175	93	19	70	123	24	1	1	12	14	5	18	2
Contained Silver (koz)	17,894		-	-	-	2,426	1,399	412	1,317	729	28	-	2	126	2,003	1,592	748	170	699	1,012	213	7	12	141	126	50	94	12
Recovery Copper (%)	81.13%		0.00%	0.00%	0.00%	77.50%	81.62%	83.96%	82.31%	79.83%	83.33%	0.00%	88.00%	84.16%	82.79%	83.39%	84.64%	83.81%	79.19%	83.86%	86.07%	86.21%	86.21%	87.36%	81.08%	84.95%	81.82%	76.09%
Recovery Molybdenum (%)	52.30%		0.00%	0.00%	0.00%	52.30%	52.30%	52.30%	52.30%	52.30%	52.30%	52.30%	52.30%	52.30%	52.30%	52.30%	52.30%	52.30%	52.30%	52.30%	52.30%	52.30%	52.30%	52.30%	52.30%	52.30%	52.30%	52.30%
Recovery Gold (%)	69.00%		0.00%	0.00%	0.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%
Recovery Silver (%)	60.00%		0.00%	0.00%	0.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%
Copper Concentrate (kt)	2,264		-	-	-	313	291	67	166	84	3	-	0	12	228	239	121	21	91	135	23	0	0	5	14	6	5	0
Copper Concentrate Grade (%)	28.00%		0.00%	0.00%	0.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%
Recovered Copper (klbs)	1,397,439		-	-	-	193,406	179,460	41,658	102,774	51,882	1,754	-	50	7,279	140,828	147,343	74,816	13,266	55,950	83,170	14,302	43	214	2,832	8,413	3,893	3,311	121
Recovered Gold (koz)	1,701		-	-	-	291	196	51	141	70	2	-	1	11	148	121	64	13	48	85	17	1	1	8	10	4	12	1
Recovered Silver (koz)	10,737		-	-	-	1,456	839	247	790	437	17	-	1	76	1,202	955	449	102	420	607	128	4	7	85	76	30	56	7
Molybdenum Concentrate (kt)	55		-	-	-	7	8	2	3	2	0	-	0	0	9	6	3	1	1	3	1	0	0	0	0	0	0	0
Molybdenum Concentrate Grade (%)	56.00%		0.00%	0.00%	0.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%
Recovered Molybdenum (klbs)	67,502		-	-	-	8,074	9,499	2,742	3,290	2,028	86	-	0	428	10,720	7,921	4,259	966	1,722	3,792	791	1	6	64	103	191	49	0
Hypogene Mineralized Material																												
Beginning Mineralized Material Inventory (kt)	-		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
Mined Mineralized Material to Concentrator (kt)	1,684,158	-	-	-	-	3,401	22,187	38,917	16,462	28,057	41,519	42,026	42,498	39,836	7,810	14,893	27,616	39,031	28,466	25,170	42,088	45,803	45,154	43,472	42,866	44,750	43,633	46,422
Mined Mineralized Material - Processed (kt)	1,684,158	-	-	-	-	3,401	22,187	38,917	16,462	28,057	41,519	42,026	42,498	39,836	7,810	14,893	27,616	39,031	28,466	25,170	42,088	45,803	45,154	43,472	42,866	44,750	43,633	46,422
Ending Mineralized Material Inventory		-				-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
Copper Grade (%)	0.148%		0.000%	0.000%	0.000%	0.270%	0.262%	0.281%	0.278%	0.203%	0.202%	0.203%	0.205%	0.187%	0.196%	0.187%	0.180%	0.185%	0.158%	0.174%	0.146%	0.145%	0.139%	0.150%	0.144%	0.126%	0.113%	0.149%
Molybdenum Grade (%)	0.019%		0.0000%	0.0000%	0.0000%	0.0218%	0.0254%	0.0323%	0.0280%	0.0147%	0.0185%	0.0271%	0.0305%	0.0356%	0.0264%	0.0321%	0.0294%	0.0320%	0.0268%	0.0180%	0.0118%	0.0144%	0.0194%	0.0231%	0.0266%	0.0261%	0.0208%	0.0189%
Gold Grade (g/t)	0.181		-	-	-	0.473	0.354	0.364	0.365	0.270	0.254	0.248	0.237	0.223	0.232	0.207	0.215	0.211	0.170	0.182	0.185	0.172	0.169	0.170	0.175	0.169	0.169	0.192
Silver Grade (g/t)	1.482		-	-	-	2.743	2.040	1.970	2.140	2.040	1.780	2.180	1.680	1.600	2.000	1.850	1.690	1.540	1.850	1.640	1.420	1.720	2.120	1.510	1.440	1.320	1.230	1.580
Contained Copper (klbs)	5,494,437		-	-	-	20,249	128,154	241,090	100,893	125,566	184,898	188,082	192,068	164,229	33,747	61,398	109,589	159,190	99,156	96,553	135,471	146,418	138,371	143,759	136,085	124,307	108,699	152,491
Contained Molybdenum (klbs)	688,290		-	-	-	1,636	12,424	27,712	10,162	9,093	16,934	25,109	28,576	31,265	4,546	10,540	17,900	27,536	16,819	9,988	10,949	14,541	19,312	22,139	25,138	25,749	20,008	19,343
Contained Gold (koz)	9,811		-	-	-	52	253	455	193	244	339	335	324	286	58	99	191	265	156	147	250	253	245	238	241	243	237	287
Contained Silver (koz)	80,239		-	-	-	300	1,455	2,465	1,133	1,840	2,376	2,946	2,295	2,049	502	886	1,501	1,933	1,693	1,327	1,921	2,533	3,078	2,110	1,985	1,899	1,725	2,358
Recovery Copper (%)	92.20%		0.00%	0.00%	0.00%	92.20%	92.20%	92.20%	92.20%	92.20%	92.20%	92.20%	92.20%	92.20%	92.20%	92.20%	92.20%	92.20%	92.20%	92.20%	92.20%	92.20%	92.20%	92.20%	92.20%	92.20%	92.20%	92.20%
Recovery Molybdenum (%)	78.60%		0.00%	0.00%	0.00%	78.60%	78.60%	78.60%	78.60%	78.60%	78.60%	78.60%	78.60%	78.60%	78.60%	78.60%	78.60%	78.60%	78.60%	78.60%	78.60%	78.60%	78.60%	78.60%	78.60%	78.60%	78.60%	78.60%
Recovery Gold (%)	66.00%		0.00%	0.00%	0.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%
Recovery Silver (%)	50.00%		0.00%	0.00%	0.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%
Copper Concentrate (kt)	8,207		-	-	-	30	191	360	151	188	276	281	287	245	50	92	164	238	148	144	202	219	207	215	203	186	162	228
Copper Concentrate Grade (%)	28.00%		0.00%	0.00%	0.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%
Recovered Copper (klbs)	5,065,871		-	-	-	18,670	118,158	222,285	93,023	115,772	170,476	173,412	177,087	151,420	31,115	56,609	101,041	146,773	91,421	89,022	124,904	134,998	127,578	132,546	125,470	114,612	100,221	140,597
Recovered Gold (koz)	6,475		-	-	-	34	167	301	128	161	224	221	214	189	38	65	126	175	103	97	165	167	162	157	159	160	156	189
Recovered Silver (koz)	40,120		-	-	-	150	728	1,232	566	920	1,188	1,473	1,148	1,025	251	443	750	966	847	664	961	1,266	1,539	1,055	992	950	863	1,179
Molybdenum Concentrate (kt)	438		-	-	-	1	8	18	6	6	11	16	18	20	3	7	11	18	11	6	7	9	12	14	16	16	13	12
Molybdenum Concentrate Grade (%)	56.00%		0.00%	0.00%	0.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%
Recovered Molybdenum (klbs)	540,996		-	-	-	1,286	9,765	21,782	7,987	7,147	13,310	19,735	22,461	24,574	3,573	8,284	14,069	21,643	13,220	7,851	8,606	11,429	15,179	17,401	19,758	20,239	15,727	15,203
Total Mill Mineralized Material																												
Beginning Mineralized Material Inventory (kt)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Mined Mineralized Material to Concentrator (kt)	2,127,790	-	-	-	-	34,500	46,882	46,472	45,612	45,478	45,507	45,626	46,361	46,597	44,545	45,785	46,256	46,939	46,084	45,437	45,838	45,881	45,348	45,162	45,410	45,846	46,145	46,579
Mined Mineralized Material - Processed (kt)	2,127,790	-	-	-	-	34,500	46,882	46,472	45,612	45,478	45,507	45,626	46,361	46,597	44,545	45,785	46,256	46,939	46,084	45,437	45,838	45,881	45,348	45,162	45,410	45,846	46,145	46,579
Ending Mineralized Material Inventory	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Copper Grade (%)	0.163%	0.000%	0.000%	0.000%	0.000%	0.355%	0.338%	0.286%	0.280%	0.234%	0.209%	0.207%	0.203%	0.187%	0.229%	0.253%	0.209%	0.192%	0.189%	0.196%	0.150%	0.145%	0.139%	0.148%	0.146%	0.128%	0.111%	0.149%
Molybdenum Grade (%)	0.018%	0.000%	0.000%	0.000%	0.000%	0.022%	0.030%	0.032%	0.019%	0.016%	0.019%	0.027%	0.030%	0.034%	0.028%	0.027%	0.027%	0.030%	0.020%	0.017%	0.012%	0.014%	0.019%	0.022%	0.025%	0.026%	0.020%	0.019%
Gold Grade (g/t)	0.192	-	-	-	-	0.427	0.357	0.358	0.316	0.273	0.269	0.265	0.256	0.247	0.245	0.235	0.236	0.233	0.163	0.185	0.186	0.172	0.169	0.172	0.175	0.169	0.172	0.192

	Total	-4	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23
Silver Grade (g/t)	1.507	-	-	-	-	2.458	1.901	1.951	1.953	1.932	1.799	2.167	1.725	1.698	2.053	1.940	1.742	1.615	1.687	1.603	1.449	1.722	2.119	1.551	1.446	1.323	1.232	1.583
Contained Copper (klbs)	7,654,041	-	-	-	-	269,817	349,015	292,897	281,737	234,636	209,449	207,924	207,913	191,720	225,106	255,268	212,733	198,458	191,658	196,153	152,088	146,468	138,619	147,000	146,460	129,072	113,130	152,650
Contained Molybdenum (klbs)	851,733	-	-	-	-	17,073	30,646	33,069	19,394	15,601	19,492	27,513	31,065	35,272	27,725	27,595	27,729	31,149	20,703	17,254	12,462	14,542	19,324	22,262	25,334	26,115	20,104	19,343
Contained Gold (koz)	13,136	-	-	-	-	474	538	534	463	400	394	389	382	370	351	346	351	352	241	271	274	254	247	249	255	249	256	288
Contained Silver (koz)	103,107	-	-	-	-	2,726	2,866	2,915	2,864	2,824	2,632	3,179	2,571	2,544	2,940	2,855	2,591	2,438	2,500	2,342	2,135	2,540	3,089	2,252	2,111	1,950	1,828	2,370
Recovery Copper (%)	87.94%	0.00%	0.00%	0.00%	0.00%	78.60%	85.47%	90.63%	82.38%	83.42%	88.47%	88.78%	89.77%	89.20%	82.88%	84.19%	86.99%	87.28%	83.42%	87.93%	91.53%	92.20%	92.19%	92.09%	91.41%	91.90%	91.72%	92.18%
Recovery Molybdenum (%)	73.55%	0.00%	0.00%	0.00%	0.00%	54.82%	62.96%	74.34%	66.08%	67.63%	75.15%	76.30%	76.49%	75.61%	56.61%	62.35%	69.28%	75.55%	73.67%	67.53%	75.41%	78.60%	78.58%	78.45%	78.40%	78.23%	78.48%	78.60%
Recovery Gold (%)	66.76%	0.00%	0.00%	0.00%	0.00%	68.67%	67.59%	66.44%	67.75%	67.17%	66.42%	66.41%	66.46%	66.68%	68.50%	68.14%	67.37%	66.74%	67.07%	67.37%	66.26%	66.01%	66.02%	66.14%	66.17%	66.07%	66.22%	66.02%
Recovery Silver (%)	52.22%	0.00%	0.00%	0.00%	0.00%	58.90%	54.92%	51.54%	56.05%	53.48%	50.97%	50.74%	51.07%	51.95%	58.29%	56.90%	54.21%	52.07%	53.23%	54.33%	51.00%	50.03%	50.04%	50.63%	50.60%	50.26%	50.56%	50.05%
Copper Concentrate (kt)	10,904	-	-	-	-	344	483	430	376	317	300	299	302	277	302	348	300	281	259	279	226	219	207	219	217	192	168	228
Copper Concentrate Grade (%)	28.00%	0.00%	0.00%	0.00%	0.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%
Recovered Copper (klbs)	6,731,255	-	-	-	-	212,076	298,318	265,439	232,102	195,745	185,290	184,602	186,649	171,016	186,570	214,898	185,049	173,215	159,874	172,481	139,206	135,041	127,792	135,377	133,883	118,617	103,763	140,718
Recovered Gold (koz)	8,770	-	-	-	-	325	364	355	314	268	261	258	254	247	240	236	237	235	162	183	182	168	163	165	169	164	169	190
Recovered Silver (koz)	53,840	-	-	-	-	1,606	1,574	1,503	1,605	1,511	1,342	1,613	1,313	1,322	1,714	1,624	1,405	1,269	1,331	1,273	1,089	1,270	1,546	1,140	1,068	980	924	1,186
Molybdenum Concentrate (kt)	507	-	-	-	-	8	16	20	10	9	12	17	19	22	13	14	16	19	12	9	8	9	12	14	16	17	13	12
Molybdenum Concentrate Grade (%)	56.00%	0.00%	0.00%	0.00%	0.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%
Recovered Molybdenum (klbs)	626,477	-	-	-	-	9,360	19,296	24,583	12,816	10,551	14,648	20,993	23,762	26,670	15,696	17,204	19,210	23,533	15,251	11,651	9,397	11,430	15,186	17,465	19,861	20,430	15,776	15,204
Heap Leach																												
Beginning Mineralized Material Inventory (kt)	-		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Mined Mineralized Material to Heap Leach (kt)	239,756		5,487	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	946
Mined Mineralized Material - Processed (kt)	239,756		5,487	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	946
Ending Mineralized Material Inventory (kt)	-		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Copper Grade (%)	0.034%		0.021%	0.044%	0.044%	0.037%	0.021%	0.027%	0.042%	0.040%	0.017%	0.009%	0.020%	0.021%	0.033%	0.044%	0.046%	0.044%	0.051%	0.019%	0.023%	0.019%	0.022%	0.029%	0.040%	0.045%	0.042%	0.044%
Gold Grade (g/t)	0.245		0.273	0.436	0.406	0.287	0.256	0.211	0.164	0.176	0.198	0.217	0.244	0.206	0.198	0.199	0.165	0.154	0.183	0.228	0.240	0.245	0.246	0.249	0.335	0.360	0.410	0.436
Silver Grade (g/t)	1.866		1.300	2.680	2.850	1.860	1.470	2.010	1.310	1.469	2.131	2.430	2.290	1.710	1.435	1.570	1.681	1.704	1.711	2.148	1.653	1.428	1.419	1.533	2.288	2.385	2.706	2.680
Contained Copper (klbs)	179,247		2,540	8,852	8,852	7,544	4,225	5,432	8,449	7,995	3,502	1,811	4,023	4,225	6,635	8,852	9,224	8,852	10,277	3,902	4,599	3,884	4,407	5,851	8,043	9,039	8,514	918
Contained Gold (koz)	1,892		48	128	119	84	75	62	48	52	58	64	72	60	58	58	48	45	54	67	70	72	72	73	98	106	120	13
Contained Silver (koz)	14,383		229	786	836	546	431	590	384	431	625	713	672	502	421	461	493	500	502	630	485	419	416	450	671	700	794	82
Recovery Copper (%)	18%		18%	18%	18%	18%	18%	18%	18%	18%	18%	18%	18%	18%	18%	18%	18%	18%	18%	18%	18%	18%	18%	18%	18%	18%	18%	18%
Recovery Gold (%)	70%		70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%
Recovery Silver (%)	26%		26%	26%	26%	26%	26%	26%	26%	26%	26%	26%	26%	26%	26%	26%	26%	26%	26%	26%	26%	26%	26%	26%	26%	26%	26%	26%
Copper Precipitate (kt)	24		0	1	1	1	1	1	1	1	0	0	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	0
Copper Precipitate Grade	60%		60%	60%	60%	60%	60%	60%	60%	60%	60%	60%	60%	60%	60%	60%	60%	60%	60%	60%	60%	60%	60%	60%	60%	60%	60%	60%
Recovered Copper (klbs)	32,264		457	1,593	1,593	1,358	760	978	1,521	1,439	630	326	724	760	1,194	1,593	1,660	1,593	1,850	702	828	699	793	1,053	1,448	1,627	1,533	165
Recovered Gold Dore - Heap Leach (koz)	1,325		34	90	83	59	53	43	34	36	41	45	50	42	41	41	34	32	38	47	49	50	51	51	69	74	84	9
Total Recovered Gold Dore (koz)	1,325		34	90	83	59	53	43	34	36	41	45	50	42	41	41	34	32	38	47	49	50	51	51	69	74	84	9
Recovered Silver Dore (koz)	3,740		60	204	217	142	112	153	100	112	163	185	175	130	109	120	128	130	131	164	126	109	108	117	175	182	206	21
Payable Metals																												
Copper Concentrate																												
Payable Copper (klbs)	6,495,661	-	-	-	-	204,653	287,877	256,149	223,978	188,894	178,805	178,141	180,116	165,030	180,040	207,376	178,572	167,153	154,278	166,444	134,334	130,314	123,319	130,639	129,197	114,466	100,132	135,793
Payable Gold (koz)	8,550	-	-	-	-	317	355	346	306	262	255	252	247	240	234	230	231	229	158	178	177	163	159	161	165	160	165	185
Payable Silver (koz)	51,148	-	-	-	-	1,526	1,495	1,427	1,525	1,435	1,275	1,532	1,247	1,256	1,628	1,543	1,334	1,206	1,264	1,209	1,034	1,207	1,469	1,083	1,014	931	878	1,127
Molybdenum Concentrates																												
Payable Molybdenum (klbs)	532,505	-	-	-	-	7,956	16,401	20,896	10,893	8,968	12,451	17,844	20,198	22,669	13,341	14,623	16,328	20,003	12,963	9,903	7,988	9,715	12,908	14,846	16,882	17,366	13,410	12,923
Gold/Silver Dore																												
Payable Metal Gold (koz)	1,298		33	88	82	58	52	42	33	35	40	44	49	41	40	40	33	31	37	46	48	49	50	50	67	72	82	9
Payable Metal Silver (koz)	3,665		58	200	213	139	110	150	98	110	159	182	171	128	107	117	126	127	128	161	124	107	106	115	171	178	202	21
Copper Precipitate																												
Payable Metal (klbs)	41,410		441	1,538	1,538	1,310	734	943	1,468	1,389	608	314	699	734	1,152	1,538	1,602	1,538	1,785	678	799	675	766	1,016	1,397	1,570	1,479	325
Income Statement (C$000)																												
Copper (C$/lb.)	$4.19		$ 4.19	$4.19	$4.19	$4.19	$4.19	$ 4.19	$ 4.19	$ 4.19	$ 4.19	$ 4.19	$ 4.19	$ 4.19	$ 4.19	$ 4.19	$ 4.19	$ 4.19	$ 4.19	$ 4.19	$4.19	$4.19	$4.19	$4.19	$4.19	$4.19	$4.19	$4.19
Molybdenum (C$/lb)	$15.00		$15.00	$15.00	$15.00	$15.00	$15.00	$15.00	$15.00	$15.00	$15.00	$15.00	$15.00	$15.00	$15.00	$15.00	$15.00	$15.00	$15.00	$15.00	$15.00	$15.00	$ 15.00	$ 15.00	$ 15.00	$ 15.00	$ 15.00	$ 15.00
Gold (C$/oz)	$2,000.00		$2,000.00	$2,000.00	$2,000.00	$2,000.00	$2,000.00	$ 2,000.00	$ 2,000.00	$ 2,000.00	$ 2,000.00	$ 2,000.00	$ 2,000.00	$ 2,000.00	$ 2,000.00	$ 2,000.00	$ 2,000.00	$ 2,000.00	$ 2,000.00	$ 2,000.00	$2,000.00	$2,000.00	$2,000.00	$2,000.00	$2,000.00	$2,000.00	$2,000.00	$2,000.00
Silver (C$/oz)	$30.00		$30.00	$30.00	$30.00	$30.00	$30.00	$30.00	$30.00	$30.00	$30.00	$30.00	$30.00	$30.00	$30.00	$30.00	$30.00	$30.00	$30.00	$30.00	$30.00	$30.00	$ 30.00	$ 30.00	$ 30.00	$ 30.00	$ 30.00	$ 30.00
Revenues																												
Copper Concentrates - Cu	$27,200,582		$-	$ -	$ -	$856,985	$1,205,486	$1,072,623	$937,909	$790,994	$748,747	$745,967	$754,236	$691,065	$753,918	$868,387	$747,772	$699,951	$646,041	$696,986	$562,524	$545,691	$516,399	$547,051	$541,012	$479,326	$419,302	$568,632
Copper Concentrates - Au	$17,100,589		$-	$ -	$ -	$634,682	$709,471	$692,174	$611,455	$523,551	$509,865	$503,520	$494,758	$480,975	$468,748	$459,579	$461,739	$457,863	$315,636	$355,959	$354,463	$326,966	$317,704	$321,511	$329,328	$320,641	$330,193	$370,909
Copper Concentrates - Ag	$ 1,534,445		$-	$ -	$ -	$45,766	$44,863	$42,824	$45,754	$43,051	$38,237	$45,972	$37,416	$37,665	$48,843	$46,298	$40,033	$36,175	$37,919	$36,266	$31,030	$36,209	$44,056	$32,490	$30,434	$27,931	$26,337	$33,814
Molybdenum Concentrates	$ 7,987,582		$-	$ -	$ -	$119,337	$246,019	$313,437	$163,398	$134,521	$186,758	$267,661	$302,969	$340,042	$200,119	$219,349	$244,926	$300,044	$194,448	$148,546	$119,814	$145,730	$193,617	$222,685	$253,228	$260,484	$201,149	$193,849
Dore' - Gold	$ 2,596,354		$66,076	$175,495	$163,420	$115,621	$103,043	$84,930	$66,012	$70,916	$79,647	$87,345	$98,213	$82,917	$79,797	$80,100	$66,453	$62,182	$73,731	$91,744	$96,458	$98,417	$99,025	$100,241	$134,682	$144,843	$164,878	$18,194
Dore' - Silver	$109,946		$1,753	$6,010	$6,391	$4,171	$3,297	$4,508	$2,938	$3,295	$4,779	$5,449	$5,135	$3,835	$3,218	$3,521	$3,770	$3,820	$3,837	$4,817	$3,708	$3,202	$ 3,181	$ 3,438	$ 5,131	$ 5,348	$ 6,069	$623
Copper Precipitate	$173,404		$1,848	$6,438	$6,438	$5,487	$3,073	$3,951	$6,146	$5,815	$2,547	$1,317	$2,927	$3,073	$4,826	$6,438	$6,709	$6,438	$7,475	$2,838	$3,345	$2,825	$ 3,206	$ 4,256	$ 5,850	$ 6,575	$ 6,193	$ 1,359
Total Revenues	$56,702,901		$69,677	$187,943	$176,249	$1,782,049	$2,315,252	$2,214,446	$1,833,611	$1,572,143	$1,570,581	$1,657,231	$1,695,654	$1,639,572	$1,559,468	$1,683,672	$1,571,401	$1,566,473	$1,279,088	$1,337,157	$1,171,342	$1,159,040	$ 1,177,188	$ 1,231,671	$ 1,299,665	$ 1,245,148	$ 1,154,119	$ 1,187,379
Operating Cost																												
Mining	$ 8,281,271		$31,641	$63,215	$111,057	$175,546	$177,250	$189,400	$187,741	$203,978	$204,317	$202,463	$206,014	$211,052	$201,123	$207,974	$201,276	$204,472	$221,313	$224,037	$219,799	$225,521	$237,423	$242,717	$241,126	$245,713	$244,666	$243,058
Concentrator	$12,344,310		$-	$ -	$ -	$210,558	$273,338	$266,569	$268,457	$264,533	$259,588	$260,050	$263,651	$265,552	$264,346	$266,984	$265,956	$266,157	$264,864	$262,344	$259,831	$258,613	$256,053	$255,993	$257,353	$259,053	$250,202	$252,380
Heap Leach - Gold Mineralized Material	$ 1,426,159		$33,024	$54,880	$54,878	$54,875	$54,874	$54,874	$54,875	$54,878	$54,879	$54,871	$54,875	$54,877	$54,880	$54,877	$54,874	$54,876	$54,876	$54,874	$52,791	$52,792	$52,793	$52,794	$52,796	$52,796	$52,796	$ 7,462
General Administration	$ 1,024,884		$8,300	$9,395	$10,988	$20,688	$21,751	$21,979	$21,910	$22,322	$22,306	$22,297	$22,378	$22,496	$22,267	$22,387	$22,247	$22,368	$22,747	$22,789	$22,669	$22,799	$23,111	$23,177	$23,152	$23,268	$23,205	$23,087
Treatment & Refining Charges																												
Copper Concentrates																												
Treatment Charges	$817,835		$-	$ -	$ -	$25,767	$36,245	$32,250	$28,200	$23,783	$22,512	$22,429	$22,677	$20,778	$22,668	$26,110	$22,483	$21,045	$19,424	$20,956	$16,913	$16,407	$15,526	$16,448	$16,267	$14,412	$12,607	$17,097
Copper Refining Charges	$504,844		$-	$ -	$ -	$15,906	$22,374	$19,908	$17,408	$14,681	$13,897	$13,845	$13,999	$12,826	$13,993	$16,117	$13,879	$12,991	$11,991	$12,936	$10,440	$10,128	$ 9,584	$10,153	$10,041	$ 8,896	$ 7,782	$10,554
Gold Refining Charges	$56,213		$-	$ -	$ -	$2,086	$2,332	$2,275	$2,010	$1,721	$1,676	$1,655	$1,626	$1,581	$1,541	$1,511	$1,518	$1,505	$1,038	$1,170	$1,165	$1,075	$ 1,044	$ 1,057	$ 1,083	$ 1,054	$ 1,085	$ 1,219
Silver Refining Charges	$68,915		$-	$ -	$ -	$2,055	$2,015	$1,923	$2,055	$1,934	$1,717	$2,065	$1,680	$1,692	$2,194	$2,079	$1,798	$1,625	$1,703	$1,629	$1,394	$1,626	$ 1,979	$ 1,459	$ 1,367	$ 1,254	$ 1,183	$ 1,519
Transportation	$ 1,711,990		$-	$ -	$ -	$53,938	$75,873	$67,510	$59,031	$49,785	$47,126	$46,951	$47,471	$43,495	$47,451	$54,656	$47,064	$44,055	$40,661	$43,868	$35,405	$34,346	$32,502	$34,431	$34,051	$30,169	$26,391	$35,789
Road Maintenance	$147,544		$-	$ -	$ -	$3,139	$3,139	$3,139	$3,139	$3,139	$3,139	$3,139	$3,139	$3,139	$3,139	$3,139	$3,139	$3,139	$3,139	$3,139	$3,139	$3,139	$ 3,139	$ 3,139	$ 3,139	$ 3,139	$ 3,139	$ 3,139
Molybdenum Concentrate																												
Treatment Charges	$-		$-	$ -	$ -	$ -	$ -	$-	$-	$-	$-	$-	$-	$-	$-	$-	$-	$-	$-	$-	$ -	$ -	$-	$-	$-	$-	$-	$-

	Total	-4	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23
Transportation	$-		$-	$ -	$ -	$ -	$ -	$-	$-	$-	$-	$-	$-	$-	$-	$-	$-	$-	$-	$-	$ -	$ -	$-	$-	$-	$-	$-	$-
Gold Dore																												
Gold Refining Charges	$1,722		$44	$116	$108	$ 77	$ 68	$56	$44	$47	$53	$58	$65	$55	$53	$53	$44	$41	$49	$61	$ 64	$ 65	$66	$66	$89	$96	$109	$12
Silver Refining Charges	$1,122		$18	$ 61	$ 65	$ 43	$ 34	$46	$30	$34	$49	$56	$52	$39	$33	$36	$38	$39	$39	$49	$ 38	$ 33	$32	$35	$52	$55	$62	$6
Transportation	$1,575		$29	$ 91	$ 94	$ 62	$ 51	$61	$42	$46	$63	$72	$70	$54	$47	$50	$50	$50	$52	$66	$ 55	$ 49	$49	$52	$76	$80	$90	$9
Copper Precipitate																												
Treatment Charges	$1,829		$26	$ 90	$ 90	$ 77	$ 43	$55	$86	$82	$36	$18	$41	$43	$68	$90	$94	$90	$105	$40	$ 47	$ 40	$45	$60	$82	$92	$87	$9
Refining Charges	$2,420		$34	$119	$119	$102	$ 57	$73	$114	$108	$47	$24	$54	$57	$90	$119	$125	$119	$139	$53	$ 62	$ 52	$59	$79	$109	$122	$115	$12
Transportation	$5,445		$59	$207	$207	$176	$ 99	$127	$197	$187	$82	$42	$94	$99	$155	$207	$215	$207	$240	$91	$107	$ 91	$103	$137	$188	$211	$199	$42
Total Cash Operating Cost	$26,398,079		73,175	128,175	177,607	565,095	669,543	660,247	645,340	641,256	631,487	630,035	637,888	637,835	634,047	656,389	634,802	632,780	642,380	648,101	623,918	626,776	633,511	641,799	640,970	640,409	623,718	595,395
Royalty	$ 1,387,645		$1,910	$4,992	$4,828	$43,875	$56,067	$54,032	$45,185	$39,336	$39,410	$40,979	$40,860	$39,699	$37,537	$40,285	$37,886	$37,906	$31,153	$32,443	$28,777	$28,573	$29,119	$30,363	$32,060	$30,934	$28,871	$29,087
Property Tax	$5,000		$100	$100	$100	$100	$100	$100	$100	$100	$100	$100	$100	$100	$100	$100	$100	$100	$100	$100	$100	$100	$100	$100	$100	$100	$100	$100
Carbon Tax	$ 3,385,972		$1,261	$3,911	$8,119	$13,728	$20,737	$29,273	$39,101	$50,329	$62,959	$76,989	$76,989	$76,989	$76,989	$76,989	$76,989	$76,989	$76,989	$76,989	$76,989	$76,989	$76,989	$76,989	$76,989	$76,989	$76,989	$76,989
Salvage Value	$(31,491)		$-	$ -	$ -	$ -	$ -	$-	$-	$-	$-	$-	$-	$-	$-	$-	$-	$-	$-	$-	$ -	$ -	$-	$-	$-	$-	$-	$-
Reclamation & Closure	$250,000		$-	$ -	$ -	$ -	$ -	$-	$-	$-	$-	$-	$-	$-	$-	$-	$-	$-	$-	$-	$ -	$ -	$-	$-	$-	$-	$-	$-
Total Cash Production Cost	$31,395,204		$76,447	$137,177	$190,654	$622,797	$746,447	$743,652	$729,725	$731,021	$733,955	$748,104	$755,838	$754,622	$748,673	$773,764	$749,777	$747,776	$750,623	$757,633	$729,785	$732,439	$739,719	$749,251	$750,119	$748,433	$729,678	$701,572
Operating Income	$25,307,697	$-	$(6,770)	$50,766	$(14,404)	$1,159,252	$1,568,804	$1,470,794	$1,103,886	$841,122	$836,626	$909,127	$939,816	$884,949	$810,795	$909,908	$821,624	$818,697	$528,465	$579,524	$441,557	$426,602	$437,469	$482,420	$549,546	$496,714	$424,441	$485,807
Yukon Mining Royalty	$ 3,069,178		$-	$5,740	$ -	$86,313	$137,261	$125,356	$81,318	$49,348	$50,243	$79,855	$121,993	$115,269	$106,193	$117,723	$106,416	$106,288	$70,801	$76,494	$59,202	$56,107	$57,322	$63,578	$70,457	$63,817	$54,567	$63,384
Net Income before Depreciation	$22,238,519	$-	$(6,770)	$45,026	$(14,404)	$1,072,939	$1,431,544	$1,345,438	$1,022,567	$791,774	$786,382	$829,273	$817,823	$769,681	$704,603	$792,186	$715,208	$712,409	$457,664	$503,031	$382,355	$370,494	$380,147	$418,842	$479,089	$432,897	$369,874	$422,423
Capital Cost Depreciation	$ 1,559,566		$(6,770)	$45,026	$(14,404)	$808,003	$541,384	$186,328	$-	$-	$-	$-	$-	$-	$-	$-	$-	$-	$-	$-	$ -	$ -	$-	$-	$-	$-	$-	$-
Additional Deduction (Class 41A Assets)	$ 1,696,296		$-	$ -	$ -	$258,470	$878,843	$558,983	$-	$-	$-	$-	$-	$-	$-	$-	$-	$-	$-	$-	$ -	$ -	$-	$-	$-	$-	$-	$-
Sustaining Capital Depreciation	$ 1,807,793				$ -	$6,466	$11,316	$21,318	$17,773	$28,375	$22,531	$27,226	$24,259	$26,835	$30,695	$34,463	$47,380	$31,261	$32,260	$32,519	$39,873	$58,036	$64,829	$50,015	$53,235	$51,252	$57,872	$72,854
Total Depreciation	$ 5,063,656		$(6,770)	$45,026	$(14,404)	$1,072,939	$1,431,544	$766,628	$17,773	$28,375	$22,531	$27,226	$24,259	$26,835	$30,695	$34,463	$47,380	$31,261	$32,260	$32,519	$39,873	$58,036	$64,829	$50,015	$53,235	$51,252	$57,872	$72,854
Net Income After Depreciation	$17,174,863		-	-	-	-	-	578,810	1,004,794	763,399	763,851	802,047	793,564	742,846	673,908	757,723	667,828	681,148	425,405	470,511	342,482	312,459	315,318	368,828	425,854	381,645	312,003	349,568
Tax Loss Carry Forward Applied	$(30,000)		-	-	-	-	-	(30,000)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Net Income After Tax Loss Carry Forward	$17,144,863		-	-	-	-	-	548,810	1,004,794	763,399	763,851	802,047	793,564	742,846	673,908	757,723	667,828	681,148	425,405	470,511	342,482	312,459	315,318	368,828	425,854	381,645	312,003	349,568
Taxable Income	$17,144,863		$-	$ -	$ -	$ -	$ -	$548,810	$1,004,794	$763,399	$763,851	$802,047	$793,564	$742,846	$673,908	$757,723	$667,828	$681,148	$425,405	$470,511	$342,482	$312,459	$315,318	$368,828	$425,854	$381,645	$312,003	$349,568
Taxes at 30%	$ 5,206,595		-	-	-	-	-	164,643	301,438	229,020	229,155	240,614	238,069	222,854	202,172	227,317	200,348	204,344	127,621	141,153	102,745	93,738	94,595	110,648	127,756	114,494	93,601	104,871
Net Income After Taxes	$11,968,269		-	-	-	-	-	414,167	703,356	534,379	534,696	561,433	555,495	519,992	471,736	530,406	467,479	476,804	297,783	329,358	239,738	218,721	220,722	258,179	298,098	267,152	218,402	244,698
Cash Flow																												
Net Income before Depreciation plus Tax Loss Carry Forward	$22,238,519		$(6,770)	$45,026	$(14,404)	$1,072,939	$1,431,544	$1,345,438	$1,022,567	$791,774	$786,382	$829,273	$817,823	$769,681	$704,603	$792,186	$715,208	$712,409	$457,664	$503,031	$382,355	$370,494	$380,147	$418,842	$479,089	$432,897	$369,874	$422,423
Working Capital																												
Account Receivable	$-		$(11,454)	$(19,441)	$1,922	$ (263,967)	$(87,650)	$16,571	$62,603	$42,981	$257	$(14,244)	$(6,316)	$9,219	$13,168	$(20,417)	$18,456	$810	$47,241	$(9,546)	$27,257	$2,022	$(2,983)	$(8,956)	$(11,177)	$ 8,962	$14,964	$(5,467)
Accounts Payable	$-		$6,014	$4,521	$4,063	$31,848	$8,585	$ (764)	$(1,225)	$ (336)	$ (803)	$ (119)	$645	$(4)	$ (311)	$1,836	$(1,774)	$ (166)	$789	$470	$(1,988)	$235	$554	$681	$(68)	$(46)	$(1,372)	$(2,328)
Inventory - Parts, Supplies	$-		$(500)	$ -	$(5,800)	$ -	$ -	$-	$500	$-	$-	$-	$-	$-	$-	$-	$-	$-	$-	$-	$ -	$ -	$-	$-	$-	$-	$-	$-
Total Working Capital	$-		$(5,939)	$(14,921)	$185	$ (232,119)	$(79,065)	$15,807	$61,878	$42,645	$ (546)	$(14,363)	$(5,671)	$9,215	$12,856	$(18,581)	$16,681	$644	$48,030	$(9,075)	$25,270	$2,257	$(2,430)	$(8,275)	$(11,245)	$ 8,916	$13,592	$(7,795)
Capital Expenditures																												
Initial Capital																												
Mine	$625,041	$-	$88,697	$154,186	$217,879	$164,280	$ -	$-	$-	$-	$-	$-	$-	$-	$-	$-	$-	$-	$-	$-	$ -	$ -	$-	$-	$-	$-	$-	$-
Concentrator	$ 2,578,536	$87,405	$ 758,625	$1,040,491	$652,340	$39,676	$ -	$-	$-	$-	$-	$-	$-	$-	$-	$-	$-	$-	$-	$-	$ -	$ -	$-	$-	$-	$-	$-	$-
Heap Leach	$-	$-	$-	$ -	$ -	$ -	$ -	$-	$-	$-	$-	$-	$-	$-	$-	$-	$-	$-	$-	$-	$ -	$ -	$-	$-	$-	$-	$-	$-
Owners Cost	$52,285	$ 2,091	$14,117	$21,960	$11,503	$2,614	$ -	$-	$-	$-	$-	$-	$-	$-	$-	$-	$-	$-	$-	$-	$ -	$ -	$-	$-	$-	$-	$-	$-
Sustaining Capital																												
Mine	$676,539		$-	$ -	$ -	$ -	$ -	$22,548	$-	$32,068	$-	$-	$-	$29,563	$7,546	$18,291	$59,891	$-	$-	$22,548	$45,095	$108,518	$67,643	$-	$61,590	$18,291	$49,197	$45,095
Tailing Filter & Core Zone	$658,306		$-	$ -	$ -	$20,865	$20,865	$-	$2,138	$23,112	$-	$-	$-	$-	$19,533	$-	$-	$-	$-	$20,290	$ -	$ -	$-	$-	$20,290	$-	$12,673	$66,803
Process Plant (includes Heap Leach)	$472,948		$-	$ -	$ -	$5,000	$5,000	$28,775	$5,000	$5,000	$5,000	$41,310	$5,000	$5,000	$5,000	$45,171	$5,000	$5,000	$20,717	$5,000	$5,000	$5,000	$ 5,000	$ 5,000	$ 5,000	$20,717	$ 5,000	$ 5,000
Total Capital Expenditures	$ 5,063,656	$89,496	$ 861,439	$1,216,636	$881,721	$232,436	$25,865	$51,323	$7,138	$60,181	$5,000	$41,310	$5,000	$34,563	$32,080	$63,462	$64,891	$5,000	$20,717	$47,838	$50,095	$113,518	$72,643	$ 5,000	$86,880	$39,008	$66,870	$116,898
Cash Flow before Taxes	$17,174,863	$(89,496)	$(874,148)	$(1,186,531)	$ (895,940)	$608,384	$1,326,613	$1,309,922	$1,077,307	$774,238	$780,836	$773,600	$807,152	$744,333	$685,379	$710,143	$666,998	$708,053	$484,978	$446,117	$357,530	$259,234	$305,074	$405,567	$380,964	$402,805	$316,596	$297,729
Cumulative Cash Flow before Taxes		$(89,496)	$(963,644)	$(2,150,175)	$(3,046,115)	$(2,437,730)	$(1,111,117)	$198,805	$1,276,112	$2,050,350	$2,831,186	$3,604,786	$4,411,938	$5,156,271	$5,841,650	$6,551,793	$7,218,791	$7,926,844	$8,411,822	$8,857,939	$9,215,469	$9,474,702	$ 9,779,777	$10,185,344	$10,566,308	$10,969,113	$11,285,708	$11,583,438
						1.0	1.0	0.8	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Taxes																												
Income Taxes	$ 5,206,595	$-	$-	$ -	$ -	$ -	$ -	$164,643	$301,438	$229,020	$229,155	$240,614	$238,069	$222,854	$202,172	$227,317	$200,348	$204,344	$127,621	$141,153	$102,745	$93,738	$94,595	$110,648	$127,756	$114,494	$93,601	$104,871
Cash Flow after Taxes	$11,968,269	$(89,496)	$(874,148)	$(1,186,531)	$ (895,940)	$608,384	$1,326,613	$1,145,279	$775,869	$545,219	$551,681	$532,986	$569,083	$521,479	$483,207	$482,826	$466,650	$503,708	$357,356	$304,964	$254,785	$165,496	$210,479	$294,919	$253,208	$288,311	$222,995	$192,859
Cumulative Cash Flow after Taxes		$(89,496)	$(963,644)	$(2,150,175)	$(3,046,115)	$(2,437,730)	$(1,111,117)	$34,162	$810,031	$1,355,249	$1,906,930	$2,439,916	$3,008,999	$3,530,477	$4,013,684	$4,496,510	$4,963,160	$5,466,869	$5,824,225	$6,129,189	$6,383,974	$6,549,470	$ 6,759,949	$ 7,054,868	$ 7,308,076	$ 7,596,387	$ 7,819,382	$ 8,012,241
						1.0	1.0	1.0	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Economic Indicators before Taxes																												
NPV @ 0%	0%	$17,174,863																										
NPV @ 5%	5%	$ 6,236,632																										
NPV @ 8%	8%	$ 3,700,156																										
NPV @ 10%	10%	$ 2,640,283																										
IRR		23.1%																										
Payback	Years	2.8																										
Economic Indicators after Taxes																												
NPV @ 0%	0%	$11,968,269																										
NPV @ 5%	5%	$ 4,198,445																										
NPV @ 8%	8%	$ 2,383,947																										
NPV @ 10%	10%	$ 1,623,726																										
IRR		19.3%																										
Payback	Years	3.0																										
NSR $/Sulphide Mineralized Material		$ 23.81																										

25 INTERPRETATION AND CONCLUSIONS

The economic results of the Study demonstrate that the project has positive economics and warrants development. Standard industry practices, equipment and processes were used in this study. The authors of this report are not aware of any unusual or significant risks, or uncertainties that could affect the reliability or confidence in the project based on the data and information made available.

25.1 MINING AND PROCESSING

This study has identified 1.1 billion tonnes of mineral resource at 0.20% copper, 0.23 g/t gold, 0.022% moly, and 1.70 g/t silver that is amenable to conventional milling and sulphide floatation. There are also 204 Mt of mineral resource at 0.26 g/t gold, 1.95 g/t silver, and 0.034% copper that is amenable to conventional heap leaching. The mineral resource amenable to milling and the mineral resource amenable to heap leaching are all classified as measured and indicated mineral resources. The project is based on conventional open pit mining and typical, well understood, processing methods.

The above quantities of resource are constrained by the current engineering design of the tailings management facility (TMF). There is potential to expand the TMF capacity by constructing an additional embankment south (downstream) of the current design. With this larger facility, the quantity of mineral resource amenable to milling is 2.1 billion tonnes at 0.16% copper, 0.19 g/t gold, 0.018% moly, and 1.51 g/t silver. The quantity of mineral resource amenable to heap leaching is 240 Mt at 0.24 g/t gold, 1.87 g/t silver, and 0.034% copper. Some of this additional resource is inferred mineral resource and additional exploration will be required to convert it to indicated mineral resource. Additional engineering and permitting work will be required for the expanded TMF.

The economic assessment of the proposed Phase II expansion is preliminary in nature and includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the preliminary economic assessment will be realized.

A mine plan was developed to supply mill material to a conventional copper sulphide flotation plant with the capacity to process mill material at a nominal rate of 120,000 t/d, or 43.8 Mt/y. Actual annual throughput will vary depending on the mill material hardness encountered during the period. The mine is scheduled to operate two 12 hour shifts per day, 365 days per year.

Both sulphide copper-molybdenum mill material and oxide gold leach material will be processed. Copper-molybdenum mill material will be transported from the mine to the concentrator facility and oxide gold leach material will be transported from the mine to a crushing facility ahead of a heap leaching facility and a gold recovery facility.

Copper-molybdenum mill material will be processed by crushing, grinding, and flotation to produce copper and molybdenum sulphide mineral concentrates. Copper concentrate will be loaded into highway haul trucks and transported to the Port of Skagway for ocean shipment to market. Molybdenum concentrate will be bagged and loaded onto highway haul trucks for shipment to market.

Oxide gold mill material will be leached with an aqueous leach solution. Gold in the enriched (or pregnant) leach solution will be recovered using carbon absorption technology to produce gold doré bars. The enriched leach solution will also be treated to recover copper and cyanide and produce a copper sulphide precipitate. The copper sulphide precipitate will be bagged and loaded onto highway haul trucks for shipment to market.

25.2 INFRASTRUCTURE

25.2.1 Tailings Management Facility

- The TMF will store approximately 712 Mt of tailings and 500 Mt of potentially reactive waste rock and overburden materials. The TMF will be constructed using a combination of local borrow and cyclone underflow sand produced from NAG tailings. A total of approximately 415 Mt of NAG tailings will be used for dam construction.

- The TMF will be constructed in stages. Centreline raises of the TMF dam will be completed to a final crest elevation of El. 981 m with a maximum height of approximately 280 m (crest to toe).

- Mill process water for ongoing operations and water for cyclone plant operations will be reclaimed from the TMF supernatant pond and the Yukon River fresh water supply system.

- Seepage and runoff from the TMF embankments will be collected by a Water Management System, which consists of underdrains and surface ditches that both discharge into a water management pond located downstream of the Main Embankment. Seepage and runoff collected in the pond will be pumped to the process plant using the reclaim water system during operations.

25.3 RISKS

25.3.1 Infrastructure

25.3.1.1 Tailings Management Facility

- Geotechnical Foundation Conditions – There is a risk that the extent or depth of permafrost conditions or other unsuitable materials in the foundations may be greater than currently anticipated, requiring more removal of material prior to construction of the TMF embankments. This risk can be mitigated through additional geotechnical investigations.

- Geotechnical and Hydrogeological Performance of the Cyclone Sand – There is a risk that cyclone sand within the main embankment does not meet the design criteria. This risk will be mitigated through advanced laboratory testing to determine the material parameters under high stresses and assess the influence of variability in the particle size distribution.

- Main Embankment Raises – There is a risk that the main embankment cannot be raised quickly enough using the assumed construction methods, especially in the early years of operation when the rate of rise is high. This risk will be mitigated by more detailed staging of the raises and contingency plans for raising the dam using screened sand stockpiled in the winter months.

25.3.1.2 Heap Leach Facility

Potential risks associated with the Heap Leach Facility include:

- Geotechnical Foundation Conditions – There is a risk that the extent or depth of permafrost conditions or other unsuitable materials in the foundations may be greater than currently anticipated, requiring more removal of material prior to constructing the HLF pad and composite liner systems. This risk can be mitigated through additional geotechnical investigations and site characterization.

- HLF Stability – There is a risk that the interface angle between the crushed mineralized material and the LLDPE liner in the composite liner system may result in instability of the HLF stack. This risk can be mitigated through shear interface testing with representative samples of the mineralized material and the liner.

- Crushed Mineralized Material Overliner – There is a risk that the crushed mineralized material may not be suitable as an overliner material. This risk can be mitigated by screening of the crushed mineralized material to remove finer particles, or by confirming the particle size distribution of the crushed mineralized material through testing of representative samples.

- Overliner Placement – There is a risk that the use of conveyors to place the overliner material may not be suitable. This risk can be mitigated through additional detailed sequencing. An alternate method may be to use haul trucks to place the overliner material, prior to conveyor stacking of the mineralized material in the HLF.

25.4 OPPORTUNITIES

25.4.1 Infrastructure

25.4.1.1 Tailings Management Facility

- Potential to use waste rock in Starter Dam construction and reduce the reliance on a dedicated rock quarry for initial construction.

- Potential to optimize main embankment geometry if further geotechnical investigation and characterization identify more favourable conditions than assumed.

25.4.1.2 Heap Leach Facility

Potential opportunities associated with the Heap Leach Facility include:

- Potential to add additional Heap Leach material to increase recovery of leached materials.

- Potential for sterile leach material (i.e., mineralized material in the HLF after irrigation) to be used to construct a closure cover for the TMF.

- Potential to remove downstream Events Pond if sufficient capacity can be maintained in the in-heap water management pond for storm storage and collection of pregnant solution.

26 RECOMMENDATIONS

Based on the results of this study, it is recommended that the Phase I development case be advanced to a Feasibility Study to establish a mineral reserve for the project. Concurrent with the later stages of the Feasibility Study, an application for environmental assessment under the Yukon Environmental and Socioeconomic Assessment Act should be prepared to continue the permitting process.

26.1 FACILITIES

26.1.1 Tailings Management Facility

The following is a list of recommendations for additional site investigations, testwork and design studies for the TMF:

- Additional drilling and sampling of the overburden materials is recommended in the area of the TMF to better define the composition and expected variability in material depths, particularly within the footprint of the TMF starter dam.
- Additional laboratory testing to confirm the physical characteristics of the cyclone sand (material classification, strength and permeability tests) is recommended. This should include strength and permeability tests at very high confining stresses, representative of the height of the Main Embankment, and testing to examine the influence of cyclone sand fines content on available percentage of sand recovery and the impact on permeability.
- Investigate the feasibility of using waste rock to construct the TMF starter dam.
- Laboratory testing to determine the physical characteristics of the PAG tailings (material classification, slurry settling, consolidation and permeability tests).
- Development of a detailed tailings and waste rock deposition strategy to optimize material handling, and tailings discharge line and reclaim barge locations throughout operations.
- Assessment of the sequencing of sand cell construction in relation to embankment raising requirements to optimise material demand and placement strategies.

26.1.2 Heap Leach Facility

The following is a list of recommendations for additional site investigations, testwork and design studies required to carry the HLF through to final design and construction:

- Geotechnical Investigations and Testwork:
    - Additional test pits/drill holes to confirm assumptions and characterization of foundation and permafrost conditions.
    - Additional test pits/drill holes to assess suitability, availability and quantity of borrow materials for earthworks construction.
    - Geotechnical laboratory testing of potential borrow materials for pad foundation (low permeability soil layer) and confining embankment or events pond dam construction (including particle size distribution, Atterberg limits, specific gravity, moisture-density relationship, permeability and shear strength tests).
    - Laboratory direct shear testing of liner interfaces, to determine the interface shear strength relationships for heap stability assessment. It is recommended that the shear strength tests are carried out for each of the liner interfaces within the composite liner systems once the potential material sources have been confirmed.
    - Mineralized Material testing (including particle size distribution, specific gravity, permeability under load, load-percolation, and direct shear/triaxial shear strength tests).

- Design Studies and Analyses:
  - Leach testing to determine optimal mineralized material densities, leaching rates and resulting in-heap moisture contents.
  - Detailed water balance analyses, based on results of hydrology and leach testing, to estimate solution storage area water volumes, peak storm flows for ditch design, and to confirm sizing of pump and pipework systems.
  - Heap stability assessment based on results of laboratory shear strength and liner interface strength testing.
  - Seepage analyses to predict seepage flow patterns and solution losses for the design of the LDRS.
  - Feasibility design of all civil and mechanical works, including sumps, intakes, outlets, pumps, pipe systems etc.
  - Advanced studies on the staging and sequencing of mineralized material placement and development of the HLF.

26.1.3 Additional Facilities

The following is a list of recommendations for additional site investigations, testwork and design studies required to carry the additional facilities to final design and construction:

- Additional drilling and sampling of the overburden materials is recommended in the area of the Processing facility to better define the composition and expected variability in material depths.
- Additional drilling and sampling of the overburden materials is recommended in the area of the proposed new airstrip to better define the composition and expected variability in material depths.

26.2 EXPLORATION

The current 2021 program by Casino Mining Corporation comprises metallurgical (MET) drilling utilizing PQ-size core, resource confirmation drilling and a suite of 12 exploration core drilling holes focusing on proximal areas of the Casino deposit and several geochemical and geophysical targets on the Canadian Creek block.

A grid-style soil geochemical sampling program covering the Casino claims outside of the main deposit area was initiated in 2021, and is recommended to continue until completion. Prospective areas determined from geochemical results are recommended for follow-up exploration.

26.3 FEASIBILITY STUDY

It is recommended that the current engineering be brought to the level of a Feasibility Study.

26.4 BUDGET AND SCHEDULE

Investigation of the TMF, HLF and additional facilities and the exploration and metallurgical program would cost approximately $8-$12 million. Most of the work will be completed in 2021, but one additional field season may be required to complete the work.

The budget for preparation of a Feasibility Study is $4 million, and will take approximately 9-12 months to complete.

27 REFERENCES

Bower, B., Case, T., DeLong, C., and Rebagliati, C.M., 1995: Casino Project 1994 Exploration and Geotechnical Drilling Program on the Casino Property Copper-Gold-Molybdenum Deposit, Unpublished Company Report, Pacific Sentinel Gold Corp., May, 1995.

Bower, B., DeLong, C., Payne, J., and Rebagliati, C.M., 1995a: The Oxide Gold, Supergene and Hypogene Zones at the Casino Property Copper-Gold-Molybdenum Deposit, West-Central Yukon, Special Volume 46, Canadian Institute of Mining, Metallurgy and petroleum, 1995.

Canadian Dam Association (CDA). 2013. 2007 Dam Safety Guidelines – 2013 Revision. Toronto, ON.

Canadian Dam Association (CDA). 2019. Technical Bulletin: Application of Dam Safety Guidelines to Mining Dams (2019 Edition). Toronto, ON.

FLSmidth. 2012. Casino Project Circuit Design Basis – Update on Test work and Mill Sizing/Selection. 2012.

Hart, C.J.R., and Selby, D., 1997: "The Pattison Creek Pluton – a Mineralized Casino Intrusion made bigger with gamma rays". In: Yukon Exploration and Geology 1997, Exploration and Geological Services Division, Yukon. Indian and Northern Affairs Canada, pp. 89-96, 1998

Johnston, R.L., 2018: "Summary report of the 2017 Exploration Programme on the Canadian Creek Property, Whitehorse Mining District, Yukon Territory".

Johnston, S.T., 1995: "Geological compilation with interpretation from geophysical surveys on the northern Dawson Range, central Yukon (115J/9 and 10; 115I/12) (1:100,000 scale map). Exploration and Geological Services Division, Yukon. Indian and Northern Affairs Canada, Open File 1995-2 (G), 1995.

Knight Piésold Ltd. (KP). 2012. Open Pit Geotechnical Design. October 12, 2012.

Knight Piésold Ltd. (KP). 2012a. Casino Copper-Gold Project – Feasibility Design of the Heap Leach Facility. Ref. No. VA101-325/8-9. Rev.0. December 18. Vancouver, BC.

Knight Piésold Ltd. (KP). 2012b. Casino Copper-Gold Project – 2011 Geotechnical Site Investigation Data Report – Waste Management Facilities. Ref. No. VA101-325/8-5. Rev.0. December 21. Vancouver, BC.

Knight Piésold Ltd. (KP). 2012c. Casino Copper-Gold Project – Report on Feasibility Design of the Tailings Management Facility. Ref. No. VA101-325/8/10. Rev 0. December 20, 2012. Vancouver, BC.Knight Piésold Ltd. (KP). 2013. Casino Project – Baseline Climate report. Ref. No. VA101-325/14-7. Rev.0. June 14. Vancouver, BC.

Knight Piésold Ltd. (KP). 2014. Casino Project – Geotechnical Laboratory Testing of Leach Ore. Ref. No. VA101-325/16-2. Rev.0. February 17. Vancouver, BC.

Knight Piésold Ltd. (KP). 2015. Casino Project – 2014 and 2015 Geotechnical Testing of Leach Ore. Ref. No. VA101-325/18-1. Rev.0. October 30. Vancouver, BC.

Payne, J.G., Gonzalez, R.A., Akhurst, K., and Sisson, W.G., 1987: "Geology of the Colorado Creek (115J/10), Selwyn River (115J/9), and Prospector Mountain (115I/5) Map areas, Western Dawson range, west-central Yukon. Exploration and Geological Services Division, Yukon. Indian and Northern Affairs Canada, Open File 1987-3, 1987.

Schulze, C., 2021, Aurora Geosciences Ltd: 2021 Draft report, Casino Deposit Status and Recommendations, In-house report for Western Copper and Gold Corp on the Casino Project. May 17, 2021

Selby, D, and Creaser, R.A., 2001: Late and Mid Cretaceous mineralization in the Northern Cordillera: Constraints from Re-Os molybdenum dates. Economic Geology 96, pp 1461-1467.

Templeman-Kluit, D.J., 1974: Reconnaissance geology of Aishihik lake, Snag and part of Stewart River map-areas, west-central Yukon. Department of Energy, Mines and Resources; Geological Survey of Canada Paper 73-41, 97 p.

Williams, W., 2021: Memo Re: Canadian Creek Exploration – Summary. In-house report for the Board of Directors, Western Copper and Gold Corp. Jan 29, 2021

Williams, W., 2021: Memo Re: Casino Cu & Au Porphyry Deposit – Observations. In-house report for the Board of Directors, Western Copper and Gold Corp. Jan 27, 2021

APPENDIX A – PRELIMINARY ECONOMIC ASSESSMENT (PEA) STUDY CONTRIBUTORS AND PROFESSIONAL QUALIFICATIONS

CERTIFICATE OF QUALIFIED PERSON

I, Daniel Roth, P.E., P. Eng. do hereby certify that:

1. I am currently employed as a project manager and civil engineer at M3 Engineering & Technology Corp. located at 2051 West Sunset Rd, Suite 101, Tucson, AZ 85704.
2. I graduated with a Bachelor of Science degree in Civil Engineering from The University of Manitoba in 1990.
3. I am a registered professional engineer in good standing in the following jurisdictions:
    - British Columbia, Canada (No. 38037)
    - Alberta, Canada (No. 62310)
    - Ontario, Canada (No. 100156213)
    - Yukon, Canada (No. 1998)
    - New Mexico, USA (No. 17342)
    - Arizona, USA (No. 37319)
    - Alaska, USA (No. 102317)
    - Minnesota, USA (No. 54138)
4. I have worked continuously as a design engineer, engineering and project manager since 1990, a period of 30 years. I have worked in the minerals industry as a project manager for M3 Engineering & Technology Corporation since 2003, with extensive experience in hard rock mine process plant and infrastructure design and construction, environmental permitting review, as well as development of capital cost estimates, operating cost estimates, financial analyses, preliminary economic assessments, pre-feasibility and feasibility studies.
5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
6. I am responsible for Sections 2, 3, 4, 5, 15, 18 (except 18.5 and 18.6), 19, 20, 21 (except 21.1.5, 21.3.1 and 21.3.3), 22, 24 (except 24.2) and corresponding sections of 1, 25, 26 and 27 of the technical report titled "Western Copper and Gold Corporation, Casino Project, Form 43-101F1 Technical Report, Preliminary Economic Assessment, Yukon, Canada" dated effective June 22, 2021 (the "Technical Report").
7. I have prior involvement with the property that is the subject of the Technical Report. I have developed various capital and operating cost tradeoff studies for Western Copper and Gold Corporation ("Western") from 2014 through 2020. I have not visited the project site.
8. As of the date of this certificate, 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.
9. I am independent of Western and its subsidiaries as defined by Section 1.5 of NI 43-101.
10. I have read NI 43-101 and Form 43-101F1. The sections of the Technical Report that I am responsible for have been prepared in compliance with that instrument and form.
11. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their website accessible by the public, of the Technical Report.

Dated this 2nd day of August, 2021.

"Signed"
Signature of Qualified Person

Daniel Roth
Print Name of Qualified Person

CERTIFICATE OF QUALIFIED PERSON

I, Michael G. Hester, do hereby certify that:

1. I am currently employed as Vice President and Principal Mining Engineer by Independent Mining Consultants, Inc. (“IMC”) of 3560 E. Gas Road, Tucson, Arizona, 84714, USA.
2. I graduated with a Bachelor of Science degree in Mining Engineering from the University of Arizona in 1979 and a Master of Science degree in Mining Engineering from the University of Arizona in 1982.
3. I am a Fellow of the Australasian Institute of Mining and Metallurgy (FAusIMM #221108), a professional association as defined by National Instrument 43-101 – Standards of Disclosure for Mineral Projects (“NI 43-101”).
4. I have worked in the minerals industry as an engineer continuously since 1979, a period of 42 years. I am a founding partner, Vice President, and Principal Mining Engineer for IMC, a position I have held since 1983. I have been employed as an Adjunct Lecturer at the University of Arizona (1997-1998) where I taught classes in open pit mine planning and mine economic analysis. I have also been a member of the Resources and Reserves Committee of the Society of Mining, Metallurgy, and Exploration since March 2012. During my career I have had extensive experience developing mineral resource models, developing open pit mine plans, estimating equipment requirements for open pit mining operations, developing mine capital and operating cost estimates, performing economic analysis of mining operations and managing various preliminary economic assessments, pre-feasibility, and feasibility studies.
5. I have read the definition of “qualified person” set out in NI 43-101 and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a “qualified person” for the purposes of NI 43-101.
6. I am responsible for Section 14 and corresponding sections of 1, 25, 26 and 27 of the technical report titled “Western Copper and Gold Corporation, Casino Project, Form 43-101F1 Technical Report, Preliminary Economic Assessment, Yukon, Canada” dated effective June 22, 2021 (the “Technical Report”).
7. I have prior involvement with the property that is the subject of the Technical Report. I worked on the feasibility study for Western Copper and Gold Corporation (“Western”) in or about January 2013. I also worked on preliminary feasibility studies conducted by Western in or around April 2011 and August 2008. I also worked on studies of the property for Pacific Sentinel Corporation in or around September 1995. I most recently inspected the property on July 22, 2008 for a period of one day.
8. As of the date of this certificate, 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.
9. I am independent of Western and its subsidiaries as defined by Section 1.5 of NI 43-101.
10. I have read NI 43-101 and Form 43-101F1. The sections of the Technical Report that I am responsible for have been prepared in compliance with that instrument and form.
11. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their website accessible by the public, of the Technical Report.

Dated this 2nd day of August, 2021.

“Signed”
Signature of Qualified Person

Michael G. Hester, FAusIMM
Print Name of Qualified Person

CERTIFICATE OF QUALIFIED PERSON

I, John M. Marek P.E. do hereby certify that:

a) I am currently employed as the President and a Senior Mining Engineer by:

   Independent Mining Consultants, Inc.
   3560 E. Gas Road
   Tucson, Arizona, USA 85714

b) This certificate is part of the report titled "Casino Project Form 43-101F1 Technical Report Preliminary Economic Assessment, Yukon, Canada", with an effective date of 22 June 2021.

c) I graduated with the following degrees from the Colorado School of Mines

   - Bachelors of Science, Mineral Engineering – Physics – 1974
   - Masters of Science, Mining Engineering – 1976
   - I am a Registered Professional Mining Engineer in the State of Arizona, USA – Registration # 12772
   - I am a Registered Professional Engineer in the State of Colorado, USA – Registration # 16191
   - I am a Professional Engineer, Yukon Territory, Canada.
   - I am a Registered Member of the American Institute of Mining and Metallurgical Engineers, Society of Mining Engineers

   I have worked as a mining engineer, geoscientist, and reserve estimation specialist for more than 45 years. My work experience includes mine planning, equipment selection, mine cost estimation and mine feasibility studies for base and precious metals projects worldwide for over 45 years.

d) I have not visited the Casino Project.

e) I am responsible for sections 16, 21.1.5, 21.3.3, 24.2, and the corresponding components of sections 1, 25, 26, and 27 of the Technical Report titled "Casino Project Form 43-101F1 Technical Report Preliminary Economic Assessment, Yukon, Canada", with an effective date of 22 June 2021.

f) I am independent of Casino Mining Corporation and Western Copper and Gold Corporation, applying the tests in Section 1.5 of National Instrument 43-101.

g) I have not worked on the Casino project prior to this Technical Report. Other members of Independent Mining Consultants, Inc. have worked on this project previously.

h) I have read National Instrument 43-101 and Form 43-101F1, and to my knowledge, the Technical Report has been prepared in compliance with that instrument and form.

i) As of 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: August 2, 2021

Signed and Sealed

John M. Marek
Professional Engineer
Yukon Territory

CERTIFICATE OF QUALIFIED PERSON

I, Laurie M. Tahija, MMSA-Q.P. do hereby certify that:

1. I am currently employed as Vice President by M3 Engineering & Technology Corporation, 2051 W. Sunset Road, Ste. 101, Tucson, Arizona 85704, USA.
2. I am a graduate of Montana College of Mineral Science and Technology, in Butte, Montana and received a Bachelor of Science degree in Mineral Processing Engineering in 1981.
3. I am recognized as a Qualified Professional (QP) member (#01399QP) with special expertise in Metallurgy/Processing by the Mining and Metallurgical Society of America (MMSA).
4. I have practiced mineral processing for 39 years. I have over twenty (20) years of plant operations and project management experience at a variety of mines including both precious metals and base metals. I have worked both in the United States (Nevada, Idaho, California) and overseas (Papua New Guinea, China, Chile, Mexico) at existing operations and at new operations during construction and startup. My operating experience in precious metals processing includes heap leaching, agitation leaching, gravity, flotation, Merrill-Crowe, and ADR (CIC & CIL). My operating experience in base metal processing includes copper heap leaching with SX/EW and zinc recovery using ion exchange, SX/EW, and casting. I have been responsible for process design for new plants and the retrofitting of existing operations. I have been involved in projects from construction to startup and continuing into operation. I have worked on scoping, pre-feasibility and feasibility studies for mining projects in the United States and Latin America, as well as worked on the design and construction phases of some of these projects.
5. I have read the definition of "qualified person" set out in National instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
6. I am responsible for Sections 13, 17, 21.3.1 and corresponding sections of 1, 25, 26, and 27 of the technical report titled "Western Copper and Gold Corporation, Casino Project, Form 43-101F1 Technical Report, Preliminary Economic Assessment, Yukon, Canada" dated effective June 22, 2021 (the "Technical Report").
7. I have not had prior involvement with the property that is the subject of the Technical Report and have not visited the Project site.
8. As of the date of this certificate, 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.
9. I am independent of Western Copper and Gold Corporation and its subsidiaries as defined by Section 1.5 of NI 43-101.
10. I have read NI 43-101 and Form 43-101F1. The sections of the Technical Report that I am responsible for have been prepared in compliance with that instrument and form.
11. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their website accessible by the public, of the Technical Report.

Dated this 2nd day of August, 2021.

"Signed"
Signature of Qualified Person

Laurie M. Tahija
Print Name of Qualified Person

CERTIFICATE OF QUALIFIED PERSON

I, Carl Schulze, with a business address at 34A Laberge Rd, Whitehorse, Yukon Y1A 5Y9, hereby certify that:

1. I am a Project Manager employed by: Aurora Geosciences Ltd., 34A Laberge Rd, Whitehorse, Yukon Y1A 5Y9.
2. This certificate applies to the technical report titled: "Western Copper and Gold Corporation, Casino Project, Form 43-101F1 Technical Report, Preliminary Economic Assessment, Yukon, Canada" dated effective June 22, 2021 (the "Technical Report").
3. I am a graduate of Lakehead University, Bachelor of Science Degree in Geology, 1984. I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia (EGBC), Lic. No. 25393. I have worked as a geologist for a total of 35 years since my graduation from Lakehead University. I have worked extensively in Yukon, British Columbia, northern Ontario and Alaska, as well as the Northwest Territories, Saskatchewan and Manitoba. I served as President of the Yukon Chamber of Mines, where I was also a Director from 2003 to 2015. I have acted in various capacities with numerous private and publicly-traded mining and exploration companies, and also served as the Resident Geologist for the Government of Nunavut from 2000 to 2002.
4. I performed a site visit for 18 days from Sept 9 through 26, 2020.
5. I am responsible for Sections 6, 7, 8, 9, 10, 11, 12, 23 and corresponding sections of 1, 25, 26, and 27 of the Technical Report.
6. I have had no involvement with Western Copper and Gold Corp, its predecessors or subsidiaries. I am independent of the issuer applying the test in section 1.5 of National Instrument 43-101;
7. I have not received nor expect to receive any interest, direct or indirect, in Western Copper and Gold Corp, its subsidiaries, affiliates and associates;
8. I have read "Standards of Disclosure for Mineral Projects", National Instrument 43-101 and Form 43-101F1, and the aforementioned sections of the Technical Report has been prepared in compliance with this Instrument and that Form;
9. As of the date of this certificate, to the best of my knowledge, information and belief, the aforementioned sections of the Technical Report contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading, and;
10. This certificate applies to the NI 43-101 compliant technical report titled "Western Copper and Gold Corporation, Casino Project, Form 43-101F1 Technical Report, Mineral Resource Statement, Yukon, Canada", dated effective July 3, 2020.
11. I consent to the public filing of this technical report with any stock exchange and any regulatory authority and consent to the publication for regulatory purposes, including electronic publication in the public company files of their websites accessible to the public, of extracts from the Technical Report by Western Copper and Gold Corp.

Dated at Whitehorse, Yukon this 2nd day of August, 2021.

"Signed" Carl Schulze

Signature of Qualified Person

Carl Schulze

Name of Qualified Person

CERTIFICATE OF QUALIFIED PERSON

I, Daniel Friedman, P. Eng. do hereby certify that:

1. This certificate applies to the technical report entitled, “Western Copper and Gold Corporation, Casino Project, Form 43-101F1 Technical Report, Preliminary Economic Assessment, Yukon, Canada” prepared for Western Copper and Gold Corporation (the “Issuer”) with an effective date of June 22, 2021 (the “Technical Report”)
2. I am employed as a Specialist Civil Engineer of Knight Piésold Ltd. with an office at Suite 1400 – 750 West Pender Street, Vancouver, British Columbia, V6C 2T8, Canada.
3. I am a graduate of McGill University, Montreal, Canada, B.Eng. (Civil), 2003. I have practiced my profession continuously since 2004. My principal experience is in the areas of water and waste management for mining projects and hydrotechnical engineering.
4. I am a registered professional engineer in good standing in the following jurisdictions:
   - Yukon, Canada (No. 3404)
   - British Columbia, Canada (No. 32571)
   - New Brunswick, Canada (No. L5001)
   - Arizona, USA (No. 53722)
5. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
6. I have not visited the project site.
7. I am a contributing author of the Technical Report and am responsible for Sections 18.5, 18.6, and corresponding sections of 1, 25, 26, and 27 of the technical report titled “Western Copper and Gold Corporation, Casino Project, Form 43-101F1 Technical Report, Preliminary Economic Assessment, Yukon, Canada” dated effective June 22, 2021 (the “Technical Report”), and accept professional responsibility for these sections of the Technical Report.
8. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101.
9. I have prior involvement with the property that is the subject of the Technical Report. I have developed various engineering studies related to tailings and water management for Western Copper and Gold Corporation (“Western”) from 2016 through 2020.
10. I have read National Instrument 43-101 and the sections of the Technical Report that I am responsible for have been prepared in accordance with National Instrument 43-101.

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11. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I have accepted responsibility contain all scientific and technical information required to be disclosed to make the Technical Report not misleading.

Effective Date: June 22, 2021
Signing Date: August 2, 2021

Signed

Daniel Friedman, P.Eng.

APPENDIX B – LIST OF CLAIMS

Casino Property

List of Casino Placer Claims

District: Whitehorse
Status: Active

Claim owner: Casino Mining Corp.

#	GRANT NUMBER	TENURE TYPE	CLAIM NAME	CLAIM NUMBER	RECORDED DATE	EXPIRY DATE
1	P 508065	Placer	CAS PL	4	2011-08-11	2022-02-11
2	P 508066	Placer	CAS PL	5	2011-08-11	2022-02-11
3	P 508067	Placer	CAS PL	6	2011-08-11	2022-02-11
4	P 508068	Placer	CAS PL	7	2011-08-11	2022-02-11
5	P 508069	Placer	CAS PL	8	2011-08-11	2022-02-11
6	P 508070	Placer	CAS PL	9	2011-08-11	2022-02-11
7	P 508071	Placer	CAS PL	10	2011-08-11	2022-02-11
8	P 508072	Placer	CAS PL	11	2011-08-11	2022-02-11
9	P 508073	Placer	CAS PL	12	2011-08-11	2022-02-11
10	P 508074	Placer	CAS PL	13	2011-08-11	2022-02-11
11	P 508075	Placer	CAS PL	14	2011-08-11	2022-02-11
12	P 508076	Placer	CAS PL	15	2011-08-11	2022-02-11
13	P 508077	Placer	CAS PL	16	2011-08-11	2022-02-11
14	P 508078	Placer	CAS PL	17	2011-08-11	2022-02-11
15	P 508079	Placer	CAS PL	18	2011-08-11	2022-02-11
16	P 508080	Placer	CAS PL	19	2011-08-11	2022-02-11
17	P 508081	Placer	CAS PL	20	2011-08-11	2022-02-11
18	P 508082	Placer	CAS PL	21	2011-08-11	2022-02-11
19	P 508083	Placer	CAS PL	22	2011-08-11	2022-02-11
20	P 508084	Placer	CAS PL	23	2011-08-11	2022-02-11
21	P 508085	Placer	CAS PL	24	2011-08-11	2022-02-11
22	P 508086	Placer	CAS PL	25	2011-08-11	2022-02-11
23	P 508087	Placer	CAS PL	26	2011-08-11	2022-02-11
24	P 508088	Placer	CAS PL	27	2011-08-11	2022-02-11
25	P 508089	Placer	CAS PL	28	2011-08-11	2022-02-11
26	P 508090	Placer	CAS PL	29	2011-08-11	2022-02-11
27	P 508091	Placer	CAS PL	30	2011-08-11	2022-02-11
28	P 508092	Placer	CAS PL	31	2011-08-11	2022-02-11
29	P 508093	Placer	CAS PL	32	2011-08-11	2022-02-11
30	P 508094	Placer	CAS PL	33	2011-08-11	2022-02-11
31	P 508095	Placer	CAS PL	34	2011-08-11	2022-02-11
32	P 508096	Placer	CAS PL	35	2011-08-11	2022-02-11
33	P 508097	Placer	CAS PL	36	2011-08-11	2022-02-11
34	P 508098	Placer	CAS PL	37	2011-08-11	2022-02-11
35	P 508099	Placer	CAS PL	38	2011-08-11	2022-02-11
36	P 508100	Placer	CAS PL	39	2011-08-11	2022-02-11
37	P 509301	Placer	CAS PL	40	2011-08-11	2022-02-11
38	P 509302	Placer	CAS PL	41	2011-08-11	2022-02-11
39	P 509303	Placer	CAS PL	42	2011-08-11	2022-02-11
40	P 509304	Placer	CAS PL	43	2011-08-11	2022-02-11
41	P 509305	Placer	CAS PL	44	2011-08-11	2022-02-11
42	P 509306	Placer	CAS PL	45	2011-08-11	2022-02-11
43	P 509307	Placer	CAS PL	46	2011-08-11	2022-02-11
44	P 509308	Placer	CAS PL	47	2011-08-11	2022-02-11
45	P 509309	Placer	CAS PL	48	2011-08-11	2022-02-11
46	P 509310	Placer	CAS PL	49	2011-08-11	2022-02-11
47	P 509311	Placer	CAS PL	50	2011-08-11	2022-02-11
48	P 509312	Placer	CAS PL	51	2011-08-11	2022-02-11
49	P 509313	Placer	CAS PL	52	2011-08-11	2022-02-11
50	P 509314	Placer	CAS PL	53	2011-08-11	2022-02-11
51	P 509315	Placer	CAS PL	54	2011-08-11	2022-02-11
52	P 509316	Placer	CAS PL	55	2011-08-11	2022-02-11
53	P 509317	Placer	CAS PL	56	2011-08-11	2022-02-11
54	P 509318	Placer	CAS PL	57	2011-08-11	2022-02-11
55	P 509319	Placer	CAS PL	58	2011-08-11	2022-02-11

Casino Property

List of Casino Quartz Claims

District: Whitehorse
Status: Active
Claim owner: Casino Mining Corp.

#	Grant Number	Claim Name	Claim Number	Staking date	Expiry Date	NTS Map	Non Standard Size
1	95740	CAT	63	1965-12-05	2025-03-25	115J10	
2	95741	CAT	64	1965-12-05	2025-03-25	115J10	
3	95742	CAT	65	1965-12-05	2025-03-25	115J10	
4	95743	CAT	66	1965-12-05	2025-03-25	115J10	
5	95745	CAT	68	1965-12-05	2025-03-25	115J10	
6	95747	CAT	70	1965-12-05	2025-03-25	115J10	
7	Y 35195	MOUSE	4	1969-06-04	2025-03-25	115J10	
8	Y 35197	MOUSE	6	1969-06-04	2025-03-25	115J10	
9	Y 35484	MOUSE	90	1969-06-22	2025-03-25	115J10	
10	YD04376	FLY	2	2011-02-02	2025-03-25	115J10	
11	YD04377	FLY	3	2011-02-02	2025-03-25	115J10	
12	YD04378	FLY	4	2011-02-02	2025-03-25	115J10	
13	YD04379	FLY	5	2011-02-02	2025-03-25	115J10	
14	YD04380	FLY	6	2011-02-02	2025-03-25	115J10	
15	YD04381	FLY	7	2011-02-02	2025-03-25	115J10	
16	YD04382	FLY	8	2011-02-02	2025-03-25	115J10	
17	YD04383	FLY	9	2011-02-02	2025-03-25	115J10	
18	YD04384	FLY	10	2011-02-02	2025-03-25	115J10	
19	YD04385	FLY	11	2011-02-02	2025-03-25	115J10	
20	YD04386	FLY	12	2011-02-02	2025-03-25	115J10	
21	YD04387	FLY	13	2011-02-02	2025-03-25	115J10	
22	YD04388	FLY	14	2011-02-02	2025-03-25	115J10	
23	YD04399	FLY	15	2011-02-02	2025-03-25	115J10	
24	YD04400	FLY	16	2011-02-02	2025-03-25	115J10	
25	YD04401	FLY	17	2011-02-02	2025-03-25	115J10	
26	YD04402	FLY	18	2011-02-02	2025-03-25	115J10	
27	YD04375	FLY	1	2011-02-02	2025-03-26	115J10	
28	YC82855	BL	1	2008-07-31	2025-08-01	105 E12	
29	YC82856	BL	2	2008-07-31	2025-08-01	105 E12	
30	YC82857	BL	3	2008-07-31	2025-08-01	105 E12	
31	YC82858	BL	4	2008-07-31	2025-08-01	105 E12	
32	YC82859	BL	5	2008-07-31	2025-08-01	105 E12	
33	YC82860	BL	6	2008-07-31	2025-08-01	105 E12	
34	YC82861	BL	7	2008-07-31	2025-08-01	105 E12	
35	YC82862	BL	8	2008-07-31	2025-08-01	105 E12	
36	YE94141	CAS19	1	2019-08-28	2025-09-03	115J10	
37	YE94142	CAS19	2	2019-08-28	2025-09-03	115J10	
38	YE94143	CAS19	3	2019-08-28	2025-09-03	115J10	
39	YE94144	CAS19	4	2019-08-28	2025-09-03	115J10	
40	YE94145	CAS19	5	2019-08-28	2025-09-03	115J10	
41	YE94146	CAS19	6	2019-08-28	2025-09-03	115J10	
42	YE94147	CAS19	7	2019-08-28	2025-09-03	115J10	
43	YE94148	CAS19	8	2019-08-28	2025-09-03	115J10	
44	YE94149	CAS19	9	2019-08-28	2025-09-03	115J10	
45	YE94150	CAS19	10	2019-08-28	2025-09-03	115J10	
46	YE94151	CAS19	11	2019-08-28	2025-09-03	115J10	
47	YE94152	CAS19	12	2019-08-28	2025-09-03	115J10	
48	YE94153	CAS19	13	2019-08-28	2025-09-03	115J10	
49	YE94154	CAS19	14	2019-08-28	2025-09-03	115J10	Full Quartz fraction (25+ acres)
50	YE94155	CAS19	15	2019-08-28	2025-09-03	115J10	Full Quartz fraction (25+ acres)
51	YE94156	CAS19	16	2019-08-28	2025-09-03	115J10	
52	YE94157	CAS19	17	2019-08-28	2025-09-03	115J10	
53	YE94158	CAS19	18	2019-08-28	2025-09-03	115J10	
54	YE94159	CAS19	19	2019-08-28	2025-09-03	115J10	

List of Casino Quartz Claims

District: Whitehorse
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#	Grant Number	Claim Name	Claim Number	Staking date	Expiry Date	NTS Map	Non Standard Size
55	YE94160	CAS19	20	2019-08-28	2025-09-03	115J10	
56	YE94161	CAS19	21	2019-08-28	2025-09-03	115J10	
57	YE94162	CAS19	22	2019-08-28	2025-09-03	115J10	
58	YE94163	CAS19	23	2019-08-28	2025-09-03	115J10	
59	YE94164	CAS19	24	2019-08-28	2025-09-03	115J10	
60	YE94165	CAS19	25	2019-08-28	2025-09-03	115J10	
61	YE94166	CAS19	26	2019-08-28	2025-09-03	115J10	
62	YE94167	CAS19	27	2019-08-28	2025-09-03	115J10	
63	YE94168	CAS19	28	2019-08-28	2025-09-03	115J10	
64	YE94169	CAS19	29	2019-08-28	2025-09-03	115J10	
65	YE94170	CAS19	30	2019-08-28	2025-09-03	115J10	
66	YE94171	CAS19	31	2019-08-28	2025-09-03	115J10	
67	YE94172	CAS19	32	2019-08-28	2025-09-03	115J10	Full Quartz fraction (25+ acres)
68	YE94173	CAS19	33	2019-08-28	2025-09-03	115J10	Full Quartz fraction (25+ acres)
69	YE94174	CAS19	34	2019-08-28	2025-09-03	115J10	
70	YE94175	CAS19	35	2019-08-28	2025-09-03	115J10	
71	YE94176	CAS19	36	2019-08-28	2025-09-03	115J10	
72	YE94177	CAS19	37	2019-08-27	2025-09-03	115J10	
73	YE94178	CAS19	38	2019-08-27	2025-09-03	115J10	
74	YE94179	CAS19	39	2019-08-27	2025-09-03	115J10	
75	YE94180	CAS19	40	2019-08-27	2025-09-03	115J10	
76	YE94181	CAS19	41	2019-08-27	2025-09-03	115J10	
77	YE94182	CAS19	42	2019-08-27	2025-09-03	115J10	
78	YE94183	CAS19	43	2019-08-27	2025-09-03	115J10	
79	YE94184	CAS19	44	2019-08-27	2025-09-03	115J10	
80	YE94185	CAS19	45	2019-08-27	2025-09-03	115J10	
81	YE94186	CAS19	46	2019-08-27	2025-09-03	115J10	
82	YE94187	CAS19	47	2019-08-27	2025-09-03	115J10	
83	YE94188	CAS19	48	2019-08-27	2025-09-03	115J10	
84	YE94189	CAS19	49	2019-08-27	2025-09-03	115J10	Full Quartz fraction (25+ acres)
85	YE94190	CAS19	50	2019-08-27	2025-09-03	115J10	
86	YE94191	CAS19	51	2019-08-27	2025-09-03	115J10	
87	YE94192	CAS19	52	2019-08-27	2025-09-03	115J10	
88	YE94193	CAS19	53	2019-08-27	2025-09-03	115J10	Full Quartz fraction (25+ acres)
89	YC99925	KANA	46	2010-06-05	2026-06-08	115J15	Full Quartz fraction (25+ acres)
90	YB37540	AZTEC	1	1992-09-12	2026-09-21	115J10	
91	YB37541	AZTEC	2	1992-09-12	2026-09-21	115J10	
92	YB37542	AZTEC	3	1992-09-12	2026-09-21	115J10	
93	YB37543	AZTEC	4	1992-09-12	2026-09-21	115J10	
94	YB37544	AZTEC	5	1992-09-12	2026-09-21	115J10	
95	YB37545	AZTEC	6	1992-09-12	2026-09-21	115J10	
96	YB37546	AZTEC	7	1992-09-12	2026-09-21	115J10	
97	YB37547	AZTEC	8	1992-09-12	2026-09-21	115J10	
98	YB37548	AZTEC	9	1992-09-12	2026-09-21	115J10	
99	YB37549	AZTEC	10	1992-09-12	2026-09-21	115J10	
100	YB37622	MAYA	31	1992-09-12	2026-09-21	115J10	
101	YB37623	MAYA	32	1992-09-12	2026-09-21	115J10	
102	YB37624	MAYA	33	1992-09-12	2026-09-21	115J10	
103	YB37625	MAYA	34	1992-09-12	2026-09-21	115J10	
104	YB37626	MAYA	35	1992-09-12	2026-09-21	115J10	
105	YB37627	MAYA	36	1992-09-12	2026-09-21	115J10	
106	YB37628	MAYA	37	1992-09-12	2026-09-21	115J10	
107	YB37629	MAYA	38	1992-09-12	2026-09-21	115J10	
108	YB37630	MAYA	39	1992-09-12	2026-09-21	115J10	
109	YB37631	MAYA	40	1992-09-12	2026-09-21	115J10	
110	YC99915	KANA	37	2009-09-01	2026-09-29	115J10	Partial Quartz fraction (<25 acres)

List of Casino Quartz Claims

District: Whitehorse

Status: Active

Claim owner: Casino Mining Corp.

#	Grant Number	Claim Name	Claim Number	Staking date	Expiry Date	NTS Map	Non Standard Size
111	YC99916	KANA	38	2009-09-01	2026-09-29	115J10	Partial Quartz fraction (<25 acres)
112	YC99917	KANA	39	2009-09-01	2026-09-29	115J10	Partial Quartz fraction (<25 acres)
113	YC99918	KANA	40	2009-09-01	2026-09-29	115J10	Partial Quartz fraction (<25 acres)
114	YC99919	KANA	41	2009-09-01	2026-09-29	115J10	Partial Quartz fraction (<25 acres)
115	YC99920	KANA	42	2009-09-01	2026-09-29	115J10	Partial Quartz fraction (<25 acres)
116	YC99921	KANA	43	2009-09-01	2026-09-29	115J10	Partial Quartz fraction (<25 acres)
117	YC99922	KANA	44	2009-09-01	2026-09-29	115J10	Partial Quartz fraction (<25 acres)
118	YC99923	KANA	45	2009-09-01	2026-09-29	115J10	Partial Quartz fraction (<25 acres)
119	YB37830	ICE	30	1993-01-22	2027-01-27	115J11	
120	YB37831	ICE	31	1993-01-22	2027-01-27	115J11	
121	YB37832	ICE	32	1993-01-22	2027-01-27	115J11	
122	YB37833	ICE	33	1993-01-22	2027-01-27	115J11	
123	YB37841	ICE	41	1993-01-22	2027-01-27	115J10	
124	YB37842	ICE	42	1993-01-22	2027-01-27	115J10	
125	YB37843	ICE	43	1993-01-22	2027-01-27	115J11	
126	YB37844	ICE	44	1993-01-22	2027-01-27	115J11	
127	YB37845	ICE	45	1993-01-22	2027-01-27	115J11	
128	YB37846	ICE	46	1993-01-22	2027-01-27	115J11	
129	YB37847	ICE	47	1993-01-22	2027-01-27	115J11	
130	YD17559	AXS	1	2009-10-05	2027-03-25	115J15	
131	YD17560	AXS	2	2009-10-05	2027-03-25	115J15	
132	YD17561	AXS	3	2009-10-05	2027-03-25	115J15	
133	YD17562	AXS	4	2009-10-05	2027-03-25	115J15	
134	YD17563	AXS	5	2009-10-05	2027-03-25	115J15	
135	YD17564	AXS	6	2009-10-05	2027-03-25	115J15	
136	YD17565	AXS	7	2009-10-05	2027-03-25	115J10	
137	YD17566	AXS	8	2009-10-05	2027-03-25	115J10	
138	YD17567	AXS	9	2009-10-05	2027-03-25	115J10	
139	YD17568	AXS	10	2009-10-05	2027-03-25	115J10	
140	YD17569	AXS	11	2009-10-06	2027-03-25	115J10	
141	YD17570	AXS	12	2009-10-06	2027-03-25	115J10	
142	YD17571	AXS	13	2009-10-06	2027-03-25	115J10	
143	YD17572	AXS	14	2009-10-06	2027-03-25	115J10	
144	YD17573	AXS	15	2009-10-06	2027-03-25	115J10	
145	YD17574	AXS	16	2009-10-06	2027-03-25	115J10	
146	YD17575	AXS	17	2009-10-06	2027-03-25	115J10	
147	YD17576	AXS	18	2009-10-06	2027-03-25	115J10	
148	YD17577	AXS	19	2009-10-05	2027-03-25	115J10	
149	YD17578	AXS	20	2009-10-05	2027-03-25	115J10	
150	YD17579	AXS	21	2009-10-05	2027-03-25	115J10	
151	YD17580	AXS	22	2009-10-05	2027-03-25	115J10	
152	YD17581	AXS	23	2009-10-05	2027-03-25	115J10	
153	YD17582	AXS	24	2009-10-05	2027-03-25	115J10	
154	YD17583	AXS	25	2009-10-05	2027-03-25	115J10	
155	YD17584	AXS	26	2009-10-05	2027-03-25	115J10	
156	YD17585	AXS	27	2009-10-05	2027-03-25	115J10	
157	YD17586	AXS	28	2009-10-05	2027-03-25	115J10	
158	YD17587	AXS	29	2009-10-05	2027-03-25	115J10	
159	YD17588	AXS	30	2009-10-05	2027-03-25	115J10	
160	YD17589	AXS	31	2009-10-05	2027-03-25	115J10	
161	YD17590	AXS	32	2009-10-05	2027-03-25	115J10	
162	YD17591	AXS	33	2009-10-05	2027-03-25	115J10	
163	YD17592	AXS	34	2009-10-05	2027-03-25	115J10	
164	YD17593	AXS	35	2009-10-05	2027-03-25	115J10	
165	YD17594	AXS	36	2009-10-05	2027-03-25	115J10	
166	YD17595	AXS	37	2009-10-05	2027-03-25	115J10	

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Claim owner: Casino Mining Corp.

#	Grant Number	Claim Name	Claim Number	Staking date	Expiry Date	NTS Map	Non Standard Size
167	YD17596	AXS	38	2009-10-05	2027-03-25	115J10	
168	YD17597	AXS	39	2009-10-05	2027-03-25	115J10	
169	YD17598	AXS	40	2009-10-05	2027-03-25	115J10	
170	YD17599	AXS	41	2009-10-05	2027-03-25	115J10	
171	YD17600	AXS	42	2009-10-05	2027-03-25	115J10	
172	YD17601	AXS	43	2009-10-05	2027-03-25	115J10	
173	YD17602	AXS	44	2009-10-05	2027-03-25	115J10	
174	YD17603	AXS	45	2009-10-05	2027-03-25	115J10	
175	YD17604	AXS	46	2009-10-05	2027-03-25	115J10	
176	YD17605	AXS	47	2009-10-05	2027-03-25	115J10	
177	YD17606	AXS	48	2009-10-05	2027-03-25	115J10	
178	YD17607	AXS	49	2009-10-06	2027-03-25	115J10	
179	YD17608	AXS	50	2009-10-06	2027-03-25	115J10	
180	YD17609	AXS	51	2009-10-06	2027-03-25	115J10	
181	YD17610	AXS	52	2009-10-06	2027-03-25	115J10	
182	YD17611	AXS	53	2009-10-06	2027-03-25	115J10	
183	YD17612	AXS	54	2009-10-06	2027-03-25	115J10	
184	YD17613	AXS	55	2009-10-05	2027-03-25	115J10	
185	YD17614	AXS	56	2009-10-05	2027-03-25	115J09	
186	YD17615	AXS	57	2009-10-05	2027-03-25	115J10	
187	YD17616	AXS	58	2009-10-05	2027-03-25	115J10	
188	YD17617	AXS	59	2009-10-05	2027-03-25	115J10	
189	YD17618	AXS	60	2009-10-05	2027-03-25	115J10	
190	YD17619	AXS	61	2009-10-05	2027-03-25	115J09	
191	YD17620	AXS	62	2009-10-05	2027-03-25	115J09	
192	YD17621	AXS	63	2009-10-05	2027-03-25	115J09	
193	YD17622	AXS	64	2009-10-05	2027-03-25	115J09	
194	YD17623	AXS	65	2009-10-05	2027-03-25	115J09	
195	YD17624	AXS	66	2009-10-05	2027-03-25	115J09	
196	YD17625	AXS	67	2009-10-05	2027-03-25	115J09	
197	YD17626	AXS	68	2009-10-05	2027-03-25	115J09	
198	YD17627	AXS	69	2009-10-06	2027-03-25	115J09	
199	YD17628	AXS	70	2009-10-06	2027-03-25	115J09	
200	YD17629	AXS	71	2009-10-06	2027-03-25	115J09	
201	YD17630	AXS	72	2009-10-06	2027-03-25	115J09	
202	YD17631	AXS	73	2009-10-06	2027-03-25	115J09	
203	YD17632	AXS	74	2009-10-06	2027-03-25	115J09	
204	YD17633	AXS	75	2009-10-05	2027-03-25	115J09	
205	YD17634	AXS	76	2009-10-05	2027-03-25	115J09	
206	YD17635	AXS	77	2009-10-05	2027-03-25	115J09	
207	YD17636	AXS	78	2009-10-05	2027-03-25	115J09	
208	YD17637	AXS	79	2009-10-05	2027-03-25	115J09	
209	YD17638	AXS	80	2009-10-05	2027-03-25	115J09	
210	YD17639	AXS	81	2009-10-05	2027-03-25	115J09	
211	YD17640	AXS	82	2009-10-05	2027-03-25	115J09	
212	YD17641	AXS	83	2009-10-05	2027-03-25	115J09	
213	YD17642	AXS	84	2009-10-05	2027-03-25	115J09	
214	YD17643	AXS	85	2009-10-05	2027-03-25	115J09	
215	YD17644	AXS	86	2009-10-05	2027-03-25	115J09	
216	YD17645	AXS	87	2009-10-06	2027-03-25	115J09	
217	YD17646	AXS	88	2009-10-06	2027-03-25	115J09	
218	YD17647	AXS	89	2009-10-06	2027-03-25	115J09	
219	YD17648	AXS	90	2009-10-06	2027-03-25	115J09	
220	YD17649	AXS	91	2009-10-06	2027-03-25	115J09	
221	YD17650	AXS	92	2009-10-06	2027-03-25	115J09	
222	YD17651	AXS	103	2009-10-07	2027-03-25	115J09	

List of Casino Quartz Claims

District: Whitehorse
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Claim owner: Casino Mining Corp.

#	Grant Number	Claim Name	Claim Number	Staking date	Expiry Date	NTS Map	Non Standard Size
223	YD17652	AXS	102	2009-10-07	2027-03-25	115J16	
224	YD17653	AXS	101	2009-10-07	2027-03-25	115J16	
225	YD17654	AXS	100	2009-10-06	2027-03-25	115J16	
226	YD17655	AXS	99	2009-10-06	2027-03-25	115J16	
227	YD17656	AXS	98	2009-10-06	2027-03-25	115J16	
228	YD17657	AXS	97	2009-10-06	2027-03-25	115J16	
229	YD17658	AXS	96	2009-10-06	2027-03-25	115J16	
230	YD17659	AXS	95	2009-10-06	2027-03-25	115J16	
231	YD17660	AXS	94	2009-10-06	2027-03-25	115J16	
232	YD17661	AXS	93	2009-10-06	2027-03-25	115J16	
233	YD17662	AXS	104	2009-10-07	2027-03-25	115J09	
234	YD17663	AXS	105	2009-10-07	2027-03-25	115J09	
235	YD17664	AXS	106	2009-10-07	2027-03-25	115J09	
236	YD17665	AXS	107	2009-10-07	2027-03-25	115J09	
237	YD17666	AXS	108	2009-10-07	2027-03-25	115J09	
238	YD17667	AXS	109	2009-10-07	2027-03-25	115J09	
239	YD17668	AXS	110	2009-10-07	2027-03-25	115J09	
240	YD17669	AXS	111	2009-10-07	2027-03-25	115J09	
241	YD17670	AXS	112	2009-10-07	2027-03-25	115J09	
242	YD17694	AXS	136	2009-10-06	2027-03-25	115J09	
243	YD17671	AXS	113	2009-10-06	2027-03-25	115J09	
244	YD17672	AXS	114	2009-10-06	2027-03-25	115J09	
245	YD17673	AXS	115	2009-10-06	2027-03-25	115J09	
246	YD17674	AXS	116	2009-10-06	2027-03-25	115J09	
247	YD17675	AXS	117	2009-10-06	2027-03-25	115J09	
248	YD17676	AXS	118	2009-10-06	2027-03-25	115J09	
249	YD17677	AXS	119	2009-10-06	2027-03-25	115J09	
250	YD17678	AXS	120	2009-10-06	2027-03-25	115J09	
251	YD17679	AXS	121	2009-10-06	2027-03-25	115J09	
252	YD17680	AXS	122	2009-10-06	2027-03-25	115J09	
253	YD17681	AXS	123	2009-10-06	2027-03-25	115J09	
254	YD17682	AXS	124	2009-10-06	2027-03-25	115J09	
255	YD17683	AXS	125	2009-10-06	2027-03-25	115J09	
256	YD17684	AXS	126	2009-10-06	2027-03-25	115J09	
257	YD17685	AXS	127	2009-10-06	2027-03-25	115J09	
258	YD17686	AXS	128	2009-10-06	2027-03-25	115J09	
259	YD17687	AXS	129	2009-10-06	2027-03-25	115J09	
260	YD17688	AXS	130	2009-10-06	2027-03-25	115J09	
261	YD17689	AXS	131	2009-10-06	2027-03-25	115J09	
262	YD17690	AXS	132	2009-10-06	2027-03-25	115J09	
263	YD17691	AXS	133	2009-10-06	2027-03-25	115J09	
264	YD17692	AXS	134	2009-10-06	2027-03-25	115J09	
265	YD17693	AXS	135	2009-10-06	2027-03-25	115J09	
266	YD08825	BERG	3	2010-06-06	2027-06-08	115J15	
267	YD08824	BERG	4	2010-06-06	2027-06-08	115J15	
268	YD08823	BERG	5	2010-06-06	2027-06-08	115J15	
269	YD08822	BERG	6	2010-06-06	2027-06-08	115J15	
270	YD08821	BERG	7	2010-06-06	2027-06-08	115J14	
271	YD08820	BERG	8	2010-06-06	2027-06-08	115J14	
272	YD08819	BERG	9	2010-06-06	2027-06-08	115J14	
273	YD08818	BERG	10	2010-06-06	2027-06-08	115J14	
274	YD08817	BERG	11	2010-06-06	2027-06-08	115J14	
275	YD08816	BERG	12	2010-06-06	2027-06-08	115J14	
276	YD08815	BERG	13	2010-06-06	2027-06-08	115J14	
277	YD08814	BERG	14	2010-06-06	2027-06-08	115J14	
278	YD08813	BERG	15	2010-06-06	2027-06-08	115J14	

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#	Grant Number	Claim Name	Claim Number	Staking date	Expiry Date	NTS Map	Non Standard Size
279	YD08812	BERG	16	2010-06-06	2027-06-08	115J14	
280	YD08811	BERG	17	2010-06-06	2027-06-08	115J14	
281	YD08810	BERG	18	2010-06-06	2027-06-08	115J14	
282	YD08809	BERG	19	2010-06-07	2027-06-08	115J14	
283	YD08808	BERG	20	2010-06-07	2027-06-08	115J14	
284	YD08807	BERG	21	2010-06-07	2027-06-08	115J14	
285	YD08806	BERG	22	2010-06-07	2027-06-08	115J14	
286	YD08827	BERG	27	2010-06-06	2027-06-08	115J15	
287	YD08828	BERG	28	2010-06-06	2027-06-08	115J15	
288	YD08829	BERG	29	2010-06-06	2027-06-08	115J15	
289	YD08830	BERG	30	2010-06-06	2027-06-08	115J15	
290	YD08831	BERG	31	2010-06-06	2027-06-08	115J14	
291	YD08832	BERG	32	2010-06-06	2027-06-08	115J14	
292	YD08833	BERG	33	2010-06-06	2027-06-08	115J14	
293	YD08834	BERG	34	2010-06-06	2027-06-08	115J14	
294	YD08835	BERG	35	2010-06-06	2027-06-08	115J14	
295	YD08836	BERG	36	2010-06-06	2027-06-08	115J14	
296	YD08837	BERG	37	2010-06-06	2027-06-08	115J14	
297	YD08838	BERG	38	2010-06-06	2027-06-08	115J14	
298	YD08839	BERG	39	2010-06-06	2027-06-08	115J14	
299	YD08840	BERG	40	2010-06-06	2027-06-08	115J14	
300	YD08841	BERG	41	2010-06-06	2027-06-08	115J14	
301	YD08842	BERG	42	2010-06-06	2027-06-08	115J14	
302	YD08847	BERG	47	2010-06-05	2027-06-08	115J11	
303	YD08848	BERG	48	2010-06-05	2027-06-08	115J11	
304	YD08849	BERG	49	2010-06-05	2027-06-08	115J11	
305	YD08850	BERG	50	2010-06-05	2027-06-08	115J11	
306	YD08854	BERG	54	2010-06-05	2027-06-08	115J11	
307	YD08855	BERG	55	2010-06-05	2027-06-08	115J11	
308	YD08856	BERG	56	2010-06-05	2027-06-08	115J11	
309	YD08853	BERG	53	2010-06-05	2027-06-08	115J11	Partial Quartz fraction (<25 acres)
310	YD08802	BERG	59	2010-06-07	2027-06-08	115J11	Full Quartz fraction (25+ acres)
311	YC99926	KANA	47	2010-06-05	2027-06-08	115J15	Partial Quartz fraction (<25 acres)
312	YC99924	KANA	58	2010-06-04	2027-06-08	115J10	Partial Quartz fraction (<25 acres)
313	YC99927	KANA	48	2010-06-08	2027-06-08	115J15	
314	YC99928	KANA	49	2010-06-08	2027-06-08	115J15	
315	YC99929	KANA	50	2010-06-08	2027-06-08	115J15	
316	YC99930	KANA	51	2010-06-08	2027-06-08	115J15	
317	YC99931	KANA	52	2010-06-08	2027-06-08	115J15	
318	YC99932	KANA	53	2010-06-08	2027-06-08	115J15	
319	YC99933	KANA	54	2010-06-08	2027-06-08	115J15	
320	YC99934	KANA	55	2010-06-08	2027-06-08	115J15	
321	YC99935	KANA	56	2010-06-05	2027-06-08	115J15	
322	YC99936	KANA	57	2010-06-05	2027-06-08	115J15	
323	YC99879	KANA	1	2009-06-20	2027-06-22	115J15	
324	YC99880	KANA	2	2009-06-20	2027-06-22	115J15	
325	YC99881	KANA	3	2009-06-20	2027-06-22	115J15	
326	YC99882	KANA	4	2009-06-20	2027-06-22	115J15	
327	YC99883	KANA	5	2009-06-20	2027-06-22	115J15	
328	YC99884	KANA	6	2009-06-20	2027-06-22	115J15	
329	YC99885	KANA	7	2009-06-20	2027-06-22	115J15	
330	YC99886	KANA	8	2009-06-20	2027-06-22	115J15	
331	YC99887	KANA	9	2009-06-20	2027-06-22	115J15	
332	YC99888	KANA	10	2009-06-20	2027-06-22	115J15	
333	YC99889	KANA	11	2009-06-20	2027-06-22	115J15	
334	YC99890	KANA	12	2009-06-20	2027-06-22	115J15	

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335	YC99891	KANA	13	2009-06-20	2027-06-22	115J15	
336	YC99892	KANA	14	2009-06-20	2027-06-22	115J15	
337	YC99893	KANA	15	2009-06-20	2027-06-22	115J15	
338	YC99894	KANA	16	2009-06-20	2027-06-22	115J15	
339	YC99895	KANA	17	2009-06-20	2027-06-22	115J15	
340	YC99896	KANA	18	2009-06-20	2027-06-22	115J15	
341	YC99897	KANA	19	2009-06-20	2027-06-22	115J15	
342	YC99898	KANA	20	2009-06-20	2027-06-22	115J15	
343	YC99899	KANA	21	2009-06-20	2027-06-22	115J15	
344	YC99900	KANA	22	2009-06-20	2027-06-22	115J15	
345	YC99901	KANA	23	2009-06-20	2027-06-22	115J15	
346	YC99902	KANA	24	2009-06-20	2027-06-22	115J15	
347	YC99903	KANA	25	2009-06-20	2027-06-22	115J15	
348	YC99904	KANA	26	2009-06-20	2027-06-22	115J15	
349	YC99905	KANA	27	2009-06-20	2027-06-22	115J15	
350	YC99906	KANA	28	2009-06-20	2027-06-22	115J15	
351	YC99907	KANA	29	2009-06-20	2027-06-22	115J15	
352	YC99908	KANA	30	2009-06-20	2027-06-22	115J15	
353	YC99909	KANA	31	2009-06-20	2027-06-22	115J15	
354	YC99910	KANA	32	2009-06-20	2027-06-22	115J15	
355	YC99911	KANA	33	2009-06-20	2027-06-22	115J15	
356	YC99912	KANA	34	2009-06-20	2027-06-22	115J15	
357	YC99913	KANA	35	2009-06-20	2027-06-22	115J15	
358	YD08861	BERG F	61	2010-08-09	2027-08-13	115J10	Full Quartz fraction (25+ acres)
359	YD08862	BERG F	62	2010-08-09	2027-08-13	115J10	Full Quartz fraction (25+ acres)
360	YD08863	BERG F	63	2010-08-09	2027-08-13	115J10	Full Quartz fraction (25+ acres)
361	YD08864	BERG F	64	2010-08-09	2027-08-13	115J10	Full Quartz fraction (25+ acres)
362	YD08865	BERG F	65	2010-08-09	2027-08-13	115J11	Full Quartz fraction (25+ acres)
363	YD08866	BERG F	66	2010-08-09	2027-08-13	115J11	Full Quartz fraction (25+ acres)
364	YD08867	BERG F	67	2010-08-09	2027-08-13	115J14	Partial Quartz fraction (<25 acres)
365	YB37482	KOFFEE	1	1992-09-12	2027-09-21	115J10	
366	YB37483	KOFFEE	2	1992-09-12	2027-09-21	115J10	
367	YB37484	KOFFEE	3	1992-09-12	2027-09-21	115J10	
368	YB37485	KOFFEE	4	1992-09-12	2027-09-21	115J10	
369	YB37486	KOFFEE	5	1992-09-12	2027-09-21	115J10	
370	YB37487	KOFFEE	6	1992-09-12	2027-09-21	115J10	
371	YB37488	KOFFEE	7	1992-09-12	2027-09-21	115J10	
372	YB37489	KOFFEE	8	1992-09-12	2027-09-21	115J10	
373	YB37490	KOFFEE	9	1992-09-12	2027-09-21	115J10	
374	YB37491	KOFFEE	10	1992-09-12	2027-09-21	115J10	
375	YB37492	KOFFEE	11	1992-09-12	2027-09-21	115J10	
376	YB37493	KOFFEE	12	1992-09-12	2027-09-21	115J10	
377	YB37494	KOFFEE	13	1992-09-12	2027-09-21	115J10	
378	YB37495	KOFFEE	14	1992-09-12	2027-09-21	115J10	
379	YB37496	KOFFEE	15	1992-09-12	2027-09-21	115J10	
380	YB37497	KOFFEE	16	1992-09-12	2027-09-21	115J10	
381	YB37498	KOFFEE	17	1992-09-12	2027-09-21	115J10	
382	YB37499	KOFFEE	18	1992-09-12	2027-09-21	115J10	
383	YB37500	KOFFEE	19	1992-09-12	2027-09-21	115J10	
384	YB37501	KOFFEE	20	1992-09-12	2027-09-21	115J10	
385	YB37502	KOFFEE	21	1992-09-12	2027-09-21	115J10	
386	YB37503	KOFFEE	22	1992-09-12	2027-09-21	115J10	
387	YB37504	KOFFEE	23	1992-09-12	2027-09-21	115J10	
388	YB37505	KOFFEE	24	1992-09-12	2027-09-21	115J10	
389	YB37506	KOFFEE	25	1992-09-12	2027-09-21	115J10	
390	YB37507	KOFFEE	26	1992-09-12	2027-09-21	115J10	

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#	Grant Number	Claim Name	Claim Number	Staking date	Expiry Date	NTS Map	Non Standard Size
391	YB37508	KOFFEE	27	1992-09-12	2027-09-21	115J10	
392	YB37509	KOFFEE	28	1992-09-12	2027-09-21	115J10	
393	YB37510	KOFFEE	29	1992-09-13	2027-09-21	115J10	
394	YB37511	KOFFEE	30	1992-09-13	2027-09-21	115J10	
395	YB37512	KOFFEE	31	1992-09-13	2027-09-21	115J10	
396	YB37513	KOFFEE	32	1992-09-13	2027-09-21	115J10	
397	YB37514	KOFFEE	33	1992-09-13	2027-09-21	115J10	
398	YB37515	KOFFEE	34	1992-09-13	2027-09-21	115J10	
399	YB37516	KOFFEE	35	1992-09-13	2027-09-21	115J10	
400	YB37517	KOFFEE	36	1992-09-13	2027-09-21	115J10	
401	YB37518	KOFFEE	37	1992-09-13	2027-09-21	115J10	
402	YB37519	KOFFEE	38	1992-09-13	2027-09-21	115J10	
403	YB37520	KOFFEE	39	1992-09-13	2027-09-21	115J10	
404	YB37521	KOFFEE	40	1992-09-13	2027-09-21	115J10	
405	YB37522	KOFFEE	41	1992-09-13	2027-09-21	115J10	
406	YB37523	KOFFEE	42	1992-09-13	2027-09-21	115J10	
407	YB37524	KOFFEE	43	1992-09-13	2027-09-21	115J10	
408	YB37525	KOFFEE	44	1992-09-13	2027-09-21	115J10	
409	YB37526	KOFFEE	45	1992-09-13	2027-09-21	115J10	
410	YB37527	KOFFEE	46	1992-09-13	2027-09-21	115J10	
411	YB37528	KOFFEE	47	1992-09-13	2027-09-21	115J10	
412	YB37529	KOFFEE	48	1992-09-13	2027-09-21	115J10	
413	YB37530	KOFFEE	49	1992-09-13	2027-09-21	115J10	
414	YB37531	KOFFEE	50	1992-09-13	2027-09-21	115J10	
415	YB37532	KOFFEE	51	1992-09-13	2027-09-21	115J10	
416	YB37533	KOFFEE	52	1992-09-13	2027-09-21	115J10	
417	YB37534	KOFFEE	53	1992-09-13	2027-09-21	115J10	
418	YB37535	KOFFEE	54	1992-09-13	2027-09-21	115J10	
419	YB37536	KOFFEE	55	1992-09-13	2027-09-21	115J10	
420	YB37537	KOFFEE	56	1992-09-13	2027-09-21	115J10	
421	YB37538	KOFFEE	57	1992-09-13	2027-09-21	115J10	
422	YB37539	KOFFEE	58	1992-09-13	2027-09-21	115J10	
423	YC99914	KANA	36	2009-09-01	2027-09-29	115J15	Partial Quartz fraction (<25 acres)
424	YB37801	ICE	1	1993-01-21	2028-01-27	115J10	
425	YB37802	ICE	2	1993-01-21	2028-01-27	115J11	
426	YB37803	ICE	3	1993-01-21	2028-01-27	115J11	
427	YB37804	ICE	4	1993-01-21	2028-01-27	115J11	
428	YB37805	ICE	5	1993-01-21	2028-01-27	115J11	
429	YB37809	ICE	9	1993-01-22	2028-01-27	115J10	
430	YB37810	ICE	10	1993-01-22	2028-01-27	115J10	
431	YB37811	ICE	11	1993-01-22	2028-01-27	115J11	
432	YB37812	ICE	12	1993-01-22	2028-01-27	115J11	
433	YB37813	ICE	13	1993-01-22	2028-01-27	115J11	
434	YB37814	ICE	14	1993-01-22	2028-01-27	115J11	
435	YB37815	ICE	15	1993-01-22	2028-01-27	115J11	
436	YB37816	ICE	16	1993-01-22	2028-01-27	115J11	
437	YB37817	ICE	17	1993-01-22	2028-01-27	115J11	
438	YB37818	ICE	18	1993-01-22	2028-01-27	115J11	
439	YB37825	ICE	25	1993-01-22	2028-01-27	115J10	
440	YB37826	ICE	26	1993-01-22	2028-01-27	115J10	
441	YB37827	ICE	27	1993-01-22	2028-01-27	115J11	
442	YB37828	ICE	28	1993-01-22	2028-01-27	115J11	
443	YB37829	ICE	29	1993-01-22	2028-01-27	115J11	
444	YA86735	ANA	1	1985-04-25	2028-02-17	115J10	
445	YA86736	ANA	2	1985-04-25	2028-02-17	115J10	
446	YA86737	ANA	3	1985-04-25	2028-02-17	115J10	

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447	YA86738	ANA	4	1985-04-25	2028-02-17	115J10	
448	YA86739	ANA	5	1985-04-25	2028-02-17	115J10	
449	YA86740	ANA	6	1985-04-25	2028-02-17	115J10	
450	YA86741	ANA	7	1985-04-25	2028-02-17	115J10	
451	YA86742	ANA	8	1985-04-25	2028-02-17	115J10	
452	YA86743	ANA	9	1985-04-25	2028-02-17	115J10	
453	YA86744	ANA	10	1985-04-25	2028-02-17	115J10	
454	YA86749	ANA	15	1985-04-25	2028-02-17	115J10	
455	YA86750	ANA	16	1985-04-25	2028-02-17	115J10	
456	YA86751	ANA	17	1985-04-25	2028-02-17	115J10	
457	YA86752	ANA	18	1985-04-25	2028-02-17	115J10	
458	YA86753	ANA	19	1985-04-25	2028-02-17	115J10	
459	YA86754	ANA	20	1985-04-25	2028-02-17	115J10	
460	YA86755	ANA	21	1985-04-25	2028-02-17	115J10	
461	YA86756	ANA	22	1985-04-25	2028-02-17	115J10	
462	YA86757	ANA	23	1985-04-25	2028-02-17	115J10	
463	YA86758	ANA	24	1985-04-25	2028-02-17	115J10	
464	YA86759	ANA	25	1985-04-25	2028-02-17	115J10	
465	YA86760	ANA	26	1985-04-25	2028-02-17	115J10	
466	YA86763	ANA	29	1985-04-25	2028-02-17	115J10	
467	YA86764	ANA	30	1985-04-25	2028-02-17	115J10	
468	YA86765	ANA	31	1985-04-25	2028-02-17	115J10	
469	YA86766	ANA	32	1985-04-25	2028-02-17	115J10	
470	YA86767	ANA	33	1985-04-25	2028-02-17	115J10	
471	YA86768	ANA	34	1985-04-25	2028-02-17	115J10	
472	YA86769	ANA	35	1985-04-25	2028-02-17	115J10	
473	YA86770	ANA	36	1985-04-25	2028-02-17	115J10	
474	YA86771	ANA	37	1985-04-25	2028-02-17	115J10	
475	YA86772	ANA	38	1985-04-25	2028-02-17	115J10	
476	YA86773	ANA	39	1985-04-25	2028-02-17	115J10	
477	YA86774	ANA	40	1985-04-25	2028-02-17	115J10	
478	YA86777	ANA	43	1985-04-25	2028-02-17	115J10	
479	YA86778	ANA	44	1985-04-25	2028-02-17	115J10	
480	YA86779	ANA	45	1985-04-25	2028-02-17	115J10	
481	YA86780	ANA	46	1985-04-25	2028-02-17	115J10	
482	YA86781	ANA	47	1985-04-25	2028-02-17	115J10	
483	YA86782	ANA	48	1985-04-25	2028-02-17	115J10	
484	YA86783	ANA	49	1985-04-25	2028-02-17	115J10	
485	YA86784	ANA	50	1985-04-25	2028-02-17	115J10	
486	YA86785	ANA	51	1985-04-25	2028-02-17	115J10	
487	YA86786	ANA	52	1985-04-25	2028-02-17	115J10	
488	YA86787	ANA	53	1985-04-25	2028-02-17	115J10	
489	YA86788	ANA	54	1985-04-25	2028-02-17	115J10	
490	YE32245	PAL	1	2016-05-17	2028-03-25	115J10	
491	YE32246	PAL	2	2016-05-17	2028-03-25	115J10	
492	YE32247	PAL	3	2016-05-17	2028-03-25	115J10	
493	YE32248	PAL	4	2016-05-17	2028-03-25	115J10	
494	YE32249	PAL	5	2016-05-17	2028-03-25	115J10	
495	YE32138	PAL	6	2016-05-16	2028-03-25	115J10	
496	YE32139	PAL	7	2016-05-16	2028-03-25	115J10	
497	YE32140	PAL	8	2016-05-16	2028-03-25	115J10	
498	YE32141	PAL	9	2016-05-16	2028-03-25	115J10	
499	YE32142	PAL	10	2016-05-16	2028-03-25	115J10	
500	YE32143	PAL	11	2016-05-16	2028-03-25	115J10	
501	YE32144	PAL	12	2016-05-16	2028-03-25	115J10	
502	YE32145	PAL	13	2016-05-16	2028-03-25	115J10	

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#	Grant Number	Claim Name	Claim Number	Staking date	Expiry Date	NTS Map	Non Standard Size
503	YE32146	PAL	14	2016-05-17	2028-03-25	115J10	
504	YE32147	PAL	15	2016-05-17	2028-03-25	115J10	
505	YE32148	PAL	16	2016-05-17	2028-03-25	115J10	
506	YE32149	PAL	17	2016-05-17	2028-03-25	115J10	
507	YE32150	PAL	18	2016-05-17	2028-03-25	115J10	
508	YE32151	PAL	19	2016-05-17	2028-03-25	115J10	
509	YE32152	PAL	20	2016-05-17	2028-03-25	115J10	
510	YE32153	PAL	21	2016-05-17	2028-03-25	115J10	
511	YE32154	PAL	22	2016-05-17	2028-03-25	115J10	
512	YE32155	PAL	23	2016-05-17	2028-03-25	115J10	
513	YE32156	PAL	24	2016-05-17	2028-03-25	115J10	
514	YE32157	PAL	25	2016-05-17	2028-03-25	115J10	
515	YE32196	PAL	26	2016-05-17	2028-03-25	115J10	
516	YE32197	PAL	27	2016-05-17	2028-03-25	115J10	
517	YE32178	PAL	28	2016-05-17	2028-03-25	115J10	
518	YE32179	PAL	29	2016-05-17	2028-03-25	115J10	
519	YE32180	PAL	30	2016-05-16	2028-03-25	115J10	
520	YE32181	PAL	31	2016-05-16	2028-03-25	115J10	
521	YE32182	PAL	32	2016-05-16	2028-03-25	115J10	
522	YE32183	PAL	33	2016-05-16	2028-03-25	115J10	
523	YE32184	PAL	34	2016-05-16	2028-03-25	115J10	
524	YE32185	PAL	35	2016-05-16	2028-03-25	115J10	
525	YE32186	PAL	36	2016-05-16	2028-03-25	115J10	
526	YE32187	PAL	37	2016-05-16	2028-03-25	115J10	
527	YE32101	PAL	38	2016-05-16	2028-03-25	115J10	
528	YE32102	PAL	39	2016-05-16	2028-03-25	115J10	
529	YE32103	PAL	40	2016-05-16	2028-03-25	115J10	
530	YE32104	PAL	41	2016-05-16	2028-03-25	115J10	
531	YE32106	PAL	42	2016-05-16	2028-03-25	115J10	
532	YE32105	PAL	43	2016-05-16	2028-03-25	115J10	
533	YE32108	PAL	44	2016-05-16	2028-03-25	115J10	
534	YE32107	PAL	45	2016-05-16	2028-03-25	115J10	
535	YE32110	PAL	46	2016-05-16	2028-03-25	115J10	
536	YE32109	PAL	47	2016-05-16	2028-03-25	115J10	
537	YE32112	PAL	48	2016-05-17	2028-03-25	115J10	
538	YE32111	PAL	49	2016-05-17	2028-03-25	115J10	
539	YE32114	PAL	50	2016-05-17	2028-03-25	115J10	
540	YE32113	PAL	51	2016-05-17	2028-03-25	115J10	
541	YE32115	PAL	52	2016-05-17	2028-03-25	115J10	
542	YE32116	PAL	53	2016-05-17	2028-03-25	115J10	
543	YE32118	PAL	54	2016-05-17	2028-03-25	115J10	
544	YE32117	PAL	55	2016-05-17	2028-03-25	115J10	
545	YE32232	PAL	56	2016-05-17	2028-03-25	115J10	
546	YE32231	PAL	57	2016-05-17	2028-03-25	115J10	
547	YE32233	PAL	58	2016-05-17	2028-03-25	115J10	
548	YE32234	PAL	60	2016-05-17	2028-03-25	115J10	
549	YE32235	PAL	62	2016-05-17	2028-03-25	115J10	
550	YE32236	PAL	64	2016-05-17	2028-03-25	115J10	
551	YE32237	PAL	65	2016-05-17	2028-03-25	115J10	
552	YE32135	PAL	66	2016-05-17	2028-03-25	115J10	
553	YE32136	PAL	67	2016-05-17	2028-03-25	115J10	
554	YE32133	PAL	68	2016-05-17	2028-03-25	115J10	
555	YE32134	PAL	69	2016-05-17	2028-03-25	115J10	
556	YE32119	PAL	70	2016-05-16	2028-03-25	115J10	
557	YE32120	PAL	71	2016-05-16	2028-03-25	115J10	
558	YE32121	PAL	72	2016-05-16	2028-03-25	115J10	

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#	Grant Number	Claim Name	Claim Number	Staking date	Expiry Date	NTS Map	Non Standard Size
559	YE32122	PAL	73	2016-05-16	2028-03-25	115J10	
560	YE32123	PAL	74	2016-05-16	2028-03-25	115J10	
561	YE32124	PAL	75	2016-05-16	2028-03-25	115J10	
562	YE32125	PAL	108	2016-05-17	2028-03-25	115J10	
563	YE32126	PAL	109	2016-05-17	2028-03-25	115J10	
564	YE32127	PAL	110	2016-05-17	2028-03-25	115J10	
565	YE32128	PAL	111	2016-05-17	2028-03-25	115J10	
566	YE32129	PAL	112	2016-05-17	2028-03-25	115J10	
567	YE32130	PAL	113	2016-05-17	2028-03-25	115J10	
568	YE32131	PAL	114	2016-05-17	2028-03-25	115J10	
569	YE32132	PAL	115	2016-05-17	2028-03-25	115J10	
570	YE32188	PAL	143	2016-05-17	2028-03-25	115J10	
571	YE32200	PAL	150	2016-05-17	2028-03-25	115J10	
572	YE32189	PAL	151	2016-05-17	2028-03-25	115J10	
573	YE32190	PAL	152	2016-05-17	2028-03-25	115J10	
574	YE32191	PAL	153	2016-05-17	2028-03-25	115J10	
575	YE32192	PAL	154	2016-05-17	2028-03-25	115J10	
576	YE32193	PAL	155	2016-05-17	2028-03-25	115J10	
577	YD60030	AXS	137	2010-05-11	2028-03-25	115J09	
578	YD60031	AXS	138	2010-05-11	2028-03-25	115J09	
579	YD60032	AXS	139	2010-05-11	2028-03-25	115J09	
580	YD60033	AXS	140	2010-05-11	2028-03-25	115J09	
581	YD60034	AXS	141	2010-05-11	2028-03-25	115J09	
582	YD60035	AXS	142	2010-05-11	2028-03-25	115J09	
583	YD60036	AXS	143	2010-05-11	2028-03-25	115J10	
584	YD60037	AXS	144	2010-05-11	2028-03-25	115J10	
585	YD60038	AXS	145	2010-05-11	2028-03-25	115J10	
586	YD60039	AXS	146	2010-05-11	2028-03-25	115J10	
587	YD60040	AXS	147	2010-05-11	2028-03-25	115J10	
588	YD60041	AXS	148	2010-05-11	2028-03-25	115J10	
589	YD60042	AXS	149	2010-05-11	2028-03-25	115J10	
590	YD60043	AXS	150	2010-05-11	2028-03-25	115J10	
591	YD60044	AXS	151	2010-05-11	2028-03-25	115J10	
592	YD60045	AXS	152	2010-05-11	2028-03-25	115J10	
593	YD60046	AXS	154	2010-05-11	2028-03-25	115J10	
594	YD60047	AXS	153	2010-05-11	2028-03-25	115J10	
595	YD60048	AXS	155	2010-05-11	2028-03-25	115J10	
596	YD60049	AXS	156	2010-05-11	2028-03-25	115J10	
597	YD60050	AXS	157	2010-05-11	2028-03-25	115J10	
598	YD60051	AXS	158	2010-05-11	2028-03-25	115J10	
599	YD60052	AXS	159	2010-05-11	2028-03-25	115J10	
600	YD60053	AXS	160	2010-05-11	2028-03-25	115J10	
601	YD60054	AXS	161	2010-05-11	2028-03-25	115J10	
602	YD60055	AXS	162	2010-05-11	2028-03-25	115J10	
603	YD60056	AXS	163	2010-05-11	2028-03-25	115J10	
604	YD60057	AXS	164	2010-05-11	2028-03-25	115J10	
605	YD60058	AXS	165	2010-05-11	2028-03-25	115J10	
606	YD60059	AXS	166	2010-05-11	2028-03-25	115J10	
607	YD60060	AXS	167	2010-05-11	2028-03-25	115J10	
608	YD60061	AXS	168	2010-05-11	2028-03-25	115J10	
609	YD60062	AXS	169	2010-05-11	2028-03-25	115J10	
610	YD60063	AXS	170	2010-05-11	2028-03-25	115J10	
611	YD60064	AXS	171	2010-05-11	2028-03-25	115J10	
612	YD60065	AXS	172	2010-05-11	2028-03-25	115J10	
613	YD60066	AXS	173	2010-05-11	2028-03-25	115J09	
614	YD60067	AXS	174	2010-05-11	2028-03-25	115J09	

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#	Grant Number	Claim Name	Claim Number	Staking date	Expiry Date	NTS Map	Non Standard Size
615	YD60068	AXS	175	2010-05-11	2028-03-25	115J09	
616	YD60069	AXS	176	2010-05-11	2028-03-25	115J09	
617	YD60070	AXS	177	2010-05-11	2028-03-25	115J09	
618	YD60071	AXS	178	2010-05-11	2028-03-25	115J09	
619	YD60072	AXS	179	2010-05-11	2028-03-25	115J09	
620	YD60073	AXS	180	2010-05-11	2028-03-25	115J09	
621	YD60074	AXS	181	2010-05-11	2028-03-25	115J09	
622	YD60075	AXS	182	2010-05-11	2028-03-25	115J09	
623	YD60076	AXS	183	2010-05-11	2028-03-25	115J09	
624	YD60077	AXS	184	2010-05-11	2028-03-25	115J09	
625	YD60078	AXS	185	2010-05-11	2028-03-25	115J09	
626	YD60079	AXS	186	2010-05-11	2028-03-25	115J09	
627	YD61120	AXS	187	2010-05-11	2028-03-25	115J10	
628	YD61121	AXS	188	2010-05-11	2028-03-25	115J10	
629	YD61122	AXS	189	2010-05-11	2028-03-25	115J10	
630	YD61123	AXS	190	2010-05-11	2028-03-25	115J10	
631	YD61124	AXS	191	2010-05-11	2028-03-25	115J10	
632	YD61125	AXS	192	2010-05-11	2028-03-25	115J10	
633	YD61126	AXS	193	2010-05-11	2028-03-25	115J10	
634	YD61127	AXS	194	2010-05-11	2028-03-25	115J10	
635	YD61128	AXS	196	2010-05-11	2028-03-25	115J10	
636	YD61129	AXS	195	2010-05-11	2028-03-25	115J10	
637	YD61130	AXS	197	2010-05-11	2028-03-25	115J10	
638	YD61131	AXS	198	2010-05-11	2028-03-25	115J10	
639	YD61132	AXS	199	2010-05-11	2028-03-25	115J10	
640	95744	CAT	67	1965-12-05	2031-03-25	115J10	
641	95746	CAT	69	1965-12-05	2031-03-25	115J10	
642	Y 35194	MOUSE	3	1969-06-04	2031-03-25	115J10	
643	Y 35196	MOUSE	5	1969-06-04	2031-03-25	115J10	
644	Y 35198	MOUSE	7	1969-06-04	2031-03-25	115J10	
645	Y 35199	MOUSE	8	1969-06-04	2031-03-25	115J10	
646	Y 35200	MOUSE	9	1969-06-04	2031-03-25	115J10	
647	Y 35201	MOUSE	10	1969-06-04	2031-03-25	115J10	
648	Y 35202	MOUSE	11	1969-06-04	2031-03-25	115J10	
649	Y 35203	MOUSE	12	1969-06-04	2031-03-25	115J10	
650	Y 35204	MOUSE	13	1969-06-04	2031-03-25	115J10	
651	Y 35205	MOUSE	14	1969-06-04	2031-03-25	115J10	
652	Y 35206	MOUSE	15	1969-06-04	2031-03-25	115J10	
653	Y 35207	MOUSE	16	1969-06-04	2031-03-25	115J10	
654	Y 35483	MOUSE	89	1969-06-22	2031-03-25	115J10	
655	Y 35491	MOUSE	97	1969-06-22	2031-03-25	115J10	
656	Y 35492	MOUSE	98	1969-06-22	2031-03-25	115J10	
657	Y 35517	MOUSE	123	1969-06-22	2031-03-25	115J10	
658	Y 35518	MOUSE	124	1969-06-22	2031-03-25	115J10	
659	Y 35519	MOUSE	125	1969-06-22	2031-03-25	115J10	
660	Y 35520	MOUSE	126	1969-06-22	2031-03-25	115J10	
661	Y 35521	MOUSE	127	1969-06-22	2031-03-25	115J10	
662	Y 35522	MOUSE	128	1969-06-22	2031-03-25	115J10	
663	YB36618	CAS	31	1991-11-28	2031-03-25	115J10	
664	YB36619	CAS	32	1991-11-28	2031-03-25	115J10	
665	YB36620	CAS	33	1991-11-28	2031-03-25	115J15	
666	YB36621	CAS	34	1991-11-28	2031-03-25	115J15	
667	YB36622	CAS	35	1991-11-28	2031-03-25	115J15	
668	YB36623	CAS	36	1991-11-28	2031-03-25	115J15	
669	YB37242	E	23	1992-09-01	2031-03-25	115J15	
670	YB37243	E	24	1992-09-01	2031-03-25	115J15	

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#	Grant Number	Claim Name	Claim Number	Staking date	Expiry Date	NTS Map	Non Standard Size
671	YB37244	E	25	1992-09-01	2031-03-25	115J15	
672	YB37246	E	27	1992-09-01	2031-03-25	115J10	
673	YB37247	E	28	1992-09-01	2031-03-25	115J10	
674	YB37248	E	29	1992-09-01	2031-03-25	115J15	
675	YB37249	E	30	1992-09-01	2031-03-25	115J15	
676	YB37250	E	31	1992-09-01	2031-03-25	115J15	
677	YB37251	E	32	1992-09-01	2031-03-25	115J15	
678	YB37278	F	27	1992-08-30	2031-03-25	115J10	
679	YB37279	F	28	1992-08-30	2031-03-25	115J10	
680	YB37640	I	1	1992-09-09	2031-03-25	115J10	
681	YB37641	I	2	1992-09-09	2031-03-25	115J10	
682	YB37642	I	3	1992-09-09	2031-03-25	115J10	
683	YB37643	I	4	1992-09-09	2031-03-25	115J10	
684	YB37658	I	19	1992-09-09	2031-03-25	115J10	
685	YB37659	I	20	1992-09-09	2031-03-25	115J10	
686	YC81316	BRIT	1	2008-06-10	2034-03-05	115J15	
687	YC81317	BRIT	2	2008-06-10	2034-03-05	115J15	
688	YC81318	BRIT	3	2008-06-10	2034-03-05	115J15	
689	YC81319	BRIT	4	2008-06-10	2034-03-05	115J15	
690	YC81320	BRIT	5	2008-06-10	2034-03-05	115J15	
691	YC81321	BRIT	6	2008-06-10	2034-03-05	115J15	
692	YC81322	BRIT	7	2008-06-10	2034-03-05	115J15	
693	YC81323	BRIT	8	2008-06-10	2034-03-05	115J15	
694	YC81324	BRIT	9	2008-06-10	2034-03-05	115J15	
695	YC81325	BRIT	10	2008-06-10	2034-03-05	115J15	
696	YC81326	BRIT	11	2008-06-10	2034-03-05	115J15	
697	YC81327	BRIT	12	2008-06-10	2034-03-05	115J15	
698	YC81328	BRIT	13	2008-06-10	2034-03-05	115J15	
699	YC81329	BRIT	14	2008-06-10	2034-03-05	115J15	
700	YC81330	BRIT	15	2008-06-10	2034-03-05	115J15	
701	YC81331	BRIT	16	2008-06-10	2034-03-05	115J15	
702	YC81332	BRIT	17	2008-06-10	2034-03-05	115J15	
703	YC81333	BRIT	18	2008-06-10	2034-03-05	115J15	
704	YC81334	BRIT	19	2008-06-10	2034-03-05	115J15	
705	YC81335	BRIT	20	2008-06-10	2034-03-05	115J15	
706	YC81336	BRIT	21	2008-06-10	2034-03-05	115J15	
707	YC81337	BRIT	22	2008-06-10	2034-03-05	115J15	
708	YC81338	BRIT	23	2008-06-10	2034-03-05	115J15	
709	YC81339	BRIT	24	2008-06-10	2034-03-05	115J15	
710	YC81340	BRIT	25	2008-06-10	2034-03-05	115J15	
711	YC81341	BRIT	26	2008-06-10	2034-03-05	115J15	
712	YC81342	BRIT	27	2008-06-10	2034-03-05	115J15	
713	YC81343	BRIT	28	2008-06-10	2034-03-05	115J15	
714	YC81344	BRIT	29	2008-06-10	2034-03-05	115J15	
715	YC81345	BRIT	30	2008-06-10	2034-03-05	115J15	
716	YC81346	BRIT	31	2008-06-10	2034-03-05	115J15	
717	YC81347	BRIT	32	2008-06-10	2034-03-05	115J15	
718	YC81348	BRIT	33	2008-06-10	2034-03-05	115J15	
719	YC81349	BRIT	34	2008-06-10	2034-03-05	115J15	
720	YC81350	BRIT	35	2008-06-10	2034-03-05	115J15	
721	YC81351	BRIT	36	2008-06-10	2034-03-05	115J15	
722	YC81352	BRIT	37	2008-06-10	2034-03-05	115J15	
723	YC81353	BRIT	38	2008-06-10	2034-03-05	115J15	
724	YC81354	BRIT	39	2008-06-10	2034-03-05	115J15	
725	YC81355	BRIT	40	2008-06-10	2034-03-05	115J15	
726	YC81356	BRIT	41	2008-06-10	2034-03-05	115J15	

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#	Grant Number	Claim Name	Claim Number	Staking date	Expiry Date	NTS Map	Non Standard Size
727	YC81357	BRIT	42	2008-06-10	2034-03-05	115J15	
728	YC81358	BRIT	43	2008-06-10	2034-03-05	115J15	
729	YC81359	BRIT	44	2008-06-10	2034-03-05	115J15	
730	YC81360	BRIT	45	2008-06-10	2034-03-05	115J15	
731	YC81361	BRIT	46	2008-06-10	2034-03-05	115J15	
732	YC81362	BRIT	47	2008-06-10	2034-03-05	115J15	
733	YC81363	BRIT	48	2008-06-10	2034-03-05	115J15	
734	YC81364	BRIT	49	2008-06-10	2034-03-05	115J15	
735	YC81365	BRIT	50	2008-06-10	2034-03-05	115J15	
736	YC81366	BRIT	51	2008-06-10	2034-03-05	115J15	
737	YC81367	BRIT	52	2008-06-10	2034-03-05	115J15	
738	YC81368	BRIT	53	2008-06-10	2034-03-05	115J15	
739	YC81369	BRIT	54	2008-06-10	2034-03-05	115J15	
740	YC81370	BRIT	55	2008-06-10	2034-03-05	115J15	
741	YC81371	BRIT	56	2008-06-10	2034-03-05	115J15	
742	YC81372	BRIT	57	2008-06-10	2034-03-05	115J15	
743	YC81373	BRIT	58	2008-06-10	2034-03-05	115J15	
744	YC81374	BRIT	59	2008-06-10	2034-03-05	115J15	
745	YC81375	BRIT	60	2008-06-10	2034-03-05	115J15	
746	YC81376	BRIT	61	2008-06-10	2034-03-05	115J15	
747	YC81377	BRIT	62	2008-06-10	2034-03-05	115J15	
748	YC81378	BRIT	63	2008-06-10	2034-03-05	115J15	
749	YC81379	CC	1	2008-06-12	2034-03-05	115J10	
750	YC81380	CC	2	2008-06-12	2034-03-05	115J10	
751	YC81381	CC	3	2008-06-12	2034-03-05	115J10	
752	YC81382	CC	4	2008-06-12	2034-03-05	115J10	
753	YC81383	CC	5	2008-06-12	2034-03-05	115J10	
754	YC81384	CC	6	2008-06-12	2034-03-05	115J10	
755	YC81385	CC	7	2008-06-11	2034-03-05	115J10	
756	YC81386	CC	8	2008-06-11	2034-03-05	115J10	
757	YC81387	CC	9	2008-06-11	2034-03-05	115J10	
758	YC81388	CC	10	2008-06-11	2034-03-05	115J10	
759	YC81389	CC	11	2008-06-11	2034-03-05	115J10	
760	YC81390	CC	12	2008-06-11	2034-03-05	115J10	
761	YC81391	CC	13	2008-06-11	2034-03-05	115J10	
762	YC81392	CC	14	2008-06-11	2034-03-05	115J10	
763	YC81393	CC	15	2008-06-11	2034-03-05	115J10	
764	YC81394	CC	16	2008-06-11	2034-03-05	115J10	
765	YC81395	CC	17	2008-06-11	2034-03-05	115J10	
766	YC81396	CC	18	2008-06-11	2034-03-05	115J10	
767	YC81397	CC	19	2008-06-11	2034-03-05	115J10	
768	YC81398	CC	20	2008-06-11	2034-03-05	115J10	
769	YC81399	CC	21	2008-06-12	2034-03-05	115J10	
770	YC81400	CC	22	2008-06-12	2034-03-05	115J10	
771	YC81401	CC	23	2008-06-12	2034-03-05	115J10	
772	YC81402	CC	24	2008-06-12	2034-03-05	115J10	
773	YC81403	CC	25	2008-06-12	2034-03-05	115J10	
774	YC81404	CC	26	2008-06-12	2034-03-05	115J10	
775	YC81405	CC	27	2008-06-12	2034-03-05	115J10	
776	YC81406	CC	28	2008-06-12	2034-03-05	115J10	
777	YC81407	CC	29	2008-06-12	2034-03-05	115J10	
778	YC81408	CC	30	2008-06-11	2034-03-05	115J10	
779	YC81409	CC	31	2008-06-11	2034-03-05	115J10	
780	YC81410	CC	32	2008-06-11	2034-03-05	115J10	
781	YC81411	CC	33	2008-06-11	2034-03-05	115J10	
782	YC81412	CC	34	2008-06-11	2034-03-05	115J10	

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#	Grant Number	Claim Name	Claim Number	Staking date	Expiry Date	NTS Map	Non Standard Size
783	YC81413	CC	35	2008-06-11	2034-03-05	115J10	
784	YC81414	CC	36	2008-06-11	2034-03-05	115J10	
785	YC81415	CC	37	2008-06-11	2034-03-05	115J10	
786	YC81416	CC	38	2008-06-11	2034-03-05	115J10	
787	YC81417	CC	39	2008-06-11	2034-03-05	115J10	
788	YC81418	CC	40	2008-06-12	2034-03-05	115J10	
789	YC81419	CC	41	2008-06-12	2034-03-05	115J10	
790	YC81420	CC	42	2008-06-12	2034-03-05	115J10	
791	YC81421	CC	43	2008-06-12	2034-03-05	115J10	
792	YC81422	CC	44	2008-06-12	2034-03-05	115J10	
793	YC81423	CC	45	2008-06-12	2034-03-05	115J10	
794	YC81424	CC	46	2008-06-12	2034-03-05	115J10	
795	YC81425	CC	47	2008-06-12	2034-03-05	115J10	
796	YC81426	CC	48	2008-06-12	2034-03-05	115J10	
797	YC81427	CC	49	2008-06-12	2034-03-05	115J10	
798	YC81428	CC	50	2008-06-12	2034-03-05	115J10	
799	YC81429	CC	51	2008-06-12	2034-03-05	115J10	
800	YC81430	CC	52	2008-06-12	2034-03-05	115J10	
801	YC81431	CC	53	2008-06-12	2034-03-05	115J10	
802	YC81432	CC	54	2008-06-12	2034-03-05	115J10	
803	YC81433	CC	55	2008-06-12	2034-03-05	115J10	
804	YC81434	CC	56	2008-06-12	2034-03-05	115J10	
805	YC81435	CC	57	2008-06-11	2034-03-05	115J10	
806	YC81436	CC	58	2008-06-11	2034-03-05	115J10	
807	YC81437	CC	59	2008-06-11	2034-03-05	115J10	
808	YC81438	CC	60	2008-06-11	2034-03-05	115J10	
809	YC81439	CC	61	2008-06-11	2034-03-05	115J10	
810	YC81440	CC	62	2008-06-11	2034-03-05	115J10	
811	YC81441	CC	63	2008-06-12	2034-03-05	115J10	
812	YC81442	CC	64	2008-06-12	2034-03-05	115J10	
813	YC81443	CC	65	2008-06-12	2034-03-05	115J10	
814	YC81444	CC	66	2008-06-12	2034-03-05	115J10	
815	YC81445	CC	67	2008-06-12	2034-03-05	115J10	
816	YC81446	CC	68	2008-06-12	2034-03-05	115J10	
817	YC81447	CC	69	2008-06-12	2034-03-05	115J10	
818	YC81448	CC	70	2008-06-12	2034-03-05	115J10	
819	YC81449	CC	71	2008-06-12	2034-03-05	115J10	
820	YC81450	CC	72	2008-06-12	2034-03-05	115J10	
821	YC81451	CC	73	2008-06-12	2034-03-05	115J10	
822	YC81452	CC	74	2008-06-12	2034-03-05	115J10	
823	YC81453	CC	75	2008-06-12	2034-03-05	115J10	
824	YC81454	CC	76	2008-06-12	2034-03-05	115J10	
825	YC81455	CC	77	2008-06-12	2034-03-05	115J10	
826	YC81456	CC	78	2008-06-12	2034-03-05	115J10	
827	YC81457	CC	79	2008-06-12	2034-03-05	115J10	
828	YC81458	CC	80	2008-06-11	2034-03-05	115J10	
829	YC81459	CC	81	2008-06-11	2034-03-05	115J10	
830	YC81460	CC	82	2008-06-11	2034-03-05	115J10	
831	YC81461	CC	83	2008-06-13	2034-03-05	115J10	
832	YC81462	CC	84	2008-06-13	2034-03-05	115J10	
833	YC81463	CC	85	2008-06-13	2034-03-05	115J10	
834	YC81464	CC	86	2008-06-13	2034-03-05	115J10	
835	YC81465	CC	87	2008-06-13	2034-03-05	115J10	
836	YC81466	CC	88	2008-06-13	2034-03-05	115J10	
837	YC81467	CC	89	2008-06-13	2034-03-05	115J10	
838	YC81468	CC	90	2008-06-13	2034-03-05	115J10	

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#	Grant Number	Claim Name	Claim Number	Staking date	Expiry Date	NTS Map	Non Standard Size
839	YC81469	CC	91	2008-06-13	2034-03-05	115J10	
840	YC81470	CC	92	2008-06-13	2034-03-05	115J10	
841	YC81471	CC	93	2008-06-13	2034-03-05	115J10	
842	YC81472	CC	94	2008-06-13	2034-03-05	115J10	
843	Y 10693	JOE	89	1966-09-24	2036-03-05	115J10	
844	Y 10694	JOE	90	1966-09-24	2036-03-05	115J10	
845	Y 10695	JOE	91	1966-09-24	2036-03-05	115J10	
846	Y 10696	JOE	92	1966-09-24	2036-03-05	115J10	
847	Y 10697	JOE	93	1966-09-24	2036-03-05	115J10	
848	Y 10698	JOE	94	1966-09-24	2036-03-05	115J10	
849	Y 10699	JOE	95	1966-09-24	2036-03-05	115J10	
850	Y 10700	JOE	96	1966-09-24	2036-03-05	115J10	
851	Y 10702	JOE	98	1966-09-24	2036-03-05	115J10	
852	Y 10703	JOE	99	1966-09-24	2036-03-05	115J10	
853	Y 10705	JOE	101	1966-09-24	2036-03-05	115J10	
854	Y 10706	JOE	102	1966-09-24	2036-03-05	115J10	
855	Y 10707	JOE	103	1966-09-24	2036-03-05	115J15	
856	Y 10708	JOE	104	1966-09-24	2036-03-05	115J15	
857	Y 35192	MOUSE	1	1969-06-04	2036-03-05	115J10	
858	Y 35193	MOUSE	2	1969-06-04	2036-03-05	115J10	
859	Y 51850	JOE	91	1970-03-29	2036-03-05	115J10	Partial Quartz fraction (<25 acres)
860	Y 51851	JOE	92	1970-03-29	2036-03-05	115J10	Partial Quartz fraction (<25 acres)
861	Y 51852	JOE	93	1970-03-29	2036-03-05	115J10	Partial Quartz fraction (<25 acres)
862	Y 51853	JOE	94	1970-03-29	2036-03-05	115J10	Partial Quartz fraction (<25 acres)
863	Y 51854	JOE	95	1970-03-29	2036-03-05	115J10	Partial Quartz fraction (<25 acres)
864	Y 51855	JOE	96	1970-03-29	2036-03-05	115J10	Partial Quartz fraction (<25 acres)
865	YC64893	VIK	1	2007-05-27	2036-03-05	115J10	
866	YC64894	VIK	2	2007-05-27	2036-03-05	115J10	
867	YC64895	VIK	3	2007-05-27	2036-03-05	115J10	
868	YC64896	VIK	4	2007-05-27	2036-03-05	115J10	
869	YC64897	VIK	5	2007-05-27	2036-03-05	115J10	
870	YC64898	VIK	6	2007-05-27	2036-03-05	115J10	
871	YC64899	VIK	7	2007-05-27	2036-03-05	115J10	
872	YC64900	VIK	8	2007-05-27	2036-03-05	115J10	
873	YC64901	VIK	9	2007-05-27	2036-03-05	115J10	
874	YC64902	VIK	10	2007-05-27	2036-03-05	115J10	
875	YC64903	VIK	11	2007-05-27	2036-03-05	115J10	
876	YC64904	VIK	12	2007-05-27	2036-03-05	115J10	
877	YC64905	VIK	13	2007-05-27	2036-03-05	115J10	
878	YC64906	VIK	14	2007-05-27	2036-03-05	115J10	
879	YC64907	VIK	15	2007-05-27	2036-03-05	115J10	
880	YC64908	VIK	16	2007-05-27	2036-03-05	115J10	
881	YC64909	VIK	17	2007-05-27	2036-03-05	115J10	
882	YC64910	VIK	18	2007-05-28	2036-03-05	115J10	
883	YC64911	VIK	19	2007-05-28	2036-03-05	115J10	
884	YC64912	VIK	20	2007-05-28	2036-03-05	115J10	
885	YC64913	VIK	21	2007-05-28	2036-03-05	115J10	
886	YC64914	VIK	22	2007-05-25	2036-03-05	115J10	
887	YC64915	VIK	23	2007-05-25	2036-03-05	115J10	
888	YC64916	VIK	24	2007-05-25	2036-03-05	115J10	
889	YC64917	VIK	25	2007-05-25	2036-03-05	115J10	
890	YC64918	VIK	26	2007-05-25	2036-03-05	115J10	
891	YC64919	VIK	27	2007-05-25	2036-03-05	115J10	
892	YC64920	VIK	28	2007-05-25	2036-03-05	115J10	
893	YC64921	VIK	29	2007-05-25	2036-03-05	115J10	
894	YC64922	VIK	30	2007-05-25	2036-03-05	115J10	

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#	Grant Number	Claim Name	Claim Number	Staking date	Expiry Date	NTS Map	Non Standard Size
895	YC64923	VIK	31	2007-05-25	2036-03-05	115J10	
896	YC64924	VIK	32	2007-05-25	2036-03-05	115J10	
897	YC64925	VIK	33	2007-05-25	2036-03-05	115J10	
898	YC64926	VIK	34	2007-05-25	2036-03-05	115J10	
899	YC64927	VIK	35	2007-05-25	2036-03-05	115J10	
900	YC64928	VIK	36	2007-05-25	2036-03-05	115J10	
901	YC64929	VIK	37	2007-05-25	2036-03-05	115J10	
902	YC64930	VIK	38	2007-05-25	2036-03-05	115J10	
903	YC64931	VIK	39	2007-05-25	2036-03-05	115J10	
904	YC64932	VIK	40	2007-05-25	2036-03-05	115J10	
905	YC64933	VIK	41	2007-05-31	2036-03-05	115J10	
906	YC64934	VIK	42	2007-05-31	2036-03-05	115J10	
907	YC64935	VIK	43	2007-06-05	2036-03-05	115J10	
908	YC64936	VIK	44	2007-06-05	2036-03-05	115J10	
909	YC64937	VIK	45	2007-06-05	2036-03-05	115J10	
910	YC64938	VIK	46	2007-06-05	2036-03-05	115J10	
911	YC64939	VIK	47	2007-06-05	2036-03-05	115J10	
912	YC64940	VIK	48	2007-06-05	2036-03-05	115J10	
913	YC64941	VIK	49	2007-05-25	2036-03-05	115J10	
914	YC64942	VIK	50	2007-05-25	2036-03-05	115J10	
915	YC64943	VIK	51	2007-05-25	2036-03-05	115J10	
916	YC64944	VIK	52	2007-05-25	2036-03-05	115J10	
917	YC64945	VIK	53	2007-05-25	2036-03-05	115J10	
918	YC64946	VIK	54	2007-05-25	2036-03-05	115J10	
919	YC64947	VIK	55	2007-05-25	2036-03-05	115J10	
920	YC64948	VIK	56	2007-05-24	2036-03-05	115J10	
921	YC64949	VIK	57	2007-05-24	2036-03-05	115J10	
922	YC64950	VIK	58	2007-05-24	2036-03-05	115J10	
923	YC64951	VIK	59	2007-05-24	2036-03-05	115J10	
924	YC64952	VIK	60	2007-05-24	2036-03-05	115J10	
925	YC64953	VIK	61	2007-05-24	2036-03-05	115J10	
926	YC64954	VIK	62	2007-05-24	2036-03-05	115J10	
927	YC64955	VIK	63	2007-05-24	2036-03-05	115J10	
928	YC64956	VIK	64	2007-05-24	2036-03-05	115J10	
929	YC64958	VIK	66	2007-05-24	2036-03-05	115J10	
930	YC64959	VIK	67	2007-05-24	2036-03-05	115J10	
931	YC64960	VIK	68	2007-05-24	2036-03-05	115J10	
932	YC64961	VIK	69	2007-05-24	2036-03-05	115J10	
933	YC64962	VIK	70	2007-05-24	2036-03-05	115J10	
934	YC64963	VIK	71	2007-05-24	2036-03-05	115J10	
935	YC64964	VIK	72	2007-05-24	2036-03-05	115J10	
936	YC64965	VIK	73	2007-05-24	2036-03-05	115J10	
937	YC64966	VIK	74	2007-05-24	2036-03-05	115J10	
938	YC64967	VIK	75	2007-05-24	2036-03-05	115J10	
939	YC64968	VIK	76	2007-05-24	2036-03-05	115J10	
940	YC64969	VIK	77	2007-05-24	2036-03-05	115J10	
941	YC64970	VIK	78	2007-05-24	2036-03-05	115J10	
942	YC64971	VIK	79	2007-05-24	2036-03-05	115J10	
943	YC64972	VIK	80	2007-05-27	2036-03-05	115J10	
944	YC64973	VIK	81	2007-05-25	2036-03-05	115J10	
945	YC64974	VIK	82	2007-05-26	2036-03-05	115J10	
946	YC64975	VIK	83	2007-05-26	2036-03-05	115J10	
947	YC64976	VIK	84	2007-05-26	2036-03-05	115J10	
948	YC64977	VIK	85	2007-05-26	2036-03-05	115J10	
949	YC64978	VIK	86	2007-05-26	2036-03-05	115J10	
950	YC64979	VIK	87	2007-05-26	2036-03-05	115J10	

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#	Grant Number	Claim Name	Claim Number	Staking date	Expiry Date	NTS Map	Non Standard Size
951	YC64980	VIK	88	2007-05-26	2036-03-05	115J10	
952	YC64981	VIK	89	2007-05-26	2036-03-05	115J10	
953	YC64982	VIK	90	2007-05-26	2036-03-05	115J10	
954	YC64983	VIK	91	2007-05-26	2036-03-05	115J10	
955	YC64984	VIK	92	2007-05-26	2036-03-05	115J10	
956	YC64985	VIK	93	2007-05-26	2036-03-05	115J10	
957	YC64986	VIK	94	2007-05-26	2036-03-05	115J10	
958	YC64987	VIK	95	2007-05-26	2036-03-05	115J10	
959	YC64988	VIK	96	2007-05-28	2036-03-05	115J10	
960	YC64989	VIK	97	2007-05-28	2036-03-05	115J10	
961	YC64990	VIK	98	2007-05-28	2036-03-05	115J10	
962	YC64991	VIK	99	2007-05-28	2036-03-05	115J10	
963	YC64992	VIK	100	2007-05-28	2036-03-05	115J10	
964	YC64993	VIK	101	2007-05-28	2036-03-05	115J10	
965	YC64994	VIK	102	2007-05-28	2036-03-05	115J10	
966	YC64995	VIK	103	2007-05-28	2036-03-05	115J10	
967	YC64996	VIK	104	2007-05-26	2036-03-05	115J10	
968	YC64997	VIK	105	2007-05-26	2036-03-05	115J10	
969	YC64998	VIK	106	2007-05-26	2036-03-05	115J10	
970	YC64999	VIK	107	2007-05-26	2036-03-05	115J10	
971	YC65000	VIK	108	2007-05-26	2036-03-05	115J10	
972	YC65001	VIK	109	2007-05-26	2036-03-05	115J10	
973	YC65002	VIK	110	2007-05-26	2036-03-05	115J10	
974	YC65003	VIK	111	2007-05-26	2036-03-05	115J10	
975	YC65004	VIK	112	2007-05-26	2036-03-05	115J10	
976	YC65005	VIK	113	2007-05-26	2036-03-05	115J10	
977	YC65006	VIK	114	2007-05-26	2036-03-05	115J10	
978	YC65007	VIK	115	2007-05-26	2036-03-05	115J10	
979	YC65008	VIK	116	2007-05-26	2036-03-05	115J10	
980	YC65009	VIK	117	2007-05-26	2036-03-05	115J10	
981	YC65010	VIK	118	2007-05-26	2036-03-05	115J15	
982	YC65011	VIK	119	2007-05-26	2036-03-05	115J15	
983	YC65012	VIK	120	2007-05-26	2036-03-05	115J15	
984	YC65013	VIK	121	2007-05-26	2036-03-05	115J15	
985	YC65014	VIK	122	2007-05-26	2036-03-05	115J15	
986	YC65015	VIK	123	2007-05-26	2036-03-05	115J15	
987	YC65016	VIK	124	2007-05-26	2036-03-05	115J15	
988	YC65017	VIK	125	2007-05-26	2036-03-05	115J15	
989	YC65018	VIK	126	2007-05-29	2036-03-05	115J15	
990	YC65019	VIK	127	2007-05-29	2036-03-05	115J15	
991	YC65020	VIK	128	2007-05-29	2036-03-05	115J15	
992	YC65021	VIK	129	2007-05-29	2036-03-05	115J15	
993	YC65022	VIK	130	2007-05-29	2036-03-05	115J15	
994	YC65023	VIK	131	2007-05-29	2036-03-05	115J15	
995	YC65024	VIK	132	2007-05-29	2036-03-05	115J15	
996	YC65025	VIK	133	2007-05-29	2036-03-05	115J15	
997	YC65026	VIK	134	2007-05-29	2036-03-05	115J15	
998	YC65027	VIK	135	2007-05-29	2036-03-05	115J15	
999	YC65028	VIK	136	2007-05-29	2036-03-05	115J15	
1000	YC65029	VIK	137	2007-05-29	2036-03-05	115J15	
1001	YC65030	VIK	138	2007-05-29	2036-03-05	115J15	
1002	YC65031	VIK	139	2007-05-29	2036-03-05	115J15	
1003	YC65032	VIK	140	2007-05-29	2036-03-05	115J15	
1004	YC65033	VIK	141	2007-05-29	2036-03-05	115J15	
1005	YC65034	VIK	142	2007-05-29	2036-03-05	115J15	
1006	YC65035	VIK	143	2007-05-29	2036-03-05	115J15	

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#	Grant Number	Claim Name	Claim Number	Staking date	Expiry Date	NTS Map	Non Standard Size
1007	YC65036	VIK	144	2007-05-29	2036-03-05	115J15	
1008	YC65037	VIK	145	2007-05-29	2036-03-05	115J15	
1009	YC65038	VIK	146	2007-05-29	2036-03-05	115J15	
1010	YC65039	VIK	147	2007-05-29	2036-03-05	115J15	
1011	YC65040	VIK	148	2007-05-29	2036-03-05	115J15	
1012	YC65041	VIK	149	2007-05-29	2036-03-05	115J15	
1013	YC65042	VIK	150	2007-05-29	2036-03-05	115J15	
1014	YC65043	VIK	151	2007-05-29	2036-03-05	115J15	
1015	YC65044	VIK	152	2007-05-29	2036-03-05	115J15	
1016	YC65045	VIK	153	2007-05-29	2036-03-05	115J15	
1017	YC65046	VIK	154	2007-05-29	2036-03-05	115J15	
1018	YC65047	VIK	155	2007-05-29	2036-03-05	115J15	
1019	YC65048	VIK	156	2007-05-29	2036-03-05	115J15	
1020	YC65049	VIK	157	2007-05-29	2036-03-05	115J15	
1021	YC65050	VIK	158	2007-05-29	2036-03-05	115J15	
1022	YC65051	VIK	159	2007-05-29	2036-03-05	115J15	
1023	YC65052	VIK	160	2007-05-29	2036-03-05	115J15	
1024	YC65053	VIK	161	2007-05-29	2036-03-05	115J15	
1025	YC65054	VIK	162	2007-05-29	2036-03-05	115J15	
1026	YC65055	VIK	163	2007-05-29	2036-03-05	115J15	
1027	YC65056	VIK	164	2007-05-29	2036-03-05	115J15	
1028	YC65057	VIK	165	2007-05-29	2036-03-05	115J15	
1029	YC65058	VIK	166	2007-05-29	2036-03-05	115J15	
1030	YC65059	VIK	167	2007-05-29	2036-03-05	115J15	
1031	YC65060	VIK	168	2007-05-29	2036-03-05	115J15	
1032	YC65061	VIK	169	2007-05-29	2036-03-05	115J15	
1033	YC65062	VIK	170	2007-05-30	2036-03-05	115J15	
1034	YC65063	VIK	171	2007-05-30	2036-03-05	115J15	
1035	YC65064	VIK	172	2007-05-30	2036-03-05	115J15	
1036	YC65065	VIK	173	2007-05-30	2036-03-05	115J15	
1037	YC65066	VIK	174	2007-05-30	2036-03-05	115J15	
1038	YC65067	VIK	175	2007-05-30	2036-03-05	115J15	
1039	YC65068	VIK	176	2007-05-30	2036-03-05	115J15	
1040	YC65069	VIK	177	2007-05-30	2036-03-05	115J15	
1041	YC65070	VIK	178	2007-05-30	2036-03-05	115J15	
1042	YC65071	VIK	179	2007-05-30	2036-03-05	115J15	
1043	YC65072	VIK	180	2007-05-30	2036-03-05	115J15	
1044	YC65073	VIK	181	2007-05-30	2036-03-05	115J15	
1045	YC65074	VIK	182	2007-05-30	2036-03-05	115J15	
1046	YC65075	VIK	183	2007-05-30	2036-03-05	115J15	
1047	YC65076	VIK	184	2007-05-28	2036-03-05	115J15	
1048	YC65077	VIK	185	2007-05-28	2036-03-05	115J15	
1049	YC65078	VIK	186	2007-05-28	2036-03-05	115J15	
1050	YC65079	VIK	187	2007-05-29	2036-03-05	115J15	
1051	YC65080	VIK	188	2007-05-29	2036-03-05	115J15	
1052	YC64957	VIK	65	2007-05-26	2036-03-05	115J10	
1053	4252	HELICOPTER		1943-09-04	2036-03-25	115J10	
1054	56979	#1 BOMBER GROUP		1947-08-07	2036-03-25	115J10	
1055	56980	#3 BOMBER GROUP		1947-08-07	2036-03-25	115J10	
1056	56981	#5 BOMBER GROUP		1947-08-07	2036-03-25	115J10	
1057	56983	#1 AIRPORT GROU		1947-08-07	2036-03-25	115J10	
1058	56984	#3 AIRPORT GROU		1947-08-07	2036-03-25	115J10	
1059	56985	#5 AIRPORT GROU		1947-08-07	2036-03-25	115J10	
1060	56987	#2 BOMBER GROUP		1947-08-07	2036-03-25	115J10	
1061	56988	#6 BOMBER GROUP		1947-08-07	2036-03-25	115J10	
1062	56990	#2 AIRPORT GROU		1947-08-07	2036-03-25	115J10	

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#	Grant Number	Claim Name	Claim Number	Staking date	Expiry Date	NTS Map	Non Standard Size
1063	56991	#4 AIRPORT GROU		1947-08-07	2036-03-25	115J10	
1064	56992	#6 AIRPORT GROU		1947-08-07	2036-03-25	115J10	
1065	56993	#8 AIRPORT GROU		1947-07-07	2036-03-25	115J10	
1066	92201	CAT	1	1965-06-29	2036-03-25	115J10	
1067	92202	CAT	2	1965-06-29	2036-03-25	115J10	
1068	92203	CAT	3	1965-06-29	2036-03-25	115J10	
1069	92204	CAT	4	1965-06-29	2036-03-25	115J10	
1070	92205	CAT	5	1965-06-29	2036-03-25	115J10	
1071	92206	CAT	6	1965-06-29	2036-03-25	115J10	
1072	92207	CAT	7	1965-06-29	2036-03-25	115J10	
1073	92208	CAT	8	1965-06-29	2036-03-25	115J10	
1074	92209	CAT	9	1965-06-29	2036-03-25	115J10	
1075	92210	CAT	10	1965-06-29	2036-03-25	115J10	
1076	92211	CAT	11	1965-06-29	2036-03-25	115J10	
1077	92212	CAT	12	1965-06-29	2036-03-25	115J10	
1078	92213	CAT	13	1965-06-29	2036-03-25	115J10	
1079	92214	CAT	14	1965-06-29	2036-03-25	115J10	
1080	92215	CAT	15	1965-06-30	2036-03-25	115J10	
1081	92216	CAT	16	1965-06-30	2036-03-25	115J10	
1082	92217	CAT	17	1965-06-30	2036-03-25	115J10	
1083	92218	CAT	18	1965-06-30	2036-03-25	115J10	
1084	92219	CAT	19	1965-06-30	2036-03-25	115J10	
1085	92220	CAT	20	1965-06-30	2036-03-25	115J10	
1086	92221	CAT	21	1965-06-30	2036-03-25	115J10	
1087	92222	CAT	22	1965-06-30	2036-03-25	115J10	
1088	92764	CAT	23	1965-09-10	2036-03-25	115J10	
1089	92765	CAT	24	1965-09-10	2036-03-25	115J10	
1090	92766	CAT	25	1965-09-10	2036-03-25	115J10	
1091	92776	CAT	35	1965-09-11	2036-03-25	115J10	
1092	92777	CAT	36	1965-09-11	2036-03-25	115J10	
1093	92778	CAT	37	1965-09-11	2036-03-25	115J10	
1094	92779	CAT	38	1965-09-11	2036-03-25	115J10	
1095	92780	CAT	39	1965-09-12	2036-03-25	115J10	
1096	92781	CAT	40	1965-09-12	2036-03-25	115J10	
1097	92782	CAT	41	1965-09-12	2036-03-25	115J10	
1098	92783	CAT	42	1965-09-12	2036-03-25	115J10	
1099	95724	CAT	47	1965-12-02	2036-03-25	115J10	
1100	95725	CAT	48	1965-12-02	2036-03-25	115J10	
1101	95726	CAT	49	1965-12-02	2036-03-25	115J10	
1102	95727	CAT	50	1965-12-02	2036-03-25	115J10	
1103	95728	CAT	51	1965-12-02	2036-03-25	115J10	
1104	95729	CAT	52	1965-12-02	2036-03-25	115J10	
1105	95730	CAT	53	1965-12-02	2036-03-25	115J10	
1106	95731	CAT	54	1965-12-02	2036-03-25	115J10	
1107	95732	CAT	55	1965-12-02	2036-03-25	115J15	
1108	95733	CAT	56	1965-12-02	2036-03-25	115J15	
1109	95734	CAT	57	1965-12-02	2036-03-25	115J10	
1110	95735	CAT	58	1965-12-02	2036-03-25	115J10	
1111	95736	CAT	59	1965-12-02	2036-03-25	115J10	
1112	95737	CAT	60	1965-12-02	2036-03-25	115J10	
1113	95738	CAT	61	1965-12-02	2036-03-25	115J10	
1114	95739	CAT	62	1965-12-02	2036-03-25	115J10	
1115	Y 10701	JOE	97	1966-09-24	2036-03-25	115J10	
1116	Y 10704	JOE	100	1966-09-24	2036-03-25	115J10	
1117	Y 35582	MOUSE	161	1969-06-25	2036-03-25	115J10	Full Quartz fraction (25+ acres)
1118	Y 35583	MOUSE	162	1969-06-25	2036-03-25	115J10	Full Quartz fraction (25+ acres)

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#	Grant Number	Claim Name	Claim Number	Staking date	Expiry Date	NTS Map	Non Standard Size
1119	Y 35584	MOUSE	163	1969-06-25	2036-03-25	115J10	Full Quartz fraction (25+ acres)
1120	Y 35585	LOST FR.	1	1969-06-25	2036-03-25	115J10	
1121	Y 35586	LOST FR.	2	1969-06-25	2036-03-25	115J10	
1122	Y 35587	LOST FR.	3	1969-06-25	2036-03-25	115J10	
1123	Y 36686	CAT	22	1969-08-12	2036-03-25	115J10	Full Quartz fraction (25+ acres)
1124	Y 36687	CAT	47	1969-08-12	2036-03-25	115J10	Full Quartz fraction (25+ acres)
1125	Y 36688	CAT	48	1969-08-12	2036-03-25	115J10	Full Quartz fraction (25+ acres)
1126	Y 36690	CAT	62	1969-08-12	2036-03-25	115J10	Full Quartz fraction (25+ acres)
1127	Y 39601	CAT	3	1969-10-23	2036-03-25	115J10	Full Quartz fraction (25+ acres)
1128	Y 39602	CAT	4	1969-10-23	2036-03-25	115J10	Full Quartz fraction (25+ acres)
1129	Y 39603	CAT	23	1969-10-23	2036-03-25	115J10	Full Quartz fraction (25+ acres)
1130	Y 51846	CAT	1	1970-03-29	2036-03-25	115J10	Partial Quartz fraction (<25 acres)
1131	Y 51847	CAT	2	1970-03-29	2036-03-25	115J10	Full Quartz fraction (25+ acres)
1132	Y 51849	CAT	26	1970-03-30	2036-03-25	115J10	Partial Quartz fraction (<25 acres)
1133	YB37280	F	29	1992-08-30	2036-03-25	115J10	
1134	YB37282	F	31	1992-08-30	2036-03-25	115J10	
1135	YB37284	F	33	1992-08-30	2036-03-25	115J10	
1136	Y 36689	CAT	57	1969-08-12	2036-06-05	115J10	Full Quartz fraction (25+ acres)