# PREFEASIBILITY STUDY TECHNICAL REPORT ON THE KUTCHO PROJECT, BRITISH COLUMBIA

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## NOTICE

JDS Energy & Mining Inc. prepared this National Instrument 43-101 Technical Report, in accordance with Form 43-101F1, for the Desert Star Resources Ltd. The quality of information, conclusions and estimates contained herein is based on: (i) information available at the time of preparation; (ii) data supplied by outside sources, and (iii) the assumptions, conditions, and qualifications set forth in this report.

Desert Star Resources Ltd. filed this Technical Report with the Canadian Securities Regulatory Authorities pursuant to provincial securities legislation. Except for the purposes legislated under provincial securities law, any other use of this report by any third party is at that party's sole risk.

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# 1 Executive Summary

## 1.1 Introduction

This Prefeasibility Study (PFS) Technical Report was compiled by JDS Energy & Mining Inc. (JDS) for Desert Star Resources Ltd. (Desert Star). Desert Star has signed a definitive agreement dated June 15, 2017 with Capstone Mining Corp. (Capstone) to acquire the Kutcho Project (the Project). Capstone owns 100% of the Project through its wholly-owned subsidiary, Kutcho Copper Corp. (Kutcho Copper).

This technical report (JDS 2017) summarizes the results of an updated PFS. In 2011, JDS conducted a PFS on the Project titled *Kutcho Copper Project Prefeasibility Study, British Columbia* with an effective date of February 15, 2011 (JDS 2011). This 2017 PFS uses the same mine, mineral processing plant, and infrastructure designs and plans as the 2011 JDS PFS. The purpose of this 2017 PFS is to show the economics of the Project based on current costs, metal prices, exchange rates, mineral resources, and metallurgical interpretations.

All dollar values in this report are Canadian dollars ($ or C$) unless otherwise stated.

## 1.2 Project Description

The Project contains three main mineralized zones: Main, Esso, and Sumac. Only Main and Esso deposits are used in the mine planning and economics of the PFS. The Sumac deposit was excluded as its resources are only at the Inferred level. The payable metals found in the deposits are copper (Cu), zinc (Zn), silver (Ag), and gold (Au) in order of economic value.

The deposits are planned to be mined predominantly by underground methods. Access to the Main and Esso deposits will be via two separate portals. The Main deposit extends from surface to about 250 metres (m) in depth. The Esso deposit is located approximately 420 m below surface and extends vertically about 200 m. The Main deposit has a strike length of about 1.5 kilometres (km) while Esso has a strike length of about 600 m. The deposits range in thickness from 3 to 20 m and have dips ranging from 30 to 70 degrees (º). Esso and Main are about 1.5 km apart.

The underground mine is envisioned to produce at an average annual rate of approximately 2,500 tonnes per day (t/d) and will operate year-round for a total of 10.4 million tonnes (Mt) of ore and a 12 year mine life. A small starter open pit will extract about 0.4 Mt of ore from the Main deposit to provide preliminary mill feed material and non-potentially acid generating (non-PAG) construction material. Planned underground mining methods are sub-level longhole (LH) stoping for steeper dipping zones, and mechanized cut and fill (MCF or C&F) for shallower dipping areas. Both methods will use paste backfill. Ore will be trucked from underground to the process plant which is adjacent to the Main portal.

The processing plant will operate year-round at an average rate of 2,500 t/d. The processing plant will consist of primary crushing, semi-autogenous grinding (SAG), a ball mill, sequential copper and zinc flotation, concentrate de-watering, tailings disposal and back-fill production. Copper rougher product will be re-ground prior to cleaning. The design Bond Work Index, a measure of grindability, is 12.2 ($P_{80}$ 58 microns [µ]).

The tailings management facility (TMF), will be fully lined and contain paste tailings. About one half of the total tailings will be placed underground. Approximately 1.0 Mt of tailings will be placed in the mined-out open pit.

Approximately 393 million pounds (Mlb) of Cu, 557 Mlbs of Zn, 5,576 thousand troy ounces (koz) of Ag and 51 koz of Au will be recovered from the deposits over the mine life from an average head grade of 2.01% Cu, 3.19% Zn, 0.37 grams per tonne (g/t) Au, and 34.6 g/t Ag.

## 1.3 Location

The Kutcho property is located approximately 100 km due east of Dease Lake in the Liard mining division of Northern British Columbia (BC). The site is located at approximately 1,500 m elevation, has an average annual temperature of -1 degrees Celsius (ºC) and experiences 0.5 m of precipitation annually, half of which is snow.

The site is accessible via a 900 m long gravel airstrip located 10 km from the deposit and a 100 km long seasonal road from Dease Lake that is only suitable for off-highway vehicles during the summer months.

## 1.4 Geology and Mineralization

Located near the eastern end of an east-west striking narrow allochthonous belt of island arc volcanic rocks of Permotriassic age, the Kutcho property contains three known Kuroko-type volcanogenic massive sulphide (VMS) deposits. They are aligned in a westerly plunging linear trend and from east to west they are called the Main, Sumac, and Esso deposits. The largest of the three, the Main deposit comes to surface near the eastern end of this trend, whereas the Esso deposit occurs at depths about 400 to 520 m below surface at the western or down plunge end of the trend as it is currently known. The trend is open down plunge but is poorly explored presumably due to the depths of any projected extension.

The mineralized zone in the Main deposit dips at an average of 45º to the north but ranges from 38º in the east to 63º in the west. Changes in foliation angles and the dip of the mineralized zone also suggest it is openly buckled. Internal stratigraphy and mineral zoning is known from drillhole interpretations and from one continuous cross-section mapped in an adit located roughly at the center of the strike length. Grade trends exhibited on long-sections suggest there are other controls to higher grade copper and zinc mineralization however these controls are not known.

The Esso lens has an elongate shape, approximately 680 m long, up to 110 m wide, and up to 21 m thick. The deposit consists of two connected lenses, the upper lens being the larger of the two. The mineralization at Esso deposit is higher grade than at Main or Sumac deposits, but displays similar mineral zonation with copper or zinc layers or zones, as well as zonation in thickness and grade from the central deposit area. Alteration at Esso is similar to the Main deposit, where sericite alteration of feldspars in the hangingwall is gradational from very weak at distances up to 50 m, to very intense with proximity to the sulphide zone.

In cross-section, the sulphide mineralization generally changes from a thick pyritic footwall zone to a copper-zinc enriched pyritic zone toward the hangingwall with the hangingwall contact often marked by a narrow, less than 1 m thick band of zinc dominated mineralization. Based upon VMS models, this is considered to be primary and syngenetic in nature. The assay contact between the largely barren footwall pyrite mineralization and the potentially economic copper-zinc-pyrite mineralization is gradational over a very short distance or often quite sharp but it does not appear to be controlled either by a change in volcanic stratigraphy or by a later structure. Visually, it is marked by the presence or absence of chalcopyrite disseminated throughout the pyrite dominated sulphide mineralization.

In contrast, the hangingwall contact is identified not only by a change in host rock but also displays a sharp break in sulphide mineralization. Often, at this upper contact, veinlets of bornite and sphalerite crosscut the contact within a confined band of about one metre or less. This zone of vein mineralization appears to be a secondary, structurally controlled remobilization of sulphide mineralization that overprints the original contact. In this zone the sulphides are texturally much coarser grained than the syngenetic VMS mineralization. This zone sometimes shows a sharp increase in copper grade due to an abundance of bornite.

## 1.5 Mineral Processing and Metallurgy

The mineralogy of the Kutcho deposits is complex and requires a similarly complex approach to produce copper and zinc concentrates at reasonable recoveries and concentrate grades.

Considerable metallurgical test work was undertaken by several of the prior owners of the Project over the past 40 years, the last of which was the Kutcho 2010 test program. The December 2010 program was completed at a pre-feasibility level by Cozamin Metallurgical Laboratory in Zacatecas, Mexico using samples assembled from drill core from the Main and Esso deposits according to the geographical locations and 2010 yearly production plan. The results from the test program, *Metallurgical Research Kutcho Project (Drill Core Samples Part 1),* and the 2017 mine plan were used to predict the grades and recoveries to the copper and zinc concentrates.

The copper concentrate is expected to grade 27.6% copper with a zinc content of 7.3%. The grade of the zinc concentrate is expected to be 55.1%. Recoveries are expected to be 84.7% for copper and 75.7% for zinc. The recovery assumptions used in this report are shown in Table 1-1.

**Table 1-1: Cash Flow Calculation Metallurgical Parameters**

| Metal | Recovery (%) | Cu Concentrate Grade (% for Cu and Zn, g/t for Au and Ag) | Zn Concentrate Grade (%) |
|---|---|---|---|
| Cu | 84.7 | 27.6 | 1.2 |
| Zn | 75.7 | 7.3 | 55.1 |
| Au | 41.2 | 2.5 | - |
| Ag | 48.0 | 268.6 | - |

Source: JDS (2017).

## 1.6 Mineral Resources Estimate

The mineral resource estimate was completed by Garth Kirkham, P. Geo., Kirkham Geosystems Ltd., using industry standard methods that conform to National Instrument 43-101 (NI 43-101) and utilizing Leapfrog® and MineSight™ Software.

The data and methodology utilized for the resource estimate is as follows:

- A total of 482 drillholes and one adit were supplied for the Kutcho property which is the combined drillholes for the Main, Esso, and Sumac zones. The database consists of all holes prior to the drilling performed by Western Keltic Mines Inc. (WKM) along with the drilling performed in 2004 for 40 drillholes, 2005 for 27 drillholes, 2006 for 23 drillholes and 81 drillholes from 2008 drilled by Kutcho Copper. In addition, there were 34 holes drilled in the Esso deposit in 2010 and 20 drillholes in 2011, one of which was drilled into the Main, one into Esso deposit and six into Sumac.
- Bulk densities were estimated on a block-by-block basis for the Main deposit based on 8,399 measurements taken from drill core.
- Sectional interpretations were created for each of the Main, Esso, and Sumac deposits. These sections were then wire-framed to form a solid which were then edited to match the drillhole intercepts precisely in 3D. The solids were used to then code the drillhole assays for subsequent geostatistical analysis and for block matching in the grade interpolation process. Solids modelling was performed in Leapfrog and then re-imported to MineSight as the domains for the interpolation of grades.
- Geostatistical analyses were performed on the assays and composites using no constraints in addition to the coded intervals within the mineralized zone solids.
- For the purpose of the mineral resource model, the solids zones were utilized to constrain the block model by matching assays to those within the solid and those outside the solid zones. The orientation and ranges (distances) utilized for search ellipsoids used in the estimation process were derived from the dimensions and orientation of the mineralized zones.
- In terms of selectivity and estimation quality, it was decided that a 2.5 m for Main and Sumac and 1.5 m for Esso composite lengths provided the best compromise between number of composites available for estimation, and a reasonable degree of dilution and regularization. Composites of the specific gravity (SG) from the drillholes were created and then interpolated into the blocks using the inverse distance to the second power.
- Grades of 15% Cu, 17.5% Zn, 100 g/t Ag and 3 g/t Au were chosen as the most reasonable threshold at which to limit grades for Main and 15% Cu, 20% Zn, 100 g/t Ag and 8 g/t Au for Sumac. The range chosen at which to limit grades greater than threshold was 12 m. The outlier strategy utilized for the Esso deposit was to cut values greater than 11% Cu, 27% Zn, 300 g/t Ag, and 2.2 g/t Au. In the case of the Esso deposit, it was determined that the best approach would be to utilize cutting for the purpose of grade limiting therefore the composite grades were cut to the threshold limits as shown above.
- The ellipsoid direction chosen for the estimation process within the Main deposit was chosen to be 10º azimuth and -45º dip for the major axis, 100º azimuth and 0º dip for the minor axis and 10º azimuth and 45º dip for the vertical axis. Sumac and Esso was chosen to be 0º azimuth and -50º dip for the major axis, 90º azimuth and 0º dip for the minor axis and 0º azimuth and 40º dip for the vertical axis.

- The block size chosen was 5 m x 5 m x 5 m oriented orthogonally in an effort to adequately discretize the mineralized zones so as not to inject an inordinate amount of internal dilution and to somewhat reflect drillhole spacing available.
- The choice of interpolator was ordinary kriging for the Main and Esso deposits whilst inverse distance to the 3rd power was used for the Sumac deposit. Nearest neighbour, inverse distance and ordinary kriging were run for all deposits for comparison and validation purposes.
- Three estimation passes were used to estimate the Resource Model because a more realistic block-by-block estimation can be achieved by using more restrictions on the blocks that are closer to drillholes, and thus better informed.
- Classification of mineral resources is based on a number of criteria namely; distance to first composite, average distance of all composites used in a block, number of composites and the number of drillholes used to estimate a block. For measured resources, 30 m was used as the distance to the nearest composite, 30 m for average distance, a minimum of 4 composites and a minimum of 3 drillholes. For indicated resources, 30 to 60 m was used as the distance to the nearest composite, 30 to 60 m for average distance, a minimum of 4 composites and a minimum of 2 drillholes. For inferred resources, greater than 60 m was used as the distance to the nearest composite, greater than 60 m for average distance, a minimum of 4 composites and a minimum of 1 drillholes.

Mineral resource estimates are tabulated at a 1.0% copper cut-off for all three deposits combined and individually and are summarized in Table 1-2.

### Table 1-2: Kutcho Project Resource Summary (Inclusive of Reserves)

| Class | Kutcho Project - Mineral Resource Estimate at a 1.0% Copper Cut-Off for All Deposits (1) | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|
| | Tonnes (kt) | Grade | | | | CuEq (%)(2) | Contained Metal | | | |
| | | Cu (%) | Zn (%) | Au (g/t) | Ag (g/t) | | Copper (M lb) | Zinc (M lb) | Au (koz) | Silver (koz) |
| Measured (M) | 7,695 | 1.89 | 2.61 | 0.31 | 28.7 | 2.60 | 320.6 | 442.8 | 77 | 7,093 |
| Indicated (I) | 9,158 | 1.89 | 3.09 | 0.39 | 36.3 | 2.80 | 381.8 | 624.2 | 116 | 10,674 |
| **M&I** | **16,853** | **1.89** | **2.87** | **0.36** | **32.8** | **2.71** | **700.8** | **1,067.6** | **195** | **17,768** |
| Inferred | 5,798 | 1.33 | 1.64 | 0.24 | 23.2 | 1.79 | 170.0 | 209.2 | 45 | 4,326 |

Notes: 1 Numbers may not total due to rounding.

2 Copper Equivalent (CuEq%) is calculated as copper equivalent recovered and based on metal price assumptions of US$2.75 per pound of copper, US$1.10 per pound of zinc, US$17 per ounce of silver and US$1,250 per ounce of gold. Recoveries are 84.7%, 75.7%, 48.0%, and 41.2% for copper, zinc, silver, and gold, respectively.

## 1.7 Mineral Reserves

The effective date for the Mineral Reserve estimate contained in this report is June 15, 2017 and was prepared by Michael Makarenko, P. Eng. of JDS. All Mineral Reserves in Table 1-3 are Probable Mineral Reserves. The Mineral Reserves are not in addition to the Mineral Resources, but are a subset thereof.

The Mineral Reserves identified in Table 1-3 comply with Canadian Institute of Mining, Metallurgy and Petroleum (CIM) classification of NI 43-101 resource and reserve definitions and standards.

**Table 1-3: Mineral Reserve Estimate**

| Category | Diluted Tonnes (kt) | Cu Grade (%) | Cu(1) (Mlbs) | Zn Grade (%) | Zn(1) (Mlbs) | Ag Grade (g/t) | Ag(1) (Moz) | Au Grade (g/t) | Au(1) (Moz) |
|---|---|---|---|---|---|---|---|---|---|
| Probable | 10,441 | 2.01 | 463 | 3.19 | 734 | 34.6 | 11.6 | 0.37 | 0.1 |
| **Total/Average** | **10,441** | **2.01** | **463** | **3.19** | **734** | **34.6** | **11.6** | **0.37** | **0.1** |

Notes: The Qualified Person for the Mineral Reserve estimate is Michael Makarenko, P. Eng., of JDS Energy & Mining Inc.

Mineral Reserves were estimated using the following metal prices. Copper: $2.75/lb Cu, $1.10/lb Zn, $1,250/oz Au, $17/oz Ag and selected cut-offs of 1.5% Cu and 1.0% Cu for the Main and Esso deposits respectively.

Other costs and factors used for copper cut-off grade determination were mining, processing and other costs of $73.72/t and recoveries of 84.7% Cu, 75.7% Zn, 48.0% Ag, and 41.2% Au.

Tonnages are rounded to the nearest 1,000 t and metal grades are rounded to two decimal places. Tonnage and grade measurements are in % and metric units.

(1) Contained metal in ore.

Source: JDS (2017).

Detailed information on mining, processing, metallurgical, and other relevant factors are contained in the followings sections of this report and demonstrate, at the time of this report, that economic extraction is justified.

The Qualified Person (QP) has not identified any risk including legal, political, or environmental that would materially affect potential Mineral Reserves development.

## 1.8 Mining

Development of the underground mine and pre-stripping of a small starter pit commences in year -1. The small starter pit ore will supplement initial production of ore in order to attain full mill capacity (2,500 t/d) in the first year of production. The underground mine will then provide all mill feed commencing in year 2 to the end of the mine life.

Two underground mining methods are proposed: MCF for the shallow dipping mineralization, and sublevel LH stoping with backfill for those blocks amenable to bulk mining. The initial pre-production development period is estimated to be 18 months (year -1 to mid-year 1). All lateral capital development is assumed to be completed by Desert Star.

The primary access for the Main mine will be a single straight incline from a starting floor elevation of 1,522 m. The cross-sectional area will be 5 m high by 5 m wide to provide clearance for equipment, ventilation and services.

Two ramp systems will be driven off the primary access ramp, one to the east and the other to the west to provide access to the other Main deposit ore zones. The east incline ramp will be driven at a maximum grade of +15%. The west ramp will split into upper and lower ramps driven at grades of +/-15%.

Access to the Esso deposit will be via a 2,600 m long decline ramp from surface to the 1,090 m elevation at the top of the Esso ore body. This ramp will also be 5 m x 5 m and will have an average grade of -15%. A central ramp will then be developed to the bottom of the Esso deposit, with sublevels and accesses driven east and west to the Esso mining zones. Although not designed for exploration purposes, the Esso access ramp could be utilized for future exploration drilling of the Sumac deposit.

During pre-production, the primary ramp in the Main zone will be established as well as secondary access ramps to the west, centre and east mining zones. Production is exclusively from the Main ore deposits in years 1 to 2, while Esso is being developed.

The access ramp to Esso begins in year -1 and is complete in year 1. Esso's pre-production period is approximately 40 months. Ore production from Esso begins in year 3 and continues at 1,500 t/d until the deposit is exhausted in year 8. While Esso is in production, Main's rate is reduced to 1,000 t/d for a total rate of 2,500 t/d from both mines. Once Esso is exhausted, Main production returns to 2,500 t/d until the end of the mine in year 12. The mine production plan is shown in Table 1-4 with tonnages round to the nearest 1,000 tonnes (t).

Backfill is an integral part of the underground mine plan and will incorporate process plant tailings as well as mine development waste. The primary purposes of the backfill are:

- Underground support and working platform in MCF mining; and
- Storage of potentially acid generating (PAG) waste rock and process plant tailings.

Waste rock will be scheduled so that material mined early in the underground development effort will more likely be classified as non-PAG and will be hauled and used on surface. As the stoping reaches a steady state underground, development rock will preferentially be used as backfill. The backfill plan calls for all waste rock generated after production year 2 to be stored underground. Therefore there are no permanent PAG or non-PAG waste rock storage facilities. Any temporary storage facilities during the initial start-up will be utilized for construction (non-PAG) or placed into the vacant open pit (PAG and non-PAG) or back underground as fill (PAG and non-PAG).

An insufficient volume of waste rock is available for the backfill requirement; hence the use of paste fill has been incorporated into the mine plan. Paste fill consists of process tailings partially dewatered and mixed with cement. This material is of a consistency that can be directed to specific locations by positive displacement pumps and pipeline. The fill plant will be operated such that all tailings required for backfill will be converted to thickened slurry and pumped to the mine for use as fill. Tailings not required for backfill will be directed to a permanent surface TMF. In general, 50% of the tailings are suitable for paste backfill.

The mine production and development plan is summarized in Table 1-4.

**Table 1-4: Mine Production Plan**

| Parameter | Unit | Production Year | | | | | | | | | | | | | Totals |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | -1 | 1 | 2 | 3 | 4 | 5 | 6 | 7 | 8 | 9 | 10 | 11 | 12 | |
| Starter Pit Production | kt | - | 446 | - | - | - | - | - | - | - | - | - | - | - | **446** |
| Main Production | kt | - | 466 | 913 | 674 | 365 | 365 | 365 | 459 | 674 | 913 | 913 | 913 | 404 | **7,660** |
| Esso Production | kt | - | - | - | 239 | 548 | 548 | 548 | 454 | 239 | - | - | - | - | **2,335** |
| Total Mine Production | kt | - | 913 | 913 | 913 | 913 | 913 | 913 | 913 | 913 | 913 | 913 | 913 | 404 | **10,441** |
| Daily Production Rate | t/d | - | 2,500 | 2,500 | 2,500 | 2,500 | 2,500 | 2,500 | 2,500 | 2,500 | 2,500 | 2,500 | 2,500 | 2,500 | **2,500** |
| Starter Pit Waste | kt | 1,533 | 1,215 | - | - | - | - | - | - | - | - | - | - | - | **2,748** |
| Copper Grade | % | - | 1.94 | 2.13 | 2.01 | 2.02 | 2.26 | 2.12 | 2.06 | 1.88 | 1.91 | 2.07 | 1.81 | 1.87 | **2.01** |
| Zinc Grade | % | - | 1.92 | 2.62 | 2.91 | 3.71 | 5.30 | 4.41 | 3.76 | 2.64 | 3.06 | 2.78 | 2.43 | 2.27 | **3.19** |
| Silver Grade | g/t | - | 26.4 | 31.0 | 42.3 | 47.2 | 41.5 | 46.9 | 35.2 | 27.6 | 29.2 | 28.0 | 27.3 | 30.1 | **34.6** |
| Gold Grade | g/t | - | 0.30 | 0.32 | 0.44 | 0.47 | 0.45 | 0.47 | 0.35 | 0.36 | 0.31 | 0.32 | 0.27 | 0.39 | **0.37** |
| Capital Development | m | 3,875 | 2,637 | 1,465 | 362 | - | - | - | - | - | - | - | - | - | **8,339** |
| Sustaining Development | m | 360 | 1,400 | 2,350 | 1,952 | 2,034 | 2,267 | 973 | 1,772 | 1,184 | 1,234 | 789 | 351 | 170 | **16,835** |
| Total Lateral Development | m | 4,235 | 4,037 | 3,815 | 2,314 | 2,034 | 2,267 | 973 | 1,772 | 1,184 | 1,234 | 789 | 351 | 170 | **24,174** |
| | m/day | 11.6 | 11.1 | 10.5 | 6.3 | 5.6 | 6.2 | 2.7 | 4.9 | 3.2 | 3.4 | 2.2 | 1.0 | 1.0 | **5.5** |
| Capital Raise Development | m | 467 | 809 | 441 | 40 | - | - | - | - | - | - | - | - | - | **1,757** |
| Mined Underground Waste | kt | 297 | 274 | 262 | 156 | 137 | 153 | 66 | 120 | 80 | 83 | 53 | 24 | 12 | **1,716** |
| Paste Backfill Placed | kt | - | 228 | 408 | 408 | 408 | 384 | 408 | 408 | 408 | 408 | 408 | 408 | 181 | **4,468** |

Note: numbers may not total due to rounding.
Source: JDS (2017).

## 1.9 Recovery Methods

The results from the metallurgical test work were used to develop the design criteria and selected flowsheet for the process facility. The Kutcho ore will be treated using sequential flotation to recover copper and zinc to saleable concentrates.

The plant will consist of the following unit operations:

- Primary Crushing – A vibrating grizzly feeder and jaw crusher in open circuit, producing a final product $P_{80}$ of 125 millimeter (mm);
- Crushed Material Storage and Reclaim – A 2,500 t live stockpile with two reclaim belt feeders feeding the Ball Mill Feed Conveyor;
- Primary Grinding – A SAG mill in open circuit, producing a transfer size $T_{80}$ of 1,000 micrometres (µm);
- Secondary Grinding – A ball mill in open circuit, producing a final product $P_{80}$ of 75 µm;
- Copper Rougher and Cleaner Flotation – Rougher flotation cells, rougher concentrate regrind and cleaner flotation cells;
- Zinc Rougher and Cleaner Flotation – Rougher flotation cells and cleaner flotation cells;
- Concentrate Dewatering and Filtration – Copper and zinc concentrate thickeners, stock tanks and filters; and
- Final Tailings Disposal – Centrifugal pumps to send slurry to the TMF and a barge reclaim system to pump reclaim water back to the process plant or to the paste plant for deposition underground.

The plant will process material at a rate of 2,500 t/d with an average life of mine (LOM) head grade of 2.01% Cu and 3.19% Zn. The two stage grinding circuit will target a product size of 80% passing ($P_{80}$) 75 µm, followed by sequential flotation to produce copper and zinc concentrates. The tailings will be pumped to a TMF or to the paste backfill plant for deposition underground. The crushing circuit will operate at an availability of 70%, while the milling and flotation circuits will operate 24-hours per day, 365 days per year at an availability of 92%.

## 1.10 Mineral Waste Management

Mineral waste will consist of tailings and waste rock including overburden materials.

The waste management plan has been developed based on the mine plan that includes an initial small starter pit in the Main deposit and underground mines in the Main and Esso deposits. The starter pit will be pre-stripped in year -1 and will provide ore in year 1 while the underground mine is being developed.

Geochemical characterization indicates that the tailings are approximately 80% pyrite and are strongly net acid generating with some acid neutralization potential. The waste rock from the mine operation includes PAG and non-PAG waste rock, all of which is stored permanently in the starter pit or underground.

The tailings from the mill will be disposed of in three areas: 1) underground mines as backfill material, 2) on-land paste tailings lined storage facility, and 3) mined-out starter pit as backfill.

The PAG and non-PAG waste rock will be separated at the sources during mine operation. The majority of waste rock generated in the underground mines will remain underground as backfill. The remaining waste rock will be hauled to the surface to the pit.

A portion of the on-surface non-PAG waste rock will be used as site construction materials during the initial site construction stage. The remaining non-PAG waste rock will be temporarily stored in a stockpile and used as site construction materials during the late stages of mine operation and at mine closure as well as underground mine backfill.

The on-surface PAG waste rock disposal will consist of three methods: 1) co-disposal with paste tailings in the on-land paste tailings storage facility, 2) co-disposal with paste tailings as backfill in the mined-out starter pit, and 3) hauled back to underground mines as backfill. The PAG waste rock generated from the starter pit in year -1 and early year 1 will be stored in a temporary PAG waste rock storage facility. The waste rock in the temporary PAG waste rock storage facility will be later moved to the mined-out starter pit or hauled down to underground as backfill.

Table 1-5 summarizes the overall mineral waste production and disposal plans.

**Table 1-5: Overall Mineral Waste Production and Disposal Plan Summary**

| Mineral Waste | Stage | Location | Total Dry Tonnage (Mt) |
|---|---|---|---|
| Tailings | Production | Mill (or process plant) | 9.35 |
| | Disposal | Underground mines as backfill | 4.90 |
| | | On-land paste tailings storage facility | 3.45 |
| | | Mined-out starter pit | 1.00 |
| non-PAG Waste Rock | Production | Starter pit | 1.22 |
| | | Underground mines | 1.61 |
| | Disposal | Remaining in underground as backfill | 0.85 |
| | | Used as on-site construction materials | 1.61 |
| | | Hauled to underground mines as backfill | 0.37 |
| PAG Waste Rock | Production | Starter pit | 1.52 |
| | | Underground mines | 0.14 |
| | Disposal | Remaining in underground as backfill | 0.13 |
| | | Co-disposed with paste tailings in on-land paste tailings storage facility | 0.62 |
| | | Co-disposed with paste tailings in mined-out starter pit | 0.80 |
| | | Hauled to underground mines as backfill | 0.11 |

Source: JDS (2017).

The proposed on-land paste tailings storage facility consists of a containment berm, a bottom liner system, and a top closure cover system. The containment berm would be a zoned earth and rock fill structure with an upstream low-permeability clay silt zone covered with a geomembrane liner to contain the paste tailings. The ground below the proposed mine waste footprint in the facility will be excavated to a depth of 4.5 m to increase the storage capacity of the facility and obtain till fill for construction. The bottom liner system consists of a geomembrane liner over a low-permeability clay silt layer and a basal drainage layer. The design storage capacity of the tailing facility is approximately 2.04 million cubic metres ($Mm^3$) or 4.08 Mt of dry mine waste.

A multiple-layer soil cover will be placed over the top of the paste tailings at mine closure to minimize the water infiltration into the facility and provide an oxygen diffusion barrier to minimize the influx of oxygen. The cover system consists of a top native soil layer, a top capillary barrier layer, a compacted low-permeability clay silt layer, and a bottom capillary barrier layer over the paste tailings. The key design objective for the low-permeability layer is to maintain a high degree of saturation under all conditions. This objective is achievable for the current cover design under the meteorological, hydrological, hydrogeological, and ground conditions of this project site.

The moisture content in the majority of the paste tailings placed in the on-land paste tailings storage facility will be maintained in a nearly saturated condition over the long-term because of the lined sides and bottom of the facility, the fine-grained nature and intrinsic low permeability of the paste tailings, the cover design that limits moisture loss of the tailings, and a gentle surface slope.

The closure cover design for the mine waste in the mined-out starter pit applies best engineering measures to minimize or prevent surface infiltration and ingress of oxygen to reduce the risk of mine waste oxidation and generation of acid drainage. The cover system consists of a top native soil layer followed by a capillary barrier layer, a low-permeability clay silt layer, and a geomembrane liner installed over the final paste tailings that is placed above the waste rock/tailings mixture.

A temporary PAG waste rock storage facility is required to store the PAG waste rock during early mine operation before the waste rock is permanently disposed. The storage capacity of the temporary PAG waste rock storage facility is approximately 0.46 $Mm^3$. The PAG waste rock will be placed in the dump in years -1 and 1. The PAG waste rock will be re-handled and placed back to the mined-out starter pit in years 2 and 3.

A temporary non-PAG waste rock stockpile is required to store a portion of the non-PAG waste rock generated during early years of mine operation and will be later used as site construction materials and as underground mine backfill. The stockpile has a maximum storage capacity of 0.36 $Mm^3$ in year 3. The storage volume will be gradually reduced to zero over the LOM when some non-PAG waste rock is used as construction materials and underground mine backfill, and through reclamation.

## 1.11 Water Management

The water management during the mine operation includes the following components:

- Diversion ditches and berms around the proposed mine waste management facilities to divert the clean surface runoff water from the undisturbed ground above the mine facilities to minimize the overall quantity of the contact water;
- A water collection pond dam to store contact water from the mine waste facility areas;
- Pumping the contact water from the water collection pond to a water treatment plant;
- Pumping contact water from underground mines and other mine site areas to the water treatment plant for treatment;

- Reclaiming a portion of the treated or untreated site contact water and process water for mineral processing; and
- Discharging the treated water to the receiving environment after the water quality meets the discharge criteria.

The proposed water collection pond dam is a zoned earth and rock fill structure with an upstream low-permeability clay silt zone to control the seepage through the dam. The ground under the upstream side of the dam and the pond will be excavated to a depth of approximately 4 m. This will increase the water storage capacity of the water collection pond, reduce the seepage through the overburden zone in the dam foundation, and obtain sufficient till fill materials for construction. The shallow bedrock below the clay silt zone could be highly fractured. A zone of curtain grout is proposed in the bedrock below the clay silt zone to reduce the potential seepage through the bedrock foundation.

The water collection pond was designed to have a sufficient capacity under various design conditions. Pumping water from the water collection pond to the water treatment plant is required during the freshet period and the following period each year to control the maximum pond water level within the design range.

A seepage collection sump located immediately downstream of the water collection pond is proposed to collect minor seepage through the dam. The water in the seepage collection sump will be regularly pumped back to the water collection pond.

After mine closure, the water from the covered on-land paste tailings storage facility and starter pit areas will be collected in the water collection pond and sumps and then pumped to the water treatment plant for treatment. The water can be discharged directly to the environment when the water quality meets the discharge criteria. The water collection pond can be decommissioned and the dam be breached after a monitoring period specified in the water use licence.

## 1.12 Environmental Considerations

The Project is subject to the British Columbia *Environmental Assessment Act* and the Canadian *Environmental Assessment Act*. The former requires that the Project undergo an environmental assessment and obtain an Environmental Assessment (EA) Certificate. The Project was initiated into the British Columbia EA process through the issuance of a Section 10 order by the British Columbia Environmental Assessment Office (BC EAO) on July 29, 2005. The provincial and federal processes will be integrated in a harmonized review, with the BC EAO taking the lead. On December 24, 2007, the Canadian Environmental Assessment Agency announced that the Project would be subject to a Comprehensive Study.

In 2005, a program of environmental and socio-economic baseline studies was initiated to provide the information necessary to prepare the EA Application and to develop management plans and monitoring programs. Studies considered both the biophysical and human environment, including meteorology, air quality, hydrology, hydrogeology, metal leaching and acid rock drainage, aquatic ecology, fish and fish habitat, soils, vegetation, ecosystem mapping, wildlife, wetlands, archaeology, socio-economics, land use, country foods and human health, and traditional use and traditional ecological knowledge.

The Project is located in the Traditional Territories of the Tahltan and Kaska Dena First Nations.

The Application Information Requirements (AIR) for the proposed Kutcho Copper-Zinc-Silver-Gold Mine Project were approved by the BC EAO on December 21, 2012. On March 29, 2016 Kutcho Copper Corp. (Gregg Bush, President and CEO) informed the BC EAO of their intention to terminate the current EA. It was requested that the Kutcho Project be withdrawn from the EA process.

A gap analysis will be completed to identify differences in environmental assessment requirements from 2005 and 2017 and to identify any survey and monitoring deficiencies.

## 1.13 Capital Cost Estimate

The capital cost (CAPEX) estimate includes all costs required to develop, sustain, and close the operation for a planned 12 year operating life starting at the detailed engineering stage, post financing and permitting. All costs up to detailed engineering are considered sunk costs, including the asset purchase price. The accuracy of this CAPEX estimate is +/-25% in accordance with the Association for the Advancement of Cost Engineering (AACE) the level of detail for a Class 4 estimate.

The summary CAPEX estimate is shown in Table 1-6. The initial or pre-production CAPEX is $220.7 million (M), with sustaining CAPEX totaling $67.1 M. Costs are expressed in Canadian dollars with no escalation (Q2-2017 dollars).

**Table 1-6: Capital Cost Estimate**

| CAPEX | Pre-Production (M$) | Sustaining (M$) | LOM Total (M$) |
|---|---|---|---|
| Underground Mine Equipment and Infrastructure | 5.5 | 15.7 | 21.2 |
| Underground Capital Development | 12.8 | 23.8 | 36.6 |
| Pre-Production | 11.5 | - | 11.5 |
| Pre-Stripping | 3.1 | - | 3.1 |
| Owner's Costs | 9.2 | - | 9.2 |
| Offsite (Road, Airstrip Ext.) | 15.7 | 4.0 | 19.7 |
| Backfill System | 8.0 | 1.3 | 9.3 |
| Waste and Water Management | 10.3 | 5.3 | 15.6 |
| Process Plant | 62.7 | - | 62.7 |
| Site Infrastructure (Camp, Roads, Fuel, Office) | 19.1 | - | 19.1 |
| Engineering, Procurement, and Construction Management (EPCM) | 6.3 | - | 6.3 |
| Indirects | 27.9 | - | 27.9 |
| Sustaining Capital (~0.5% of OPEX) | - | 2.4 | 2.4 |
| Closure | - | 6.8 | 6.8 |
| Subtotal | 191.9 | 59.2 | 251.1 |
| Contingency | 28.8 | 7.9 | 36.7 |
| **Total Capital** | **220.7** | **67.1** | **287.8** |

Source: JDS (2017).

Preparation of the capital cost estimate is based on the JDS philosophy that emphasizes accuracy over contingency, and uses defined and proven project execution strategies. The estimates were developed using first principles, applying directly-related project experience, and the use of general industry factors. Almost all of the estimates used in this project were obtained from engineers, contractors, and suppliers who have provided similar services to existing operations and have demonstrated success in executing the plans set forth in this study.

The reclamation estimate is based on a preliminary estimation of a closure plan commencing in year 12 and continuing to the end of year 15. Salvage value across the entire facility was assumed to be $5 M and was used to partially offset reclamation costs.

## 1.14 Operating Cost Estimate

The operating cost estimate (OPEX) for the Project is based on a combination of experience, reference projects, first principle calculations, budgetary quotes, and factors as appropriate for a PFS.

The LOM unit and total operating costs are summarized in Table 1-7.

**Table 1-7: Operating Cost Estimate**

| Operating Costs | Unit Cost Estimate ($/t milled) | LOM Cost (M$) |
|---|---|---|
| Underground Mining (includes mobile equipment lease) | 40.10 | 418.7 |
| Open Pit Mining (includes waste mining) | 0.31 | 3.2 |
| Process | 20.79 | 217.1 |
| Leased Equipment (power plant) | 1.66 | 17.3 |
| General and Administration | 10.86 | 113.4 |
| **Total** | **73.72** | **769.7** |

Source: JDS (2017).

Table 1-8 outlines the major assumptions used to build up the operating costs.

**Table 1-8: Major Operating Cost Assumptions**

| Item | Unit | Value |
|---|---|---|
| Electrical power cost | $/kWh | 0.15 |
| Diesel cost (delivered) | $/litre | 0.97 |

Source: JDS (2017).

## 1.15 Economic Analysis

Three price scenarios were evaluated for this pre-feasibility study. The Base Case utilized metals prices closer to current prices, while Case 2 considered lower metals prices and Case 3 used higher price assumptions. All cases were examined by use of sensitivity analysis.

The price assumptions for economic modeling are shown in Table 1-9 and analysis results are detailed in Table 1-10.

### Table 1-9: Metal Price Assumptions

| Metal | Unit | Base Case | Case 2 | Case 3 |
|---|---|---|---|---|
| Cu | US$/lb Cu | 2.75 | 2.50 | 3.00 |
| Zn | US$/lb Zn | 1.10 | 1.00 | 1.20 |
| Au | US$/oz Au | 1,250 | 1,125 | 1,375 |
| Ag | US$/oz Ag | 17.00 | 15.30 | 18.70 |
| Exchange Rate | C$/US$ | 1.33 | 1.38 | 1.29 |

Source: JDS (2017).

Table 1-10 below summarizes the results of the economic analysis.

### Table 1-10: Economic Analysis Results

| Item | Unit | Base Case | Case 2 | Case 3 |
|---|---|---|---|---|
| Unit Open Pit Mining Costs(1) | $/t milled | 0.31 | 0.31 | 0.31 |
| Unit Underground Mining Costs | $/t milled | 40.10 | 40.10 | 40.10 |
| Unit Milling Costs | $/t milled | 20.79 | 20.79 | 20.79 |
| Unit G&A and Site Services | $/t milled | 10.86 | 10.86 | 10.86 |
| Capital Leases | $/t milled | 1.66 | 1.66 | 1.66 |
| Royalties | $/t milled | 3.63 | 3.38 | 3.87 |
| Unit Total OPEX (with royalties) | $/t milled | 77.35 | 77.10 | 77.58 |
| Unit OPEX (net of trans, ref, credits) | US$/lb Cu | 0.59 | 0.66 | 0.67 |
| Total Initial Capital | M$ | 220.7 | 220.7 | 220.7 |
| NPV8% Pre Tax | M$ | 423.5 | 340.7 | 501.0 |
| NPV8% After Tax | M$ | 265.2 | 211.1 | 315.7 |
| IRR Pre Tax | % | 34.6 | 29.9 | 38.8 |
| IRR After Tax | % | 27.6 | 23.9 | 31.0 |
| Payback Period (Post-tax) | years | 3.5 | 3.9 | 3.3 |

Note: (1) Open pit mining costs in year 1 are $1.93/t moved which averaged over the LOM milled tonnes is $0.31/t.
Source: JDS (2017).

A production summary is shown in Table 1-11.

**Table 1-11: Production Summary**

| Parameter | Unit | Total | Production Year | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | 1 | 2 | 3 | 4 | 5 | 6 | 7 | 8 | 9 | 10 | 11 | 12 |
| Mill Feed | kt | 10,441 | 913 | 913 | 913 | 913 | 913 | 913 | 913 | 913 | 913 | 913 | 913 | 404 |
| Cu Grade | Cu % | 2.01 | 1.94 | 2.13 | 2.01 | 2.02 | 2.26 | 2.12 | 2.06 | 1.88 | 1.91 | 2.07 | 1.81 | 1.87 |
| Zn Grade | Zn % | 3.19 | 1.92 | 2.62 | 2.91 | 3.71 | 5.30 | 4.41 | 3.76 | 2.64 | 3.06 | 2.78 | 2.43 | 2.27 |
| Au Grade | Au g/t | 0.37 | 0.30 | 0.32 | 0.44 | 0.47 | 0.45 | 0.47 | 0.35 | 0.36 | 0.31 | 0.32 | 0.27 | 0.39 |
| Ag Grade | Ag g/t | 34.6 | 26.4 | 31.0 | 42.3 | 47.2 | 41.5 | 46.9 | 35.2 | 27.6 | 29.2 | 28.0 | 27.3 | 30.1 |
| Cu Concentrate | k dmt | 646 | 58 | 63 | 54 | 61 | 60 | 60 | 59 | 51 | 52 | 56 | 49 | 22 |
| Zn Concentrate | k dmt | 458 | 22 | 30 | 34 | 50 | 72 | 60 | 51 | 31 | 36 | 33 | 28 | 12 |
| Cu in Cu Concentrate | Mlb | 378 | 32 | 35 | 34 | 35 | 37 | 36 | 34 | 30 | 30 | 33 | 29 | 13 |
| Au in Cu Concentrate | koz | 46 | 3 | 4 | 5 | 5 | 5 | 5 | 4 | 4 | 3 | 3 | 3 | 2 |
| Ag in Cu Concentrate | koz | 5,018 | 387 | 454 | 503 | 623 | 526 | 606 | 428 | 328 | 347 | 332 | 325 | 158 |
| Zn in Zn Concentrate | Mlbs | 473 | 22 | 30 | 34 | 52 | 77 | 63 | 54 | 31 | 36 | 33 | 29 | 12 |

Source: JDS (2017).

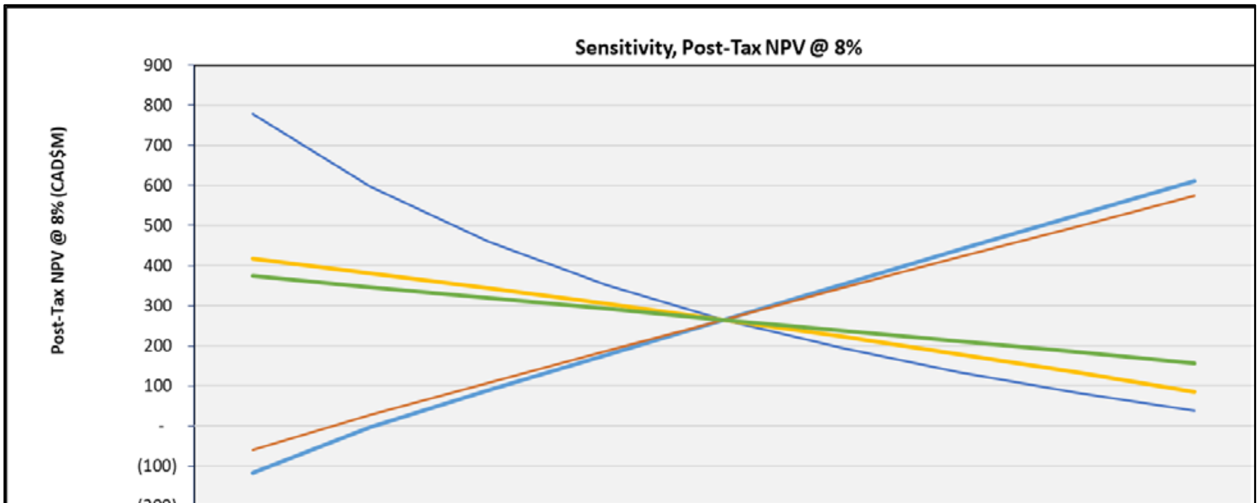
### 1.15.1 Sensitivity Analysis

Sensitivity analyses were carried out using metal prices, exchange rate, mill head grade, capital costs and operating costs as variables. Each variable was changed independently. Sensitivities were generated using the post-tax net present value (NPV) at an 8% discount rate as the measure of project performance.

The NPV of the Project is most affected by the price of metal and by the US$:C$ exchange rate. The Project is also sensitive to head grade and then OPEX and CAPEX. These results identify two areas on which to focus in order to effect positive changes to economic performance of the Project. Other than metal pricing which is out of the operator's control unless price hedging contracts are put into place, metallurgical recovery, and operating cost control have a marked effect. The significance of the capital on the NPV has been reduced by utilizing a leasing program that reduced the amount of pre-production capital and in effect moved the burdens to operating.

Sensitivity results are shown in Figure 1-1.

**Figure 1-1: Economic Sensitivity Case Graph**




| | -40% | -30% | -20% | -10% | Base | +10% | +20% | +30% | +40% |
|---|---|---|---|---|---|---|---|---|---|
| Price | (117) | (3) | 88 | 177 | 265 | 352 | 439 | 526 | 612 |
| CAD:US FX | 779 | 597 | 459 | 352 | 265 | 194 | 134 | 83 | 39 |
| Mill Feed Grade | (61) | 27 | 108 | 187 | 265 | 343 | 421 | 498 | 575 |
| OPEX | 417 | 381 | 344 | 306 | 265 | 223 | 179 | 133 | 86 |
| CAPEX | 374 | 347 | 319 | 292 | 265 | 238 | 211 | 184 | 157 |

Source: JDS (2017).

## 1.16 Conclusions

The Project contains a substantial Cu-Zn sulphide resource with Au and Ag by-product credits. The Main deposit can be selectively mined by open pit and underground mining methods while the Esso deposit can be mined using underground methods. The Project as described in this report yields positive economic results based on industry-standard methods of mining and mineral processing.

The QPs are not aware of any fatal flaws with the Project. As with most mining projects there are several risks and opportunities that could impact the Project viability.

## 1.17 Recommendations

It is recommended that the Project progress towards a feasibility study by conducting exploration, definition, and expansion drilling in an attempt to convert inferred resources to measured or indicated. The resource drilling program is estimated to be $5,400,000. Additional work around metallurgy and geotechnical work should also be conducted. These programs are estimated to cost an additional $3,000,000.

# 2 Introduction

The Qualified Persons (QPs) preparing this report are specialists in the fields of geology, exploration, mineral resource and mineral reserve estimation and classification, geotechnical, environmental, permitting, metallurgical testing, mineral processing, processing design, capital and operating cost estimation, and mineral economics.

None of the QPs or any associates employed in the preparation of this report has any beneficial interest in Desert Star and neither are they insiders, associates, or affiliates. The results of this report are not dependent upon any prior agreements concerning the conclusions to be reached, nor are there any undisclosed understandings concerning any future business dealings between Desert Star and the QPs. The QPs are being paid a fee for their work in accordance with normal professional consulting practice.

The following individuals, by virtue of their education, experience and professional association, are considered QPs as defined in the NI 43-101, and are members in good standing of appropriate professional associations. The QPs are responsible for specific sections as summarized in Table 2-1.

**Table 2-1: Qualified Person Site Visits and Areas of Responsibility**

| Qualified Person | Site Visit Date | Report Sections |
|---|---|---|
| Michael Makarenko, P. Eng. | September 27 to 29, 2010 | 1.1, 1.2, 1.3, 1.7, 1.8, 1.13, 1.14, 1.15, 1.16, 1.17, 2, 3, 12.3, 15, 16, 18 (except 18.11), 19, 20 (except 20.1.4.3), 21, 22, 23, 24, 25, 26, 27, 28, 29 |
| Kelly McLeod, P. Eng. | No Site Visit | 1.5, 1.9, 12.2, 13, 17 |
| Garth Kirkham, P. Geo. | April 30, 2008 and July 17, 2017 | 1.4, 1.6, 4, 5, 6, 7, 8, 9, 10, 11,12.1,14 |
| Daniel Jarratt, P. Eng. | No Site Visit | 1.12, 20.1.4.3 |
| Guangwen (Gordon) Zhang, P. Eng. | September 27 to 29, 2010 | 1.10, 1.11, 18.11 |

Source: JDS (2017).

The Project site is in advanced exploration stage and site visits by Kelly McLeod and Daniel Jarratt were not necessary to complete this PFS report.

## 2.1 Sources of Information

The sources of information include data and reports supplied by Capstone as well as documents cited throughout the report and referenced in Section 27. In particular, background Project information was taken directly from the most recent technical report titled *JDS Energy & Mining Inc. Kutcho Copper Project Prefeasibility Study, British Columbia* with an effective date of February 15, 2011 (JDS 2011). JDS 2011 was the basis for the mineral resource and reserve estimates, mine plan, infrastructure, and processing plant design.

Other sources of information include:

- Wardrop Kutcho Project Prefeasibility Study dated October 25, 2007;

- SRK Preliminary Economic Assessment Technical Report dated May 31, 2008;
- Consolidated Management Consultants Electrical Power Options, 2011 Prefeasibility Study Report dated June 2010;
- EBA Kutcho Project - Main Deposit Pre-feasibility Level Geotechnical Evaluation Report dated November 29, 2010;
- EBA Kutcho Project - Esso Deposit Pre-feasibility Level Geotechnical Evaluation dated November 29, 2010;
- Onsite Engineering Ltd. Onsite Kutcho Access Road, Geometric Road Design Report dated December 17, 2010;
- Stantec-Mining (Stantec) Capstone Kutcho Project, Prefeasibility Level Ventilation Planning & Design Report dated December 15, 2010; and
- JDS Energy & Mining Inc. Kutcho Copper Project Prefeasibility Study, British Columbia with an effective date of February 15, 2011.

## 2.2 Currency and Rounding

Unless otherwise specified or noted, the units used in this technical report are metric. Every effort has been made to clearly display the appropriate units being used throughout this technical report. Currency is in Canadian dollars ($ or C$).

This report includes technical information that required subsequent calculations to derive subtotals, totals and weighted averages. Such calculations inherently involve a degree of rounding and consequently introduce a margin of error. Where these occur, the QPs do not consider them to be material.

# 3 Reliance on Other Experts

This report is based on information provided by Desert Star and other non-QP experts throughout the course of the study. The QPs have taken reasonable measures to confirm information provided by others and take responsibility for the information.

Non-QP specialists relied upon for specific advice include:

- Justin Himmelright, R.P.Bio, M. Eng., provided the update and review to the socio-economic section (Section 20.4).
- Wentworth Taylor, Chartered Accountant, provided taxation advice and calculations for the economic model found in Section 23.5.

The QPs responsible for these sections used their experience to determine if the information from the non-QPs was accurate.

# 4 Property Description and Location

## 4.1 Introduction

The Kutcho property is approximately 100 km east of the Town of Dease Lake in Northern British Columbia and is shown in Figure 4-1. The property is located on national topographic system (NTS) map sheet 104I/1. The geodetic coordinates for the center of the claim area are 58°12'N and 128°22'W. The universal transverse mercator (UTM) coordinates for the centre of the Main deposit are approximately 537500E and 6452000N.

**Figure 4-1: Kutcho Project Area**


Source: Capstone (2011).

The Project area contains 46 mineral claims and 1 mineral lease covering an area of 17,060.89 hectares (ha). Mineral tenures are shown in Figure 4-2 and are listed in Table 4-1.

Kutcho Copper owns the claims through two separate purchase agreements and through claim staking. One agreement is with Barrick Gold Inc. (a subsidiary of Barrick Gold Corporation [Barrick]) and AMI Resources Inc., who had 80% and 20% ownership, respectively, in all of the claims except the 16 SMRB claims and the 30 KC claims. The other agreement is with Sumac Mines Inc., a subsidiary of Sumitomo Metal Mining Co. Ltd. In 2008, Kutcho Copper staked 11 claims. Barrick Gold Inc. and AMI Resources subsequently sold their rights to Royal Gold Inc.

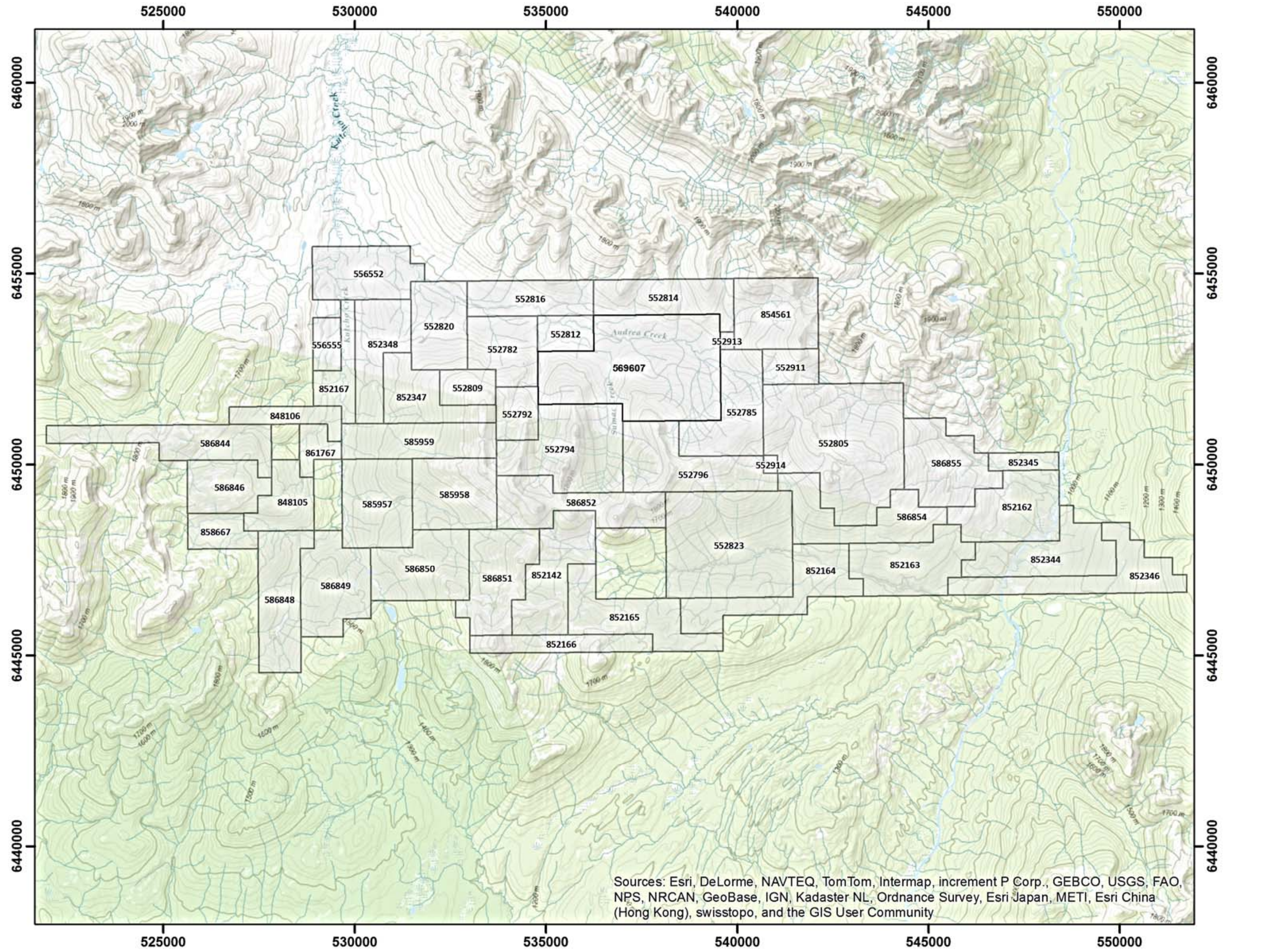
### Table 4-1: Kutcho Mineral Tenures

| Tenure Number | Tenure Claim | Claim Name | Map Number | Good to Date | Status | Area (ha) | Tag Number |
|---|---|---|---|---|---|---|---|
| 552782 | Mineral Claim | | 104I | 2022/APR/11 | GOOD | 306.87 | 552782 |
| 552785 | Mineral Claim | | 104I | 2022/APR/11 | GOOD | 409.34 | 552785 |
| 552792 | Mineral Claim | | 104I | 2022/APR/11 | GOOD | 153.50 | 552792 |
| 552794 | Mineral Claim | | 104I | 2022/APR/11 | GOOD | 597.09 | 552794 |
| 552796 | Mineral Claim | | 104I | 2022/APR/11 | GOOD | 494.79 | 552796 |
| 552805 | Mineral Claim | | 104I | 2022/APR/11 | GOOD | 1074.74 | 552805 |
| 552809 | Mineral Claim | | 104I | 2022/APR/11 | GOOD | 136.42 | 552809 |
| 552812 | Mineral Claim | | 104I | 2022/APR/11 | GOOD | 136.37 | 552812 |
| 552814 | Mineral Claim | | 104I | 2022/APR/11 | GOOD | 357.90 | 552814 |
| 552816 | Mineral Claim | | 104I | 2022/APR/11 | GOOD | 306.77 | 552816 |
| 552820 | Mineral Claim | | 104I | 2022/APR/11 | GOOD | 340.92 | 552820 |
| 552823 | Mineral Claim | | 104I | 2022/APR/11 | GOOD | 921.83 | 552823 |
| 552911 | Mineral Claim | PASS1 | 104I | 2022/APR/11 | GOOD | 136.41 | 552911 |
| 552913 | Mineral Claim | ADD1 | 104I | 2022/APR/11 | GOOD | 17.05 | 552913 |
| 552914 | Mineral Claim | ADD2 | 104I | 2022/APR/11 | GOOD | 17.06 | 552914 |
| 556552 | Mineral Claim | ADD3 | 104I | 2022/APR/11 | GOOD | 374.88 | 556552 |
| 556555 | Mineral Claim | ADD4 | 104I | 2022/APR/11 | GOOD | 102.29 | 556555 |
| 569607 | Mineral Lease | | 104I | 2017/NOV/07 | GOOD | 1090.00 | 569607 |
| 585957 | Mineral Claim | MOTHER 1 | 104I | 2022/APR/11 | GOOD | 426.64 | 585957 |
| 585958 | Mineral Claim | MOTHER 2 | 104I | 2022/APR/11 | GOOD | 409.55 | 585958 |
| 585959 | Mineral Claim | MOTHER 3 | 104I | 2022/APR/11 | GOOD | 375.29 | 585959 |
| 586844 | Mineral Claim | ACCENT 1 | 104I | 2022/APR/11 | GOOD | 426.47 | 586844 |
| 586846 | Mineral Claim | ACCENT 2 | 104I | 2022/APR/11 | GOOD | 273.02 | 586846 |
| 586848 | Mineral Claim | SOUTH FORK 1 | 104I | 2022/APR/11 | GOOD | 426.92 | 586848 |
| 586849 | Mineral Claim | SOUTH FORK 2 | 104I | 2022/APR/11 | GOOD | 426.88 | 586849 |
| 586850 | Mineral Claim | SOUTH FORK 3 | 104I | 2022/APR/11 | GOOD | 426.83 | 586850 |
| 586851 | Mineral Claim | SOUTH FORK 4 | 104I | 2022/APR/11 | GOOD | 426.86 | 586851 |
| 586852 | Mineral Claim | TRONDHJEMITE 1 | 104I | 2022/APR/11 | GOOD | 426.65 | 586852 |
| 586854 | Mineral Claim | TRONDHJEMITE 2 | 104I | 2022/APR/11 | GOOD | 426.69 | 586854 |

### Table 4-1: Kutcho Mineral Tenures (continued)

| Tenure Number | Tenure Claim | Claim Name | Map Number | Good to Date | Status | Area (ha) | Tag Number |
|---|---|---|---|---|---|---|---|
| 586855 | Mineral Claim | TRONDHJEMITE 3 | 104I | 2022/APR/11 | GOOD | 426.55 | 586855 |
| 848105 | Mineral Claim | ACCENT 3 | 104I | 2022/APR/11 | GOOD | 238.92 | 848105 |
| 848106 | Mineral Claim | ACCENT 4 | 104I | 2022/APR/11 | GOOD | 153.51 | 848106 |
| 852142 | Mineral Claim | PYRAMID PEAK | 104I | 2022/APR/11 | GOOD | 426.85 | 852142 |
| 852162 | Mineral Claim | TUCHO 1 | 104I | 2022/APR/11 | GOOD | 426.68 | 852162 |
| 852163 | Mineral Claim | TUCHO 2 | 104I | 2022/APR/11 | GOOD | 426.84 | 852163 |
| 852164 | Mineral Claim | TUCHO 3 | 104I | 2022/APR/11 | GOOD | 426.91 | 852164 |
| 852165 | Mineral Claim | THE SPHINX | 104I | 2022/APR/11 | GOOD | 426.97 | 852165 |
| 852166 | Mineral Claim | NILE RIVER | 104I | 2022/APR/11 | GOOD | 222.07 | 852166 |
| 852167 | Mineral Claim | SOUTH ROAD | 104I | 2022/APR/11 | GOOD | 187.56 | 852167 |
| 852344 | Mineral Claim | FAR EAST 1 | 104I | 2022/APR/11 | GOOD | 426.82 | 852344 |
| 852345 | Mineral Claim | FAR EAST 3 | 104I | 2022/APR/11 | GOOD | 85.31 | 852345 |
| 852346 | Mineral Claim | FAR EAST 2 | 104I | 2022/APR/11 | GOOD | 426.89 | 852346 |
| 852347 | Mineral Claim | CAMPVIEW 1 | 104I | 2022/APR/11 | GOOD | 306.97 | 852347 |
| 852348 | Mineral Claim | CAMPVIEW 2 | 104I | 2022/APR/11 | GOOD | 340.97 | 852348 |
| 854561 | Mineral Claim | KUTCHO FAULT | 104I | 2022/APR/11 | GOOD | 409.07 | 854561 |
| 858667 | Mineral Claim | ACCENT 5 | 104I | 2022/APR/11 | GOOD | 153.62 | 858667 |
| 861767 | Mineral Claim | ACCENT 6 | 104I | 2022/APR/11 | GOOD | 102.36 | 861767 |
| | | | | | **Total** | **17,060.89** | |

Source: JDS (2017).


525000
530000
535000
540000
545000
550000
6460000
6455000
6450000
6445000
6440000
Kotcho Creek
Andrea Creek
556552
552816
552814
854561
556555
852348
552820
552812
552913
552782
569607
552911
852167
852347
552809
552792
552785
848106
586844
861767
585959
552794
552805
586855
852345
552914
552796
586846
848105
585957
585958
586852
852162
586854
858667
552823
586850
586849
586851
852142
852164
852163
852344
852346
586848
852165
852166
Sources: Esri, DeLorme, NAVTEQ, TomTom, Intermap, increment P Corp., GEBCO, USGS, FAO, NPS, NRCAN, GeoBase, IGN, Kadaster NL, Ordnance Survey, Esri Japan, METI, Esri China (Hong Kong), swisstopo, and the GIS User Community

## 4.2 Issuer's Title

Desert Star has an agreement (dated June 15, 2017) to acquire 100% acquire Capstone's wholly-owned subsidiary Kutcho Copper Corp. which owns the Project. The purchase agreement includes a C$28.8 M cash payment plus the number of Desert Star common shares that, subsequent to completion of the acquisition and any concurrent financing, Capstone will own 9.9% of the issued and outstanding shares of Desert Star.

## 4.3 Royal Gold Inc. Back-In Right

Following notice by Kutcho Copper that it has completed a feasibility study on the Project, Royal Gold will have 120 days to elect to 'not back-in' for a 50% interest in the Project by paying, within two years, three times Kutcho Copper's eligible expenditures on the property to Desert Star. This applies only to that portion of the property on which Barrick previously held an interest (SRK 2008).

## 4.4 Royalty Terms

Pursuant to the Sumac Agreement, Sumac is entitled to a royalty of 2% of net smelter returns on the portion of the Project it sold to the Company, between the third anniversary and the sixth anniversary of the date of commencement of commercial production, and a royalty of 3% of net smelter returns after the sixth anniversary of the date of commencement of commercial production.

Royal Gold is entitled to a royalty of 2% of net smelter returns on the portion of the Kutcho Project they sold to the Company if Royal Gold elects to 'not back-in' on the Project (SRK 2008).

For the purpose of this study, it has been assumed that a royalty of 2% will be paid for the life of the Project.

# 5 Accessibility, Climate, Local Resources, Infrastructure and Physiography

The Kutcho property is located approximately 100 km east of Dease Lake, BC. Dease Lake is a community of about 335 people and has basic services such as an airstrip, medical clinic, school, restaurants, college extension campus, grocery store, and hotels. The Dease Lake area offers a pool of potential project employees that would be supplemented with people from outside the region.

Dease Lake is reachable via a good all weather road, Highway 37 North, from Smithers, BC (about 600 km to the south). Dease Lake is about 400 km from the Port of Stewart (Stewart, BC). A marginal, seasonal access road runs to the property but is only suitable for summer access with special equipment.

Access to the Project area is by fixed-wing aircraft and helicopter from Smithers or Dease Lake, landing at the 900 m gravel airstrip located at the junction of Kutcho and Andrea creeks. The deposit area of the property is connected to the airstrip by a 10 km road. Currently this road has had culverts removed and is only passable to four-wheel drive trucks with good ground clearance. Four-wheel drive vehicles have access to the property via the road to Dease Lake during the late summer and early fall, but this access is weather-dependent due to extensive muddy sections.

The property is located within the Cassiar Mountains, just to the north of the continental divide between the Arctic and Pacific watersheds. The area is moderately rugged with elevations ranging from 1,400 to 2,200 m. Most of the area is alpine with tree line at approximately 1,500 m. Snow cover can persist for nine months of the year, particularly on shady north-facing slopes. Winters are cold and dry, while summers are cool and moist.

Dease Lake, the nearest government weather station, gets about 0.25 m of rain and over 2 m of snowfall annually.

The starter pit, underground mine, tailings storage area, workshop, plant site, and camp accommodation complex are all planned to be located within an area outlined by Andrea, Sumac and Playboy Creeks; refer to Section 18, Figure 18-1 for more information. Power will be generated at site with liquefied natural gas (LNG) generators. Water sources for the Project have not been defined but possible options include run-off collection, wells and dewatering from underground and drawing from creeks.

# 6 History

Mineralization on what was to become the Kutcho property was first discovered in 1968 by a joint venture exploration program operated by Imperial Oil Ltd.

The property has changed hands a number of times with ownership by companies such as Esso Minerals Ltd. (EML) (formerly Imperial Oil, later Homestake Canada Ltd. [Homstake]), Sumac Mines Ltd. (SML) (a subsidiary of Sumitomo Metal Mining Co. Ltd.), Homestake, American Reserve Mining Corporation (ARMC), Teck Cominco Ltd. (TCL), Barrick, WKM, Sherwood Copper Corp. and Capstone Mining Corp.

The discovery was made by prospecting in response to anomalous stream sediment samples collected during a regional drainage survey. Twenty claims were staked by W. Melnyk directly over the as of yet undiscovered main Kutcho sulphide deposit. These claims were allowed to lapse when the other partners in the joint venture declined to fund further exploration. After the statutes of the joint venture agreement expired, Imperial Oil returned to the area in 1972 in order to re-stake the area. However, SML had conducted stream sediment sampling earlier that season, and in response to anomalous samples R. Britten staked eight “two-post” claims along the anomalous stream and eight more claims along the geological strike direction, resulting in the cruciform claim outline overlying the western part of the main Kutcho sulphide deposit. Imperial Oil staked a much larger area encompassing SML’ claims.

Beginning in 1973 both SML and EML carried out exploration work, and their early successes prompted additional staking which resulted in claim boundaries roughly as they are today. Diamond drilling commenced in 1974, and by 1982 approximately 60,000 m had been drilled by both companies, defining three sulphide lenses. During this time EML also drilled a number of exploration targets in other areas of the property with moderate success. Environmental, metallurgical, and engineering studies were initiated by both groups in 1980. A partnership agreement on engineering and development work was signed by EML and SML in 1983, made retroactive to 1981 (the year SML began work driving the adit in order to collect a 100 tonne bulk sample). The agreement was essentially a 50/50 joint venture for development work, and culminated in a prefeasibility study by Wright Engineers Limited in 1985. This study indicated an 11% internal rate of return when using a copper price of US$0.95/lb. Given the risk factors involved and long-term price projections for copper below the US$0.95/lb level, the companies put the Project on hold pending further exploration results. A limited amount of exploration work was done on EML’s claims to the south of the main mineralized trend between 1985 and 1988; however, this work and the numerous geophysical surveys that had been undertaken indicated limited potential for additional open pit mineralization.

In 1989, EML sold most of its mining assets to Homestake. In 1990, Homestake optioned the Kutcho property to ARMC, who funded a $1.1 M exploration program (Homestake remained the operator) which included 7,031 m of drilling in 28 holes, mostly in outlying target areas (thereby earning a 20% interest). Exploration successfully confirmed the presence of extensive areas of favourable geology and alteration indicative of hydrothermal activity, but failed to discover zones of potentially economic mineralization. For example, 10 km southwest of the Kutcho deposit, a narrow zone of cryptocrystalline massive pyrite with a strike length in excess of 5 km was intersected in four widely-spaced drillholes, but was barren of base or precious metals. ARMC carried out engineering studies but did no further exploration work, relinquishing the option in 1993 while retaining a 20% interest in Homestake's property.

The property was optioned to TCL in 1992. TCL carried out deep penetration EM geophysical surveys (UTEM) over the Esso zone with the goal of defining additional conductors along the Kutcho trend. Due to extensive cover of conductive argillaceous units in the hangingwall, the UTEM system was unable to detect the Esso deposit or other conductors at depth, leading TCL to drop their option.

Homestake was purchased by Barrick in 2003. Extensions of the Kutcho stratigraphy to the west have been staked and worked by various companies in the past. Shortly after the discovery of the Kutcho deposits, Noranda staked the Kutcho formation to the west of Kutcho Creek. Noranda conducted geophysical surveys and carried out a small drilling program. The claims were allowed to lapse and were re-staked in 1995 by Gary Belik. Mr. Belik carried out a detailed mapping program and optioned the claims to Atna Resources Ltd. (Atna) in 1997. Atna conducted UTEM geophysical surveys and an extensive drilling program. Results of Atna's work were mixed, and although no deposits were discovered, significant weak to moderately mineralized alteration zones were intersected. Structural complexity and lack of clear geophysical targets prevented additional work and the option was terminated.

Negotiations by WKM to purchase the property from Barrick and Sumitomo were initiated in 2003 and concluded in early 2004, and the property placed into Kutcho Copper, WKM's wholly owned subsidiary. WKM carried out diamond drilling within the Main and Esso deposits during 2004 to confirm historical results and obtain material for metallurgical studies. A second round of drilling by WKM in 2005 tested the Main deposit's potential for up-dip and down-dip extensions, as well as western extensions to the Esso deposit. The Sumac deposit was also drilled in 2005 to test for higher grade zones. A third round of drilling in 2006 focused on infill drilling within the five-year pit area of the Main deposit. The Kutcho property was entered into the Mine Development Review Process in 2006 and EA studies were initiated to provide baseline data for provincial and federal EA reviews (Wardrop 2007).

In February 2008, Sherwood acquired 93% ownership in Kutcho Copper, owner of the Kutcho property.

On May 27, 2008 Sherwood acquired 100% ownership in WKM by amalgamating WKM with a subsidiary so that Kutcho Copper now owns the Kutcho property. On November 27, 2008, Sherwood amalgamated with Capstone under a plan of arrangement that resulted in Kutcho Copper being a wholly owned subsidiary of Capstone. Kutcho Copper embarked upon a program of diamond drilling of 78 holes (81 holes were collared but three were abandoned due to technical issues) for a total of 9,905 m of drill core diameter of 63.5 mm (HQ) size drill core. The results of this program were used to update the mineral resource estimate for the Main deposit as discussed in Section 14.

The principal objectives of the 2008 drill program were to:

- Infill gaps in previous resource drilling programs and enlarge the assay database;
- Better define and test higher grade trends for expansion within the Main deposit;
- Demonstrate grade continuity to support a better resource classification;
- Provide material for extensive metallurgical testing that would relate to a revamped mine plan;

- Provide geotechnical information for mine design and for assessment of infrastructure locations; and
- Provide information to support project permitting activities and to develop a mine closure plan.

The program was designed principally to increase the assay sample density and to provide material for further metallurgical and environmental testing. The drill program in-filled on earlier work that had already defined the gross limits and overall geometry of the mineralized zone and as expected did not result in a material change to these limits or the geometry of the resource model, but it did better define higher grade trends within the deposit and provided more confidence in, and thus increased, the classification levels for the 2008 mineral resource estimate.

Kutcho Copper completed a second diamond drill program in 2010. On July 3rd, 2010, a program totalling 17,970 m of infill and step-out drilling commenced on Esso deposit which generated significant changes in the Mineral Resource Estimate of Esso deposit. The principal objectives of the 2010 drill program were to:

- Test selected undrilled perimeter areas to expand the size of the Esso deposit;
- Infill gaps in previous mineral resource drilling programs at Esso and enlarge the assay database;
- Better define and test higher grade trends for expansion within Esso deposit;
- Demonstrate grade continuity at Esso to support a better mineral resource classification;
- Provide material for extensive metallurgical testing at Esso that will relate to a mine plan;
- Provide geotechnical information for mine design and for assessment of infrastructure locations; and
- Provide information to support project permitting activities and a mine closure plan.

The 2010 drill program was designed principally to increase the assay sample density and to provide material for further metallurgical and environmental testing related to the Esso deposit. Most drillholes in-filled on earlier work that had already defined the gross limits and overall geometry of the mineralized zone at Esso and, as expected did not result in a material change to these limits or the geometry of the resource model. The 2010 program better defined higher grade trends within the deposit, eliminated an internal gap in the mineral resource model at the west end of Esso deposit and provided more confidence in, and thus increased, the classification categories.

From April 8 to 19, 2011 an airborne electromagnetic (EM) survey was conducted over the Kutcho property using Geotech Ltd.'s proprietary versatile time domain electromagnetic (VTEM) system. The VTEM survey consisted of 1,649.4 line-km (plus tie-lines) covering 147.2 km$^2$. The survey grid was oriented along flight lines at 004° azimuth, perpendicular to the strike of the host rock strata in the deposit area.

The principal objectives of the VTEM survey included:

- improved depth penetration (up to 750 m);
- a potential to see through the conductive overburden higher in the stratigraphy; and
- generation of precisely located drill-ready EM targets that did not require follow-up ground surveys.

This geophysical program was followed by a drill program that tested nine high-priority VTEM targets totalling 4,227 m of HQ size core in 20 holes (including 2 short abandoned holes). The 2011 exploration program generated the following conclusions:

- The Sumac deposit extends further south, east, and up-dip than previously recognized;
- Anomalous VTEM responses were observed over the east end of the Sumac deposit and along the up-dip edge of the Esso stratigraphic horizon. This represents deeper levels of penetration and higher levels of sensitivity than previous airborne EM systems used at Kutcho; and
- Graphitic mudstone horizons are commonly close to pyritic tuff horizons, and the mudstone itself may host exhalative sulphides.

## 6.1 Historic Resources and Reserves

The following lists various historic resources and reserves that have been reported on the Project within the Main (also known as the Kutcho zone), Sumac and Esso (also known as the Esso West zone) zones.

The first known resources or reserves were reported in 1986 by Imperial Oil. These were listed as reserves for the three zones: Kutcho - 17 Mt grading 1.62% Cu, 2.32% Zn, 29.2 g/t Ag and 0.39 g/t Au; Sumac - approximately 10 Mt grading 1.0% Cu and 1.2% Zn; and Esso West – with about 1 to 1.5 Mt of approximately double Kutcho grades (CIM 1986).

An "underground mineable reserve" was reported in 1991 for the Kutcho zone as 11.6 Mt grading 1.67% Cu, 2.30% Zn, 32.7 g/t Ag and 0.36 g/t Au; and for the Esso West zone as 2.7 Mt grading 2.14% Cu, 3.61% Zn, 44.9 g/t Ag and 0.40 g/t Au (George Cross News Letter No.54 (March 18) 1991).

In 1996, mineable reserves of the Kutcho zone is also reported as containing 14.3 Mt grading 1.76% Cu, 2.54% Zn, 35 g/t Ag and 0.37 g/t Au (BC MEMP 1996).

The 1986, 1991, and 1996 historic resource/reserves were reported prior to the advent of NI 43-101. There is limited information with respect to the classifications used; however, they are listed as reserves which suggests that these were internal company figures. The reader is cautioned not to rely upon any of these estimates or figures and that they are only supplied for historical context and to show the progression of the Project. The author has not done sufficient work to classify these historical estimates as current and the issuer is not treating these historical estimates as current.

The following include resource estimates that were performed on behalf of WKM, Sherwood Copper and Capstone, respectively. These subsequent resource and reserve estimates were performed following the advent of NI 43-101 and are compliant with the instrument at the time at which they were reported. These are supplied for historical context and comparison purposes. In addition, they serve to show progression of the Project.

The WKM resource and reserve estimates were published in the Kutcho Project Prefeasibility Study dated September 2007 authored by Wardrop. Tables 6-1 and 6-2 list the 2007 resources and reserves, respectively.

### Table 6-1: 2007 Western Keltic Mines Inc. Resources at 0.75% Copper Equivalent Cut-off

| | Tonnes (kt) | Cu (%) | Zn (%) | Ag (g/t) | Au (g/t) | CuEq (%) |
|---|---|---|---|---|---|---|
| **Main Deposit** | | | | | | |
| Measured | 2,938 | 1.83 | 2.65 | 28.0 | 0.39 | 3.06 |
| Indicated | 12,717 | 1.60 | 2.04 | 25.7 | 0.31 | 2.56 |
| Measured plus Indicated | 15,654 | 1.65 | 2.15 | 26.1 | 0.32 | 2.65 |
| Inferred | 811 | 0.95 | 1.92 | 24.2 | 0.33 | 1.86 |
| **Sumac Deposit** | | | | | | |
| Inferred | 10,615 | 0.94 | 1.45 | 14.0 | 0.14 | 1.60 |
| **Esso Deposit** | | | | | | |
| Indicated | 2,040 | 2.24 | 3.96 | 37.7 | 0.49 | 4.05 |
| Inferred | 443 | 2.47 | 4.15 | 38.1 | 0.53 | 4.37 |
| **All Deposits** | | | | | | |
| Measured and Indicated | 17,695 | 1.71 | 2.36 | 27.5 | 0.34 | 2.81 |
| Inferred | 11,868 | 1.00 | 1.58 | 15.6 | 0.17 | 1.72 |

Note: Following interpolation of the Cu, Zn, Au, and Ag grade models, a model of Copper Equivalent (CuEq) was generated using the following formula:

CuEq = (%Cu*1)+(%Zn*0.414)+(g/t Ag*0.00256)+(g/t Au*0.119)

This equivalent is calculated using prices of US$2.36/lb Cu, US$1.38/lb Zn, US$534.36/oz Au, and US$9.78/oz Ag. Metallurgical recoveries are 87% Cu, 79% Zn, 31% Au, and 37% Ag.

Source: JDS (2017).

### Table 6-2: 2007 Western Keltic Mines Inc. Reserves at $62 Net Smelter Return Cut-off

| | Tonnes (kt) | Cu (%) | Zn (%) | Ag (g/t) | Au (g/t) |
|---|---|---|---|---|---|
| **Main Deposit** | | | | | |
| Probable Mineral reserve (95% recover, 5% dilution) | 15,021 | 1.59 | 20.9 | 25.0 | 0.31 |
| **Esso Deposit** | | | | | |
| Probable Mineral reserve (90% recovery, 15% dilution) | 2,047 | 1.97 | 3.50 | 33.2 | 0.43 |
| **All Deposits** | | | | | |
| Probable Mineral Reserve | 17,068 | 1.63 | 2.26 | 26.0 | 0.32 |

Source: JDS (2017).

The Kutcho Copper (subsidiary of Sherwood Copper) estimates were published in the Preliminary Economic Assessment - Kutcho Project dated May 2008 authored by SRK Consultants. Tables 6-3 and 6-4 list the 2008 resources.

### Table 6-3: 2008 Resources for Kutcho Creek at CuEq Cut-off Grade of 0.75%

| Resource Category | Tonnes (kt) | CuEq (%) | Cu (%) | Zn (%) | Ag (g/t) | Au (g/t) |
|---|---|---|---|---|---|---|
| Indicated | 17,285 | 2.54 | 1.56 | 2.12 | 26.1 | 0.29 |
| Inferred | 367 | 2.43 | 1.62 | 1.77 | 23.6 | 0.13 |

Note: CuEq = Cu% + (0.41555 X Zn%) + (0.00257 X Ag g/t) + (0.1177 X Au g/t)

This equivalent is calculated using prices of US$2.36/lb Cu, US$1.38/lb Zn, US$534.36/oz Au, and US$9.78/oz Ag. Metallurgical recoveries were assumed to be 87% Cu, 79% Zn, 31% Au, and 37% Ag.

Source: JDS (2017).

Mine design for the Main pit was initiated with the development of a net smelter return (NSR) model. The model included estimates of metal prices, exchange rate, mining dilution, mill recovery, concentrate grade smelting and refining payables and costs, freight and marketing costs and royalties. The NSR model was based on a 5 m x 5 m x 5 m block size. The model was then used with the Gemcom Whittle - Strategic Mine Planning™ (WhittSle) software to determine the optimal mining shell. Mine planning and scheduling was then conducted on the optimal pit shell with the use of MineSight® software and mineral reserves were estimated. A $31/t NSR cut-off was used within the planned pit.

### Table 6-4: 2008 PEA Main Pit Life of Mine Resource at $31/t Net Smelter Return Cut-off

| Class | Tonnes (kt) | In situ Grade | | | | Contained Metal | | | |
|---|---|---|---|---|---|---|---|---|---|
| | | Cu (%) | Zn (%) | Au (g/t) | Ag (g/t) | Cu (Mlb) | Zn (Mlb ) | Au (koz) | Ag (koz) |
| Indicated | 10,513 | 1.73 | 2.35 | 0.27 | 26.3 | 400.7 | 543.7 | 90 | 8,900 |
| Inferred | 163 | 1.67 | 1.93 | 0.11 | 22.2 | 6.0 | 6.9 | 1 | 116 |

Source: JDS (2017).

The Kutcho Copper (subsidiary of Capstone) estimates were published in the PEA Revised Underground Mining Option, Kutcho Project dated July 2010 authored by JDS. Tables 6-5 and 6-6 list the 2010 resources.

### Table 6-5: Kutcho Project – Mineral Resource Estimate at 1.5% Copper Cut-off

| Class | Tonnes (kt) | Grade | | | | | Contained Metal | | | |
|---|---|---|---|---|---|---|---|---|---|---|
| | | Cu (%) | Zn (%) | Au (g/t) | Ag (g/t) | CuEq[1] (%) | Cu (Mlb) | Zn (Mlb) | Au (koz) | Ag (koz) |
| Measured (M) | 5,421 | 2.15 | 2.86 | 0.34 | 31.4 | 3.70 | 256.6 | 341.8 | 59 | 5,482 |
| Indicated (I) | 4,994 | 2.14 | 2.83 | 0.39 | 33.5 | 3.74 | 235.8 | 312.0 | 62 | 5,376 |
| **M&I** | **10,415** | **2.14** | **2.85** | **0.36** | **32.4** | **3.72** | **492.4** | **653.8** | **121** | **10,857** |
| Inferred | 1,893 | 2.09 | 2.93 | 0.46 | 33.6 | 3.78 | 87.3 | 122.4 | 28 | 2,047 |

Notes: Numbers may not total due to rounding.

1 Equivalent copper grade calculated using these metal prices in US$: Cu $1.50/lb, Zn $0.50/lb, Ag $12.00/oz, Au $700.00/oz.

Source: JDS (2017).

**Table 6-6: 2010 Kutcho Project – Potentially Minable Mineral Resources**

| Resource Class | Tonnes (kt) | Cu (%) | Zn (%) | Ag (g/t) | Au (g/t) | CuEq (%) | Total (%) |
|---|---|---|---|---|---|---|---|
| Measured | 2,310 | 2.06 | 2.97 | 30.6 | 0.37 | 3.38 | **21.1** |
| Indicated | 7,523 | 1.92 | 2.64 | 28.8 | 0.33 | 3.09 | **68.8** |
| Inferred | 1,109 | 2.48 | 3.99 | 35.0 | 0.57 | 4.30 | **10.0** |
| **Grand Total** | **10,942** | **2.01** | **2.80** | **29.8** | **0.36** | **3.28** | **100** |

Source: JDS (2017).

The Kutcho Copper (subsidiary of Capstone) estimates were published in the Kutcho Copper Project Prefeasibility Study dated February 2011 authored by JDS. Tables 6-7 and 6-8 list the 2011 resources.

**Table 6-7: 2011 Kutcho Project – Mineral Resource Estimate at 1.5% Copper Cut-off**

| Class | Tonnes (kt) | Grade | | | | Contained Metal | | | |
|---|---|---|---|---|---|---|---|---|---|
| | | Cu (%) | Zn (%) | Au (g/t) | Ag (g/t) | Cu (Mlb) | Zn (Mlb) | Au (koz) | Ag (koz) |
| Measured (M) | 5,421 | 2.15 | 2.86 | 0.34 | 31.4 | 256.6 | 341.8 | 59 | 5,482 |
| Indicated (I) | 5,859 | 2.24 | 3.67 | 0.45 | 41.6 | 289.2 | 473.5 | 84 | 7,831 |
| **M&I** | **11,280** | **2.19** | **3.28** | **0.39** | **36.7** | **545.8** | **815.3** | **143** | **13,313** |
| Inferred | 1,090 | 1.74 | 1.74 | 0.35 | 30.7 | 41.9 | 49.1 | 12 | 1,077 |

Source: JDS (2017).

**Table 6-8: 2011 Kutcho Project – Mineral Reserve Estimate**

| Deposit | Classification | Tonnes (kt) | Cu (%) | Zn (%) | Ag (g/t) | Au (g/t) |
|---|---|---|---|---|---|---|
| Main(1) | Probable | 8,106 | 1.92 | 2.51 | 28.0 | 0.31 |
| Esso(2) | Probable | 2,335 | 2.32 | 5.53 | 57.5 | 0.59 |
| **Total/Average** | **Probable** | **10,441** | **2.01** | **3.19** | **34.6** | **0.37** |

Notes: (1) at a 1.5% Cu cut off.
(2) at a 1.0% Cu cut off.

Source: JDS (2017).

The historical resources in the years 1986, 1991, and 1996 were reported prior to the advent of NI 43-101. There is limited information with respect to the classifications used; however, they are listed as reserves which suggests that these were internal company figures. The reader is cautioned not to rely upon any of these estimates or figures and that they are only supplied for historical context and to show the progression of the Project. The author has not done sufficient work to classify these historical estimates as current and the issuer is not treating these historical estimates as current.

The subsequent 2007, 2008, 2010, and 2011 resource/reserves were reported in compliance with NI 43-101 and can be relied on; however, they are superseded by the resource and reserves estimates which are the subject of this current report. The author has done sufficient work and was in fact the author of the 2008, 2010, and 2011 resource estimates.

All historical resource estimates are being supplied for historical context and to show the progression of the Project. In addition, the issuer is not treating these historical estimates as current.

On June 15, 2017, Desert Star announced the acquisition of 100% ownership of Kutcho Copper and by extension, the Kutcho deposit.

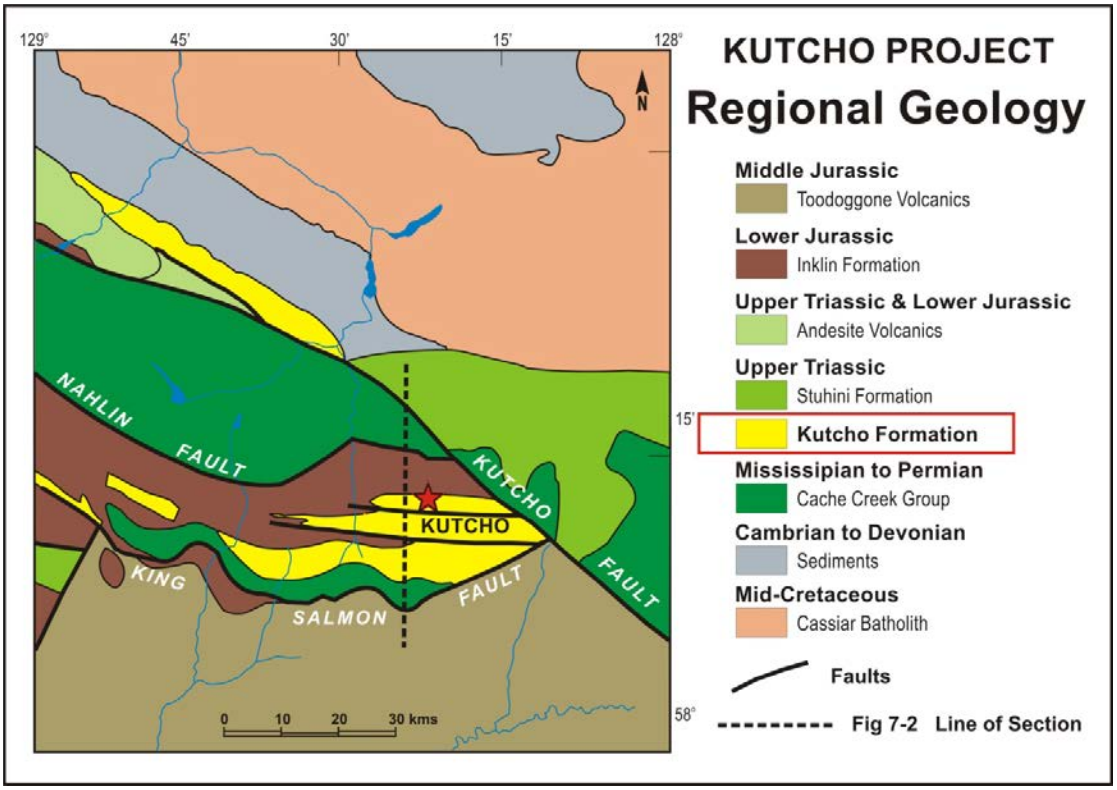
# 7 Geological Setting and Mineralization

The Kutcho property lies within the King Salmon Allochthon (KSA), a narrow belt of Permotriassic island arc volcanic rocks and Jurassic sediments, sandwiched between two northerly-dipping thrust faults: the Nahlin fault to the north, and the King Salmon fault to the south (Figure 7-1). Penetrative foliation and axial planes of major folds are parallel to these east-west trending bounding faults. The belt of volcanic rocks is thickest in the area where it hosts the VMS deposits, partly due to primary deposition, but also to stratigraphic repetition by folding and possibly thrusting. The KSA is terminated to the east (near the eastern edge of the property) by the Kutcho strike-slip fault (Mansy and Gabrielse 1978; Gabrielse 1978), but extends to the west for hundreds of kilometres. However, Kutcho Formation rocks thin to the west, and do not occur or are rarely exposed 10 km to the west of Kutcho Creek. Stratigraphy of the KSA consists primarily of the Kutcho Formation, which is overlain by the limestone of the upper Triassic Sinwa Formation, which in turn is overlain by sediments (predominately argillite) of the Lower Jurassic Inklin Formation. Major folds are delineated by the Sinwa limestone and, where the Sinwa is absent, by the contact between the Kutcho and Inklin Formations (Figures 7-1 and 7-2).

## 7.1 Stratigraphy

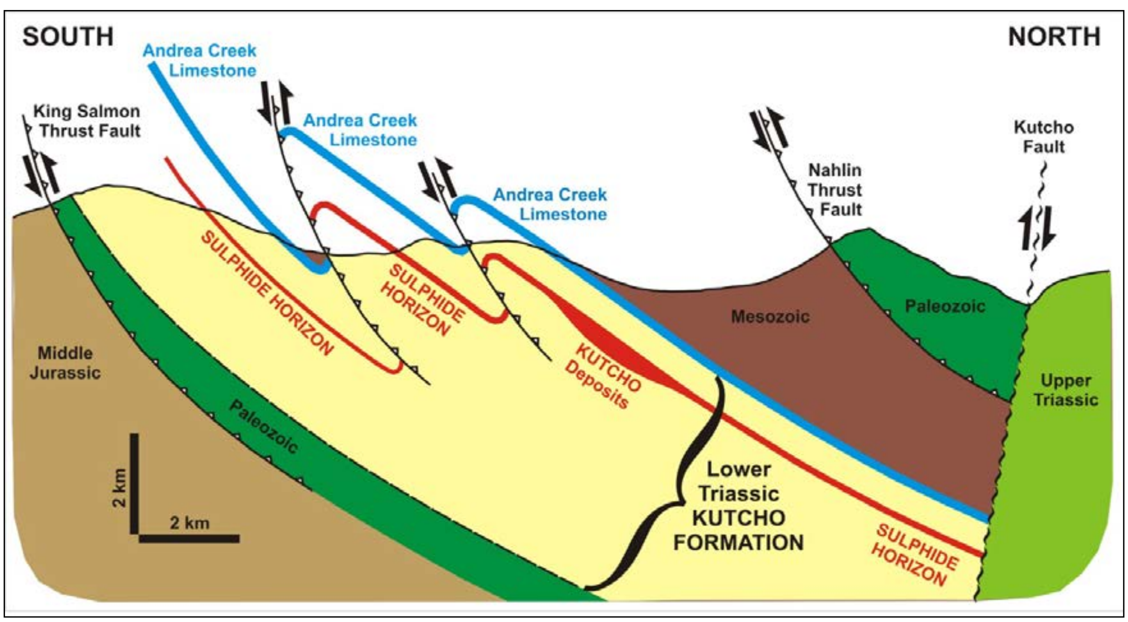
Stratigraphy of the Kutcho property has been described by CIM (1986), Thorstad (1983), and Holbek (1985) and will only be briefly reviewed here. Stratigraphy is best understood in the upper part of the Kutcho formation, where units are better exposed and drill information is available. The footwall stratigraphy (particularly away from the deposit area) is not well understood. The lowest rocks in the section are exposed on the southern ends of Imperial and Sumac ridges and include interbedded basalts, basaltic tuffs and wackes, rhyolitic lapilli tuffs, and possible trondhjemite. The mafic rocks are fine to very fine-grained, chloritic, equigranular to weakly porphyritic, and are commonly given the field term of greenstone. The lapilli tuffs are pale grey and siliceous, and commonly contain very fine quartz phenocrysts and lenticular fragments from 0.5 to 3.0 centimetres (cm) in length. Textures can only be seen on weathered (but lichen-free) surfaces. The trondhjemite unit is somewhat unclear. It is described by Pearson and Pantaleyev (1975) and CIM (1986) as a fine-grained, equigranular, plagioclase-rich unit, but it is very similar to some of the tuffaceous units as well. A weak but pervasive carbonate-chlorite-pyrite or propylitic alteration of this unit is subtle but discernible. Stratigraphy is shown in Figure 7-3.

## Figure 7-1: Regional Geology Setting



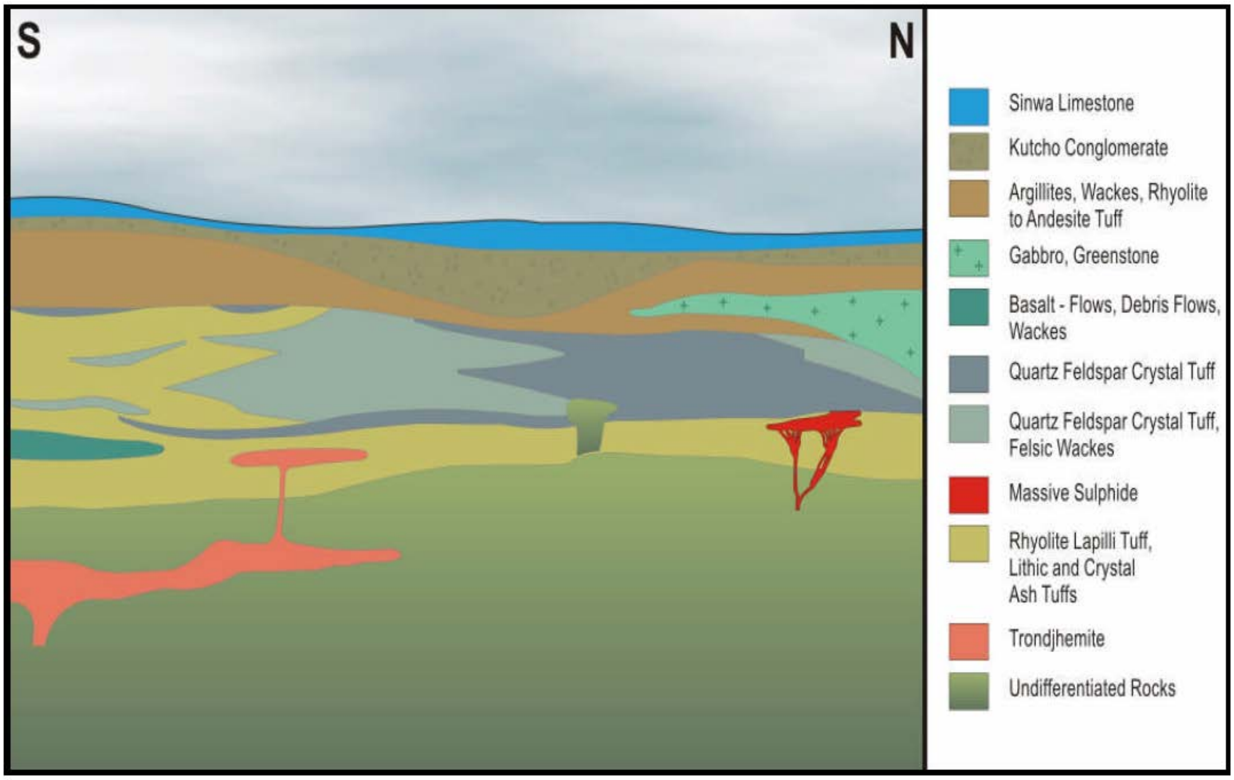

Sources: Wardrop (2007).

## Figure 7-2: Kutcho Area Schematic Cross-section




Sources: Wardrop (2007).

**Figure 7-3: Property Stratigraphy Schematic ~ 10x Vertical Exaggeration**




Sources: Wardrop (2007).

Rocks overlying the greenstone-lapilli tuff package have been termed the ore sequence, and consist of lapilli tuffs, crystal-lithic tuffs, and quartz and quartz-feldspar crystal tuffs. Away from the deposit area these units tend to be thin, interbedded, and variably but weakly altered. Fine quartz-crystal ash tuff with silica rich laminations and rare thin zones of ferroan dolomite typically mark the distal exhalative zone. The sulphide zones occur at (or near to) the contact between footwall lapilli tuff and hangingwall quartz crystal tuff. In general, both lapilli fragments and phenocrysts are much coarser grained in the vicinity of the deposits, becoming progressively finer grained to the south and west. The quartz-feldspar crystal tuff is quartz-rich near the deposits, becoming more feldspar-rich to the south.

A large zone of feldspar crystal tuff with almost no free quartz occurs a few hundred metres south of the sulphide zones, and whether this unit is footwall, hangingwall, or a facies equivalent to the quartz-feldspar crystal tuff is as yet indeterminate. An interesting feature is the occurrence of a coarse breccia texture within the quartz-feldspar crystal tuff immediately over the sulphide zones. The breccia fragments are typically sub-round, from 2 to 30 cm in size, and are identical to crystal tuff matrix except for an increase in the amount of epidote from 1 to 2% to closer to 10%. This feature has been interpreted to be a debris flow of semi-consolidated crystal tuff, shed from a flow dome complex and trapped in the graben or half-graben structure which hosts the sulphide lenses.

Rocks between the ore sequence and the overlying conglomerate unit are referred to as the Tuff-Argillite Unit (TAU), and consist of gabbroic to basaltic intrusive sills and dykes, greywackes, and argillite. In the area of the deposit the gabbroic units are commonly coarse-grained and are commonly referred to as metagabbro. Higher in the section, to the east and west of the Kutcho deposits, this mafic unit becomes much finer grained, and an intrusive origin is not so clearly identified.

The amount of argillite increases in a westerly direction, supporting the concept that this direction is towards the marine basin. The base of the TAU is interpreted to be a thrust fault, and there are numerous other fault zones within the unit as noted in drill cores and the adit. The basal thrust plane does not cause significant offset of the Sinwa limestone in the fold nose to the west, implying a scissor-type action with increasing movement to the east.

Overlying the TAU and truncating it to the west is the Kutcho conglomerate. This unit is a heterolithic, fragment-supported conglomerate composed of sub-rounded clasts, ranging in size from 1 to 38 cm (long axis), and derived from all of the underlying lithologies. The conglomerate is conformably overlain and transitional into the Sinwa limestone, which in turn appears to be conformably overlain by Jurassic aged Inklin Formation argillite. (However, it is quite possible that there could be a contact between Kutcho Formation argillite and Inklin Formation argillite higher in the section, which would be difficult to spot and could be unconformable).

The Kutcho Formation is of Upper Triassic to uppermost Permian in age. Thorstad (1983) determined an Upper Triassic age on the basis of rubidium-strontium dating of volcanic rock sand regional stratigraphic constraints. Subsequent work by F. Childe at the Mineral Deposit Research Unit of the University of British Columbia in 1996 suggests an age range from Lower Triassic to uppermost Permian (Wardrop 2007).

## 7.2 Structure

Rocks of the Kutcho formation are characterized by penetrative axial planar foliation that has a relatively constant strike direction of 270º to 290° with northerly dips from 45º to 65°. Minor but systematic changes in foliation from the east to west suggest low amplitude buckling of the fold axes. There appears to be a tendency for the dip of the foliation to decrease with structural depth, indicating that the axial planes are convex to the south.

Folds are open to tight, asymmetrical, inclined, and verging to the south. Fold plunges range from 0º to 30° in a westerly direction. Folds are most evident in well bedded competent units, and therefore spatial distribution of the fold data is heavily biased to the western property area, where these units predominate.

Two aspects of the structure that critically affect stratigraphic interpretations are the number and size of foliation parallel thrust faults, and the degree to which the folds are propagated through the stratigraphic sequence. Neither of these aspects can be determined independently, and thus there remains considerable scope to reinterpret the stratigraphic position of various units locally. Foliation parallel thrust faults are difficult to detect from surface outcrop, but can be inferred from missing stratigraphy, contact geometry, shearing, and topographic evidence. Faults of this type are consistent with the deformation style and are considered to be prevalent over the property area.

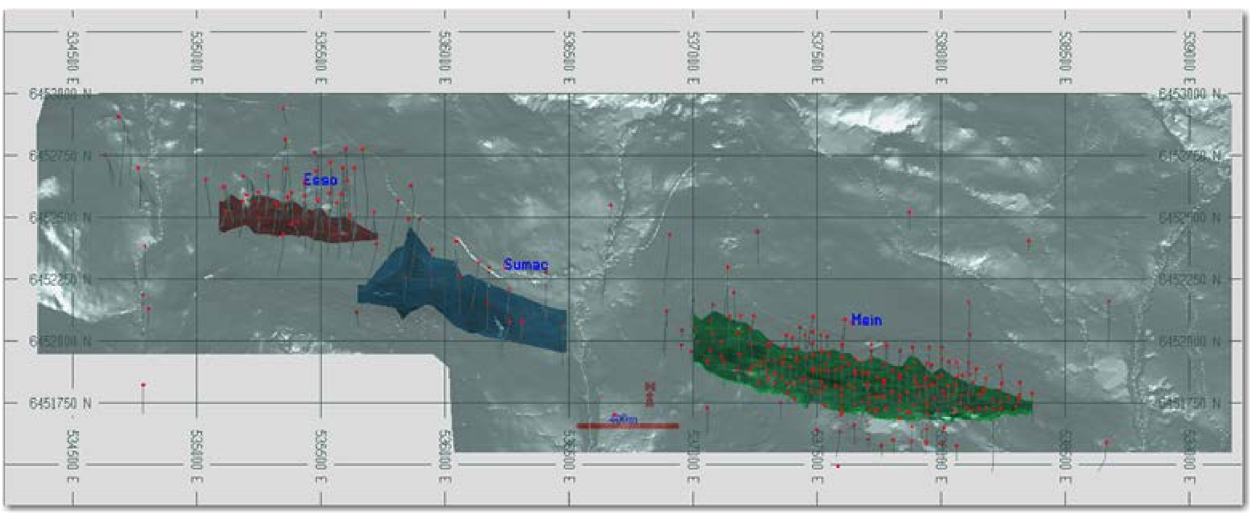
Fold hinges outlined by the Sinwa limestone unit on Conglomerate Ridge (immediately east of Kutcho Creek) are difficult to trace in an easterly direction. Structural data (Holbek 1985) indicate that the folds are cylindrical and therefore should be continuous within the depth of exposed stratigraphy. However, lithological competency contrasts are likely to result in disharmonic folding (Holbek and Heberlein 1986), causing discontinuity of the axial plane towards the core of the fold. Stratigraphically thicker units will tend to produce a series of lower amplitude folds toward the core of the structure, which may explain why the fold axes so clearly outlined by the limestone unit on the western part of the property are not at all evident to the east, in the vicinity of the Sumac and Main deposits. Therefore, a certain degree of flexibility needs to be maintained regarding structural and stratigraphic interpretations in the vicinity of the sulphide deposits (Wardrop 2007).

## 7.3 Mineralization

Mineralization occurs as three deposits along a 3.5 km trend. Metallic minerals occur in a series of massive sulphide lenses and include pyrite, sphalerite, chalcopyrite, bornite, minor chalcocite, trace tennantite, galena, digenite, djurleite, and idaite. Gangue minerals include quartz, dolomite, ankerite, sericite, gypsum, and anhydrite.

There are three known deposits that comprise the Project and form a westerly plunging linear trend. From east to west, the deposits are termed the Main, Sumac, and Esso deposits. The Main deposit comes to surface at its eastern end, whereas the Esso deposit occurs at depths about 400 m below surface and are shown with drillholes and topography in Figure 7-4.

**Figure 7-4: Main, Sumac and Esso Deposits with Drillholes and Topography**




Source: JDS (2017).

## 7.4 Main Deposit

The Main deposit has an elliptical, lenticular shape with approximate dimensions of 1,500 m long, 260 m wide (down-dip), and up to 36 m maximum thickness. The long axis of the deposit plunges to the west at about 12°, just slightly less than the regional fold axes. The deposit is conformable with stratigraphy, dipping moderately to the north. There is a gentle warping of the deposit, such that the dip of the deposit changes from east to west and north to south. The shallowest dip (about 38°) occurs at the south-eastern edge and becomes progressively steeper (to about 63°) at the northwestern edge. In general, the up-dip edge of the sulphide lens is narrow and pinches out, whereas the down-dip edge is thicker and interlayered with tuffaceous rock.

In detail, the Main deposit is composed of three depositional cycles. The cycles are interpreted to begin with pyritic mineralization, which then becomes more copper-rich and finally zinc-rich. The cycles are variably separated by siliceous or carbonate exhalative material and minor volcanic ash and detritus. Interpretation of the shape of the sulphide zone, taken together with the observed volcanic and depositional textures of the enclosing rocks, suggests that the sulphide mineralization was deposited in a structural depression, likely a half-graben type structure. Fine mineralogical layering and sulphide-ash, sulphide-silica, or carbonate inter-layering, as well as framboidal and snowball textures in both the sulphide and carbonate minerals, suggests quasi-sedimentary deposition at the seawater seafloor interface. Polished section analysis indicates that very little sulphide recrystalization has taken place.

Sulphide mineralogy of the deposit is relatively simple, consisting of pyrite, chalcopyrite, sphalerite, and bornite, with minor sulphide minerals chalcocite, tetrahedrite, digenite (and related minerals), galena, idaite, hessite, and electrum. Gangue minerals include quartz, dolomite, ankerite, sericite, gypsum, and anhydrite. Fluorite and barite have been observed, but do not occur in volumetrically significant amounts.

The internal stratigraphy of the Main deposit was determined by detailed drill core logging (Holbek and Heberlein 1986) along a single longitudinal section of drillholes. The deposit appears to have formed from three hydrothermal-depositional cycles that begin with barren pyrite which grades into a copper-rich middle and zinc-rich top.

Depositional cycles are commonly separated by layers of exhalative quartz and/or carbonate and minor volcanic ash; however, continued hydrothermal activity results in sulphide replacement mineralization. Additional features also cause complexity to the internal sulphide stratigraphy, such as an irregular depositional surface, localized slumping of sulphide mineralization or chimney collapse, and late stage (post depositional) hydrothermal activity. Areas of late overprinting by oxidized copper species and enrichment in precious metals are interpreted as indicators of vent areas, and occur along a linear trend on the down-dip side of the deposit, with two hot-spots near each end of the deposit. However, no well-defined areas of classical footwall stringer mineralization have been identified by drilling. The upper contact of the sulphide mineralization is relatively sharp, with almost no sulphide minerals occurring in the hangingwall rocks except for scattered coarse grains of porphyroblastic pyrite. However, alteration of feldspar to sericite and carbonate in the hangingwall is intense, and occurs for up to 50 m above the sulphide contact. It is common for a small shear zone to occur at the sulphide-schist contact, which varies in thickness from 20 to 200 cm, and in many drillholes, carries some grade. The base of the deposit consists of nearly barren massive pyrite with interstitial quartz. The contact between ore and the footwall pyrite zone can be either gradational or sharp. Below the footwall pyrite zone is quartz-sericite schist with bands of generally barren massive to semi-massive pyrite. The footwall pyrite content diminishes with depth away from the deposit, but extends to a maximum depth of 200 m below the central part of the deposit.

## 7.5 Sumac Deposit

The Sumac deposit was initially identified by a chargeability anomaly in the mid-1980's and is located approximately 550 m west of the Main deposit, is nearly continuous with the Esso deposit and sits within a local depression relative to the Main and Esso deposits. A total of 20 drillholes at approximately 100 m spacing define the Sumac deposit. Better intercepts include 1.45% Cu, 2.56% Zn, and 23.7 g/t Ag over 26.1 m, 1.37% Cu, 1.9% Zn, and 26.2 g/t Ag over 23.4 m, and 1.94% Cu, 2.66% Zn, and 43.2 g/t Ag over 10.1 m.

The Sumac deposit is finely banded but massive and competent, and has the highest sulphide content (>90%) of the three deposits. Alteration of the host stratigraphy around it is very similar to that of the Main and Esso Deposits.

## 7.6 Esso Deposit

The Esso deposit is the deepest and most westerly massive sulphide lens and lies between 400 to 550 m below the surface. It was discovered by following the westward trend of mineralization down plunge beyond the Main and Sumac deposit areas. The Esso deposit has an elongated lens shape with a strike length of approximately 640 m, a dip direction of 240 m and is up to 21 m thick but averages approximately 12 m thick. The deposit consists of two connected lenses, the upper lens being the larger of the two. The mineralization at Esso deposit is higher grade than at Main or Sumac deposits, but displays similar mineral zonation with copper or zinc layers or zones, as well as zonation in thickness and grade from the central deposit area. Alteration at Esso is similar to the Main deposit, where sericite alteration of feldspars in the hangingwall is gradational from very weak at distances up to 50 m, to very intense with proximity to the sulphide zone. As a result of the 35 drillholes completed in 2010 the Esso deposit is now drilled off on approximately 50 m centres; allowing reclassification of the entire Mineral Resource for Esso into the Indicated Category.

## 7.7 Other Mineralization

Other zones of mineralization include the footwall zone (FWZ), and the Jenn Area. The FWZ occurs approximately 100 m stratigraphically below the Main deposit and slightly up-dip and to the east of the centre of the Main deposit. The FWZ is relatively narrow, at 2 to 5 m thick, and relatively zinc-rich. The mineralization was only systematically drilled up to the historical Esso-Sumac property boundary, but a number of drillholes by SML (and more recently WKM) demonstrate that the FWZ does not extend for significant distances to the west and down-dip of its current position.

The Jenn area is on the eastern end of the property and received a fair amount of attention by EML. Although significant alteration and some local mineralization were intersected in a number of drillholes, no resources have been defined in the Jenn area.

# 8 Deposit Types

Mineralization of the Project is part of the VMS or volcanic hosted massive sulphide (VHMS) family of deposits. These deposits are major sources of copper, zinc, lead, silver, and gold around the world. Speculation about the origin of these deposits goes back to the mid-1850s, when various French and English scientists postulated chemical precipitation from seafloor volcanic activity (Stanton 1991). In the early 19th century, Japanese workers documented astute observations of the sulphide textures preserved in the Kuroko deposits of Japan and the association of these deposits with rhyolite domes and articulated the submarine sinter theory. However this work did not seem to attract much attention, and genetic theories or models of ore formation of this deposit type did not gain international acceptance until similar observations were published by others worldwide in the 1950s and 1960s (Ohmoto and Skinner 1983). Discovery of the Red Sea brine deposits in 1965 provided substantial impetus for the proponents of the submarine exhalative model. A certain amount of controversy between syngenetic and epigenetic theories continued through the 1970s, but with the advent of deep-sea submersibles and the filming of black and white hydrothermal vents (or smokers) in volcanic rift zones on the seafloor, scientific models went to a new level of realism and detail.

VMS deposits have been classified into various subtypes, depending upon the composition of the host rocks and the mineralization, and the tectonic setting of origin. The Kutcho deposits are VMS deposits of the Kuroko type or felsic volcanisiliciclastic depending upon the classification scheme. Mineralization is related to felsic volcanism in island arc or back-arc tectonic settings. Perhaps the most significant feature of VMS deposits from an exploration perspective is their tendency to occur in clusters. Larger VMS camps can have up to 25 discrete deposits, and mineralized districts are common.

Features of the Kutcho deposits suggest that they formed at (or very near to) the water-seafloor interface in a structurally controlled depression, likely a half-graben type structure. The Kutcho deposits have some uncommon features: the absence of lead and barite is likely due to the low potassium content of the volcanic host rocks (and presumably the associated rhyolite dome), and the abundant carbonate is probably of exhalative origin.

Alteration associated with VMS deposits is well documented, and provides a valuable exploration tool in that the area of alteration is much larger (by a factor of up to 10 to 100) than the actual sulphide deposit, thereby providing a much larger exploration target. Extensive studies of the alteration around the Kutcho deposits have been undertaken, and the chemical composition of the alteration is well zoned about the hydrothermal vent areas. This zonation allows the use of geochemical analysis of drill core within the alteration zone to provide vectors towards the hydrothermal vent area and, hopefully, the sulphide deposits.

# 9 Exploration

There are no current exploration activities to report on the property.

# 10 Drilling

## 10.1 Drilling History

The first drillholes into the Main deposit where carried out nearly simultaneously by Esso Minerals Ltd. and SML in the summer of 1974, although within different areas of the deposit. The first two seasons of drilling were experimental, with relatively wide-spaced drillholes used to determine approximate extents of the mineralization. In 1976, SML carried out a chargeability geophysical survey, which provided clear indications of the size of the conductive mineralization, as well as indications of the Sumac deposit. Shortly thereafter both companies adopted a grid approach to the location of drillholes. A common grid for both companies was implemented later, with north-south grid lines spaced at 60 m. The 60 m spacing (as opposed to the more common 50 m) was chosen as it allowed for four subdivisions (if required) and better approximated the 100 and 200 foot line spacing used by EML for initial soil geochemical surveys. Consequently, almost 90% of the drillholes within the Main deposit are located along the grid lines. Similar approaches were applied to the Sumac and Esso deposits.

In 2004, initial large diameter (HQ) core drilling in the Main deposit was carried out by WKM to verify historical data and to obtain metallurgical samples. The approach to drillhole locations for this program was to obtain a distribution of drillholes covering the entire deposit area, with specific drillhole locations placed where they would result in infilling areas of lower drillhole density. Because the HQ drill was a skid-mounted rig, drillhole locations were restricted to areas accessible to such a rig with minimal road building. The 2006 drilling program on the Main Zone was designed to infill the starter pit area on approximate 30 m centres.

Sampling methods for drill core were similar for all of the exploration phases on the property. Core size varied (as discussed in earlier sections) and sampling of the core using a mechanical splitter was initially used by both SML and EML, with SML switching to a diamond saw after the first nine drillholes, and EM switching after approximately 30 drillholes. Splitting by diamond saw has been used ever since.

In 2008, large diameter (HQ) core drilling in the Main zone was carried out by Capstone to infill gaps in previous drilling, verify historical data, obtain metallurgical samples and collect detailed geotechnical data. Drillholes covered the entire deposit area, with specific drillhole locations placed where they would result in infilling areas of lower drillhole density. The drilling was helicopter supported, facilitating access to collar locations not possible during past programs. A total of 9,897.7 m was drilled in 81 holes (78 holes for the Main Zone including 3 holes totalling 69.2 m which were lost before intersecting the ore zone). In 2010, Capstone drilled the Esso zone to increase drillhole density and confirm extents of the zone. Metallurgical samples of half core (NQ; drill core diameter of 47.6 mm), assay samples of quarter core, bulk density measurements plus geotechnical and geological core logging were completed. Core in the zone was NQ sized; HQ core drilling was done on the upper portion of the holes to roughly 200 m deep for extra control in intercepting chosen targets at depth. Helicopter support was used for the drill program, again facilitating access to collar locations not possible in previous ground-support only programs. Overall, 34 holes totalling 18,042.1 m were drilled at the Esso zone, including 5 holes totalling 1,324.3 m which were abandoned above the zone when it became apparent the hole could not hit the appropriate target at depth.

In 2011, a drill program was conducted by Kutcho Copper testing nine high-priority VTEM targets and for definition drilling at the Sumac Deposit totalling 4,227 m of HQ size core in 19 holes (including 2 short abandoned holes). In addition to this exploration drilling, single vertical groundwater monitoring well holes were drilled at Main and Esso for a total of 645 m.

## 10.2 Drilling

The Main deposit is defined by 325 drillholes, the Esso deposit is defined by 98 holes and Sumac is defined by 22 holes (Table 10-1).

### Table 10-1: Drillhole Summary (Number of Holes)

| Company | Main | Main Wedged | Sumac | Sumac Wedged | Esso | Esso Wedged | Other | Other Wedged | Total |
|---|---|---|---|---|---|---|---|---|---|
| SML | 116 | 29 | 10 | 1 | - | - | 4 | - | 130 |
| EML | 57 | 7 | 2 | - | 49 | 33 | 17 | 8 | 125 |
| WKM | 70 | 23 | 4 | - | 15 | 4 | 1 | - | 90 |
| Kutcho Copper | 82 | 43 | 6 | - | 34 | - | 17[1] | - | 139 |
| **Total** | **325** | | **22** | | **98** | | **39** | | **463** |

Notes: [1] denotes RK holes which intersected mineralization, VTEM exploration holes, and two groundwater monitoring well holes.

Adit sampling not included in total.

Wedged holes from pilot holes are listed for information purposes.

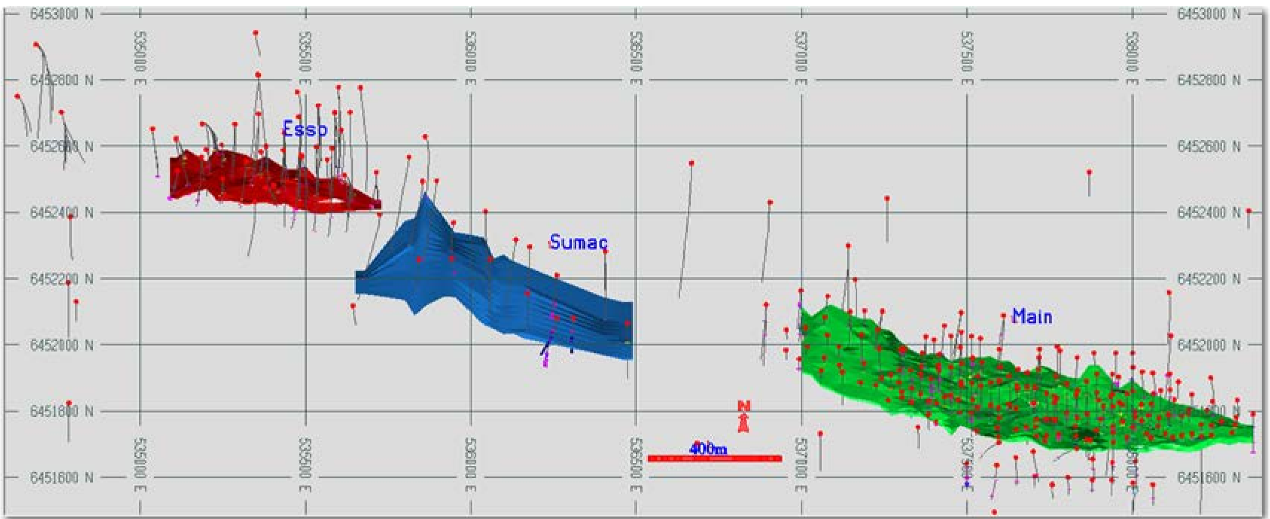
Source: JDS (2017).

Locations of drillholes that define the Esso, Sumac and Main deposits are displayed on a plan map, in Figure 10-1.

Drill collars and claim locations were surveyed periodically during exploration programs by McElhanney Engineering Services Limited (MESL) until 1983; all later WKM drillholes and many of the historical drillholes were surveyed or resurveyed by MESL in September 2006. Previous surveys were surveyed in mine-grid coordinates, whereas the recent survey was carried out in UTM coordinates, and survey points that were not resurveyed were converted to UTM grid points by MESL.

Most of the drill sites have been reclaimed, although many collars are still locatable and most of the drill core is stored on the property. A new core storage area was constructed for the Esso deposit drill core in 1985, and all of it was moved there over the following six years. The Sumac deposit core was stored on core racks in the area between the Main and Sumac deposits. Due to decomposition of these racks, the core was removed and cross-stacked nearby, and the core racks were dismantled. Approximately half of both Sumac and Esso deposits drill core was relogged in 1984 and 1985 (Holbek and Heberlein 1986) using the GEOLOG system. WKM has recovered all available data (in both digital and paper formats) and merged it with current data into two master databases; one for resource work and the other being the total exploration database.

**Figure 10-1: Esso, Sumac and Main Deposits with Drillhole Collars and Traces**




Source: JDS (2017).

Initial drilling in the Main deposit was carried out on 120 m spaced sections with drillholes spaced approximately 60 m along section lines. This spacing was subsequently reduced to approximately 30 m spaced drill intersections along 60 m spaced sections. WKM reduced the drill spacing in selected areas to 30 m or less. Historical drillhole diameters are mostly drill core diameter of 38 mm (BQ) and recoveries were generally very good with only rare core loss in minor fault zones. The more recent drilling has been a combination of HQ and NQ in the Main deposit and NQ (or BQTW in wedge branches) within the Sumac and Esso deposits. Most holes were drilled at -45 to -60° in order to intersect mineralization at close to 90°. Due to strong foliation dipping to the north, even vertical holes tend to flatten and cut the mineralization roughly perpendicular to its dip.

The historical drillhole database for the Main deposit contained assays for copper, zinc, silver, and SG, with most holes containing assays for gold and approximately 60% containing assays for sulphur. Historical drill data for the Sumac and Esso deposits contain results for copper, zinc, silver, gold, and specific gravity. There are 4,569 assay intervals within the total resource database, of which 1,589 are WKM. Of the remaining 2,978 historical assay intervals, 2,061 are in the Main deposit, 443 from the Esso side of the deposit, and 1,618 from the Sumac side. The assay intervals were generally longer at Esso, while the Sumac data commonly contained shorter intervals based on sulphide mineralogy.

The 2008 drill program was designed to infill the Main deposit resource area, principally to increase confidence in the resource classification, better define higher grade trends and provide sufficient sample material to conduct more extensive metallurgical sampling in support of a mine plan. Additional data such as measured bulk densities, fracture density, and other geotechnical data were also recorded to aid in this evaluation. The overall aim of producing a more robust geological, geotechnical and resource model that could support a more robust economic assessment was achieved.

In addition to 1/4 cylinder core samples taken for assay, a further 1/2 cylinder sample was cut and sealed in nitrogen-filled bags and stored in nitrogen filled pails for stable storage in an oxygen deprived environment for later metallurgical testing. Three metallurgical tests of this large sample set have been completed and testing continues. Another aim of the 2008 program was to increase the overall pierce-point density to a nominal 30 m x 30 m grid in the parts of the deposit that have a reasonable expectation of economic extraction based upon previous mine plans and assuming a positive feasibility study.

Most new exploration holes were drilled on an azimuth of 180º or as close as possible and range in length from 59 to 207 m, with inclinations ranging from -90º to -45º but averaging between -60º to -45º. This typical inclination ensures that most mineralized intercepts are, at or as near to, perpendicular to the enveloping hangingwall and footwall surfaces as possible and therefore the mineralized intercept can be expected to be, at or close to, true width.

The 2010 drill program was designed to infill the Esso deposit resource area and test its perimeter, principally to increase confidence in the resource classification, better define higher grade trends, and provide sufficient sample material to conduct more extensive metallurgical sampling in support of a mine plan. Additional data such as measured bulk densities, fracture density, point-load tests and other geotechnical data were also recorded to aid in this evaluation. The result produced a more robust geological, geotechnical and resource model that could support a more robust economic assessment.

As in 2008, 1/4 cylinder core samples taken for assay, a further 1/2 cylinder sample was cut and sealed in nitrogen-filled bags and stored in nitrogen filled pails for stable storage in an oxygen deprived environment for metallurgical testing. All core from this large sample set have been submitted for metallurgical testing at the laboratory at the Cozamin Mine, Mexico.

Another aim of the 2010 program was to increase the overall pierce-point density to a nominal 50 m by 50 m grid in the parts of the deposit that have a reasonable expectation of economic extraction based upon previous mine plans, and assuming a positive feasibility study.

Most holes were drilled on an azimuth of 180º, and range in length from 496 to 678 m, with inclinations ranging from -90º to -70º and averaging -77º. This inclination ensured that most mineralized intercepts are at or near to perpendicular to the enveloping hangingwall and footwall surfaces, and therefore the mineralized intercept could be expected to be at or close to true width.

The program resulted in a more robust geological model and mineral resource estimate. Infilling the undrilled gaps that were evident in previous resource models provided crucial data to support future mine plans.

The 2011 drill program was designed to infill the Sumac deposit resource area. In addition, the program tested nine high-priority VTEM targets along with single vertical groundwater monitoring well holes drilled at Main and Esso.

The Sumac holes were drilled on an azimuth of 180º, with inclinations ranging from -50º to -70º. This inclination ensured that most mineralized intercepts are at or near to perpendicular to the enveloping hangingwall and footwall surfaces, and therefore the mineralized intercept could be expected to be at or close to true width.

The program resulted in a more robust geological model and mineral resource estimate at Sumac. The exploration holes did not result in any significant new discoveries. The groundwater monitoring holes provided crucial data to support future potential mining activities.

# 11 Sample Preparation, Analyses and Security

The sampling intervals ideally ranged from 1.0 to 1.5 m in mineralized material and may be up to 3.0 m in length where waste intervals between mineralized zones occur. Two shoulder samples were taken in waste at both upper and lower contacts, consisting of a 1.5 m sample and a 1.0 m sample. Samples did not cross geological contacts.

The samples were tagged and then split in half using a rock saw on-site. Half of the core was selected for metallurgical testing. The remaining half core was cut into two quarters. One quarter cut of the core was placed into plastic sample bags and heat sealed. Sample bags, typically 6 to 10, were then packaged into rice bags with security zip seals. The sample submittal was dispatched from the site via air charter to Dease Lake, BC then by truck to the laboratory/sample preparation facility. The remaining quarter core was returned to the original boxes and remains on-site as a record of the hole.

Prior to submission for assay, the specific gravity of each quarter cut sample was measured. In non-mineralized, un-sampled intervals, SG was measured approximately every 10 m. Specific gravities were measured in the field by weighing the sample using an Ohaus scale in air and in water, with the SG calculated using the formula:

$$SG = \frac{\text{Weight in Air}}{(\text{Weight in Air - Weight in Water})}$$

Note: sample weights were accurate to the nearest tenth of a gram.

The core was photographed after the sample tags were stapled to the boxes at the down hole end of each sample. Sample tags for standards, blanks and duplicates were stapled to the box in the numerical order as inserted.

All Kutcho samples were assayed at ALS Chemex (ALS) in North Vancouver, BC. Samples were prepared at ALS in North Vancouver in 2008, then in the ALS sample processing facility in Terrace, thereafter. ALS attained ISO 17025 accreditation by Standards Council of Canada for test procedures including fire assay Au by AA, induced coupled plasma (ICP), and gravimetric finish, multi-element ICP and AA Assays for Ag, Cu, Pb, and Zn. In addition, ALS is ISO 9001:2008 registered.

Core samples including blanks were ground to more than 85% passing -75 µm. Analysis of core samples and standard reference materials included ICP methods following an aqua regia digestion. If either copper or zinc reported over 2,500 parts per million (ppm) (0.25%), ore grade analysis was conducted for copper, zinc and silver. The ore grade analysis included aqua regia digestion followed by atomic absorption spectroscopy.

A comprehensive quality assurance/quality control (QA/QC) program was used for the 2008 to 2011 drilling programs, summarized below in the Tables 11-1 and 11-2. Two standards were inserted for every 20 samples, alternating between a blank and a certified standard reference material (CRM) with placement and type at the discretion of the logging geologist. A minimum of one blank and CRM per drillhole was added. Within every 20 samples taken, a laboratory duplicate sample was added to the sample submittal.

The laboratory duplicate alternated between a pulp duplicate sample and a coarse reject duplicate sample as selected by the core logger. Comparison of the original assay values and the duplicates was the final component of the QA/QC program.

Field duplicates, the other half of the core sample, were not taken to preserve material for future studies.

The blank sample material, approximately 100 to 500 g of crushed limestone or felsic rock was purchased from a landscaping supply company.

A variety of CRM across the typical grade ranges, as summarized in the table below, were used to monitor accuracy of results. The grade ranges reflect the span of grades through the deposit and parallel values which may be significant to future operational plans such as head grade targets or cut-off grade determinations.

### Table 11-1: Reference Standards Used in Quality Assurance/Quality Control Programs

| Standard | Mean Grade of Certified Elements ± 2 Standard Deviations | | | | |
|---|---|---|---|---|---|
| | Cu (%) | Zn (%) | Ag (g/t) | Au (g/t) | Pb (%) |
| CDN-FCM-3[1] | 0.29 +/-0.02 | 0.54 +/-0.03 | 23.6 +/-3.3 | 0.40 +/-0.07 | 0.15 +/-0.01 |
| CDN-ME-2[2] | 0.48 +/-0.018 | 1.35 +/-0.10 | 14.0 +/-1.3 | 2.10 +/- 0.11 | - |
| CDN-ME-6[2] | 0.61 +/-0.034 | 0.517 +/-0.04 | 101 +/-7.1 | 0.27 +/- 0.028 | 1.02 +/- 0.08 |
| CDN-ME-11[3] | 2.44 +/- 0.11 | 0.96 +/-0.06 | 79.3 +/-6.0 | 1.38 +/- 0.10 | 0.86 +/- 0.10 |
| CDN-ME-18[3] | 1.93 +/- 0.09 | 4.60 +/-0.22 | 58.2 +/-5.1 | 0.512 +/- 0.070 | 0.098 +/- 0.012 |
| CDN-GS-P3A | - | - | - | 0.338 +/- 0.022 | - |
| CDN-HZ-3[2] | 0.61 +/-0.03 | 3.16 +/-0.16 | 27.3 +/-3.2 | provisional 0.055 +/- 0.010 | 0.707 +/-0.036 |
| CDN-HZ-2 | 1.36 +/-0.06 | 7.2 +/-0.35 | 61.1 +/-4.1 | 0.124 +/-0.024 | 1.62 +/-0.11 |
| CDN-HC-2 | 4.63 +/-0.26 | 0.259 +/-0.014 | 15.3 +/-1.4 | 1.67 +/- 0.12 | 0.48 +/-0.04 |
| CDN-HLHC | 5.07 +/-0.27 | 2.35 +/-0.11 | 111.0 +/-8.6 | 1.97 +/-0.22 | 0.17 +/-0.01 |
| CDN-HLLC | 1.49 +/-0.06 | 3.01 +/-0.17 | 65.1 +/-6.7 | 0.83 +/-0.12 | 0.29 +/-0.03 |

Notes: CDN Resource Labs, Delta BC. HL CRM are High Lake VMS deposit material. FCM are Campo Morado VMS material. HC and HZ CRM are derived from VMS deposit material. ME are Lookout, Niblack VMS material. GS-P3A is made from Bald Mountain, NV material (Carlin style mineralization).

(1) 2008 use only.

(2) Not used in 2008.

(3) 2011 use only.

Behaviour of the blanks and CRM was monitored as data returns from the lab. If more than two control samples in a work order returned unacceptable values, the entire work order was rerun. If two or fewer control samples fail, the 20 samples in the analytical batch are rerun. In the case of a blank fail, samples and the control were reprocessed from coarse reject material. CRM values outside of three standard deviations are considered failures, as are more than two consecutive values between two and three standard deviations above or below the mean.

**Table 11-2: 2008, 2010, and 2011 Kutcho Project Quality Control Data**

<table>
<tr><th colspan="2" rowspan="2"></th><th colspan="6">Year</th></tr>
<tr><th colspan="2">2008</th><th colspan="2">2010</th><th colspan="2">2011</th></tr>
<tr><td colspan="2">Total Samples Collected</td><td colspan="2">2,973</td><td colspan="2">1,171</td><td colspan="2">379</td></tr>
<tr><td colspan="2" rowspan="7">CRM Used</td><td>HLLC</td><td>34</td><td>HLLC</td><td>4</td><td>HLLC</td><td>8</td></tr>
<tr><td>HLHC</td><td>22</td><td>HLHC</td><td>7</td><td>HLHC</td><td>2</td></tr>
<tr><td>HC-2</td><td>26</td><td>HC-2</td><td>10</td><td>P3A</td><td>1</td></tr>
<tr><td>HZ-2</td><td>28</td><td>HZ-2</td><td>18</td><td>ME-18</td><td>1</td></tr>
<tr><td rowspan="3">FMC-3</td><td rowspan="3">37</td><td>ME-2</td><td>19</td><td>ME-11</td><td>2</td></tr>
<tr><td>ME-6</td><td>13</td><td>ME-6</td><td>1</td></tr>
<tr><td>HZ-3</td><td>1</td><td>HZ-3</td><td>2</td></tr>
<tr><td colspan="2">Total CRM</td><td colspan="2">147</td><td colspan="2">72</td><td colspan="2">17</td></tr>
<tr><td colspan="2">Blanks</td><td colspan="2">77</td><td colspan="2">60</td><td colspan="2">15</td></tr>
<tr><td rowspan="2">Paired Data</td><td>Coarse Reject Duplicate</td><td colspan="2">72</td><td colspan="2">44</td><td colspan="2">14</td></tr>
<tr><td>Pulp Reject Replicate</td><td colspan="2">72</td><td colspan="2">47</td><td colspan="2">18</td></tr>
<tr><td colspan="2">Total QC samples</td><td colspan="2">368</td><td colspan="2">223</td><td colspan="2">64</td></tr>
<tr><td colspan="2">Frequency (percent)</td><td colspan="2">12</td><td colspan="2">19</td><td colspan="2">17</td></tr>
<tr><td colspan="2">Umpire checks (percent)</td><td colspan="2">2</td><td colspan="2">3</td><td colspan="2">0</td></tr>
</table>

Source: JDS (2017).

The laboratory duplicate samples were used to monitor variability in the sample material through the preparation process. The sample tag is stapled to the core box and marked with a C indicating a coarse reject duplicate or a P indicating a pulp duplicate. The core cutter placed the tags for the parent sample and the duplicate sample with the sawn material, labelling the bag with both sample numbers and the duplicate letter code.

For increased confidence in the dataset, a comparison of analyses between labs was conducted periodically. At the end of the drill programs in 2008 and 2010, 2 to 3% of the sample pulp rejects at ALS were selected across a variety of grade ranges representative of the location and targets tested at the Project. The pulp rejects were submitted to IPL Inspectorate in Vancouver, BC, with one CRM added to every 10 samples. All the samples were renumbered with new, sequential sample identifications. Identical analytical methods were used. Results were shown to be reproducible.

## 11.1 2008 Drill Program

In 2008, the mineralized intervals in core were sampled in lengths ranging from 30 cm to 1.5 m, averaging 1 to 1.5 m. The sampling intervals are typically 1.5 m in mineralized material and may be as long as 3 m where waste intervals between mineralized zones occur. Two shoulder samples were taken in waste at both upper and lower contacts, consisting of a 1.5 m sample and a 1 m sample. Samples do not cross geological contacts.

Sample quality was very good. Recovery of every drill run was recorded and typically exceeded 95% in mineralized zone. The samples were taken across the entire mineralized interval, with sample boundaries respecting lithological contacts, VMS cycles and mineralization styles, including varying percentages of sulphide minerals. Higher grade intervals within low grade intersections were identified visually during core logging; the variations in mineralization were also respected during the sampling process. Two shoulder samples were added at both the up hole and down hole contacts to ensure the mineralization was fully sampled.

Samples taken from holes drilled at 180º azimuth and -45º inclination are representative of true width in the Kutcho deposits.

In the author's opinion, the sampling, sample preparation, quality of security, and analytical procedures used are adequate and suit the Kutcho deposits type and mineralization.

Performance of the copper blanks was acceptable. Two failures were found to be caused by sample data entry errors by the core logger and three failures were found to be caused by contamination from mineralized samples during processing at the lab. This was corrected and rectified by: 1) the core loggers were reminded to take care in recording sample types, 2) advising the lab of the contamination issue during the program and 3) re-communicating standard operating procedures to personnel at the lab.

Blank performance overall was acceptable for gold with very little suggestion of contamination from inadequate cleaning between samples during the crushing and pulverization stages. One failure was re-assayed from coarse reject material and returned marginal fail, suggesting the failure was due to inadequate cleaning between samples during the coarse crush. The lab was advised of the two failures and subsequently reminded staff of protocols for cleaning between samples. The blank had also failed for copper; the copper reanalysis from coarse reject was acceptable. A marginal failure after a high grade interval was not re-assayed.

Blank performance for silver was acceptable, with very little between-sample contamination evident.

Performance of the CRMs across all grade ranges was acceptable; the CRMs did show periods of slight bias both above and below the mean for was within acceptable standards.

Reproducibility of copper and silver results in duplicate samples prepared from coarse reject materials was excellent. Reproducibility of gold results in duplicate samples prepared from coarse reject materials was excellent up to 1.5 g/t Au; higher results may not be as reproducible. This characteristic is likely due to inherent properties of the Kutcho deposits. The nugget nature of gold makes it difficult to reproduce results at the higher grade ranges.

Reproducibility of copper and silver results in duplicate samples prepared from pulp reject material was very good.

Reproducibility of gold results in pulp duplicates was acceptable. The reproducibility is excellent below 0.6 g/t Au and fair in higher grade ranges. The sample preparation is sufficiently homogenizing pulp material.

At the end of the drill program in 2008, 2% of the sample pulp rejects at ALS were selected across a variety of grade ranges representative of the location and targets tested at the Project. The pulp rejects were submitted to IPL Inspectorate in Vancouver, BC, with one CRM added to every 10 samples. All the samples were renumbered with new, sequential sample identifications. Identical analytical methods were used. The target for pulp samples analyzed at different labs should have a relative difference not exceeding 15% at the 90$^{th}$ percentile. For check samples in 2008, copper results had a relative difference of 12% for the 90% of the population, gold results had a relative difference of 15% for 71% of the population and zinc had a relative difference of 15% for 79% of the population. The level of precision is excellent for copper and acceptable for gold and zinc. Results were shown to be reproducible.

For additional information regarding performance of quality control samples in 2008 please refer to *Preliminary Economic Assessment Revised Mining Option Kutcho Project British Columbia*, JDS Energy & Mining Inc., July 2010.

## 11.2 2010 Drill Program

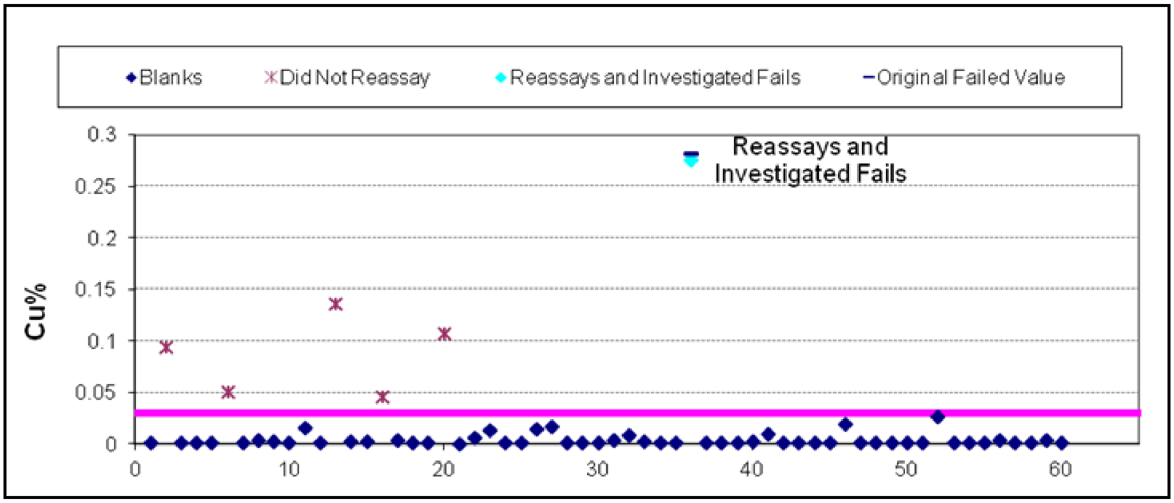
In 2010, mineralized intervals in core were sampled in lengths ranging from 20 cm to 1.5 m, averaging 1 to 1.5 m. The sampling intervals are typically 1.5 m in mineralized material and may be as long as 3 m where waste intervals between mineralized zones occur. Two shoulder samples were taken in waste at both upper and lower contacts, consisting of a 1.5 m sample and a 1.0 m sample. Samples do not cross geological contacts.

Sample quality was very good. Recovery of every drill run was recorded and typically exceeded 90 to 95% in the mineralized zone. The samples were taken across the entire mineralized interval, with sample boundaries respecting lithological contacts, VMS cycles and mineralization styles, including varying percentages of sulphide minerals. Higher grade intervals within low grade intersections were identified visually during core logging; the variations in mineralization were also respected during the sampling process. Two shoulder samples were added at both the up hole and downhole contacts to ensure the mineralization was fully sampled.

Samples taken from holes drilled at 180º azimuth and -45º inclination are representative of true width in the Esso deposit.

Blank performance for copper was excellent (Figure 11-1). Five values exceeding the blank warning performance gate were not reanalyzed; the results were attributed to minor contamination after very high grade samples exceeding 4% copper. One failure was reassayed and returned a similar value, indicating the contamination occurred at the coarse crush stage of sample preparation after an 8% copper sample. These instances of contamination are not considered systematic, however, as there were many other very high samples submitted with well-performing blanks. The lab was notified of the instances and consequently reminded workers to take care during sample preparation, following all standard operating procedures.

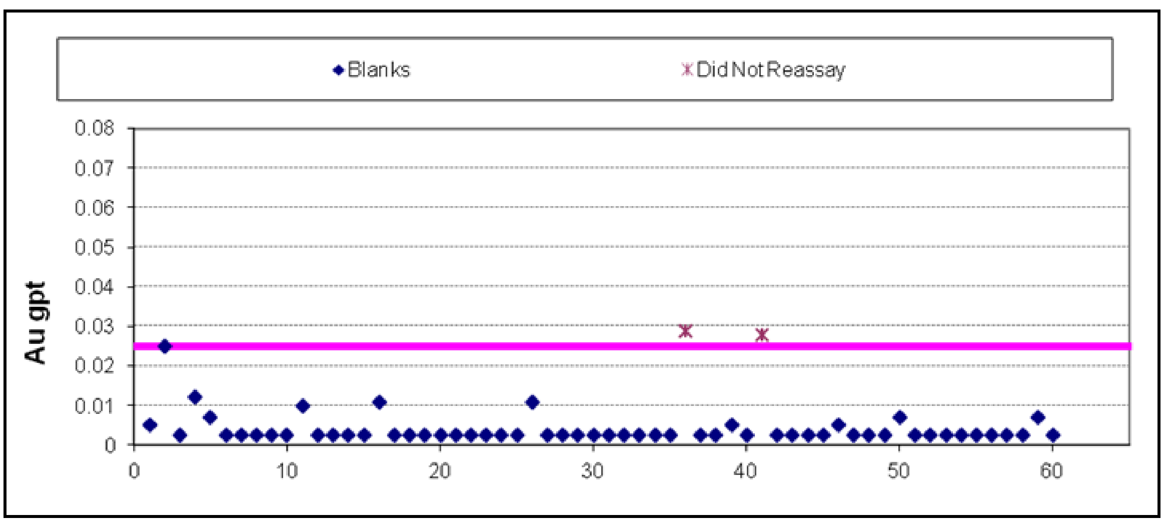
**Figure 11-1: Copper Blank Performance**




Source: JDS (2011).

Blank performance for gold shown in Figure 11-2 was excellent. Two values marginally exceeded the blank warning performance gate and were not reassayed. Note that one of the two marginal fails is the same sample reassayed for a copper blank failure.

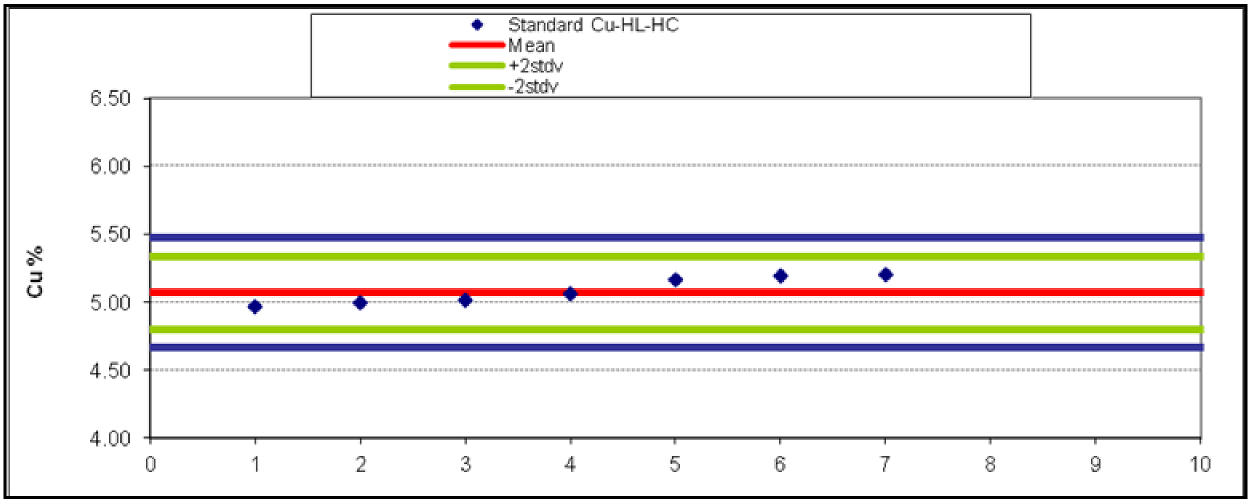
**Figure 11-2: Gold Blank Performance**




Source: JDS (2011).

Performance of CRM HL-HC (Figure 11-3) was excellent for copper. Periods of slight bias both above and below the mean exist for copper but are within acceptable limits.

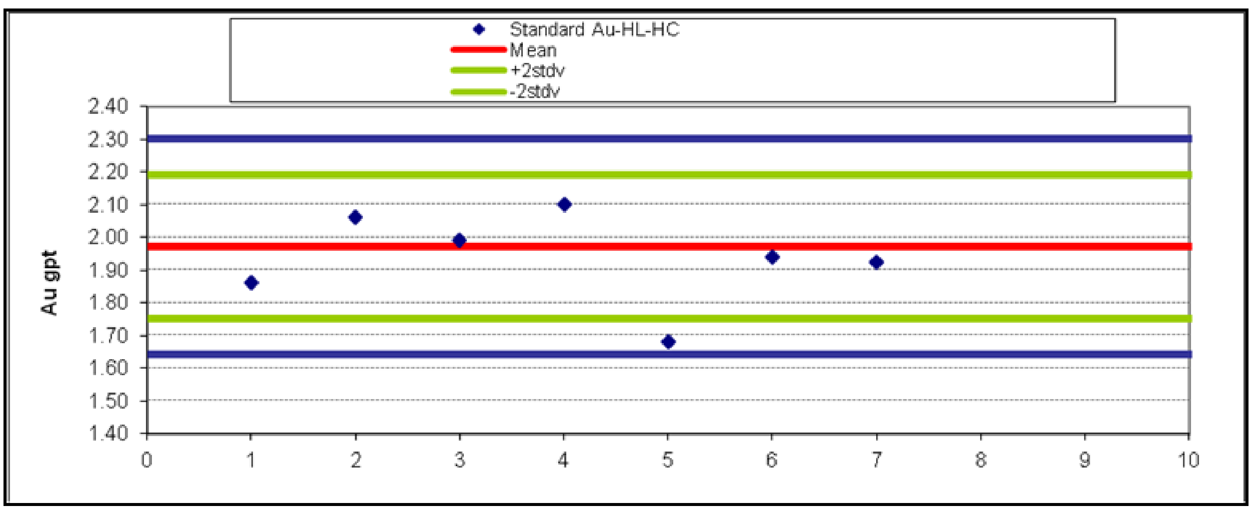
**Figure 11-3: Copper CRM HL-HC Performance**




Source: JDS (2011).

Performance of CRM HL-HC shown in Figure 11-4 was excellent for gold; values are distributed about the mean.

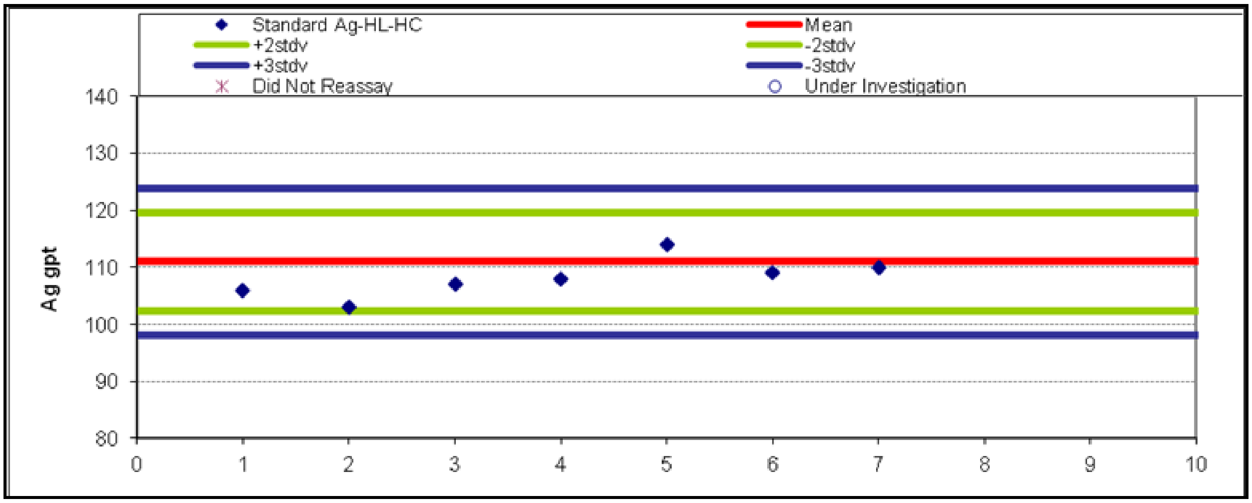
**Figure 11-4: Gold CRM HL-HC Performance**




Source: JDS (2011).

Performance of CRM HL-HC (Figure 11-5) was excellent for silver. Silver values are biased slightly below the mean, within acceptable limits.

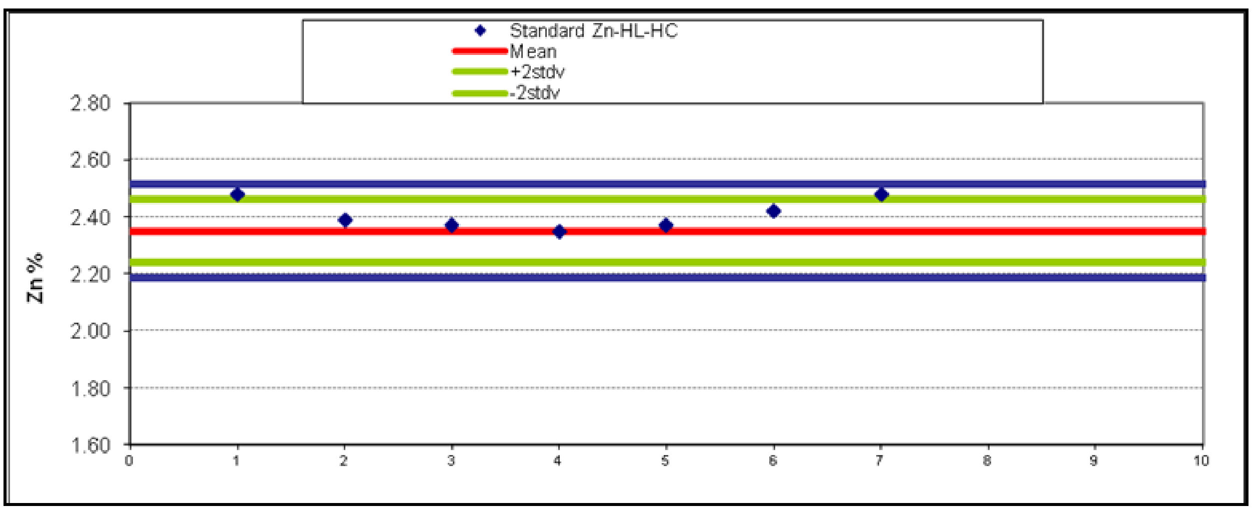
**Figure 11-5: Silver CRM HL-HC Performance**




Source: JDS (2011).

Performance of CRM HL-HC (Figure 11-6) was excellent for zinc. Zinc values are biased slightly above the mean, within acceptable limits.

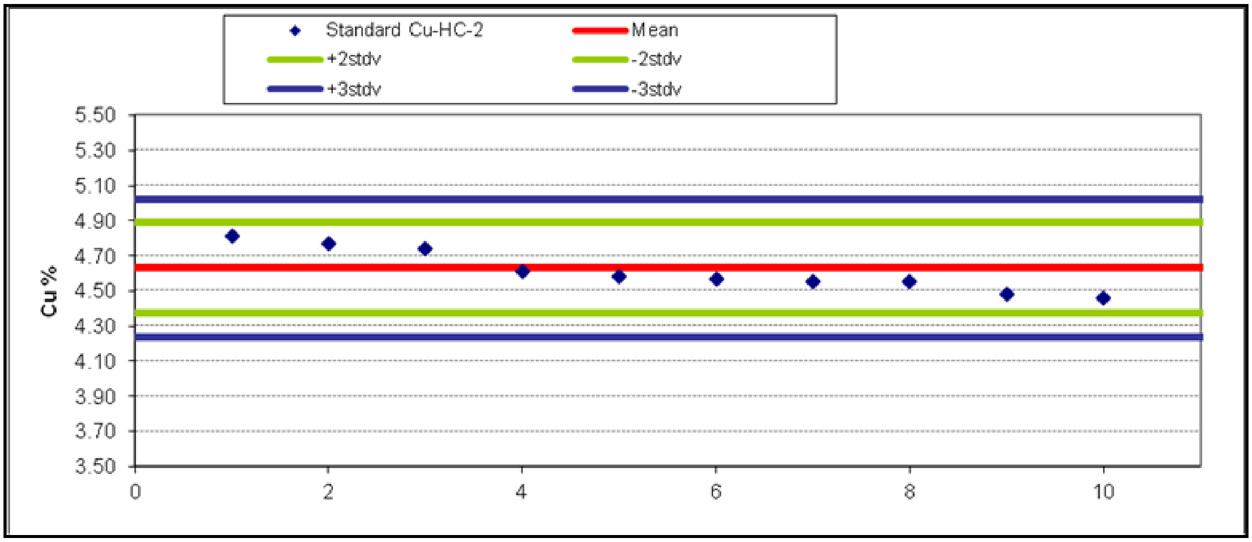
**Figure 11-6: Zinc CRM HL-HC Performance**




Source: JDS (2011).

Performance of CRM HC-2 (Figure 11-7) was excellent for copper. Copper values exhibited a slight bias below the mean through most of the drill campaign; however the bias was within acceptable limits.

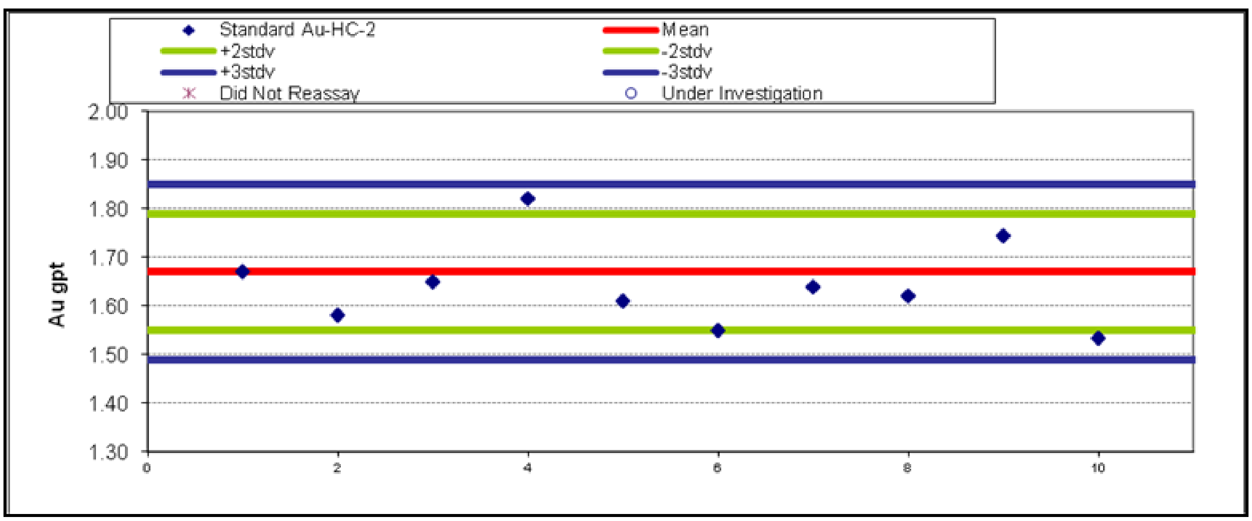
### Figure 11-7: Copper CRM HC-2 Performance




Source: JDS (2011).

Performance of CRM HC-2 (Figure 11-8) was excellent for gold. Gold values exhibited a slight bias below the mean through most of the drill campaign; however the bias was within acceptable limits.

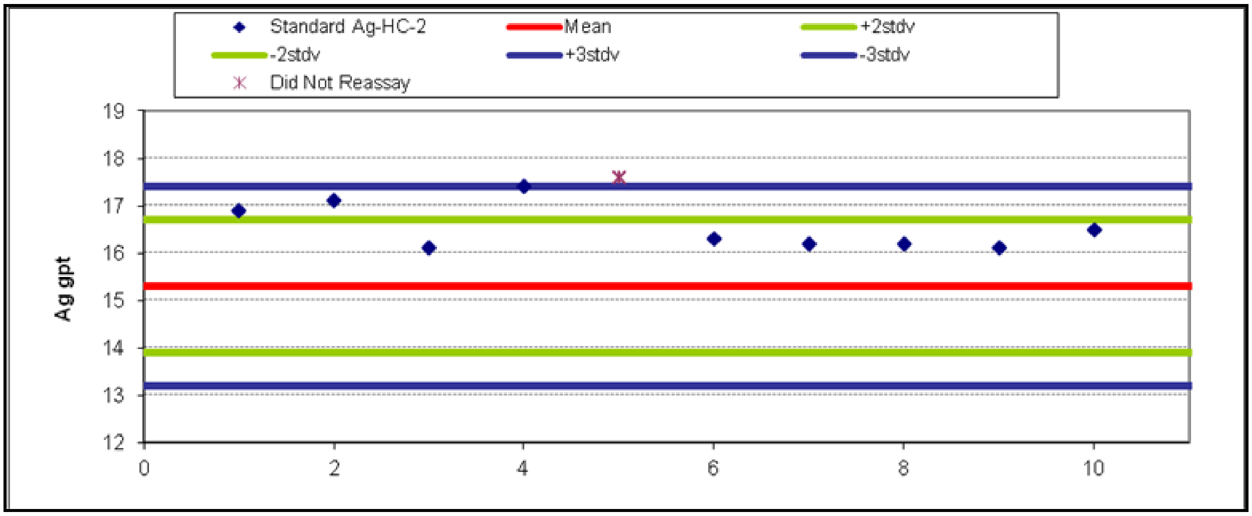
### Figure 11-8: Gold CRM HC-2 Performance




Source: JDS (2011).

Performance of CRM HC-2 (Figure 11-9) was acceptable for silver. One marginal fail just outside the three standard deviations above the mean was not reanalyzed. The CRM exhibited consistent bias above the mean within acceptable limits.

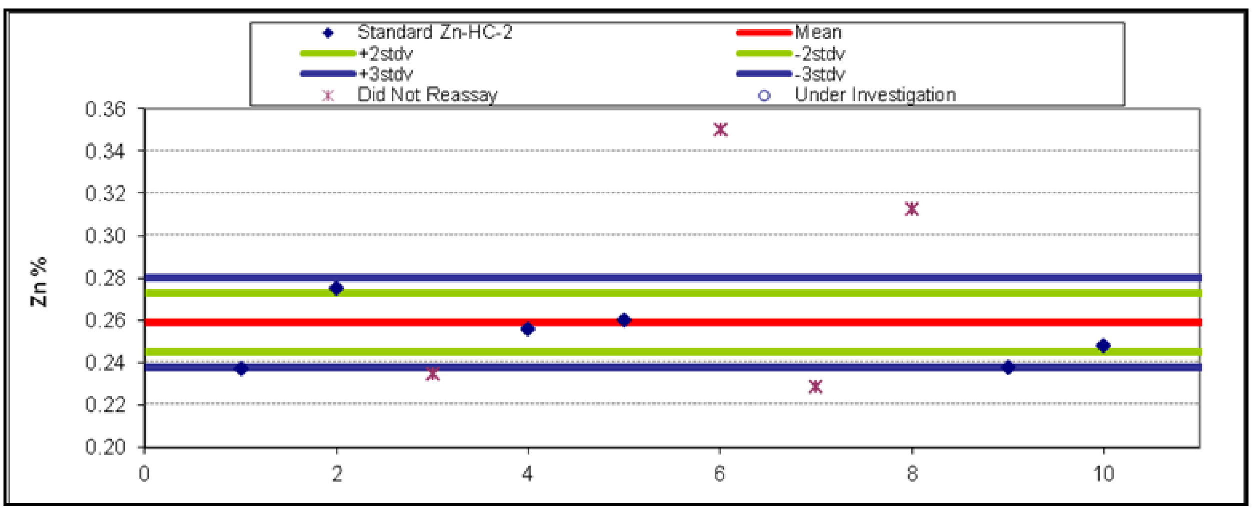
**Figure 11-9: Silver CRM HC-2 Performance**




Source: JDS (2011).

Performance of CRM HC-2 (Figure 11-10) for zinc was acceptable overall. Four failures outside of three standard deviations from the mean occurred, two above and two below. Two marginal fails just at the lower performance gate were not reassayed. Two remaining failures were not reassayed as the analytical batch contained an additional CRM better suited the high grade zinc values of the core samples. Values are well distributed about the mean; no bias is apparent.

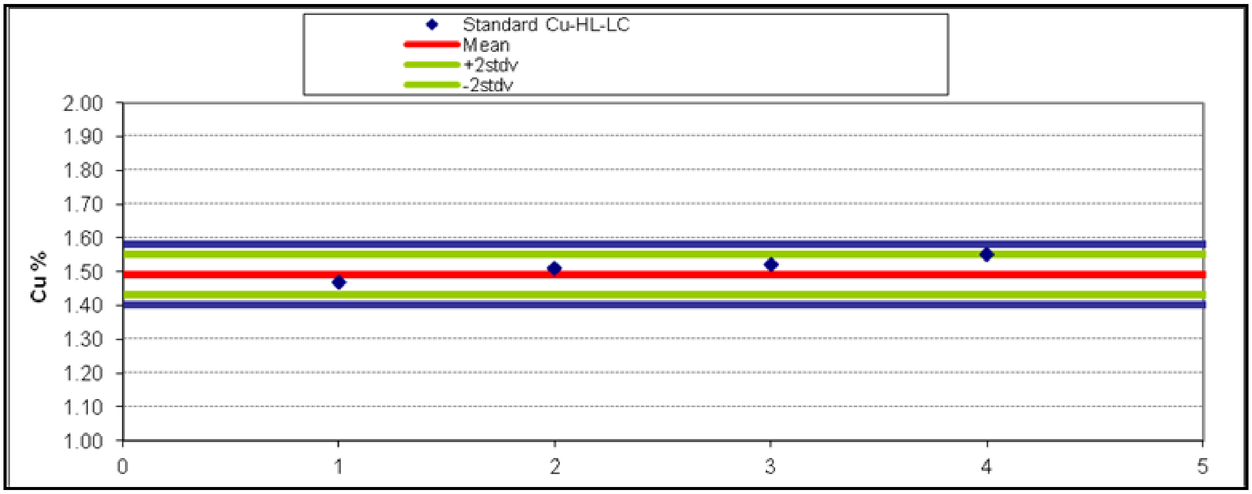
**Figure 11-10: Zinc CRM HC-2 Performance**




Source: JDS (2011).

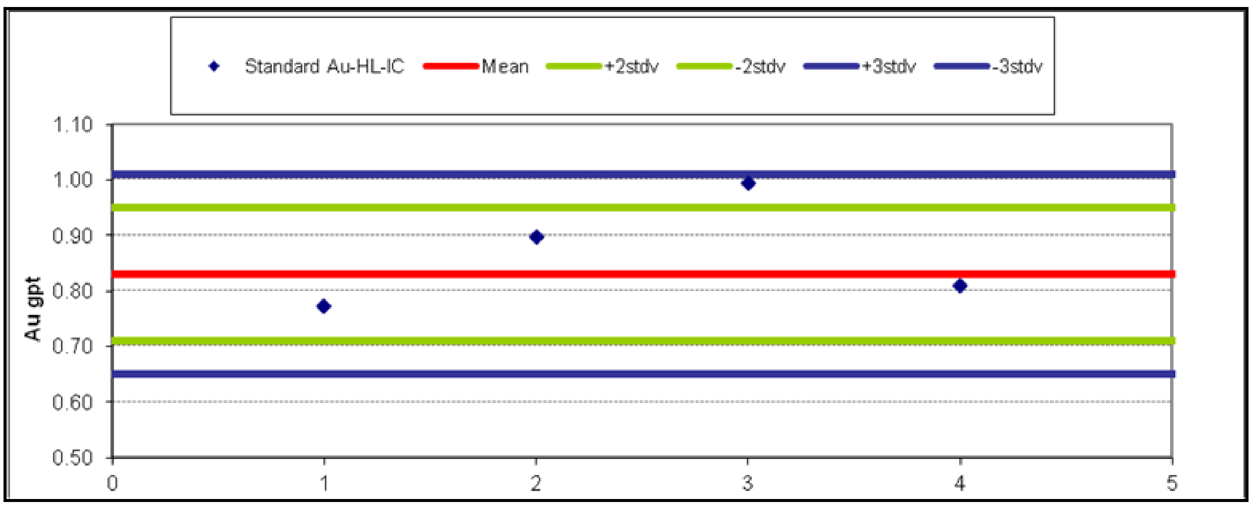
Performance of CRM HL-LC was excellent for copper, gold, silver and zinc as shown in Figures 11-11 to 11-14. Values for copper, gold and zinc are distributed about the mean. Bias above the mean, within acceptable limits, is shown in the silver chart.

**Figure 11-11: Copper CRM HL-LC Performance**



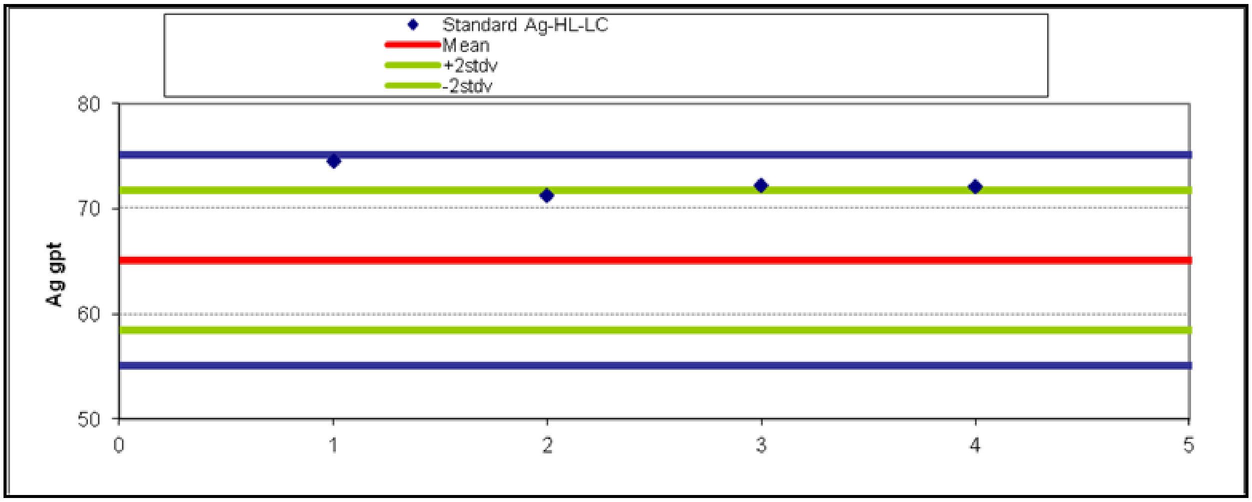

Source: JDS (2011).

**Figure 11-12: Gold CRM HL-LC Performance**



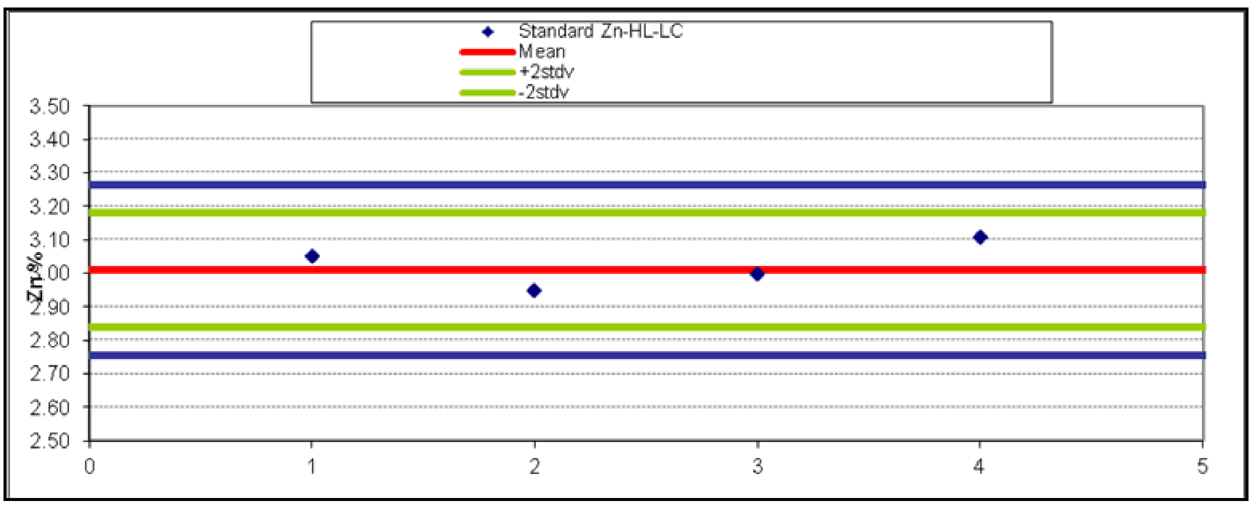

Source: JDS (2011).

**Figure 11-13: Silver CRM HL-LC Performance**




Source: JDS (2011).

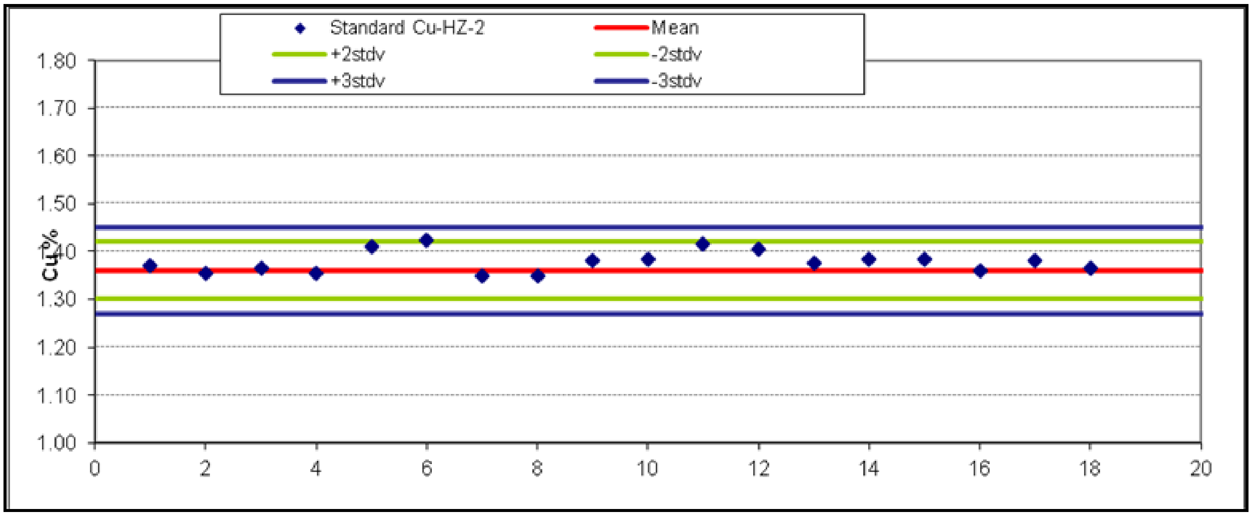
**Figure 11-14: Zinc CRM HL-LC Performance**




Source: JDS (2011).

Performance of CRM HZ-2 (Figure 11-15) was excellent for copper. Weak bias above the mean, within acceptable limits, was exhibited through most of the drill program.

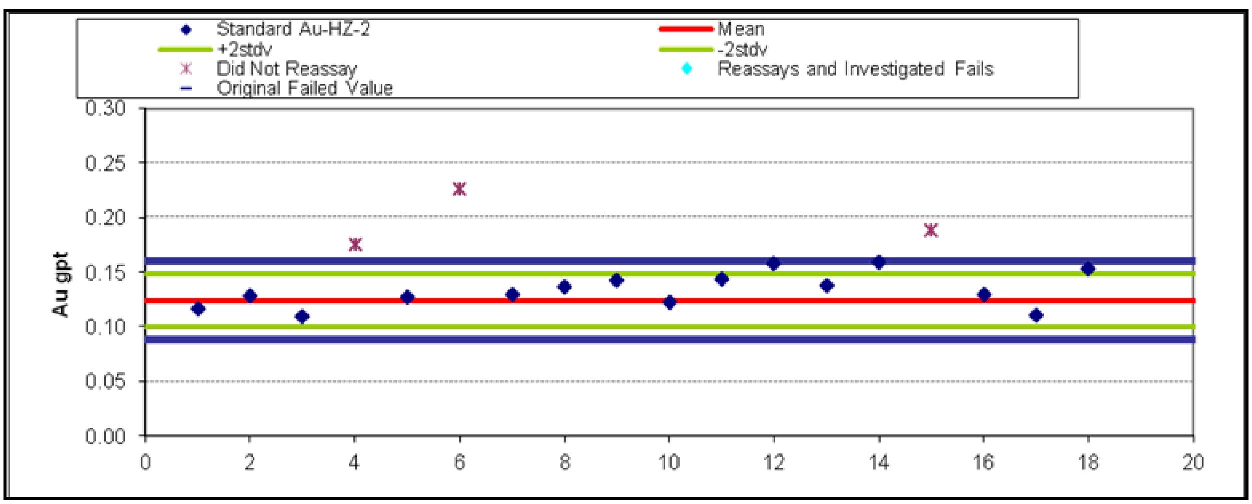
### Figure 11-15: Copper CRM HZ-2 Performance




Source: JDS (2011).

Performance of CRM HZ-2 (Figure 11-16) was very good for gold. Three marginal failures above the mean were not reassayed. Weak bias above the mean, within acceptable limits, was seen throughout the drill program.

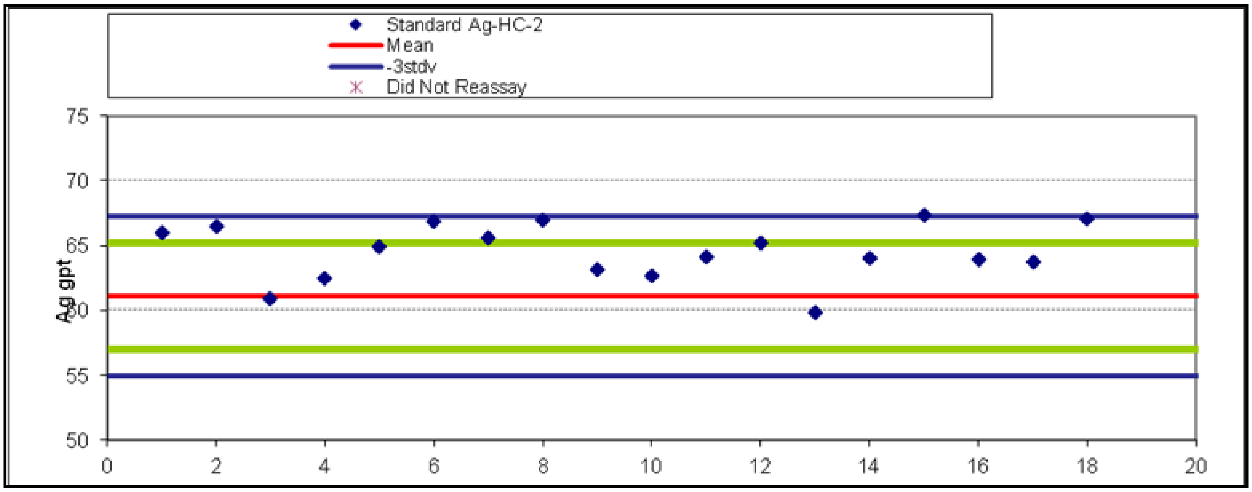
### Figure 11-16: Gold CRM HZ-2 Performance




Source: JDS (2011).

Performance of CRM HZ-2 (Figure 11-17) was excellent for silver, although consistent weak bias above the mean, within acceptable limits, was seen throughout the drill program.

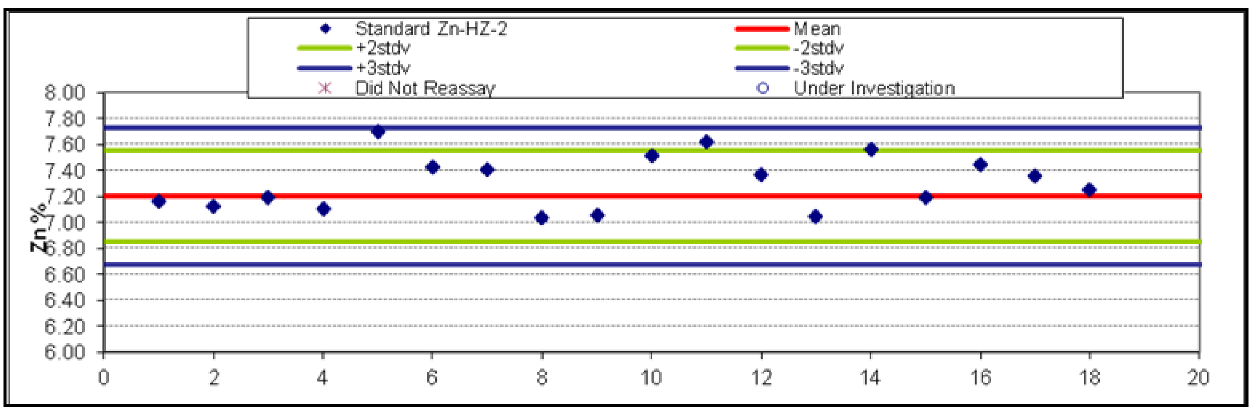
**Figure 11-17: Silver CRM HZ-2 Performance**




Source: JDS (2011).

Performance of CRM HZ-2 (Figure 11-18) was nearly perfect for zinc. Sporadic weak bias above the mean, within acceptable limits, is evident on the chart.

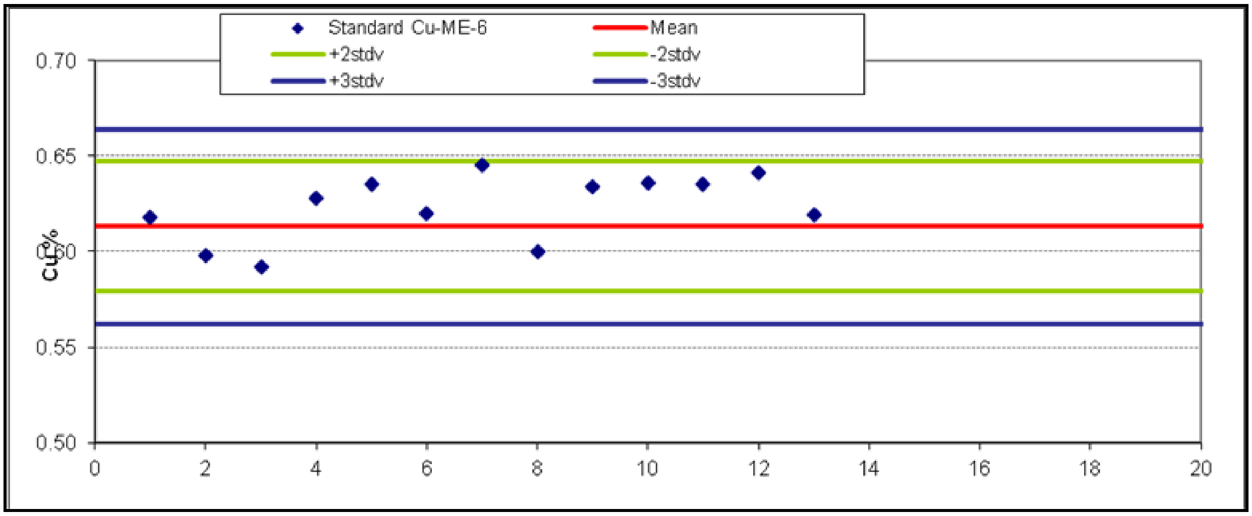
**Figure 11-18: Zinc CRM HZ-2 Performance**




Source: JDS (2011).

Performance of CRM ME-6 (Figure 11-19) is excellent for copper, with the exception of weak (within acceptable limits), above-mean bias for the majority of the drill program.

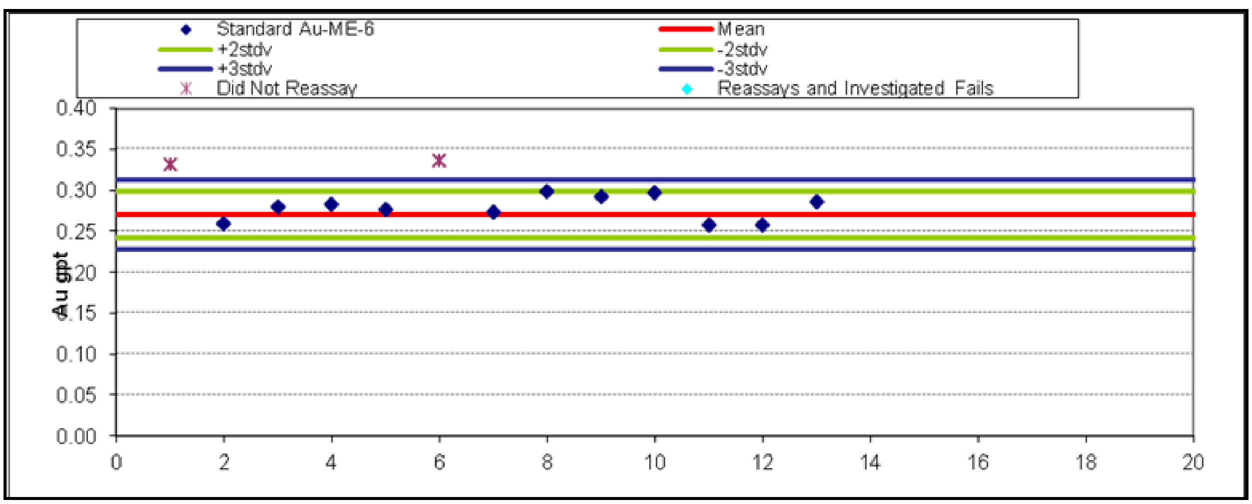
**Figure 11-19: Copper CRM ME-6 Performance**




Source: JDS (2011).

Performance of CRM ME-6 (Figure 11-20) for gold was very good. Two marginal fails close to the three standard deviations above the mean were not reanalyzed. In both cases, the batches containing the inserted CRM were outside of the main Esso zone and were not integrated into the resource calculation.

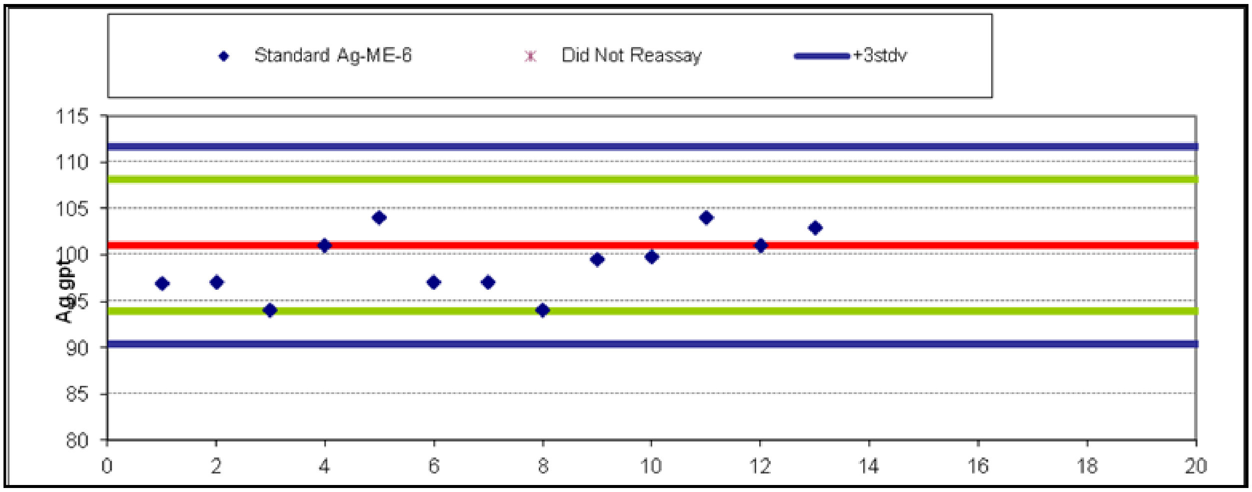
**Figure 11-20: Gold CRM ME-6 Performance**




Source: JDS (2011).

Performance of the CRM ME-6 (Figure 11-21) for silver was excellent. Weak bias below the mean, within acceptable limits, was seen throughout the drill program.

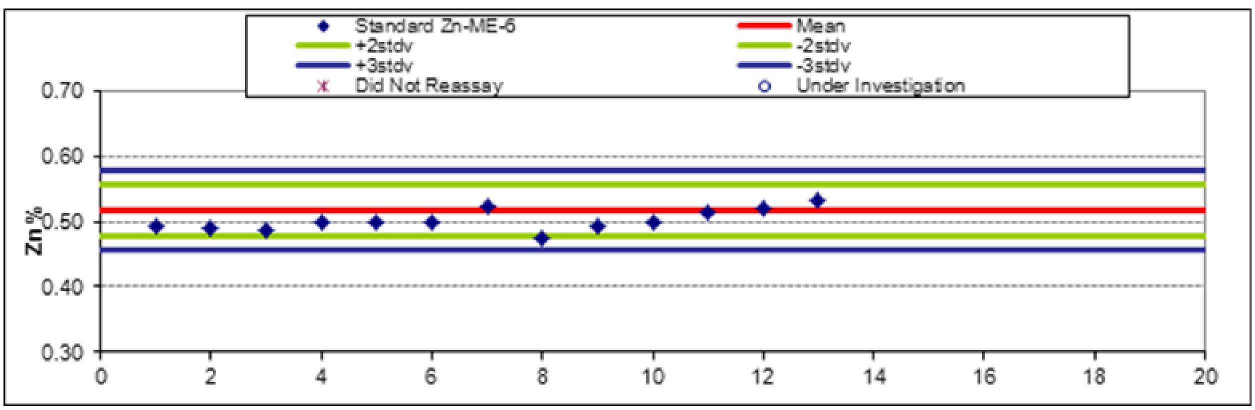
**Figure 11-21: Silver CRM ME-6 Performance**




Source: JDS (2011).

Performance of CRM ME-6 (Figure 11-22) for zinc was perfect, marred only by a very slight below-mean bias throughout the drill program. The bias was within acceptable limits.

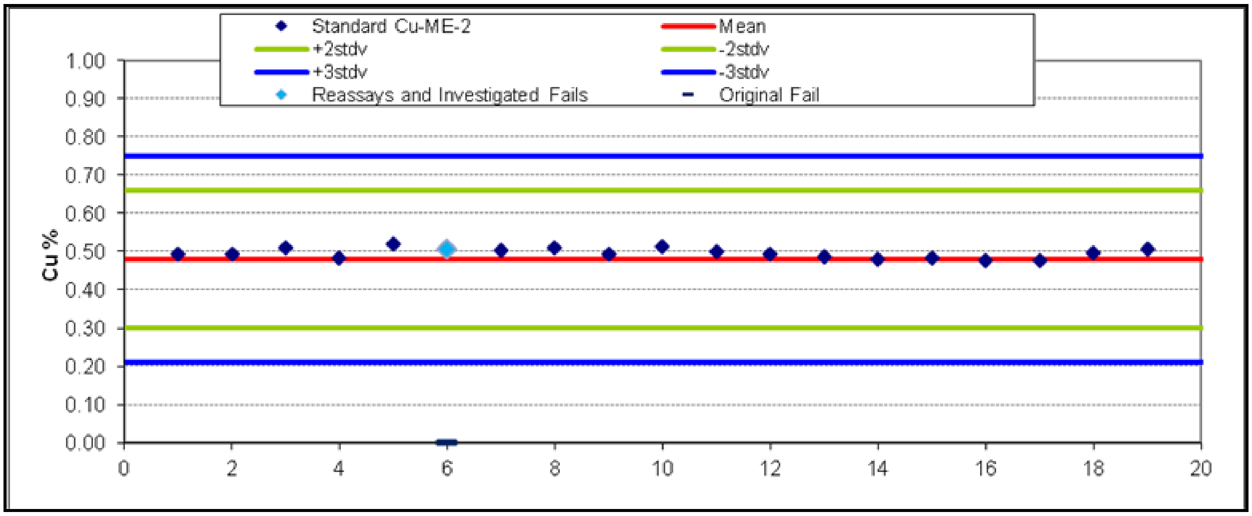
**Figure 11-22: Zinc CRM ME-6 Performance**




Source: JDS (2011).

Performance of CRM ME-2 (Figure 11-23) for copper was excellent, with all final values clustered at the mean. A batch of 20 samples containing the single failure passed upon reanalysis. Investigation of the failure revealed an isolated error where the instrument skipped four samples in the batch, affecting the copper, silver and zinc results. The ALS quality analyst followed up with the laboratory team regarding data review processes to ensure the error is not repeated.

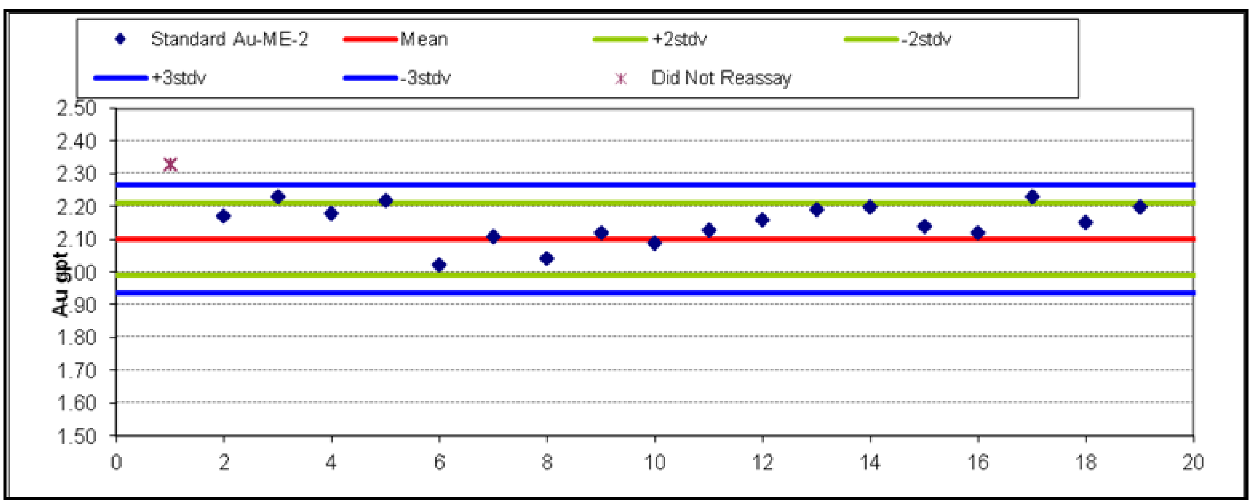
**Figure 11-23: Copper CRM ME-2 Performance**




Source: JDS (2011).

Performance of the CRM ME-2 (Figure 11-24) for gold was very good. One marginally high failure occurred just outside the three standard deviation above the mean performance gate was not reanalyzed. A slight high bias within acceptable limits was seen through most of the drill program.

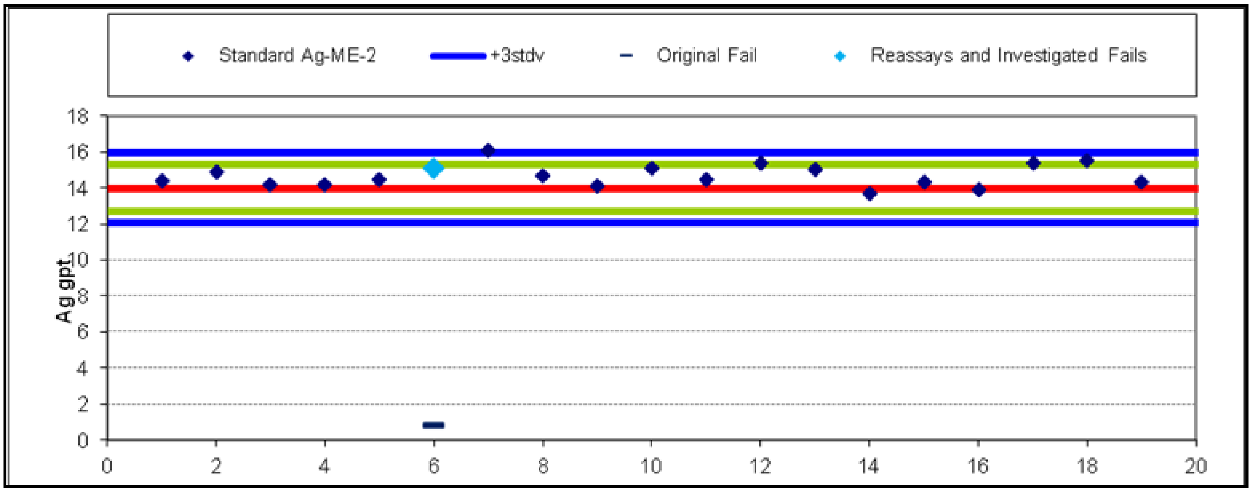
**Figure 11-24: Gold CRM ME-2 Performance**




Source: JDS (2011).

Performance of the CRM ME-2 (Figure 11-25) for silver was excellent, showing a slight high bias within acceptable limits. A batch of 20 samples containing the single failure passed upon reanalysis; the sample also failed for copper and zinc. The failure was attributed to the analytical instrument bypassing four samples in the batch. The ALS quality analyst followed up with the laboratory team regarding data review processes to ensure the error is not repeated.

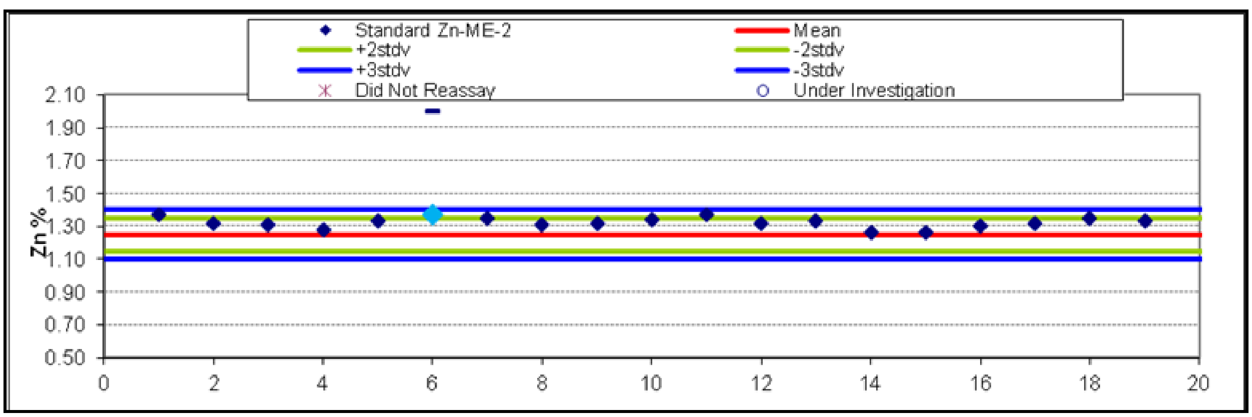
**Figure 11-25: Silver CRM ME-2 Performance**




Source: JDS (2011).

Performance of the CRM ME-2 as shown in Figure 11-26 for zinc was excellent; a consistent high bias within the acceptable limits occurred through the entire drill program. A single failed value was corrected upon reanalysis of a batch of 20 samples (the CRM also failed for copper and silver). Investigation into the failure revealed the analytical instrument had skipped four samples in the batch; ALS internal data review processes were reviewed with lab staff to ensure the error did not recur.

**Figure 11-26: Zinc CRM ME-2 Performance**



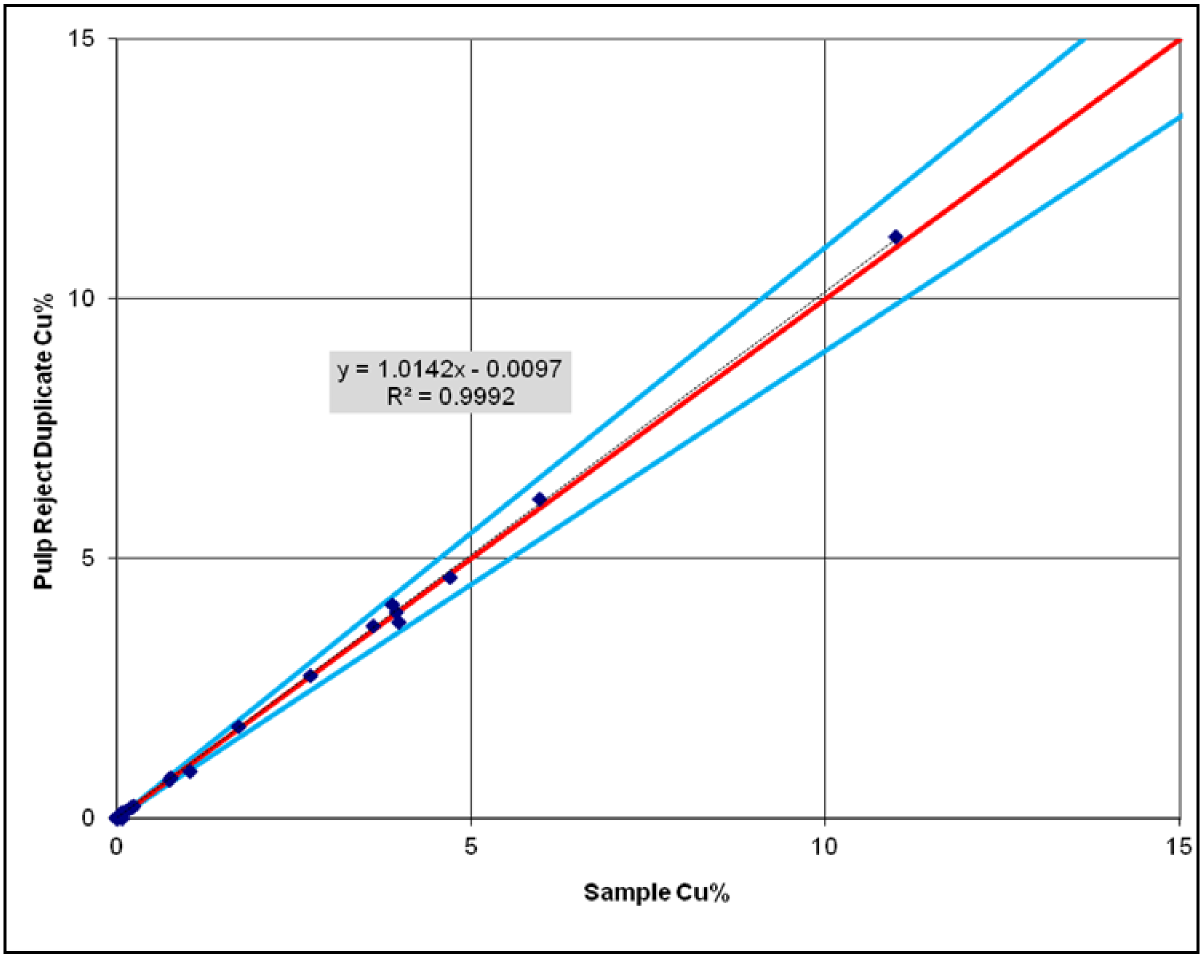

Source: JDS (2011).

#### 11.2.1.1 Duplicate Analysis of Pulp Reject Materials

The core logger designated 1 out of every 20 samples as a pulp reject duplicate by marking a P on the sample tag; the corer cutter placed the original tag and the P tag in the sample bag of the original sample. ALS made up two pulps from this core material and assigned the first sample number to the original and the second sample number to the duplicate. The duplicate sample was analyzed in sequence with the original sample.

Reproducibility of copper in pulp duplicates in Figure 11-27 was excellent for the 2010 drill campaign. The duplicates were marginally higher overall compared to the original samples.

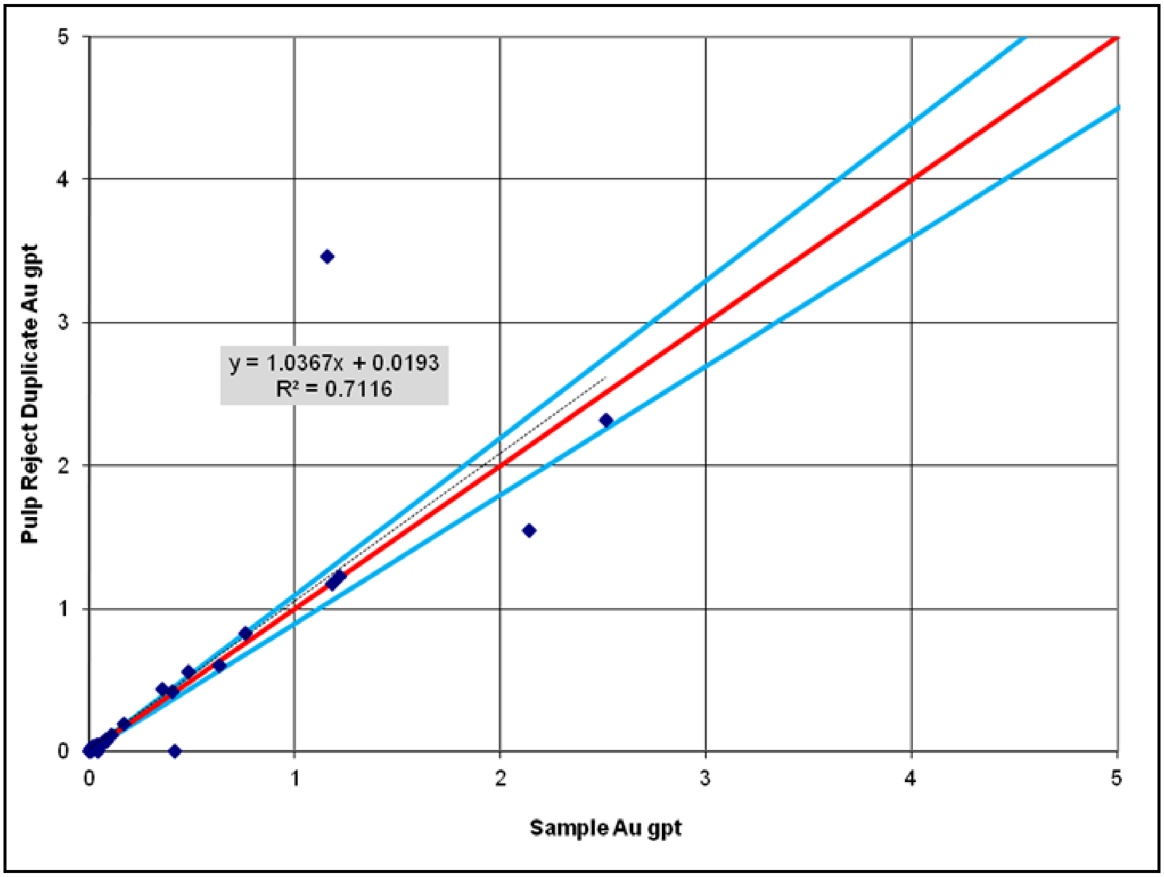
**Figure 11-27: Sample Cu% versus Pulp Reject Duplicate Cu%**




Source: JDS (2011).

Reproducibility of gold in pulp duplicates was very good for the 2010 drill campaign as shown in Figure 11-28. The duplicates were marginally higher overall compared to the original samples and there are three occurrences significantly outside the +/-10% performance gate, two below (the original result was higher grade than the duplicate) and one above (the original result was lower grade than the duplicate).

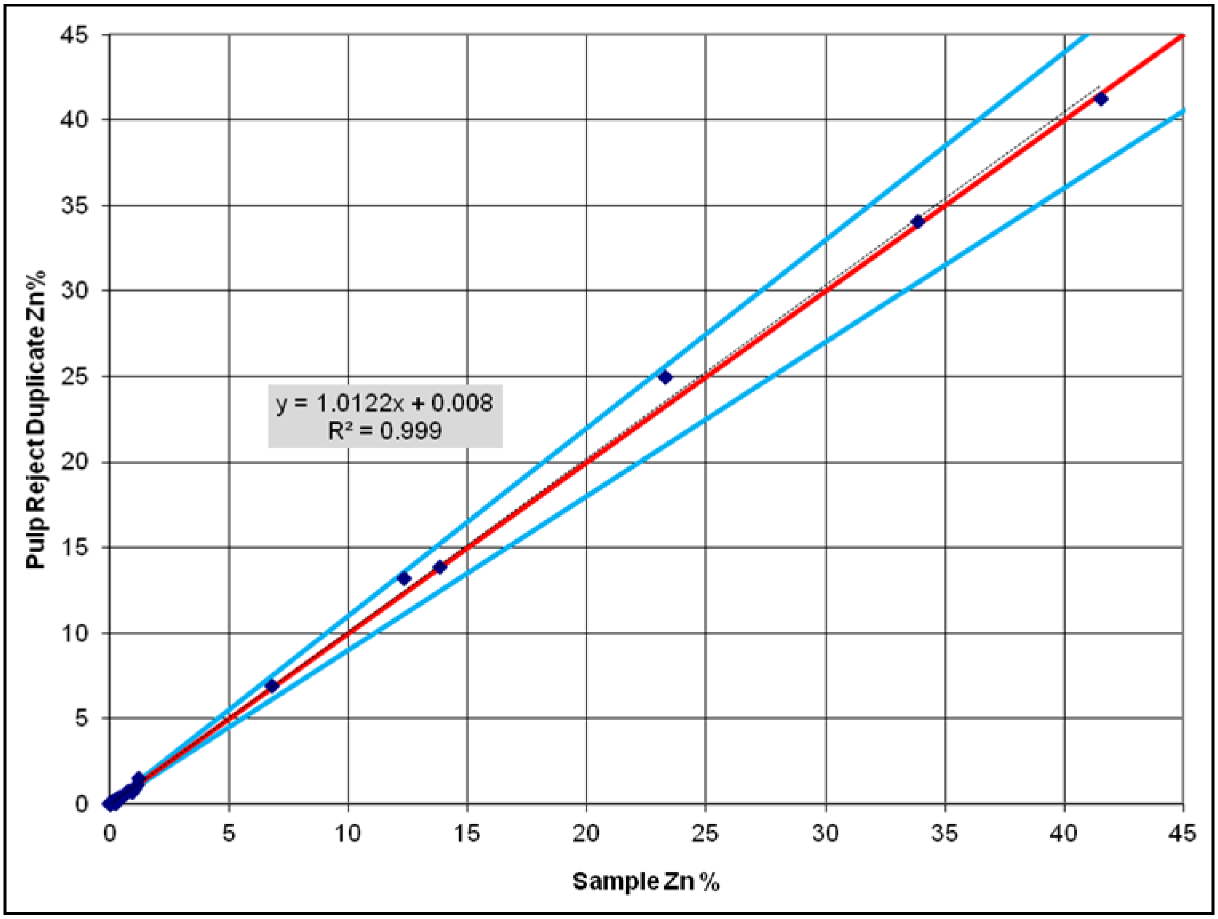
**Figure 11-28: Sample Au g/t versus Pulp Reject Duplicate Au g/t**




Source: JDS (2011).

Reproducibility of zinc in pulp duplicates was excellent for the 2010 drill campaign (Figure 11-29). The duplicates were marginally higher overall compared to the original samples; none of the duplicate pairs plots outside the +/-10% performance gate.

**Figure 11-29: Sample Zn% versus Pulp Reject Duplicate Zn%**



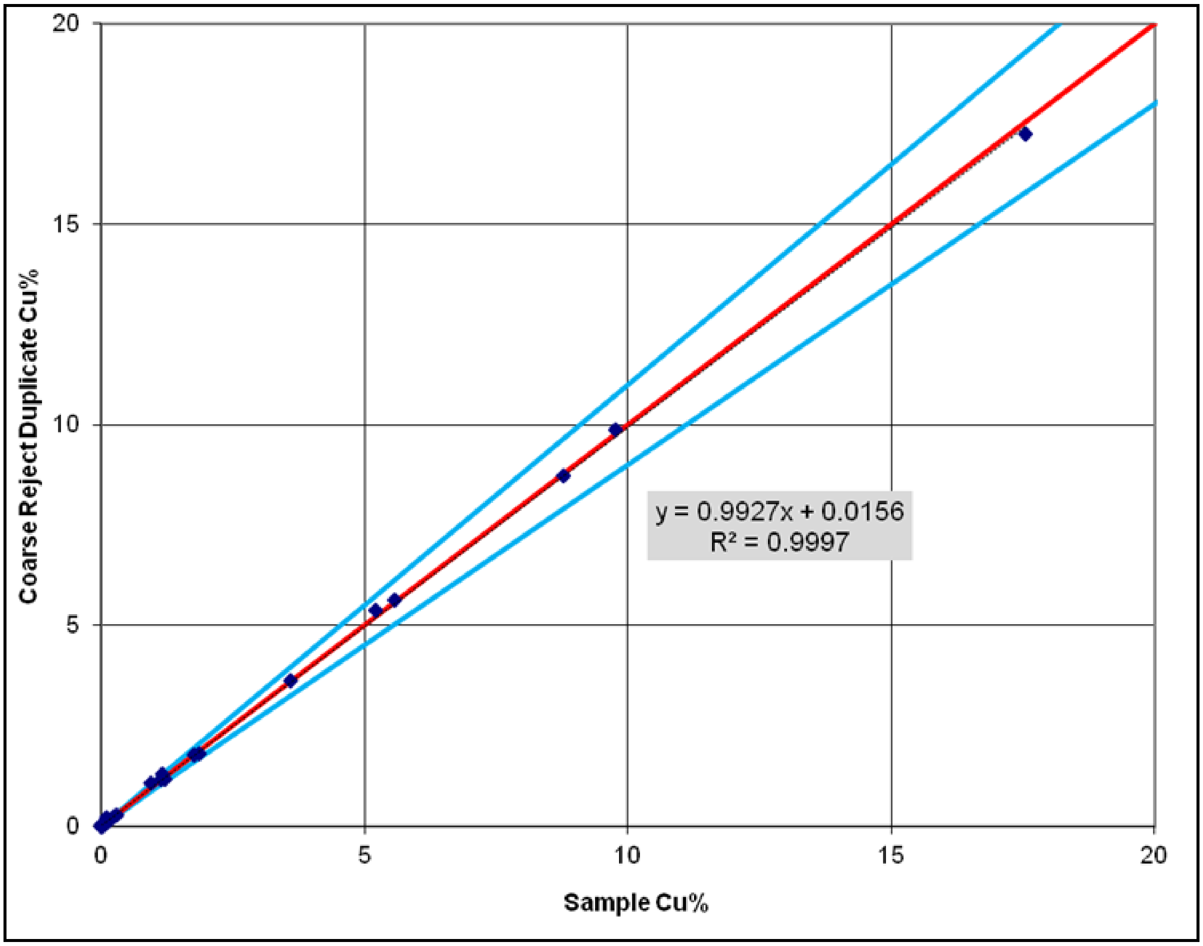

Source: JDS (2011).

### 11.2.1.2 Duplicate Analysis of Coarse Reject Materials

The core logger designated 1 out of every 20 samples as a coarse reject duplicate by marking a P on the sample tag; the corer cutter placed the original tag and the P tag in the sample bag of the original sample. ALS made up two pulps after sub-sampling the coarse crush phase of the core material, and then assigned the first sample number to the original and the second sample number to the duplicate. The duplicate sample was analyzed in sequence with the original sample.

Reproducibility of copper in coarse reject duplicates was excellent for the 2010 drill campaign as shown in Figure 11-30. All duplicate pairs plot within the +/-10% performance gate.

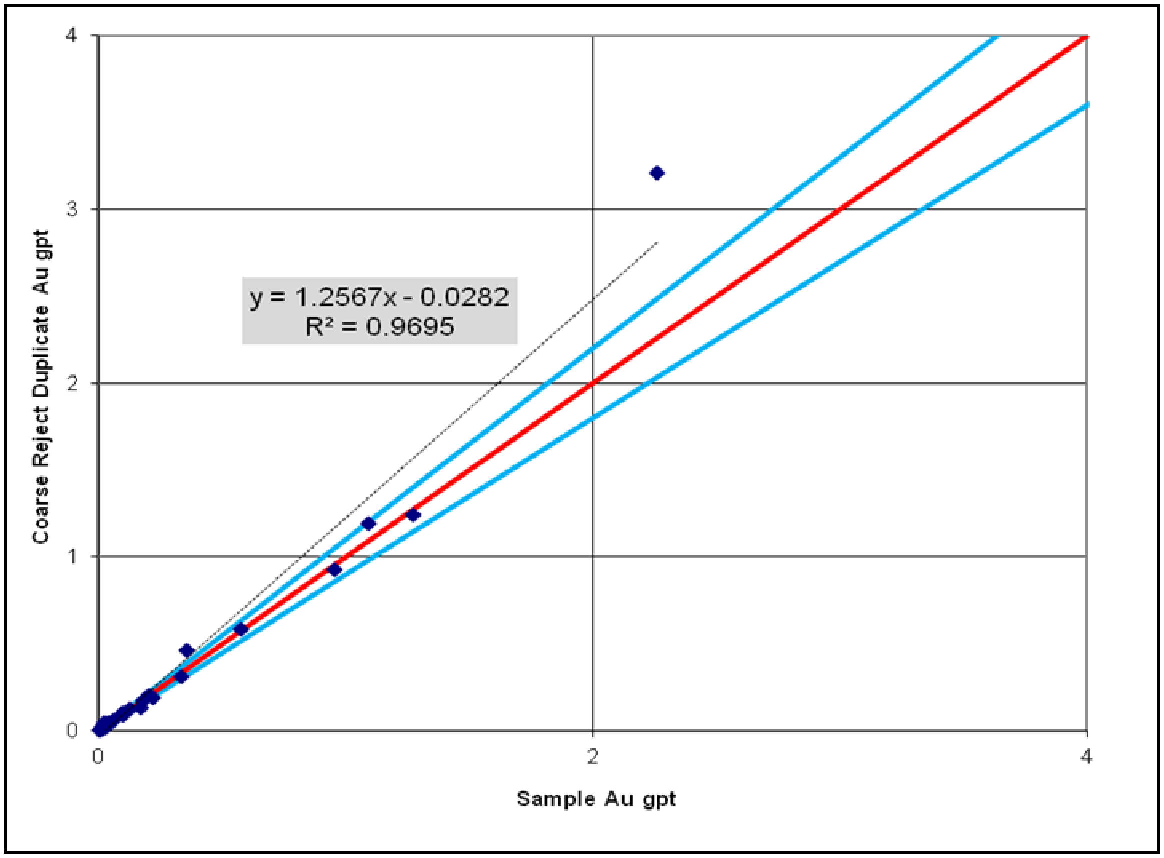
**Figure 11-30: Sample Cu% versus Coarse Reject Duplicate Cu%**




Source: JDS (2011).

Reproducibility of gold in coarse reject duplicates was fair for the 2010 drill campaign (Figure 11-31). The duplicates were higher overall compared to the original samples and there is 1 occurrence significantly above the +/-10% performance gate where the original result was lower grade than the duplicate.

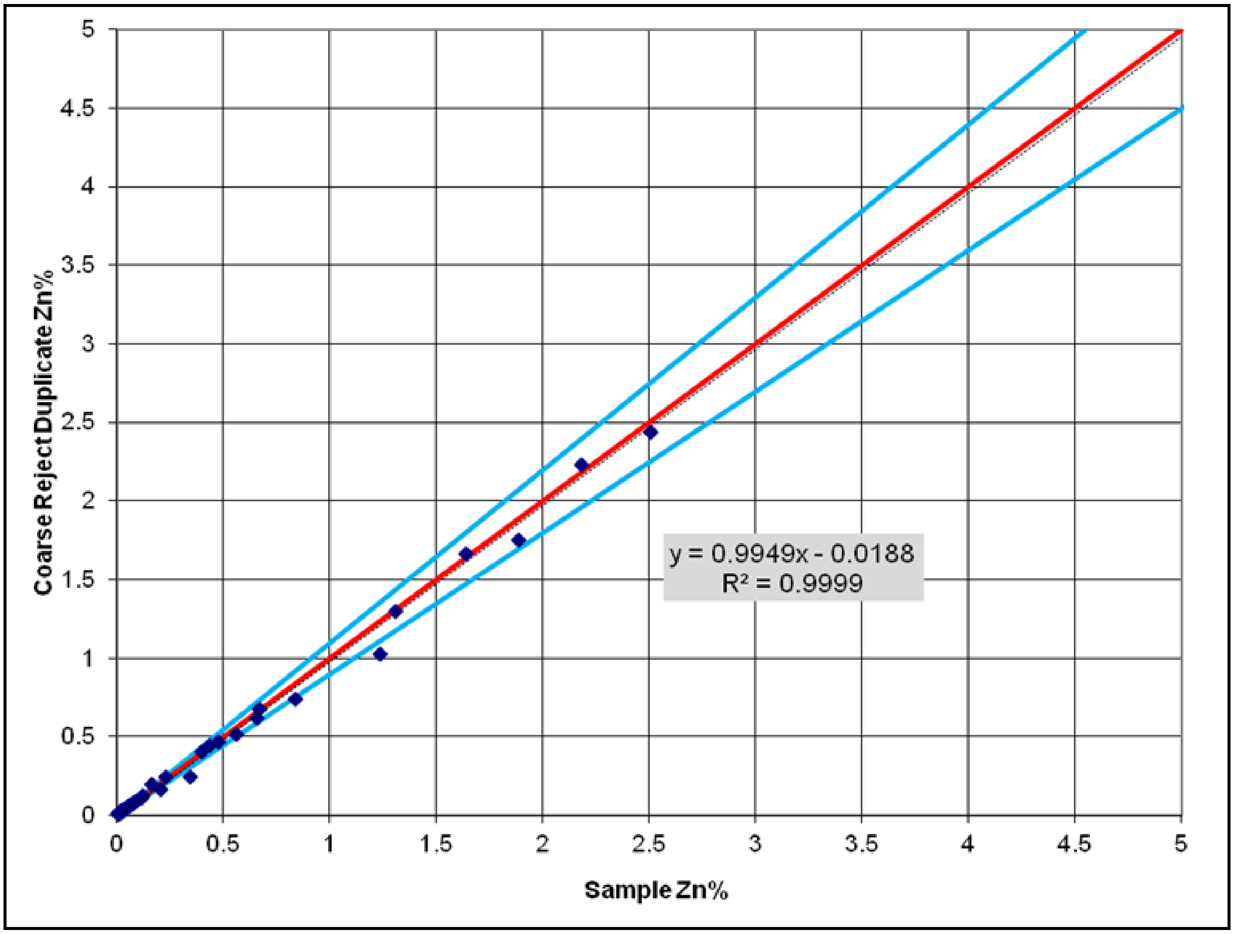
**Figure 11-31: Sample Au g/t versus Coarse Reject Duplicate Au g/t**




Source: JDS (2011).

Reproducibility of zinc in coarse duplicates was very good for the 2010 drill campaign as shown in Figure 11-32. The duplicates were marginally lower overall compared to the original samples. Two of the duplicate pairs plots below the +/-10% performance gate, indicating the original results were higher grade than the duplicate results.

**Figure 11-32: Sample Zn% versus Coarse Reject Duplicate Zn%**




Source: JDS (2011).

For increased confidence in the dataset, a comparison of analyses between labs was conducted. At the end of the drill program, 3% of the sample pulp rejects at ALS were selected across a variety of grade ranges in the Esso zone. The pulp rejects were submitted to IPL Inspectorate in Vancouver, BC, with one CRM added every 15 samples. All the samples were renumbered with new, sequential sample identifications. Identical analytical methods were used. Results for the check analysis demonstrated a very strong correlation between the original sample and duplicate sample for copper, zinc, lead and silver. Gold values in the between-lab check samples correlated well but were not as reproducible, particularly over 1 g/t Au, due to the nugget behaviour of gold in the Esso lens.

## 11.3 2011 Drill Program

In 2011, mineralized intervals in core were sampled in lengths ranging from 30 cm to 2.6 m, averaging 1 to 1.5 m. The sampling intervals are typically 0.8 to 1.3 m in mineralized material with a maximum sample of 3 m allowed where waste intervals between mineralized zones occur. Two shoulder samples were taken in waste at both upper and lower contacts, consisting of a 1.5 m sample and a 1.0 m sample. Samples do not cross geological contacts.

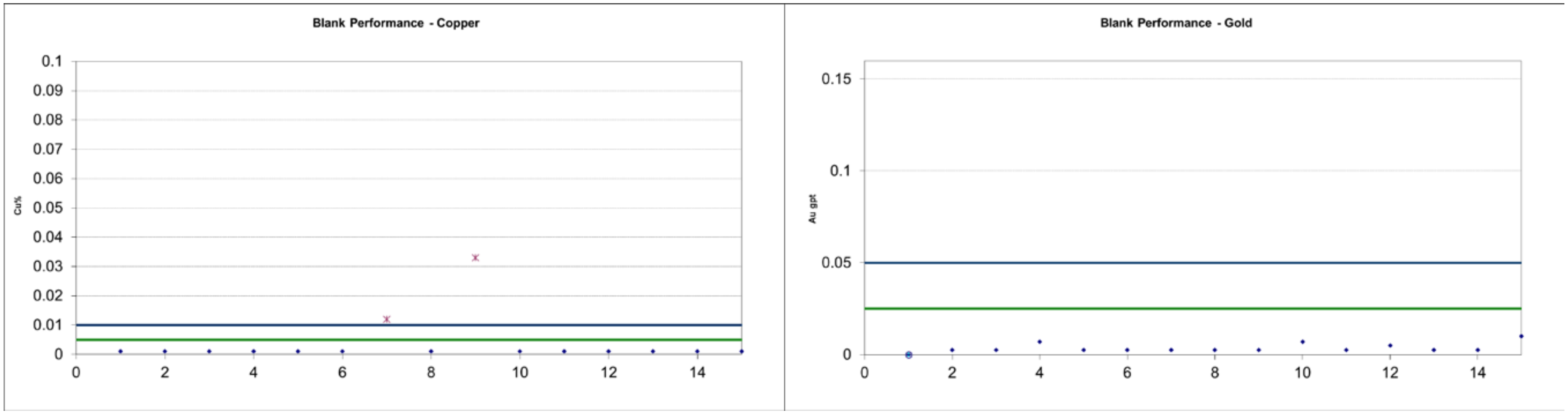
Sample quality was very good. Recovery of every drill run for 5 of 6 holes was recorded and typically exceeded 90 to 95% in the mineralized zone. The samples were taken across the entire mineralized interval, with sample boundaries respecting lithological contacts, VMS cycles and mineralization styles, including varying percentages of sulphide minerals. Higher grade intervals within low grade intersections were identified visually during core logging; the variations in mineralization were also respected during the sampling process. Two shoulder samples were added at both the up hole and downhole contacts to ensure the mineralization was fully sampled.

Samples taken from holes drilled at approximately 190º azimuth and -55º inclination are representative of true width in the Sumac Deposit, although the steeply dipping extents require more drilling.

Performance of blanks was acceptable for copper and gold. Figure 11-33 shows QA/QC performance of blanks with the warning line at in green (5 times the detection limit) and the failures line in blue (10 times the detection limit). Failures that were not reassayed are shown as red stars. Minor, sporadic contamination events occurred for copper but were not reassayed. Concerns were communicated to ALS, who reviewed the between-sample cleaning protocols with preparation personnel.

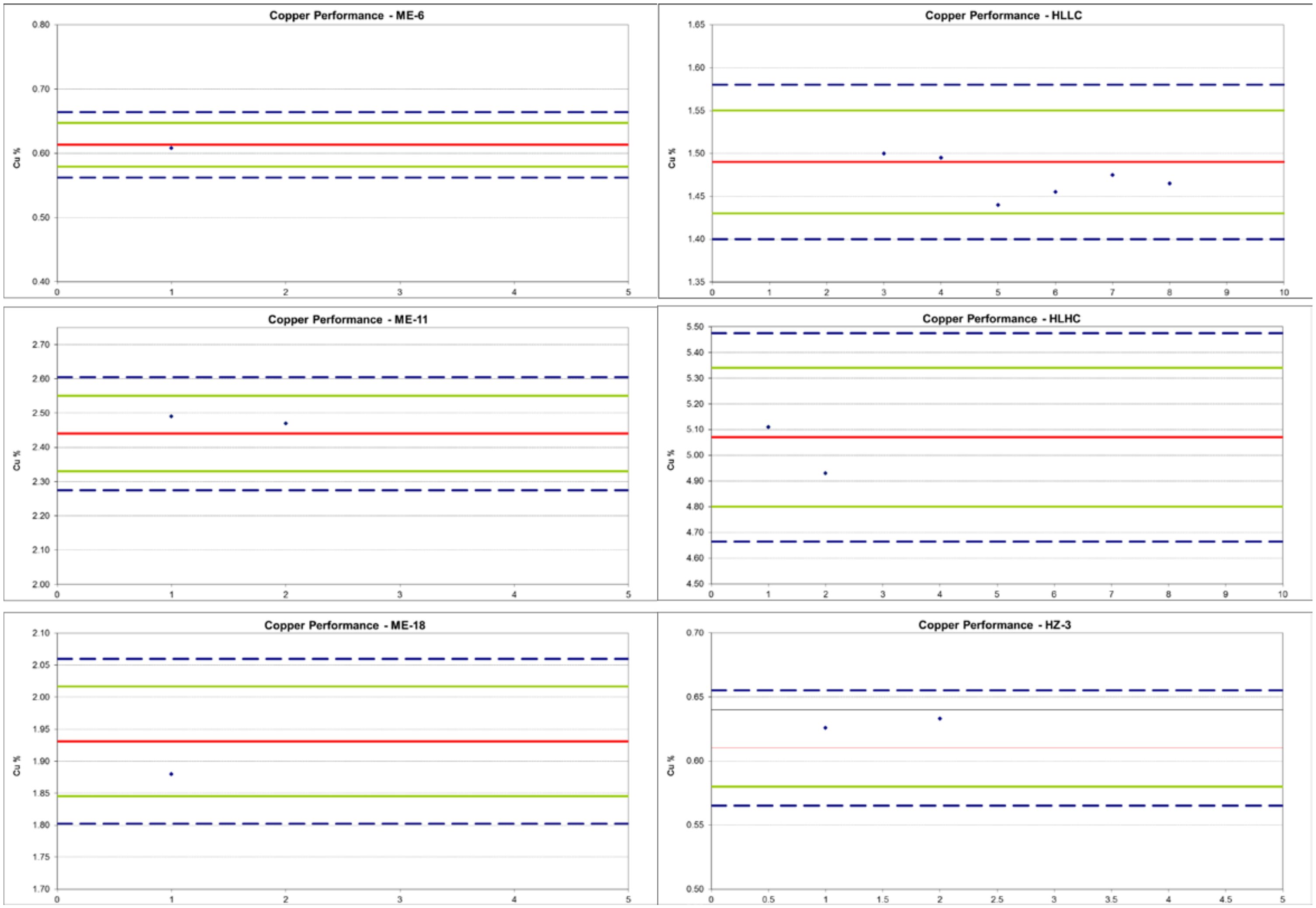
Performance of CRM was acceptable for copper (Figure 11-34), zinc (Figure 11-35), silver (Figure 11-36), and gold (Figure 11-37). One failure for zinc in CRM HZ-3 was resolved upon reassay of the batch of affected samples. Figures 11-34 through 11-37 are shown with the mean in red, two standard deviations from the mean as a green line and three standard deviations from the mean as a dashed blue line.

## Figure 11-33: QA/QC Performance of Blanks (Copper and Gold)



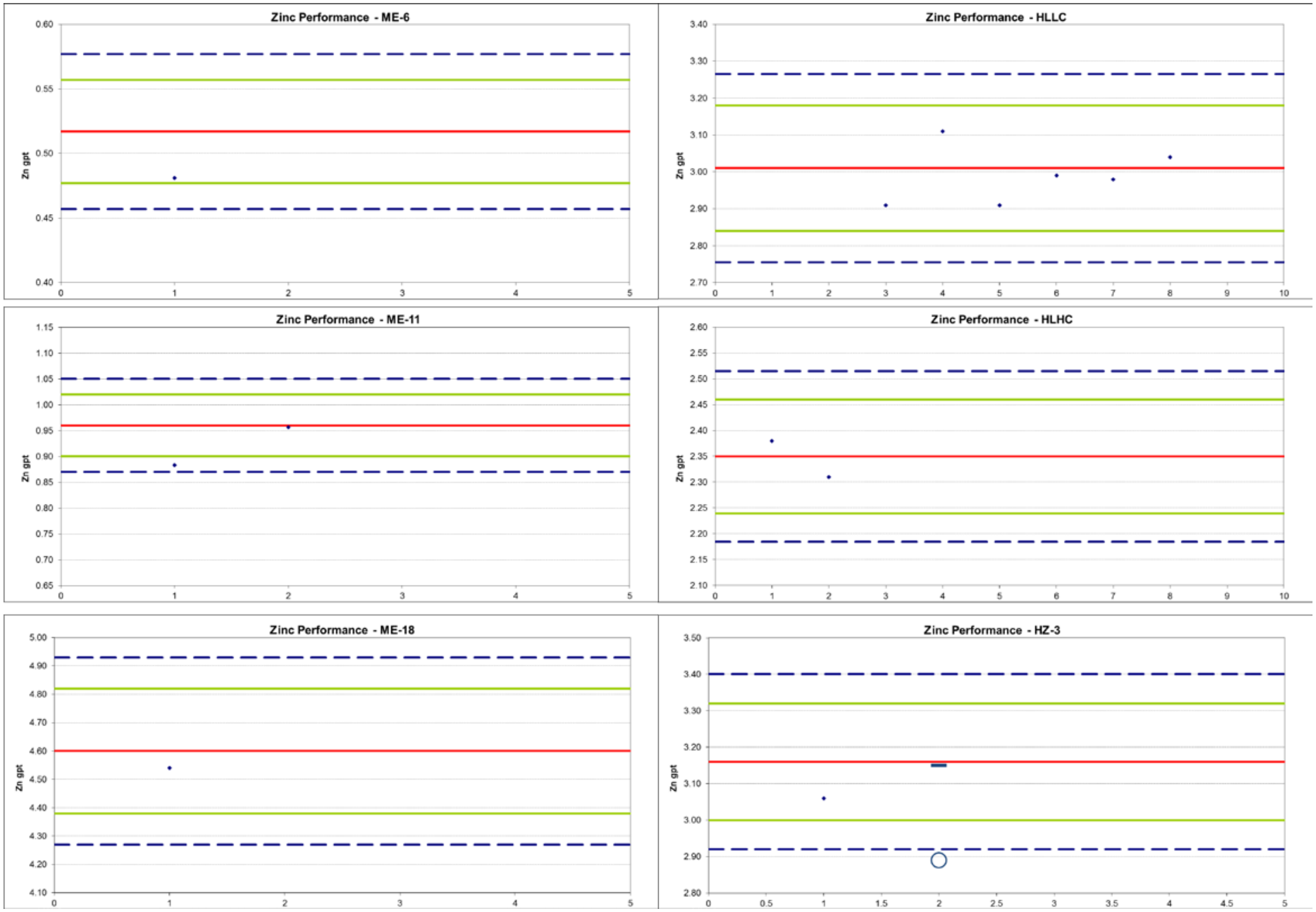

Source: JDS (2017).

## Figure 11-34: QA/QC Performance of Copper CRM



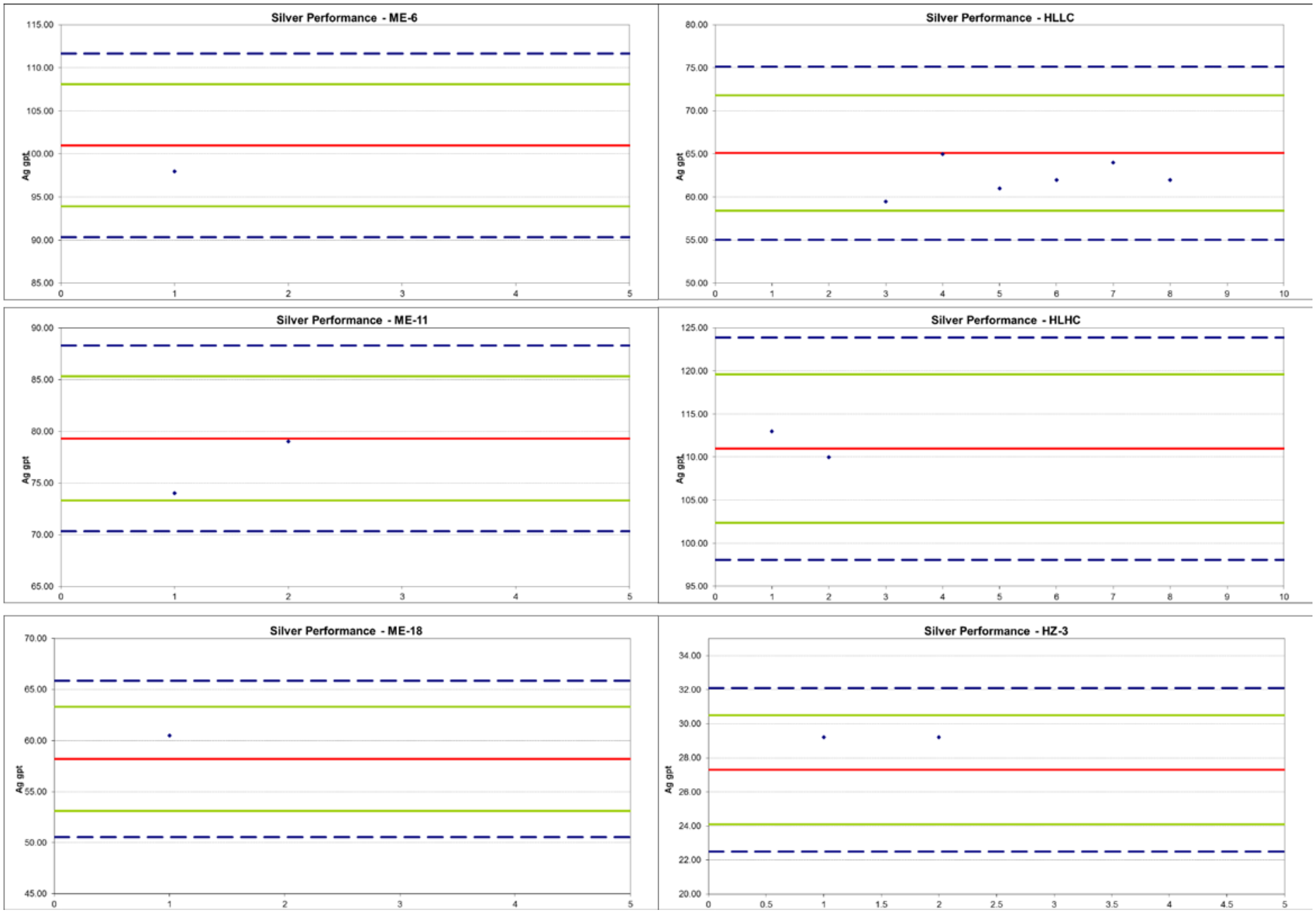

Source: JDS (2017).

## Figure 11-35: QA/QC Performance of Zinc CRM



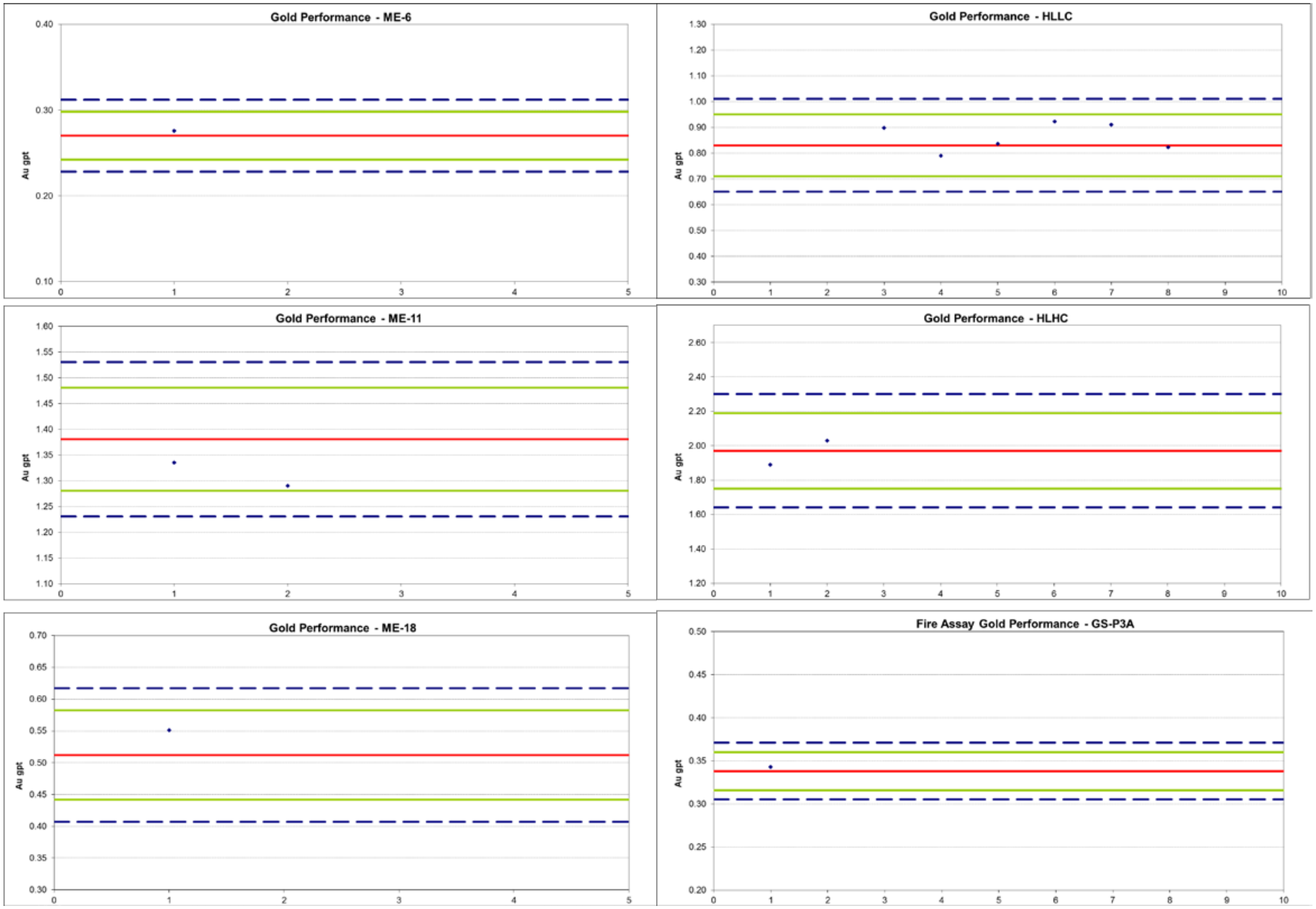

Source: JDS (2017).

### Figure 11-36: QA/QC Performance of Silver CRM




Source: JDS (2017).

## Figure 11-37: QA/QC Performance of Gold CRM




Source: JDS (2017).

## 11.4 Qualified Professional Statement of Adequacy

In the authors' opinion, the sample preparation, analysis QA/QC and security protocols follow accepted standards and industry best practices. Based on the data and results, it is the authors opinion that the complied database is valid and of sufficient quality to be used for this mineral resource estimate.

# 12 Data Verification

## 12.1 Geology, Drilling and Assaying

Garth Kirkham, P. Geo., has been involved with the Kutcho property since 2008 as Independent Qualified Person. This included being author of the Technical Reports beginning in 2008 and then subsequent update in 2011. The author was also involved in the drill program planning focussed on resource expansion and in-fill drilling.

Garth Kirkham, P. Geo., visited the property on April 30, 2008, viewed and inspected core and drill sites, reviewed procedures, and confirmed data collection techniques. The site visit included a tour of major centres and surrounding villages most likely to be affected by any potential mining operations along with aerial reconnaissance of the Project site, deposit areas and location of planned facilities. On-site staff led the author through the chain of custody and methods used at each stage of the logging and sampling process. All methods and processes are up to industry standards and reflect best practices, and no issues were identified. The core is accessible and stored in covered racks. It is the author's opinion that the methods and procedures at that time met industry standards and best practices. No individual sample validation and verification was employed by the author due to the history of the property and it is believed that the level of workmanship and professionalism is at a high level and therefore not warranted.

Garth Kirkham, P. Geo., visited the property most recently on July 17, 2017. The tour of the office and camp showed a clean, well-organized, professional environment. The majority of the core from the pre-2010 drill program is accessible and the core is stored in covered racks a short distance from the camp. Core from the more recent 2010 and 2011 programs is stored at the end of the air strip and is stacked.

Eight drillholes were laid out from various vintages of programs and inspected. Logs and assay sheets were verified against the core and the logged intervals. The data correlated with the physical core and no issues were identified. In addition, the author toured the complete core storage areas, selecting and reviewing core throughout. No issues were identified.

The author is confident that the data and results are valid based on the site visit and inspection of all aspects of the Project, including methods and procedures used. It is the opinion of the independent author that all work, procedures, and results have adhered to best practices and industry standards required by NI 43-101.

No duplicate samples were taken by the author to verify assay results, but the author is of the opinion that the work was being performed by a well-respected, western company that employs competent professionals that adhere to industry best practices and standards. There were no issues or concerns that prevented data validation and verification.

The author is satisfied that the assay data are of suitable quality to be used as the basis for this resource estimate.

## 12.2 Metallurgy

The first steps taken to verify the metallurgical data was to review the metallurgical reports prepared by SGS Canada Inc. and Metallurgical Lab Cozamin (Cozamin) based on metallurgical samples from Kutcho. The drill core provided to Cozamin was clearly identified by drillhole and interval of each sample allowing the material to be identified within the two zones; Main and Esso. The metallurgical drillhole locations and segments used to create the composites were summarized by Cozamin in excel spreadsheets. The samples for Main zone, eight drillholes, ranged in head grade from 1.5% up to 2.7% Cu and 2.0% up to 10.0% Zn. The samples for Esso zone, four drillholes, ranged in head grade from 1.9% up to 9.9% Cu and 3.6% up to 22.5% Zn. The drillholes used for metallurgical test work were plotted against the planned area to be mined and were found to be spatially representative of the majority of the two zones. Drillholes did not include the extents of the two zones and the Esso composites used for the years 3, 4, and 5 composites were higher in grade for zinc and silver compared to the mine plan. To predict the recoveries the QP worked with the geologist to determine the composites and corresponding test work that best fit with the areas to be mined. It is the QP's opinion that there is sufficient data and test work to estimate metallurgical recoveries and define the flowsheet at a at a PFS level.

## 12.3 Mining

Mining design data was verified through review of previous studies and reports. Any studies referred to were thoroughly reviewed and summarized in this report and align with the PFS mine design and mine plan. All mining data was verified and is adequate for this PFS Technical Report as required by NI 43-101 guidelines.

There were no limitations to conducting data verification by the QPs in preparation of this Technical Report.

# 13 Mineral Processing and Metallurgical Testing

## 13.1 Introduction

The metallurgical test work carried out by SGS and Metallurgical Lab Cozamin were reviewed for this updated report. No additional test work has been completed since JDS 2011.

Metallurgical test work was carried out in 2010 to investigate the metallurgical response of Kutcho material and to develop the process flowsheet and design criteria. The test program was aimed at developing a conventional copper-zinc flowsheet comprising comminution and sequential flotation circuits. Tests were conducted on samples from the Main and Esso deposits.

Comminution tests showed that the ore has low levels of hardness and abrasiveness with a Bond Rod Mill Work Index of 8.9 kilowatt hours per tonne (kWh/t), Bond Ball Mill Work Index of 12.2 kWh/t and abrasion index of 0.16 g. Target primary grind is a $P_{80}$ of 75 µm.

The preliminary updated projections for the recoveries to the copper and zinc concentrates were determined based on the updated mine plan and the December 2010 locked cycle tests (LCT) results. The LOM average copper recovery of 84.7% at 27.6% Cu concentrate grade and an average zinc recovery of 75.7% at 55.1% Zn concentrate grade are predicted. Gold and silver reporting to the copper concentrate are predicted to have recoveries of 41.2% and 48% recovery respectively. The preceding recoveries and concentrate grades were used in the economic model.

## 13.2 Historical Metallurgical Testing

Extensive metallurgical test work (including process mineralogy) has been conducted since 1975 by Esso Minerals Canada Ltd., Coastech Research, Sumitomo Mining's Niihama Laboratory and Lakefield Research (SGS), and included batch tests, locked cycle tests and a pilot plant campaign. These programs investigated copper-zinc differential flotation as well as a copper-zinc bulk flotation followed by copper-zinc separation. The results obtained from the main test work programs were not consistent, but similar in some degree of range (SRK 2008).

Reports from previous test work are documented and summarized in the 2008 SRK Consulting Preliminary Economic Assessment (PEA) and JDS 2011. The reports reviewed for this report include:

- SGS Lakefield Research Limited, 2006. Project No. LR 10933-001 Report 1: The Recovery of Copper and Zinc from Kutcho Creek Project Samples;
- SGS Lakefield Research, 2008. Project No.10933-001,002 and 004 Final Report: The Grindability Characteristics of Samples from the Kutcho Creek deposit;
- SGS Lakefield Research Limited, 2009. Project No. LR 11904-001 Report 1: The Recovery of Copper and Zinc from Kutcho Creek Project Samples;
- Capstone, Metallurgical Lab Cozamin, May, 2010a. Metallurgical Research Kutcho Project;
- SGS Canada Inc., 2010a. Project 12284-001 – Final Report: The Recovery of Cu and Zn from the Kutcho Creek deposit;

- SGS Canada Inc., 2010b. Project 12284-002 – Final Report: The Deportment Study of Gold in the Lock Cycle F Tail Sample from the Kutcho Copper; and
- Capstone, Metallurgical Lab Cozamin, December, 2010b. Metallurgical Research Kutcho Project – Drill Core Samples Part 1.

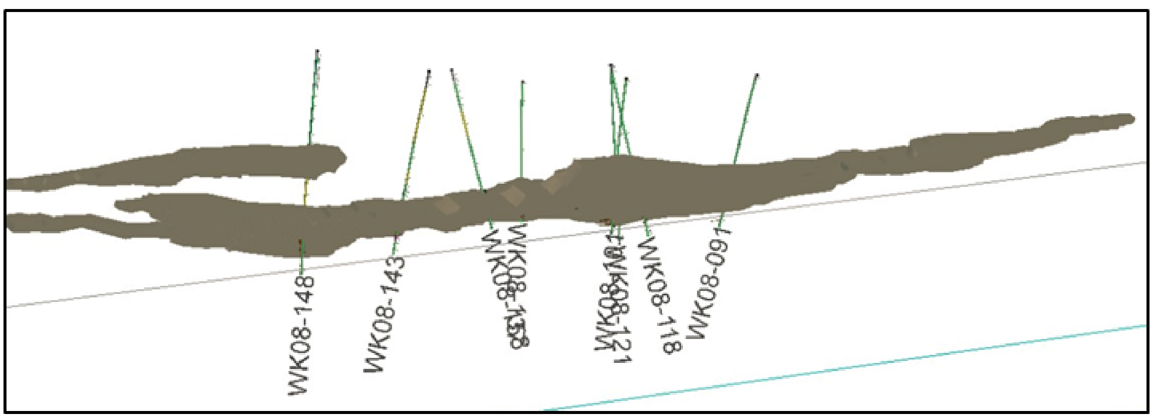
## 13.3 Summary of Metallurgical Research Kutcho Project Test Program, December 2010

Various samples from the Project were delivered to Cozamin for the 2011 PFS metallurgical test program. The primary objective of the test program was to determine process design criteria for crushing, grinding and flotation. The test work details are summarized in the *Metallurgical Research Kutcho Project (Drill Core Samples Part 1)* report from the Cozamin lab dated December 2010 (Capstone 2010b). The source of all figures and tables are from the December 2010 report and backup data unless otherwise specified.

### 13.3.1 Sample Selection and Head Assay

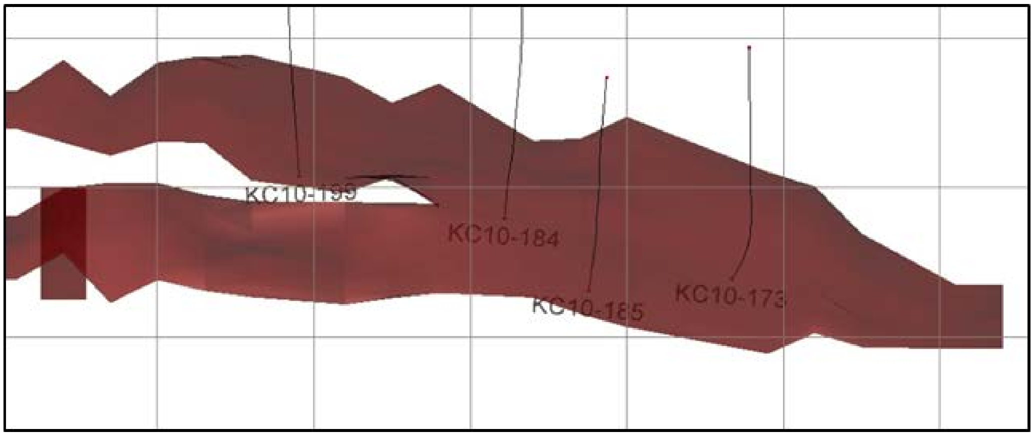
The samples prepared for the test program were based on drill core from Main and Esso zones. Spatially the drillholes are representative of the majority of the two zones. During the next phase of engineering additional test work to include the extents of Main and Esso is recommended. The location of the drillholes are shown below in Figure 13-1 and 13-2.

**Figure 13-1: Main Zone Sample Drillholes**




Source: Capstone (2010b).

### Figure 13-2: Esso Zone Sample Drillholes




Source: Capstone (2010b).

The lists of the composites and the head assay of each composite are summarized in Table 13-1.

### Table 13-1: Composite Summary and Head Assays

| Composite Identity | kg | Au (g/t) | Ag (g/t) | Cu (%) | Pb (%) | Zn (%) | Fe (%) |
|---|---|---|---|---|---|---|---|
| Main Zone Geographic Composite #1 | 38 | 0.35 | 43 | 2.55 | 0.029 | 3.17 | 30.08 |
| Main Zone Geographic Composite #2 | 39 | 0.27 | 29.2 | 2.46 | 0.032 | 2.21 | 28.83 |
| Main Zone Geographic Composite #3 | 26 | 0.59 | 48.8 | 2.68 | 0.034 | 2.03 | 21.22 |
| Main Zone Geographic Composite #4 | 38 | 0.34 | 34.3 | 1.95 | 0.13 | 3.19 | 24.23 |
| Main Zone Geographic Composite #5 | 45 | 0.34 | 30.3 | 1.83 | 0.038 | 3.27 | 33.38 |
| Main Zone Geographic Composite #6 | 22 | 0.29 | 44.5 | 2.31 | 0.065 | 3.43 | 27.27 |
| Main Zone Geographic Composite #7 | 15 | 0.53 | 33.3 | 2.26 | 0.093 | 5.33 | 10.96 |
| Main Zone Geographic Composite #8 | 16 | 0.56 | 31.9 | 2.51 | 0.044 | 3.69 | 30.6 |
| Calculated Main Zone Global Composite | 240 | 0.38 | 36.4 | 2.27 | 0.056 | 3.1 | 27.21 |
| Main Zone Global Composite | 240 | 0.48 | 33.9 | 2.38 | 0.06 | 3.35 | 28.77 |
| Year 1 Time Variation Composite | 105 | 0.41 | 25.5 | 1.45 | 0.134 | 3.52 | 6.52 |
| Year 2 Time Variation Composite | 16 | 0.38 | 40.2 | 2.29 | 0.05 | 3.19 | 22.22 |
| Year 3 Time Variation Composite | 16 | 0.69 | 245 | 2.56 | 0.19 | 5.58 | 24.25 |
| Year 4 Time Variation Composite | 16 | 1.35 | 222.7 | 2.6 | 0.19 | 4.58 | 17.71 |
| Year 5 Time Variation Composite | 16 | 0.31 | 32.5 | 2.08 | 0.25 | 10.05 | 21.41 |
| Year 6 to 10 Time Variation Composite | | | | | | | |
| Esso Zone Geographic Composite #1 | 16 | 1.03 | 56.6 | 2.06 | 0.311 | 20.1 | 16.58 |
| Esso Zone Geographic Composite #2 | 16 | 1.43 | 554.5 | 9.9 | 0.533 | 22.45 | 13.79 |
| Esso Zone Geographic Composite #3 | 16 | 0.85 | 146.2 | 4.03 | 0.097 | 3.65 | 14.44 |
| Esso Zone Geographic Composite #4 | 16 | 0.82 | 71.5 | 1.91 | 0.243 | 22 | 10.77 |
| Esso Zone Global Composite | 16 | 1.17 | 177.7 | 4.56 | 0.22 | 11.1 | 15.66 |

Source: Capstone (2010b).

### 13.3.2 Mineralogy

Cozamin lab commissioned the University of San Luis Potosi to carry out the studies on liberation and mineralogical characterization of Kutcho global composite samples. The main copper mineral species and their percentages are summarized in Table 13-2. The only species of zinc is wurtzite.

**Table 13-2: Copper Mineral Species**

| Copper Species | Percent (%) |
|---|---|
| Bornite | 65.74 |
| Chalcopyrite | 30.20 |
| Chalcocite | 4.06 |

Source: Capstone (2010b).

Liberation studies revealed that bornite has a high degree of liberation at 80.3% and its main association is with pyrite in simple union at an average size of 10 µ.

The 58.5% of the chalcopyrite is free and the main association of chalcopyrite is with pyrite, the liberation average size is 20 µ. Chalcocite demonstrated low degree of liberation.

Wurtzite is presented free in a range of 69.8% and associated with pyrite at 30.2%, the liberation average size is 20 µ.

### 13.3.3 Ore Grindability Testing

Cozamin lab commissioned SGS Mineral Services, Durango, Mexico to carry out comminution testing for the 2011 PFS (JDS 2011).

Samples made up from drill core samples of the global composite samples were used to represent the overall Kutcho material. The composites were submitted to SGS to determine the bond rod and ball mill work indexes and abrasion index. The results are summarized in Table 13-3.

**Table 13-3: Bond Mill Work Index Test Results**

| Sample | RWI (kWh/t) | BWI (kWh/t) | AI (g) |
|---|---|---|---|
| Global Composite | 8.9 | 12.19 | 0.16 |

Source: Capstone (2010b).

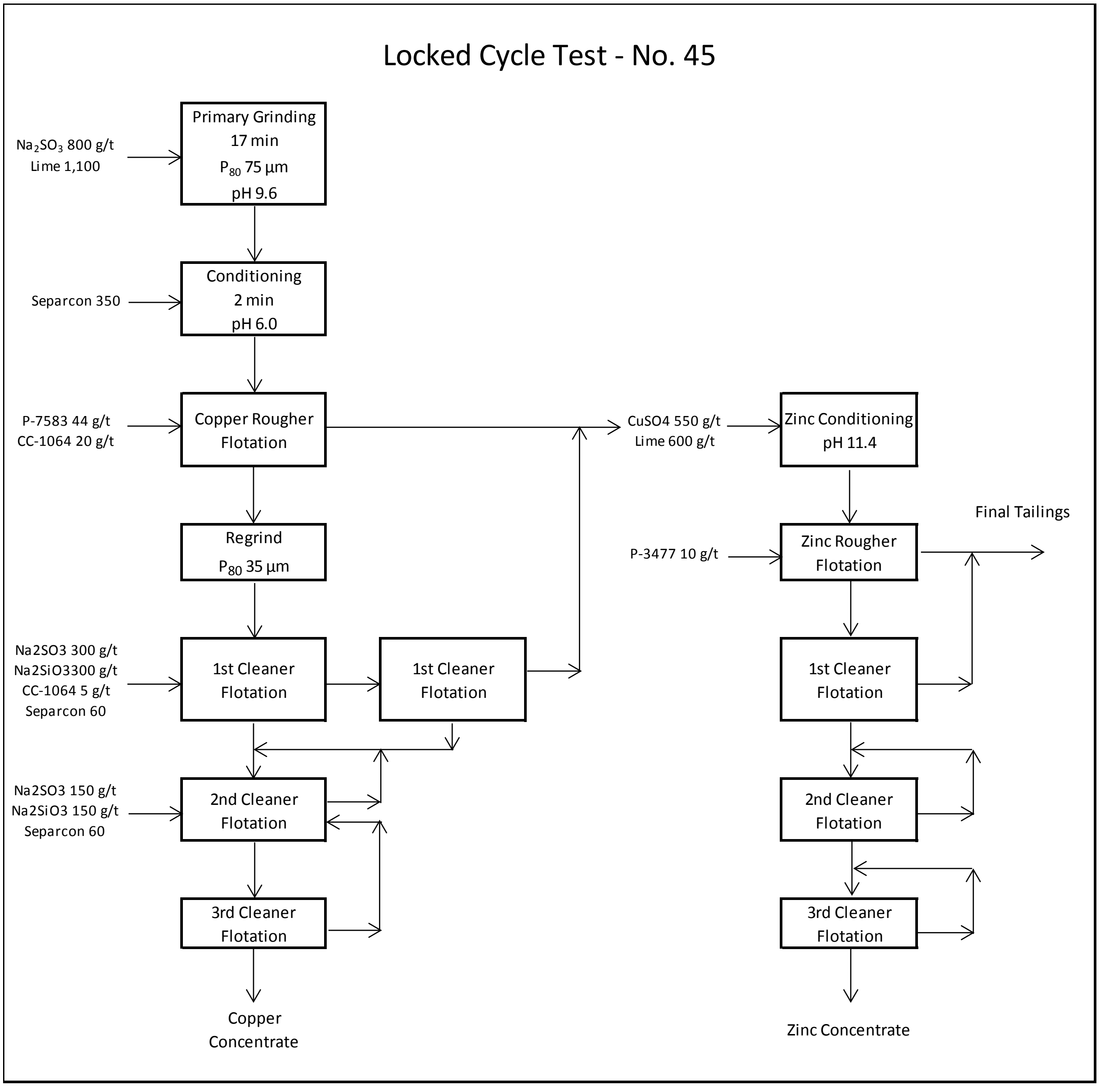
The grinding test results were then used to size the grinding circuit. The basic assumptions for the mill sizing are:

- Plant throughput 2,500 mt/d at 92% availability;
- Primary grinding circuit will consist of one SAG mill and one Ball mill;
- Feed to the grinding circuit is $F_{80}$: 125 mm and product size is $P_{80}$: 75 µm; and
- No pebble crusher - it will be considered during operation and installed if necessary.

### 13.3.4 Flotation Test Work

Flotation tests were completed to investigate the effect of primary grind size, regrinding, and reagents on copper and zinc recovery and their concentrate grade. From the results the test parameters and flowsheet for the LCT were determined. The LCT flowsheet and results of the Main global composite and time variation composites for years 1, 3, 4, and 5 are shown below in Figure 13-3 and Table 13-4. Time Variation Composite year 2 results were not reported.

**Figure 13-3: Locked Cycle Test Flow Sheet**




Source: Capstone (2010b).

### Table 13-4: Locked Cycle Test Results

| Product | Weight (g) | Weight (%) | Assays | | | | | | Distribution % | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | Au (g/t) | Ag (g/t) | Cu (%) | Pb (%) | Zn (%) | Fe (%) | Au | Ag | Cu | Pb | Zn | Fe |
| **Global Composite 1** | | | | | | | | | | | | | | |
| Copper Concentrate | 107.4 | 5.5 | 1.91 | 224.1 | 27.7 | 0.7 | 8.5 | 19.6 | 41.1 | 45.0 | 82.1 | 70.2 | 16.2 | 4.3 |
| Zinc concentrate | 72.5 | 3.7 | 0.52 | 36.8 | 1.6 | 0.1 | 54.0 | 7.0 | 7.6 | 5.0 | 3.3 | 4.2 | 69.5 | 1.0 |
| Tailing Zinc cleaner | 165.4 | 8.4 | 0.37 | 37.6 | 0.5 | 0.0 | 2.4 | 27.9 | 12.4 | 11.6 | 2.4 | 6.3 | 6.9 | 9.5 |
| Tailing Zinc rougher | 1,613.3 | 82.4 | 0.12 | 12.7 | 0.3 | 0.0 | 0.3 | 25.6 | 38.8 | 38.4 | 12.2 | 19.3 | 7.4 | 85.1 |
| Flotation Feed | 1,958.6 | 100 | 0.25 | 27.3 | 1.85 | 0.05 | 2.88 | 24.77 | 100 | 100 | 100 | 100 | 100 | 100 |
| **Year 1 Time Variation Composite** | | | | | | | | | | | | | | |
| Copper Concentrate | 96.1 | 4.9 | 2.53 | 306.7 | 25 | 1.9 | 12.9 | 12.4 | 43.3 | 66 | 90 | 84 | 26.3 | 9.6 |
| Zinc concentrate | 58.3 | 3.0 | 2.88 | 50.9 | 1.4 | 0.1 | 51.8 | 7.1 | 29.9 | 6.7 | 3.1 | 3.8 | 64.0 | 3.4 |
| Tailing Zinc cleaner | 91.8 | 4.7 | 0.24 | 37.3 | 0.8 | 0.1 | 2.8 | 14 | 3.9 | 7.7 | 2.8 | 3.6 | 5.3 | 10.4 |
| Tailing Zinc rougher | 1,719.9 | 87.5 | 0.07 | 5.1 | 0.1 | 0.0 | 0.1 | 5.5 | 22.9 | 19.6 | 4.1 | 8.6 | 4.4 | 76.6 |
| Flotation Feed | 1,966.1 | 100 | 0.29 | 22.7 | 1.35 | 0.11 | 2.4 | 6.28 | 100 | 100 | 100 | 100 | 100 | 100 |
| **Year 3 Time Variation Composite** | | | | | | | | | | | | | | |
| Copper Concentrate | 143.9 | 7.1 | 5.45 | 2,173.5 | 29.2 | 2.3 | 4.7 | 21.5 | 40.6 | 73.9 | 86.3 | 75.2 | 6.8 | 6.7 |
| Zinc concentrate | 148 | 7.3 | 3.14 | 163.6 | 0.7 | 0.3 | 57.8 | 3.4 | 24.0 | 5.7 | 2.2 | 9.5 | 85.1 | 1.1 |
| Tailing Zinc cleaner | 150.6 | 7.4 | 1.42 | 202.4 | 1.3 | 0.2 | 3.1 | 29.0 | 11.1 | 7.2 | 4.0 | 5.1 | 4.6 | 9.5 |
| Tailing Zinc rougher | 1,591.8 | 78.2 | 0.30 | 35.2 | 0.2 | 0.0 | 0.2 | 23.9 | 24.3 | 13.2 | 7.5 | 10.1 | 3.5 | 82.7 |
| Flotation Feed | 2,034.3 | 100 | 0.95 | 208.2 | 2.39 | 0.22 | 4.94 | 22.58 | 100 | 100 | 100 | 100 | 100 | 100 |
| **Year 4 Time Variation Composite** | | | | | | | | | | | | | | |
| Copper Concentrate | 152.02 | 7.6 | | 1,522.4 | 26.8 | 0.9 | 4.8 | 29.7 | | 76.7 | 89.2 | 31.3 | 7.5 | 12.9 |
| Zinc concentrate | 140.8 | 7.1 | | 147.8 | 0.5 | 1.4 | 55.2 | 4.3 | | 6.9 | 1.7 | 47.0 | 81.1 | 1.7 |
| Tailing Zinc cleaner | 140.5 | 7.1 | | 145.2 | 1.3 | 0.3 | 5.8 | 29.1 | | 6.8 | 4.0 | 11.8 | 8.5 | 11.7 |
| Tailing Zinc rougher | 1,559 | 78.3 | | 18.7 | 0.2 | 0.0 | 0.2 | 16.4 | | 9.7 | 5.2 | 9.8 | 2.9 | 73.6 |
| Flotation Feed | 1,992.32 | 100 | | 151.48 | 2.29 | 0.21 | 4.8 | 17.48 | | 100 | 100 | 100 | 100 | 100 |
| **Year 5 Time Variation Composite** | | | | | | | | | | | | | | |
| Copper Concentrate | 105.5 | 5.3 | 1.73 | 198.6 | 28.5 | 2.7 | 4.9 | 26.9 | 22.3 | 38.8 | 83.5 | 62.8 | 3.1 | 6.8 |
| Zinc concentrate | 247.1 | 12.4 | 0.38 | 34.0 | 0.3 | 0.4 | 61.0 | 2.0 | 11.5 | 15.6 | 2.2 | 19.9 | 92.2 | 1.2 |
| Tailing Zinc cleaner | 173.4 | 8.7 | 1.06 | 46.0 | 1.3 | 0.2 | 2.3 | 32.9 | 22.5 | 14.8 | 6.4 | 7.6 | 2.4 | 13.77 |
| Tailing Zinc rougher | 1,458.8 | 73.5 | 0.25 | 11.5 | 0.2 | 0.0 | 0.2 | 22.2 | 43.7 | 30.9 | 7.9 | 9.7 | 2.2 | 78.18 |
| Flotation Feed | 1,984.8 | 100 | 0.41 | 27.24 | 1.81 | 0.23 | 8.23 | 20.88 | 100 | 100 | 100 | 100 | 100 | 100 |

Source: Capstone (2010b).

The reagent types and usages used in each of the locked cycle tests are shown in Table 13-5.

**Table 13-5: Locked Cycle Reagent Usage Summary**

| Reagent | Usage (g/t) |
|---|---|
| Lime | 1,700 |
| Promoter 7583 | 50 |
| Promoter 3477 | 10 |
| Copper Sulphate ($CuSO_4$) | 550 |
| $Na_2SO_3$ | 1,250 |
| Frother 1064 | 30 |
| Sodium Silicate | 450 |
| Ammonium Bisulphate | 500 |

Source: Capstone (2010b).

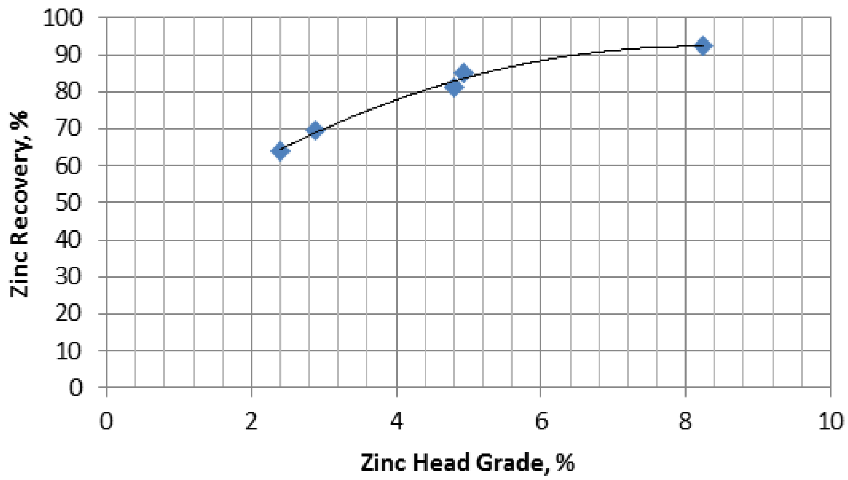
## 13.4 Analysis of Results

### 13.4.1 December 2010 Locked Cycle Tests

Time variation composites for project years 1 to 5, the Main global composite, and the corresponding LCT results were analyzed and the pertinent information is summarized below.

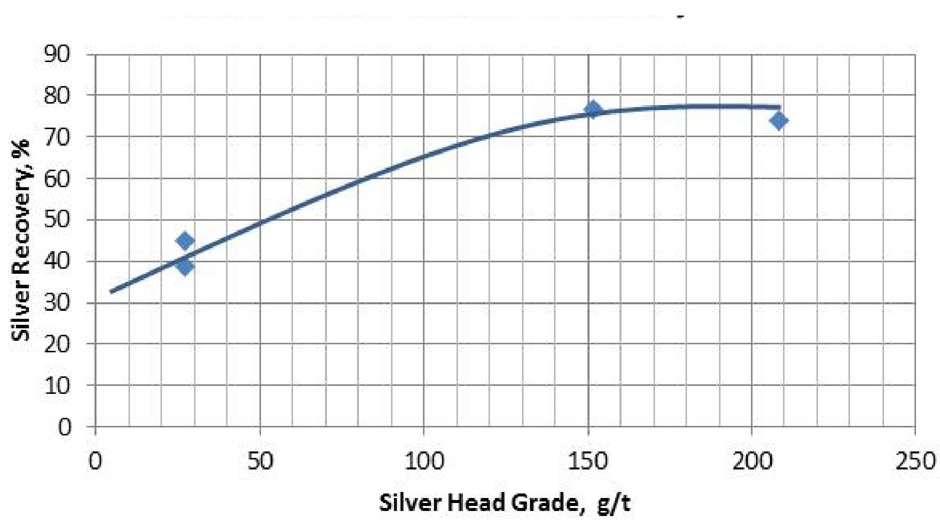
- The year 3 to 5 time variation composites have higher head grades than anticipated in the mine plan.
- A statistically strong relationship between head grade and recovery was not determined for copper and gold based on the LCT.
- The LCT show a correlation between head grade and zinc and silver recoveries and the graphs are shown Figures 13-4 and 13-5.

**Figure 13-4: Zinc Head Grade versus Recovery**




Source: JDS (2017).

**Figure 13-5: Silver Head Grade versus Recovery**




Source: JDS (2017).

## 13.4.2 Life of Mine Grades and Recoveries

The results from the LCT were used to predict the yearly recoveries based on the LOM plan and are shown in Table 13-6.

**Table 13-6: Life of Mine Grade and Recovery Projection**

| | Unit | Year | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | -1 | 1 | 2 | 3 | 4 | 5 | 6 | 7 | 8 | 9 | 10 | 11 | 12 |
| **Days per Year** | - | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 162 |
| Pit Production | kt | - | 446 | - | - | - | - | - | - | - | - | - | - | - |
| Main Production | kt | - | 466 | 913 | 674 | 365 | 365 | 365 | 459 | 913 | 913 | 913 | 913 | 404 |
| Esso Production | kt | - | - | - | 239 | 548 | 548 | 548 | 454 | - | - | - | - | - |
| **Total Ore Production** | kt | **-** | **913** | **913** | **913** | **913** | **913** | **913** | **913** | **913** | **913** | **913** | **913** | **404** |
| **Tonnes per Day** | t/d | - | 2,500 | 2,500 | 2,500 | 2,500 | 2,500 | 2,500 | 2,500 | 2,500 | 2,500 | 2,500 | 2,500 | 2,492 |
| **Cu Grade** | % | - | 1.94 | 2.13 | 2.01 | 2.02 | 2.26 | 2.12 | 2.06 | 1.88 | 1.91 | 2.07 | 1.81 | 1.87 |
| **Zn Grade** | % | - | 1.92 | 2.62 | 2.91 | 3.71 | 5.30 | 4.41 | 3.76 | 2.64 | 3.06 | 2.78 | 2.43 | 2.27 |
| **Ag Grade** | g/t | - | 26.4 | 31.0 | 42.3 | 47.2 | 41.5 | 46.9 | 35.2 | 27.6 | 29.2 | 28.0 | 27.3 | 30.1 |
| **Au Grade** | g/t | - | 0.30 | 0.32 | 0.44 | 0.47 | 0.45 | 0.47 | 0.35 | 0.36 | 0.31 | 0.32 | 0.27 | 0.39 |
| Cu Recovery | % | - | 86.1 | 86.1 | 86.3 | 89.2 | 83.5 | 86.4 | 86.4 | 82.1 | 82.1 | 82.1 | 82.1 | 82.1 |
| Au Recovery | % | - | 42.2 | 42.2 | 40.6 | 41.1 | 41.1 | 41.1 | 41.1 | 41.1 | 41.1 | 41.1 | 41.1 | 41.1 |
| Ag Recovery | % | - | 55.5 | 55.5 | 45.0 | 50.0 | 48.0 | 49.0 | 46.0 | 45.0 | 45.0 | 45.0 | 45.0 | 45.0 |
| Zn Recovery | % | - | 66.8 | 66.8 | 69.1 | 81.1 | 85.4 | 83.3 | 83.3 | 69.5 | 69.5 | 69.5 | 69.5 | 69.5 |
| **Cu Concentrate** | % Cu | - | 26.4 | 26.4 | 29.2 | 26.8 | 28.5 | 27.7 | 27.7 | 27.7 | 27.7 | 27.7 | 27.7 | 27.7 |
| | % Zn | - | 10.7 | 10.7 | 4.7 | 4.8 | 4.9 | 4.9 | 4.9 | 8.5 | 8.5 | 8.5% | 8.5% | 8.5 |
| | **Ag g/t (calc)** | | 231 | 248 | 320 | 351 | 301 | 347 | 252 | 223 | 232 | 205 | 229 | 245 |
| | **Au g/t (calc)** | - | 1.97 | 1.97 | 3.01 | 2.91 | 2.82 | 2.90 | 2.26 | 2.62 | 2.22 | 2.15 | 2.09 | 2.86 |
| | **Cu tonnes** | - | 57,880 | 63,377 | 54,324 | 61,260 | 60,401 | 60,450 | 58,701 | 50,949 | 51,651 | 56,107 | 48,887 | 22,376 |
| | **Ag kg** | - | 13,376 | 15,701 | 17,386 | 21,531 | 18,171 | 20,957 | 14,785 | 11,351 | 11,996 | 11,484 | 11,217 | 5,476 |
| | **Au kg** | - | 114 | 125 | 164 | 178 | 170 | 176 | 133 | 133 | 115 | 120 | 102 | 64 |
| **Zn Concentrate** | % Zn | - | 52.9% | 52.9% | 54.0% | 55.2% | 57.5% | 56.3% | 56.3% | 54.0% | 54.0% | 54.0% | 54.0% | 54.0% |
| | % Cu | - | 1.5% | 1.5% | 1.6% | 0.5% | 0.3% | 0.4% | 0.4% | 1.6% | 1.6% | 1.6% | 1.6% | 1.6% |
| | **Zn tonnes** | - | 22,162 | 30,147 | 33,994 | 49,791 | 71,857 | 59,526 | 50,720 | 31,034 | 35,908 | 32,619 | 28,486 | 11,792 |

Source: JDS (2017).

The following assumptions were used to predict the grades and recoveries shown in Table 13-6:

- In the mine plan year 1 includes open pit and underground material and year 2 underground material both from Main zone. years 1 and 2 are best represented by main global composite and time variation composite - year 1. An average of grades and recoveries from the corresponding LCT were used for years 1 and 2.
- Years 3 to 5 include Main and Esso material. The zinc and silver head grades of the time variation composites used for years 3 to 4 were higher than anticipated in the mine plan. The correlation between head grade and recovery derived from the LCT were used to predict the new recoveries.
- Test work was not carried out for years 6 and 7. The grades and recoveries are based on an average of years 4 and 5.
- The main global composite LCT results were used for the years 8 through 12 grades and recoveries.

## 13.5 Life of Mine Average Metallurgical Recoveries

The results from the December 2010 LCT were used to determine the LOM average grades and recoveries and are summarized in Table 13-7.

**Table 13-7: Preliminary Recover Projections**

| Metal | Unit | Recovery | Cu Concentrate Grade | Zn Concentrate Grade |
|---|---|---|---|---|
| Cu | % | 84.7 | 27.6 | 1.2 |
| Zn | % | 75.7 | 7.3 | 55.1 |
| Au | g/t | 41.2 | 2.5 | - |
| Ag | g/t | 48.0 | 268.6 | - |

Note: Zinc in the copper concentrate will likely incur a minor smelter penalty. Additional test work to reduce the zinc in the copper concentrate and a multi-element ICP scan to identify any deleterious elements in the concentrates that may encounter smelter penalties is recommended in the next stage of engineering.

Source: JDS (2017).

# 14 Mineral Resource Estimates

## 14.1 Introduction

The following sections detail the methods, processes and strategies employed in creating the mineral resource estimate for the Kutcho deposits. This mineral resource estimate entails an update of the mineral resource estimate for the Main and Esso zones which includes all historic drilling along with the most recent data from the 2008, 2010 and 2011 drilling campaigns. In addition, the resources for the Sumac zone are included in the report to cover the complete Kutcho deposit area.

## 14.2 Data Evaluation

A total of 482 drillholes and one adit were supplied for the Kutcho property which is the combined drillholes for the Main, Esso, and Sumac zones. The database consists of all holes prior to the drilling performed by WKM along with the drilling performed in 2004 for 40 drillholes, 2005 for 27 drillholes, 2006 for 23 drillholes and 81 drillholes from 2008 drilled by Kutcho Copper. In addition, there were 34 holes drilled in the Esso deposit in 2010 and 20 drillholes in 2011, one of which was drilled into the Main, one into Esso deposit and six into Sumac. The drillholes within the database included collars, downhole surveys, assays, and lithology.

The drillhole database (metric) was supplied in electronic format by Kutcho Copper (subsidiary of Capstone). This included collars, downhole surveys, lithology data and assay data (e.g., Au g/t, Cu%, Zn%, Ag g/t, S%, SG, and an alpha-numeric lithology code) with downhole from and to intervals in metric units. The assay database included 33 element ICP analysis following an aqua regia digestion. In addition to multi-element ICP and AA assays for Cu, Zn, Ag, and Pb which was used unless values exceeded ore grade thresholds. If either copper or zinc reported over 0.25%, then ore grade analysis was performed using aqua regia digestion followed by atomic absorption and this value took precedence over the prior sample analysis. The analysis for Au included fire assay by AA, ICP and gravimetric finish which was used as the Au value.

The drillhole database was numerically coded by mineralized zone solid; Main zone ore = 3, Esso zone ore = 4, Sumac zone ore = 5. The database was then manually adjusted drillhole by drillhole to ensure accuracy of zonal intercepts.

Simple statistics for the assay data are shown in Table 14-1, which shows statistics for copper, zinc, silver, and gold assays along with sulphur and SG values weighted by assay interval.

## Table 14-1: Weighted Copper, Zinc, Silver, and Gold Assay Statistics

| | Number | Length (m) | Max | Mean | Std. Devn. | CV |
|---|---|---|---|---|---|---|
| **Main** | | | | | | |
| Cu grade (%) | 3,623 | 4338.4 | 35.50 | 1.63 | 2.05 | 1.1 |
| Zn grade (%) | 3,623 | 4338.4 | 47.15 | 2.16 | 3.23 | 1.5 |
| Ag grade (g/t) | 3,623 | 4338.4 | 942.86 | 27.83 | 38.00 | 1.4 |
| Au grade (g/t) | 3,623 | 4338.4 | 17.00 | 0.30 | 0.69 | 2.3 |
| S grade (%) | 3,623 | 4338.4 | 56.31 | 18.88 | 15.59 | 0.8 |
| SG | 3,575 | 4290.3 | 6.00 | 3.71 | 0.72 | 0.2 |
| **Esso** | | | | | | |
| Cu grade (%) | 639 | 779.0 | 21.80 | 2.24 | 3.10 | 1.2 |
| Zn grade (%) | 639 | 779.0 | 49.64 | 4.49 | 8.50 | 1.9 |
| Ag grade (g/t) | 639 | 779.0 | 4,470.00 | 66.59 | 225.87 | 3.4 |
| Au grade (g/t) | 639 | 779.0 | 160.00 | 0.92 | 7.07 | 7.7 |
| S grade (%) | 305 | 471.2 | 49.00 | 9.30 | 11.15 | 1.2 |
| SG | 294 | 455.7 | 4.78 | 3.26 | 0.48 | 0.1 |
| **Sumac** | | | | | | |
| Cu grade (%) | 265 | 317.3 | 5.18 | 0.82 | 0.88 | 1.0 |
| Zn grade (%) | 265 | 317.3 | 11.50 | 1.19 | 1.27 | 1.1 |
| Ag grade (g/t) | 265 | 317.3 | 122.00 | 12.90 | 16.43 | 1.3 |
| Au grade (g/t) | 265 | 317.3 | 1.42 | 0.13 | 0.15 | 1.2 |
| S grade (%) | 265 | 317.3 | 55.00 | 24.64 | 22.86 | 0.9 |
| SG | 213 | 250.9 | 4.91 | 4.12 | 1.06 | 0.3 |
| **All Min Zones** | | | | | | |
| Cu grade (%) | 4,527 | 5434.6 | 35.50 | 1.67 | 2.21 | 1.2 |
| Zn grade (%) | 4,527 | 5434.6 | 49.64 | 2.44 | 4.42 | 1.8 |
| Ag grade (g/t) | 4,527 | 5434.6 | 4,470.00 | 32.51 | 93.21 | 2.9 |
| Au grade (g/t) | 4,527 | 5434.6 | 160.00 | 0.38 | 2.76 | 7.2 |
| S grade (%) | 4,193 | 5126.9 | 56.31 | 18.35 | 16.11 | 0.9 |
| SG | 4,082 | 4996.9 | 6.00 | 3.69 | 0.74 | 0.2 |
| **All Assays** | | | | | | |
| Cu grade (%) | 9,652 | 13407.5 | 35.50 | 0.72 | 1.75 | 2.1 |
| Zn grade (%) | 9,652 | 13407.5 | 49.64 | 1.08 | 3.10 | 2.9 |
| Ag grade (g/t) | 9,652 | 13407.5 | 4470.00 | 14.40 | 61.52 | 4.3 |
| Au grade (g/t) | 9,652 | 13407.5 | 160.00 | 0.17 | 1.77 | 10.3 |
| S grade (%) | 8,399 | 11683.2 | 56.31 | 12.01 | 13.95 | 1.2 |
| SG | 7,640 | 10438.1 | 6.00 | 3.30 | 0.77 | 0.2 |

Source: JDS (2017).

The assay Au, Cu, Zn, and Ag database (Au, Cu, Zn, and Ag values) shows that Au, Cu, Zn, and Ag distributions are very well behaved, still with several samples in each case representing an outlier population. The respective mean for Cu, Zn, Ag, and Au grade (weighted by sample length) for the Main zone is 1.63% Cu, 2.16% Zn, 27.83 g/t Ag, and 0.30 g/t Au, with modest to low coefficient of variations of 1.1, 1.5, 1.4, and 2.3, respectively. This indicates a relatively modest scatter of the raw data values. Sulphur values are very high up to a maximum of 56.3% and averaging 18.9%. SGs at Main average 3.17 with the maximum being 6.00.

The coefficient of variation is defined as CV=σ/m (standard deviation/mean), and represents a measure of variability that is unit-independent. This is a variability index that can be used to compare different and unrelated distributions. Au has a CV that is relatively high; however, this is not too unexpected given the nuggety nature of the gold.

The Esso zone has mean Cu, Zn, Ag, and Au grades of 2.24%, 4.49%, 66.6 g/t, 0.92 g/t and low CVs of 1.2, 1.9, 3.4, and 7.7, respectively. This is a significant increase in relation to the previous drilling within the Esso zone. This is a result of infill drilling targeting a higher-grade region within the Esso zone which resulted in the delineation of a high grade core. In addition, it is important to note that the CV for Ag and Au has increased significantly within the Esso deposit as a result of a number of relatively high-grade silver and gold assays from the 2010 drilling campaign. Sulphur content remains relatively high with the mean being 9.3% and maximum of 49%. In addition, SGs range up to 4.78 and average at 3.26.

The Sumac zone has mean Cu, Zn, Ag, and Au grades of 0.82%, 1.19%, 12.9 g/t, 0.13 g/t, and very low CV's of 1.0, 1.1, 1.3, and 1.2, respectively. Sulphur content remains relatively high with the mean being 24.6% and maximum of 55%. In addition, SGs range up to 4.91 and average at 4.12.

## 14.3 Topography

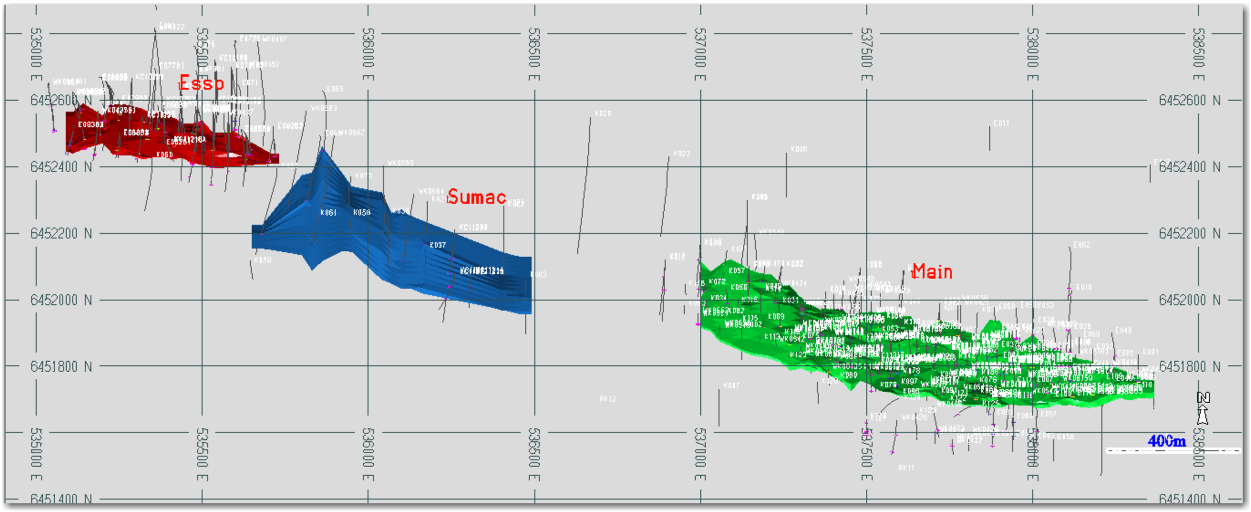
Topography was imported from an AutoCAD topographic map supplied by Kutcho Copper in DXF format. The surfaces were generated from a 1 m topographic map created by McElhanney Engineering Services Limited. The topography was surveyed and is believed to be accurate. Checks against drillhole collars illustrate accuracy to within 1 m.

## 14.4 Computerized Geologic Modeling

A solid model of the Main and Esso ore zones within the Kutcho deposit were created from sections and based on a combination of lithology, copper grades and site knowledge. It is important to note that the 2008, 2010, and subsequent 2011 drilling resulted in new insights into the mineralization and grade distribution which assisted in the creation of the solids. These ore zone solids were then used for constraining the interpolation procedure.

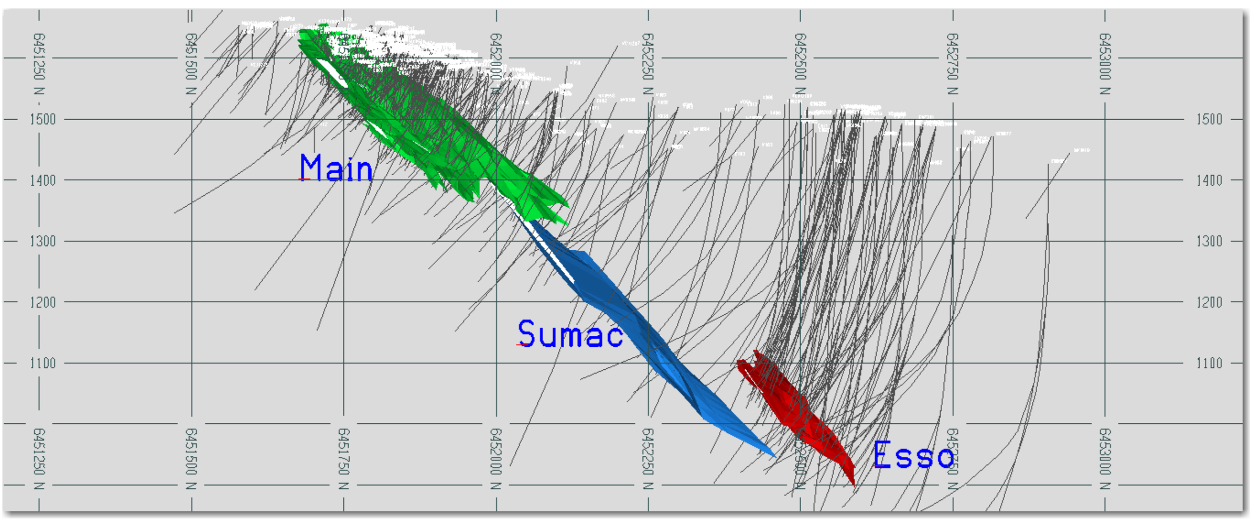
All intersections were inspected and the solids were then manually adjusted to match exactly the interval intercepts. Once the solids models were created, they were used to code the drillhole assays and composites for subsequent statistical and geostatistical analysis. For the purpose of the resource model, the solid zone was utilized to constrain the block model by matching assays to those within the zones in a process called geologic matching so that only composites that lie within a particular zone are used to interpolate the blocks within that zone. Figures 14-1 and 14-2 illustrate the completed zone solids model.

**Figure 14-1: 3D Plan View of Drillholes and Geology Solids**




Source: JDS (2017).

**Figure 14-2: 3D Section View of Geology Solids Looking West**




Source: JDS (2017).

## 14.5 Composites

It was confirmed that the 1.5 m composites for the Esso deposit and 2.5 m composite lengths for the Main and the Sumac deposits offered the best balance between supplying common support for samples and minimizing the smoothing of the grades in addition to reducing the effect of high grades to a small extent. Tables 14-2 and 14-3 shows the basic statistics for the 1.5 m and 2.5 m composites for Cu, Zn, Ag, and Au in addition to sulphur and SG, respectively for the Main, Esso, and Sumac zones. Note that in all cases the CV is substantially reduced indicating less variability in the composite data which is to be expected when smoothing.

Figures 14-3 through 14-26 illustrate the histograms and cumulative distribution plots for copper, zinc, silver, and gold grade by ore zone. For the most part, the mineralization within the zones demonstrates a normal distribution.

### Table 14-2: 1.5 m Composite Statistics Weighted by Length

| | # | Length (m) | Max | Mean | CV |
|---|---|---|---|---|---|
| **Main** | | | | | |
| Cu grade (%) | 2,901 | 4,338.4 | 14.34 | 1.63 | 0.9 |
| Zn grade (%) | 2,901 | 4,338.4 | 38.40 | 2.16 | 1.2 |
| Ag grade (g/t) | 2,901 | 4,338.4 | 367.20 | 27.83 | 1.1 |
| Au grade (g/t) | 2,901 | 4,338.4 | 13.75 | 0.30 | 1.9 |
| S grade (%) | 2,901 | 4,338.4 | 54.86 | 18.88 | 0.8 |
| SG | 2,877 | 4,299.4 | 5.11 | 3.71 | 0.2 |
| **Esso** | | | | | |
| Cu grade (%) | 531 | 779.0 | 16.08 | 2.21 | 1.1 |
| Zn grade (%) | 531 | 779.0 | 49.25 | 4.49 | 1.7 |
| Ag grade (g/t) | 531 | 779.0 | 1310.69 | 63.18 | 1.9 |
| Au grade (g/t) | 531 | 779.0 | 160.00 | 0.92 | 7.7 |
| S grade (%) | 323 | 471.2 | 48.70 | 9.30 | 1.1 |
| SG | 313 | 455.7 | 4.78 | 3.26 | 0.1 |
| **Sumac** | | | | | |
| Cu grade (%) | 220 | 317.3 | 4.23 | 0.81 | 0.9 |
| Zn grade (%) | 220 | 317.3 | 6.93 | 1.19 | 1.0 |
| Ag grade (g/t) | 220 | 317.3 | 87.13 | 12.90 | 1.1 |
| Au grade (g/t) | 220 | 317.3 | 0.86 | 0.13 | 1.1 |
| S grade (%) | 220 | 317.3 | 55.00 | 24.64 | 0.9 |
| SG | 167 | 242.3 | 4.91 | 4.26 | 0.2 |
| **All Min Zones** | | | | | |
| Cu grade (%) | 3,652 | 0.0 | 16.08 | 1.67 | 1.0 |
| Zn grade (%) | 3,652 | 5,434.6 | 49.25 | 2.44 | 1.6 |
| Ag grade (g/t) | 3,652 | 5,434.6 | 1310.69 | 32.02 | 1.7 |
| Au grade (g/t) | 3,652 | 5,434.6 | 160.00 | 0.38 | 7.2 |
| S grade (%) | 3,444 | 5,126.9 | 55.00 | 18.35 | 0.8 |
| SG | 3,357 | 4,997.4 | 5.11 | 3.70 | 0.2 |

### Table 14-2: Composite Statistics Weighted by Length (continued)

| | # | Length (m) | Max | Mean | CV |
|---|---|---|---|---|---|
| **All Composites** | | | | | |
| Cu grade (%) | 9,260 | 13,271.8 | 16.08 | 0.71 | 1.9 |
| Zn grade (%) | 9,260 | 13,271.8 | 49.25 | 1.08 | 2.5 |
| Ag grade (g/t) | 9,260 | 13,271.8 | 1310.69 | 14.31 | 2.7 |
| Au grade (g/t) | 9,260 | 13,271.8 | 160.00 | 0.17 | 10.2 |
| S grade (%) | 8,156 | 11,683.2 | 55.00 | 12.01 | 1.1 |
| SG | 7,325 | 10,432.5 | 5.11 | 3.30 | 0.2 |

Source: JDS (2017).

### Table 14-3: 2.5 m Composite Statistics Weighted by Length

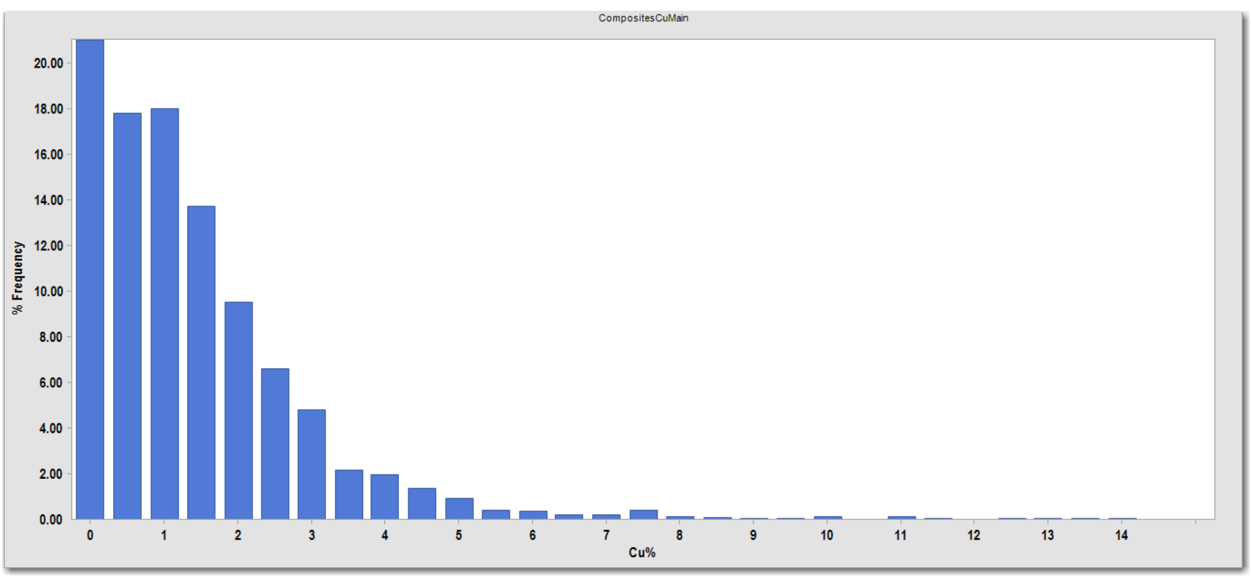
| | # | Length (m) | Max | Mean | CV |
|---|---|---|---|---|---|
| **Main** | | | | | |
| Cu grade (%) | 1,753 | 4,338 | 13.71 | 1.632 | 0.8 |
| Zn grade (%) | 1,753 | 4,338 | 25.08 | 2.159 | 1.1 |
| Ag grade (g/t) | 1,753 | 4,338 | 355.34 | 27.826 | 1.0 |
| Au grade (g/t) | 1,753 | 4,338 | 13.501 | 0.3028 | 1.8 |
| S grade (%) | 1,753 | 4,338 | 52.3 | 18.876 | 0.8 |
| SG | 1,741 | 4,338 | 4.93 | 3.709 | 0.2 |
| **Esso** | | | | | |
| Cu grade (%) | 324 | 779 | 15.2 | 2.238 | 1.0 |
| Zn grade (%) | 324 | 779 | 49.26 | 4.485 | 1.6 |
| Ag grade (g/t) | 324 | 779 | 1,310.69 | 63.229 | 1.7 |
| Au grade (g/t) | 324 | 779 | 96.223 | 0.9168 | 6.0 |
| S grade (%) | 198 | 471 | 37.02 | 9.298 | 1.1 |
| SG | 192 | 456 | 4.47 | 3.259 | 0.1 |
| **Sumac** | | | | | |
| Cu grade (%) | 135 | 317 | 2.91 | 0.817 | 0.8 |
| Zn grade (%) | 135 | 317 | 6.86 | 1.189 | 0.9 |
| Ag grade (g/t) | 135 | 317 | 73.16 | 12.903 | 1.1 |
| Au grade (g/t) | 135 | 317 | 0.698 | 0.1284 | 1.0 |
| S grade (%) | 135 | 317 | 55 | 24.641 | 0.9 |

### Table 14-3: 2.5 m Composite Statistics Weighted by Length (continued)

| | # | Length (m) | Max | Mean | CV |
|---|---|---|---|---|---|
| SG | 108 | 251 | 4.91 | 4.118 | 0.2 |
| **All Min Zones** | | | | | |
| Cu grade (%) | 2,212 | 5,435 | 15.2 | 1.672 | 0.9 |
| Zn grade (%) | 2,212 | 5,435 | 49.26 | 2.436 | 1.4 |
| Ag grade (g/t) | 2,212 | 5,435 | 1,310.69 | 32.029 | 1.6 |
| Au grade (g/t) | 2,212 | 5,435 | 96.223 | 0.3806 | 5.6 |
| S grade (%) | 2,086 | 5,435 | 55 | 18.353 | 0.8 |
| SG | 2,041 | 5,012 | 4.93 | 3.689 | 0.2 |
| **All Composites** | | | | | |
| Cu grade (%) | 5,724 | 13,272 | 15.2 | 0.729 | 1.7 |
| Zn grade (%) | 5,724 | 13,272 | 49.26 | 1.083 | 2.3 |
| Ag grade (g/t) | 5,724 | 13,272 | 1,310.69 | 14.315 | 2.5 |
| Au grade (g/t) | 5,724 | 13,272 | 96.223 | 0.1719 | 8.0 |
| S grade (%) | 5,043 | 13,272 | 55 | 12.013 | 1.1 |
| SG | 4,548 | 13,272 | 4.93 | 3.299 | 0.2 |

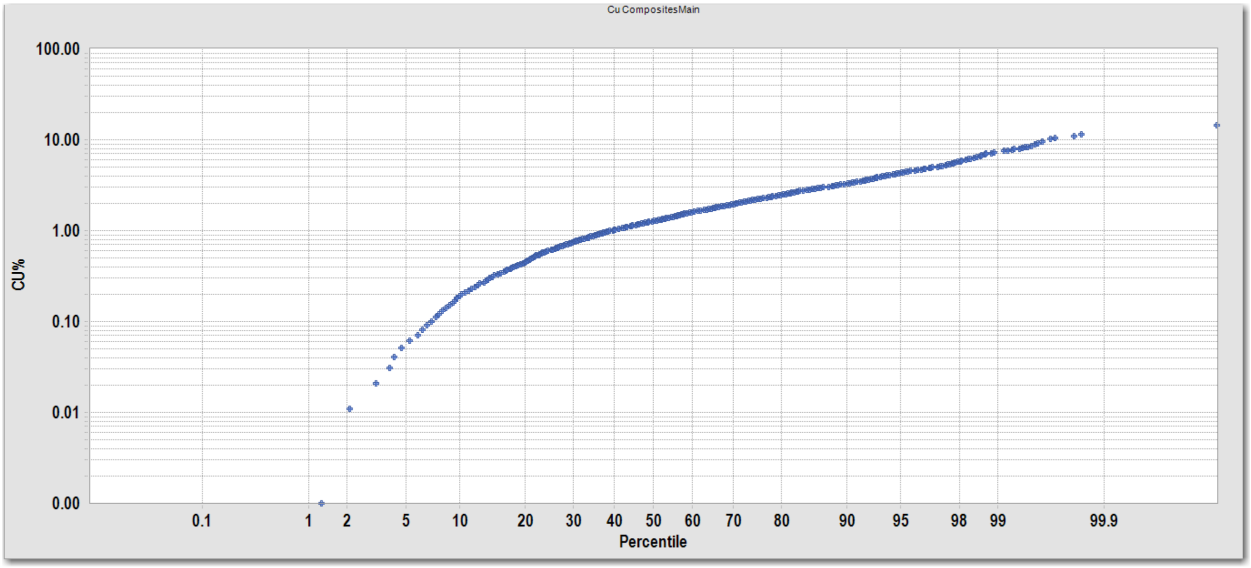
Source: JDS (2017).

### Figure 14-3: Histogram for Cu, 2.5 m Composites for Main Zone



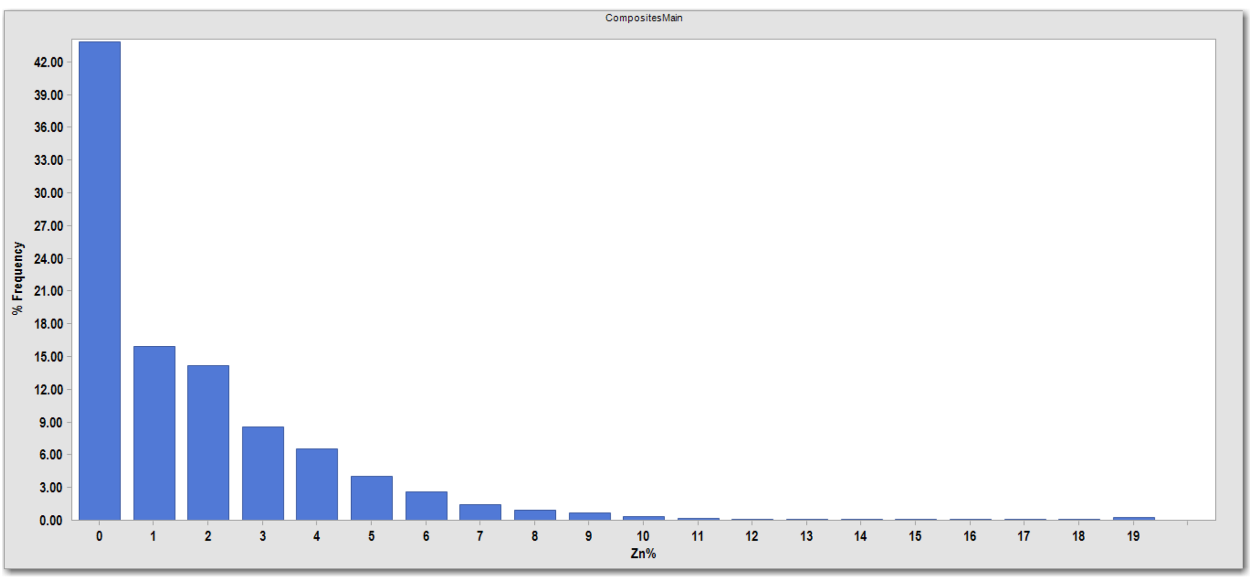

Source: JDS (2017).

## Figure 14-4: Cumulative Distribution Plot for Cu, 2.5 m Composites for Main Zone



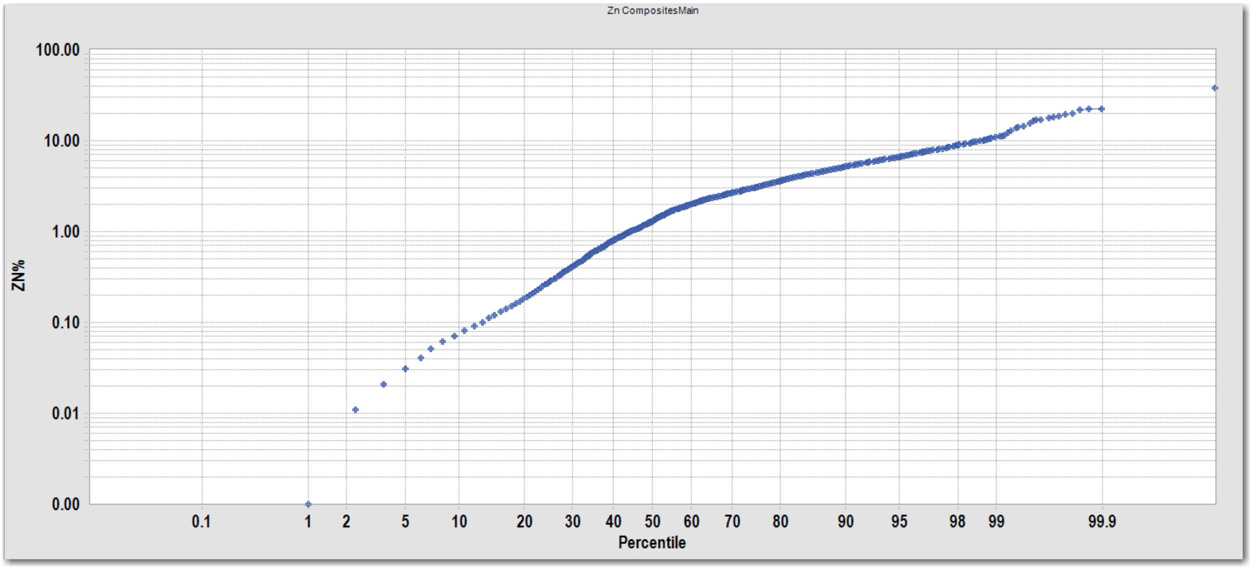

Source: JDS (2017).

## Figure 14-5: Histogram for Zn, 2.5 m Composites for Main Zone



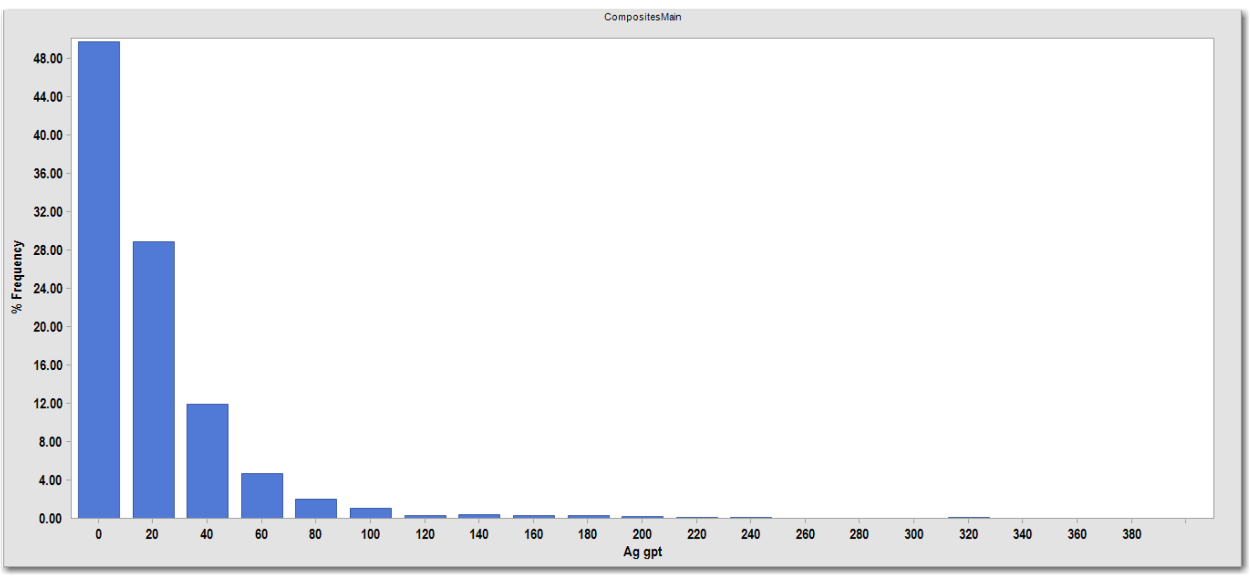

Source: JDS (2017).

**Figure 14-6: Cumulative Distribution Plot of Zn, 2.5 m Composites for Main Zone**



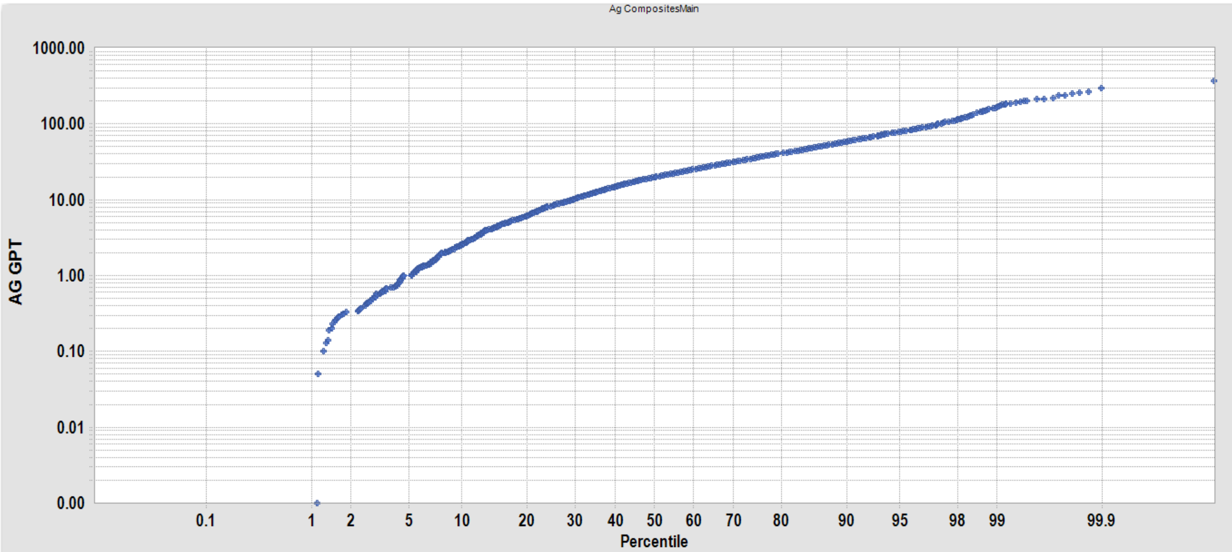

Source: JDS (2017).

**Figure 14-7: Histogram for Ag, 2.5 m Composites for Main Zone**



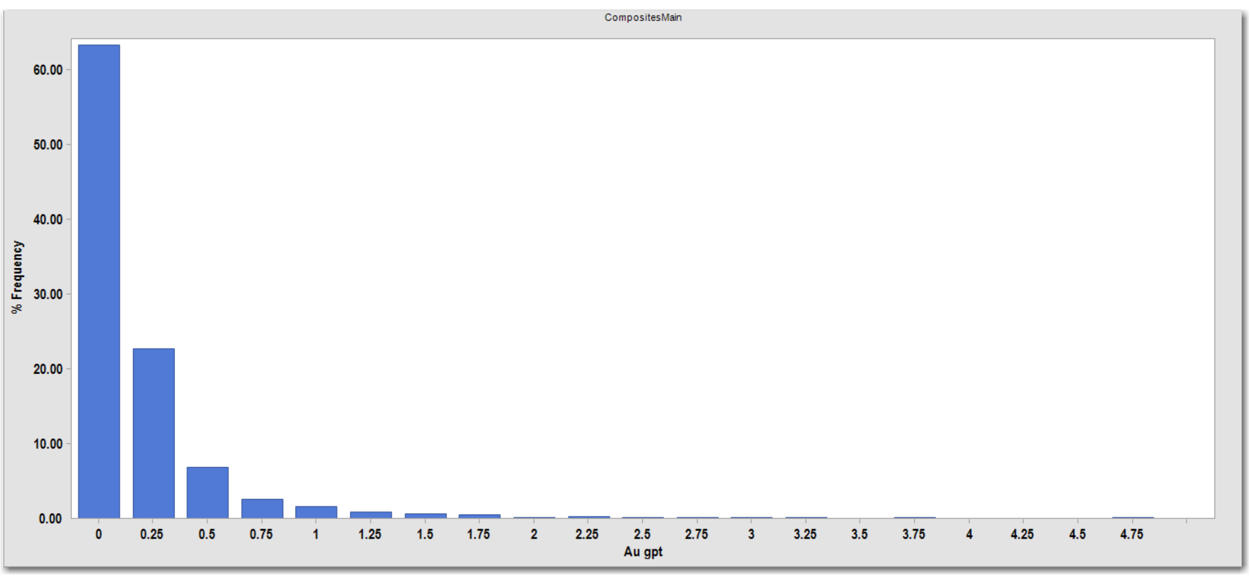

Source: JDS (2017).

**Figure 14-8: Cumulative Distribution Plot for Ag, 2.5 m Composites for Main Zone**



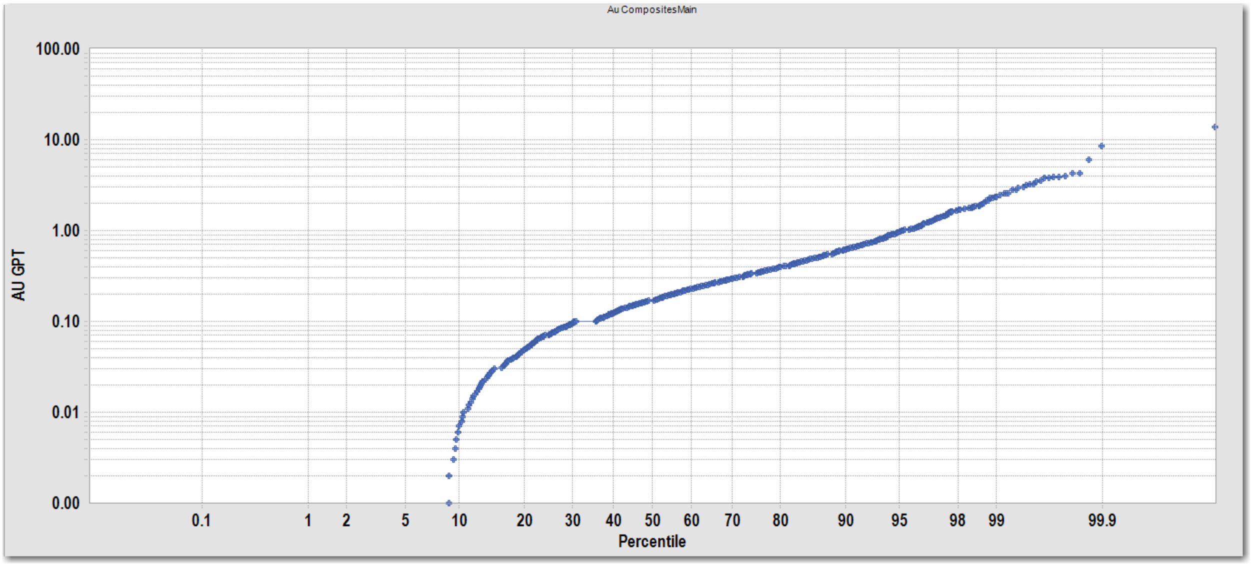

Source: JDS (2017).

**Figure 14-9: Histogram for Au, 2.5 m Composites for Main Zone**



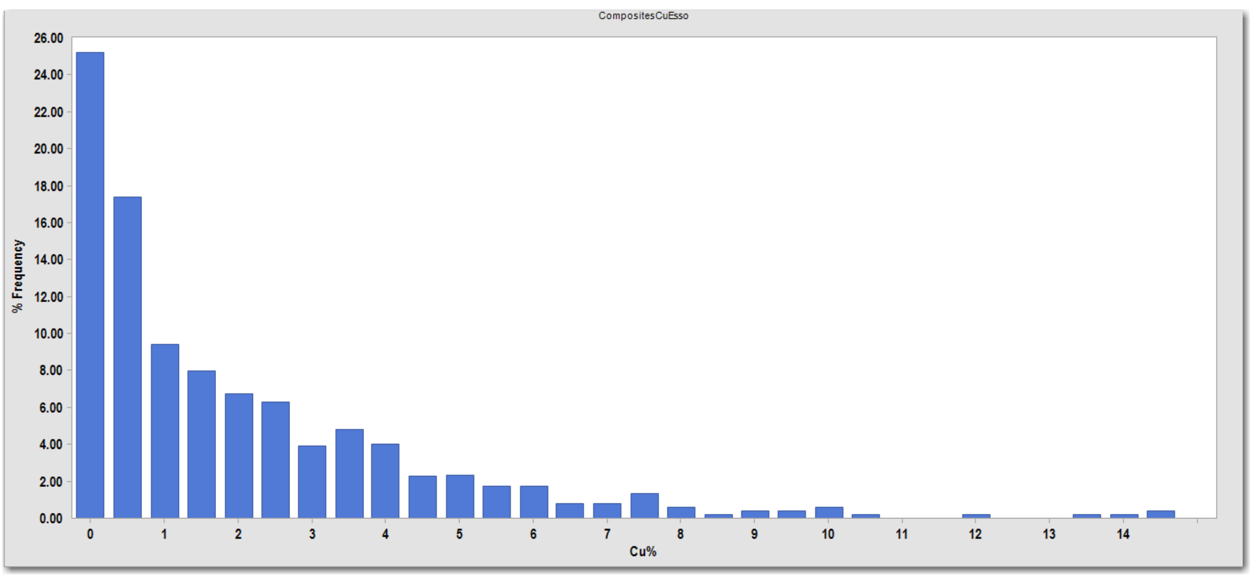

Source: JDS (2017).

### Figure 14-10: Cumulative Distribution Plot for Au, 2.5 m Composites for Main Zone



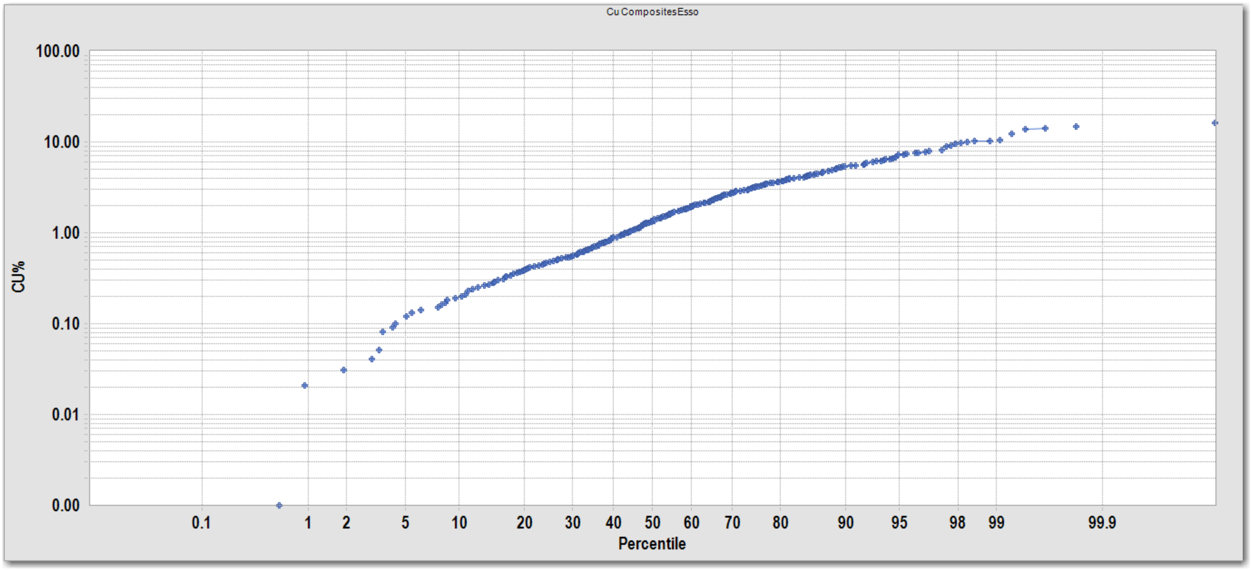

Source: JDS (2017).

### Figure 14-11: Histogram for Cu, 1.5 m Composites for Esso Zone



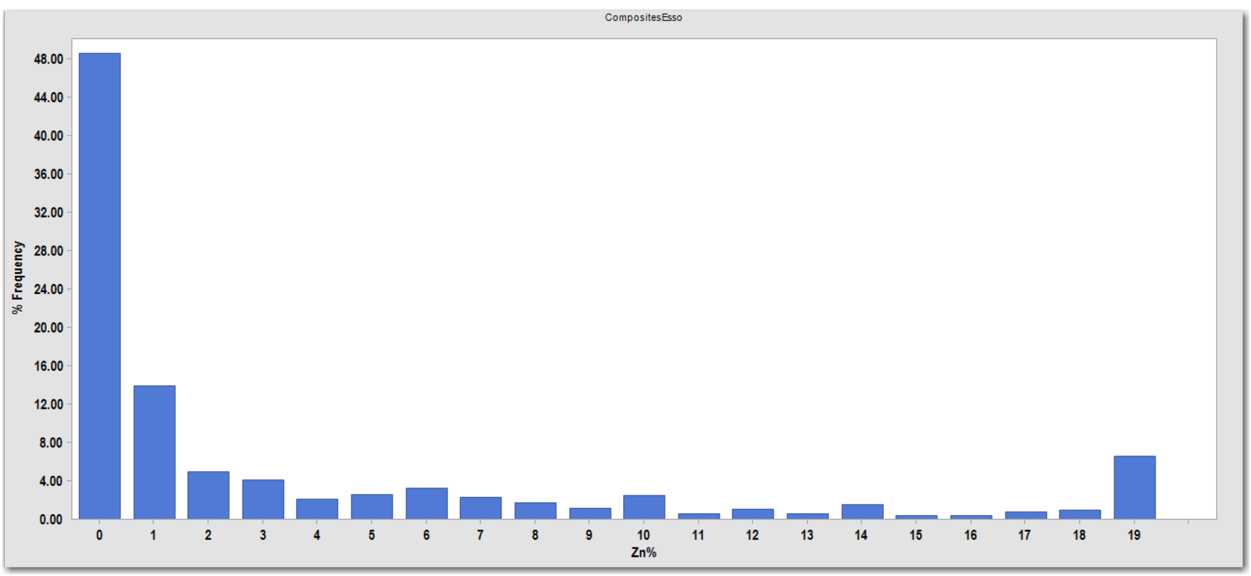

Source: JDS (2017).

## Figure 14-12: Cumulative Distribution Plot for Cu, 1.5 m Composites for Esso Zone



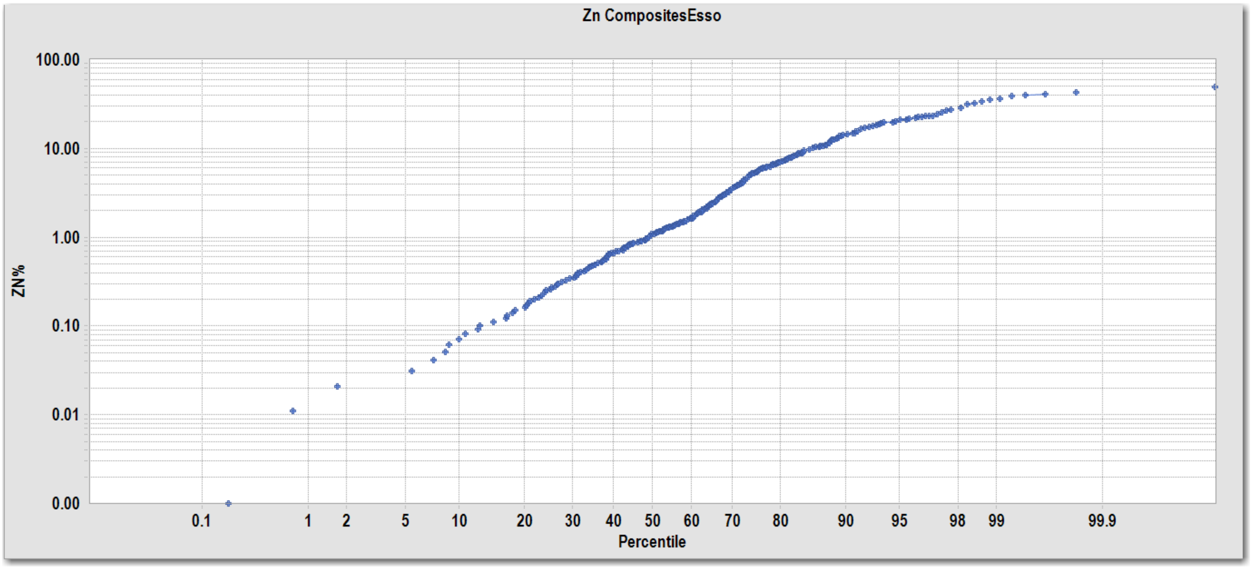

Source: JDS (2017).

## Figure 14-13: Histogram for Zn, 1.5 m Composites for Esso Zone



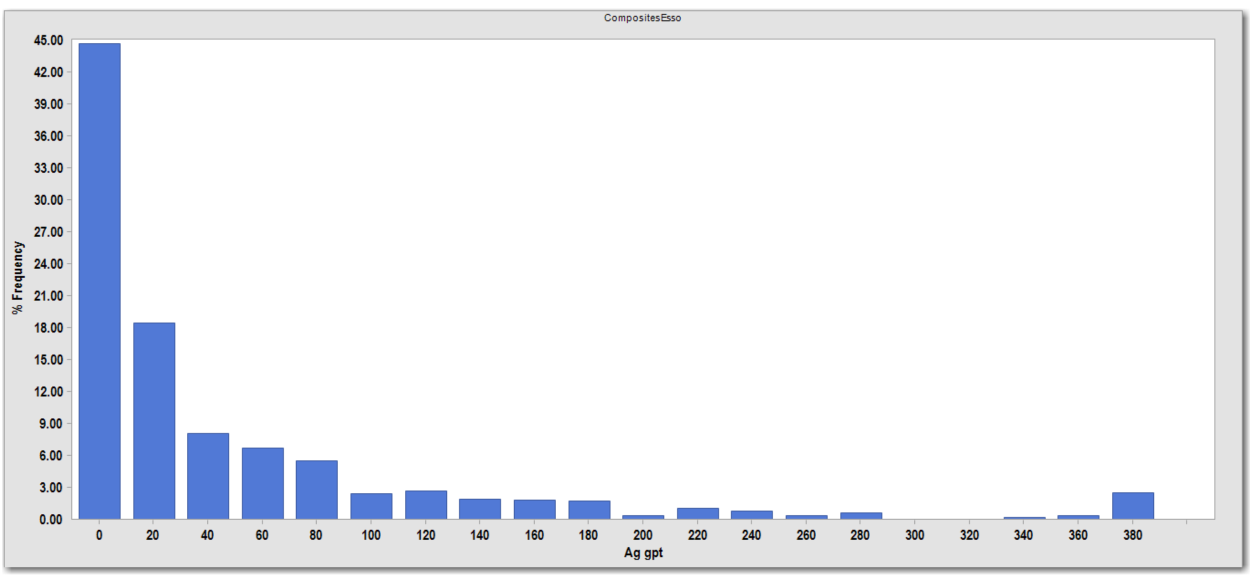

Source: JDS (2017).

## Figure 14-14: Cumulative Distribution Plot of Zn, 1.5 m Composites for Esso Zone



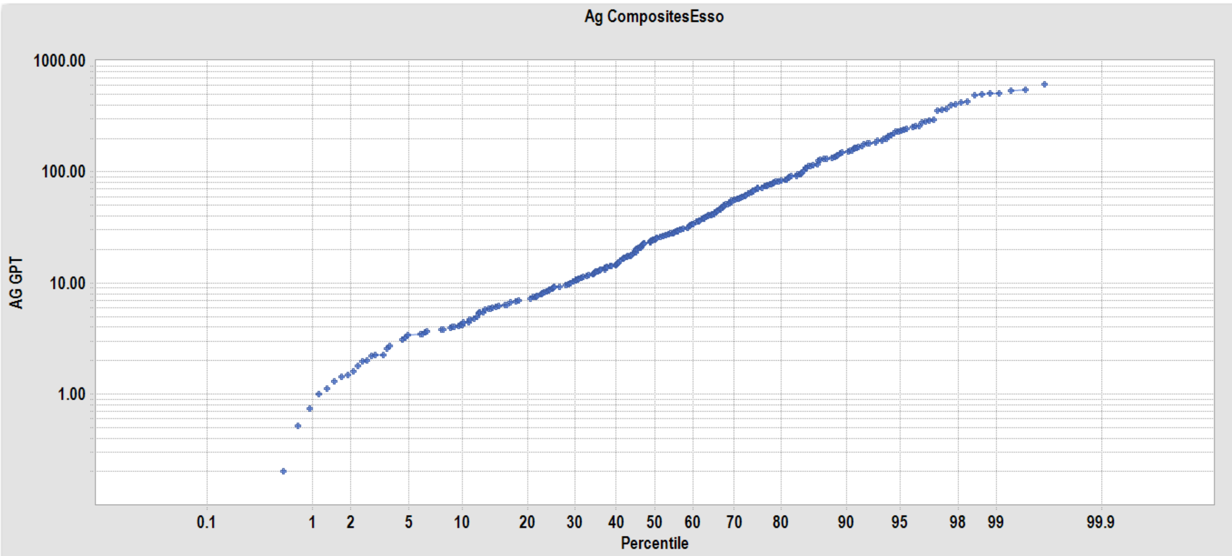

Source: JDS (2017).

## Figure 14-15: Histogram for Ag, 1.5 m Composites for Esso Zone



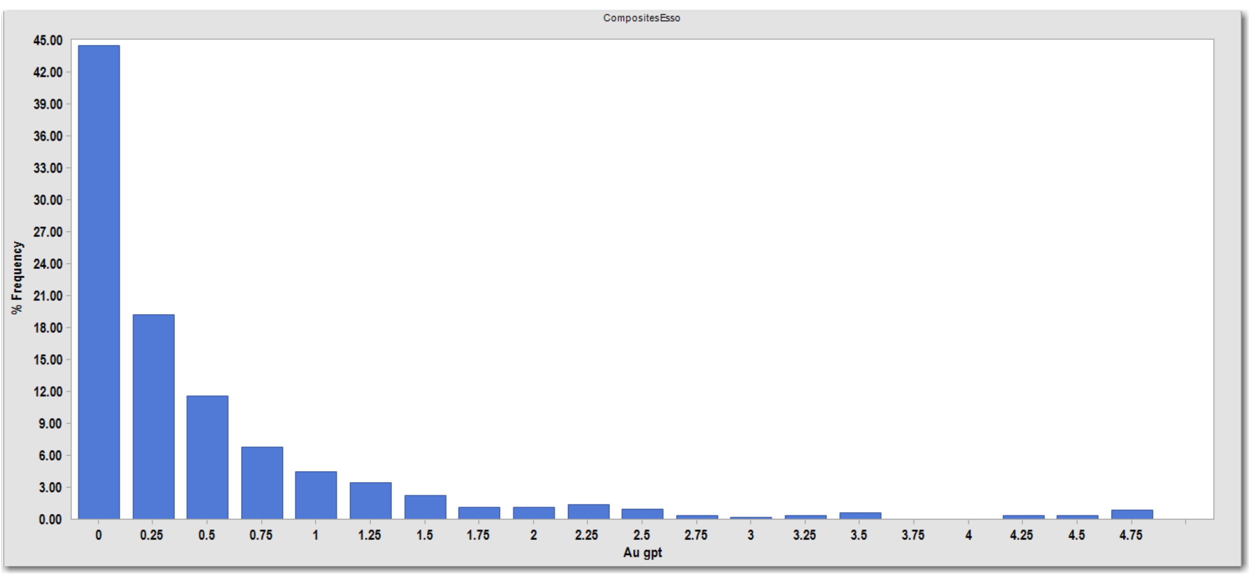

Source: JDS (2017).

**Figure 14-16: Cumulative Distribution Plot for Ag, 1.5 m Composites for Esso Zone**



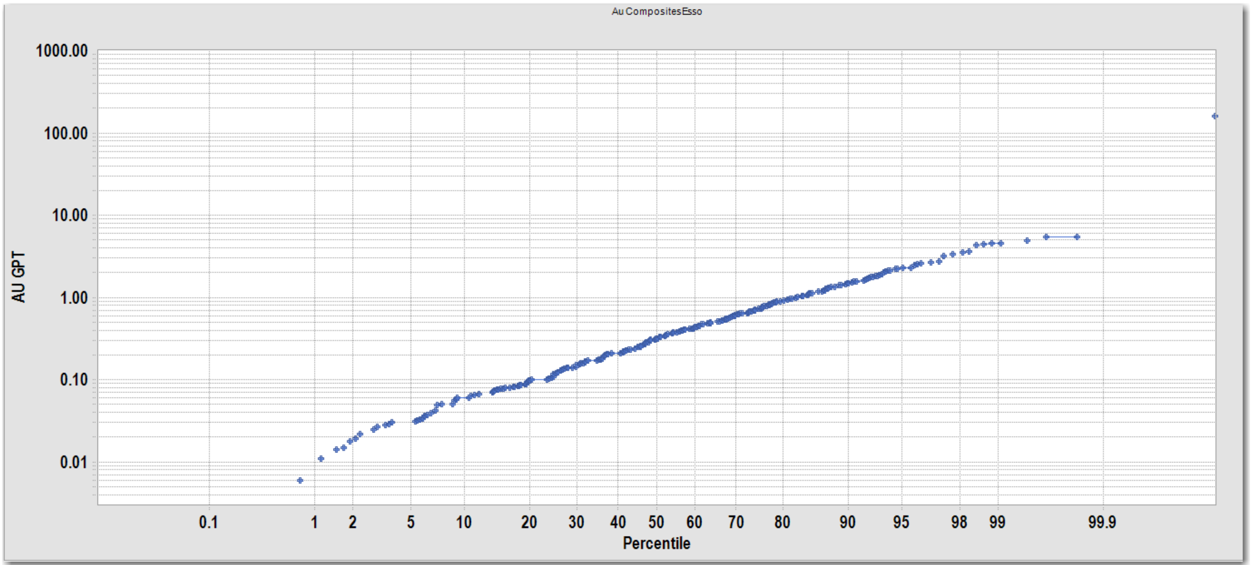

Source: JDS (2017).

**Figure 14-17: Histogram for Au, 1.5 m Composites for Esso Zone**



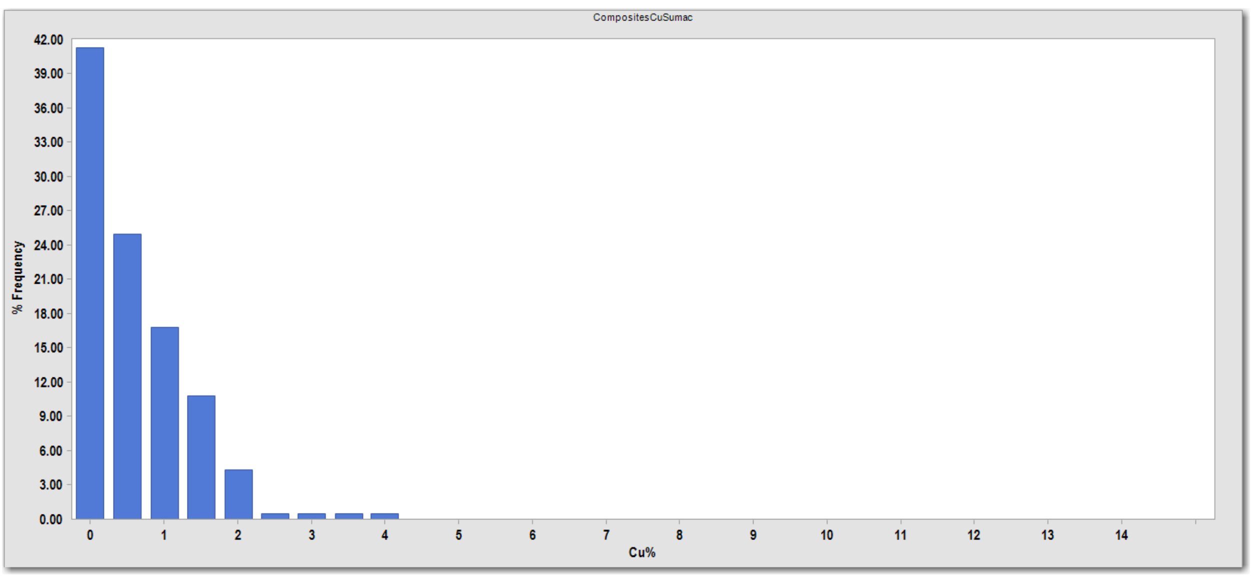

Source: JDS (2017).

### Figure 14-18: Cumulative Distribution Plot for Au, 1.5 m Composites for Esso Zone



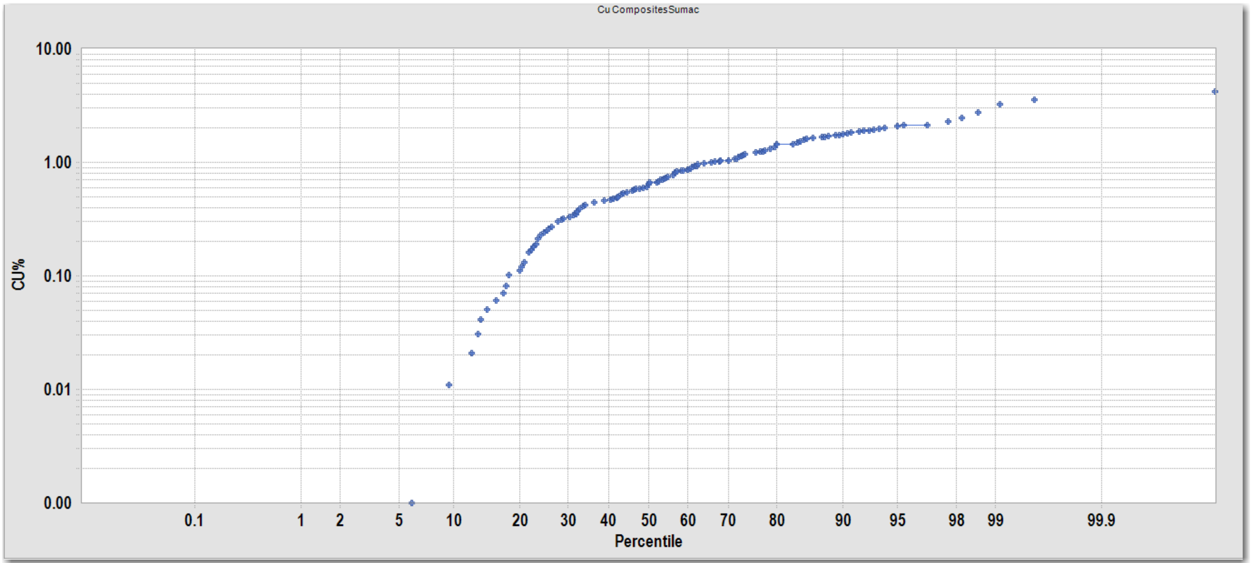

Source: JDS (2017).

### Figure 14-19: Histogram for Cu, 2.5 m Composites for Sumac Zone



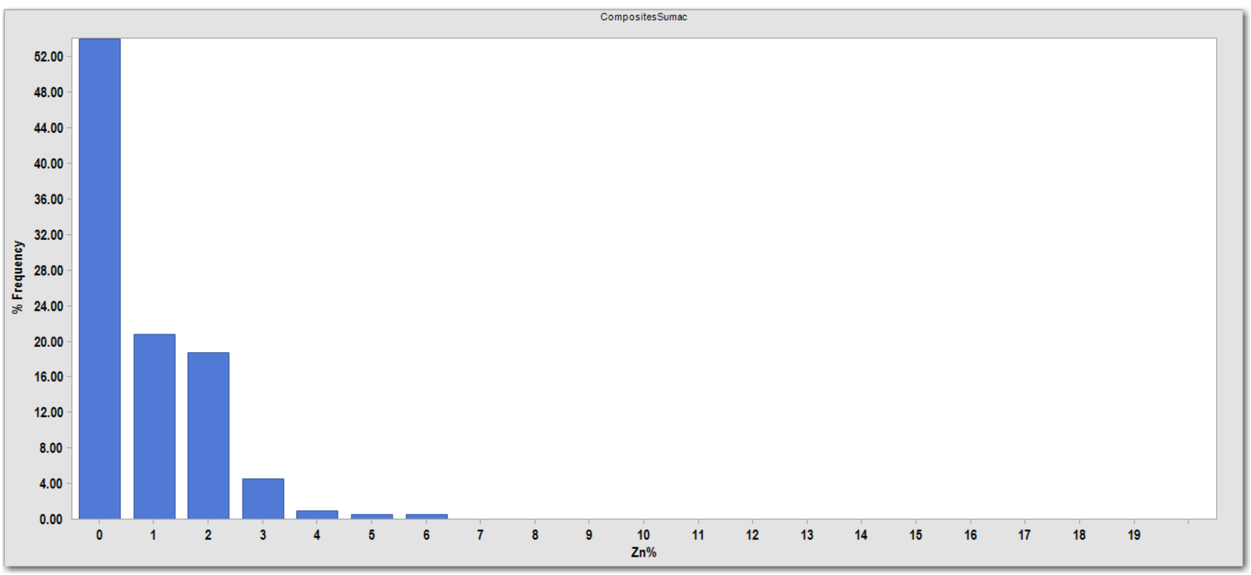

Source: JDS (2017).

## Figure 14-20: Cumulative Distribution Plot for Cu, 2.5 m Composites for Sumac Zone



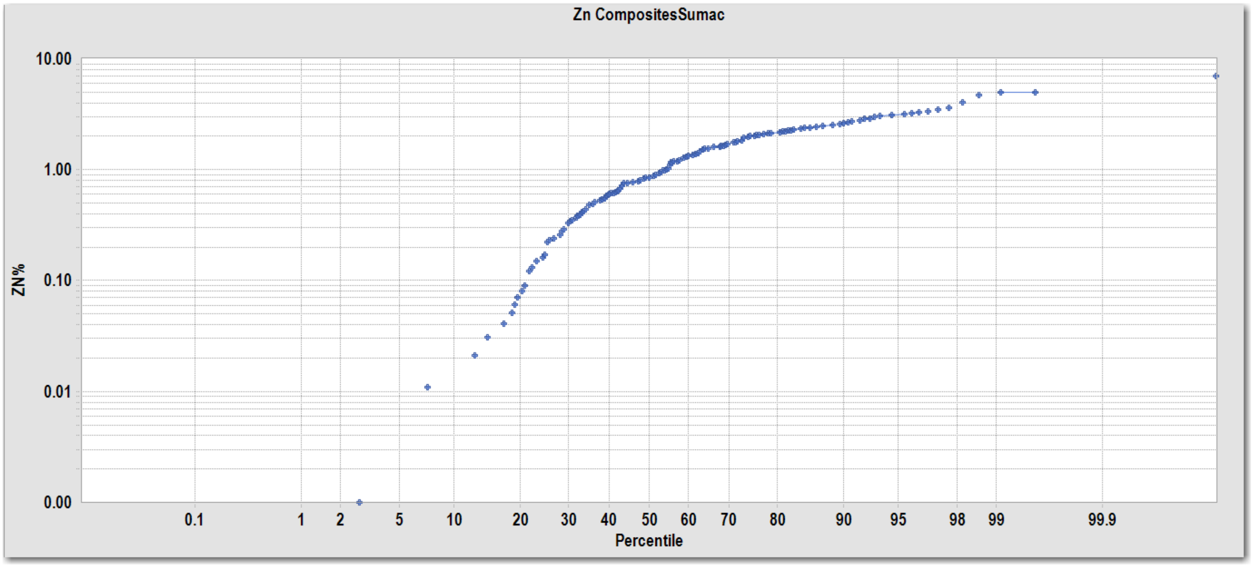

Source: JDS (2017).

## Figure 14-21: Histogram for Zn, 2.5 m Composites for Sumac Zone



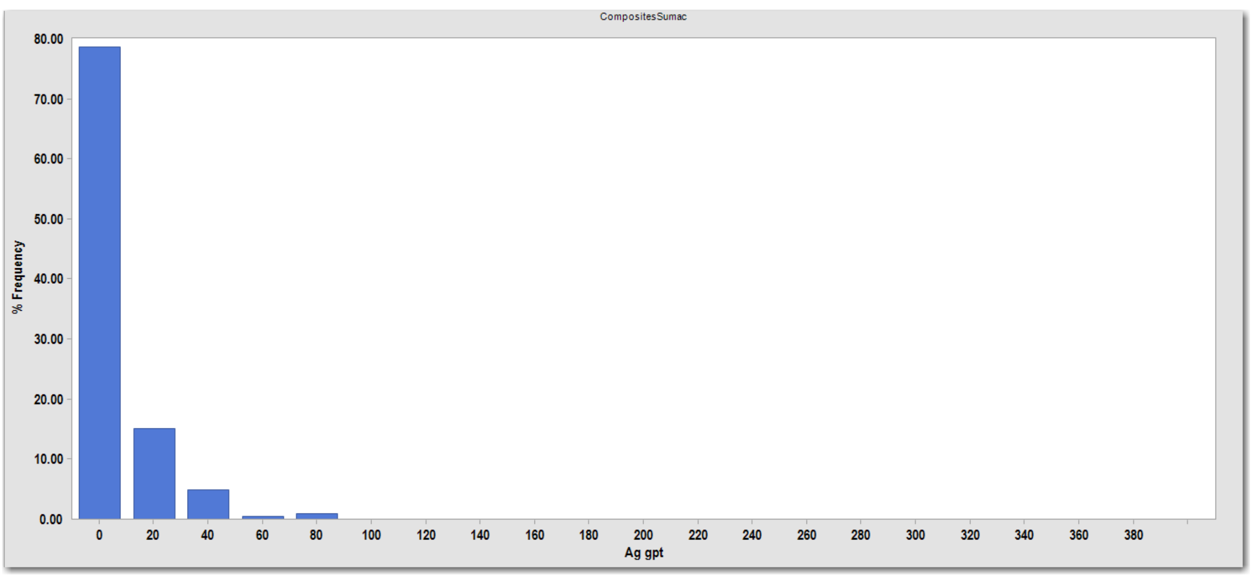

Source: JDS (2017).

**Figure 14-22: Cumulative Distribution Plot of Zn, 2.5 m Composites for Sumac Zone**



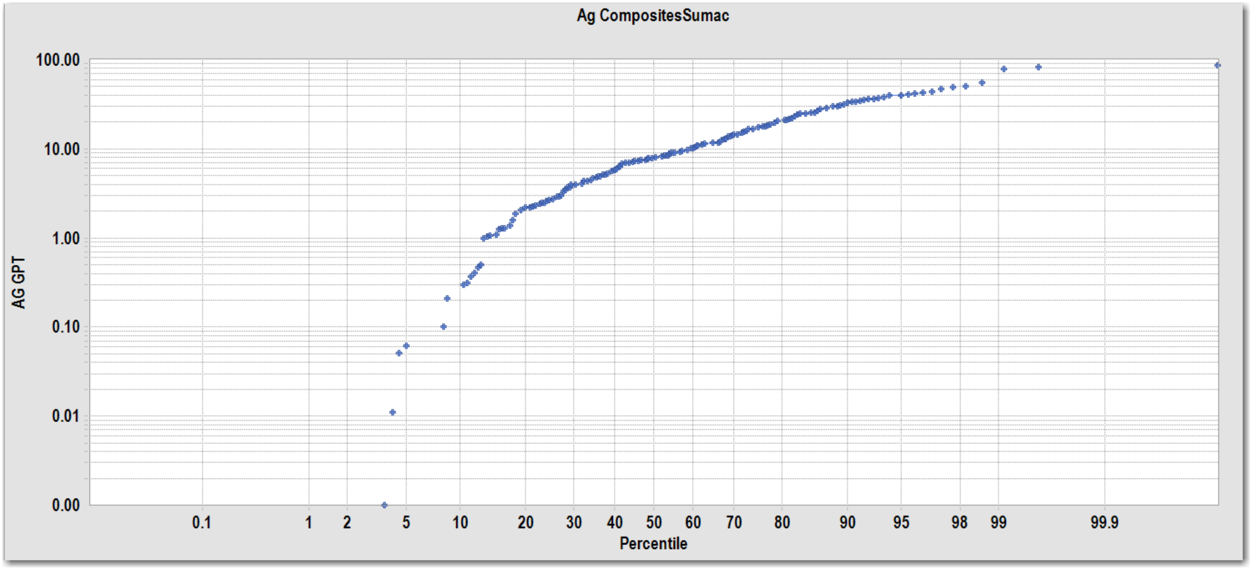

Source: JDS (2017).

**Figure 14-23: Histogram for Ag, 2.5 m Composites for Sumac Zone**



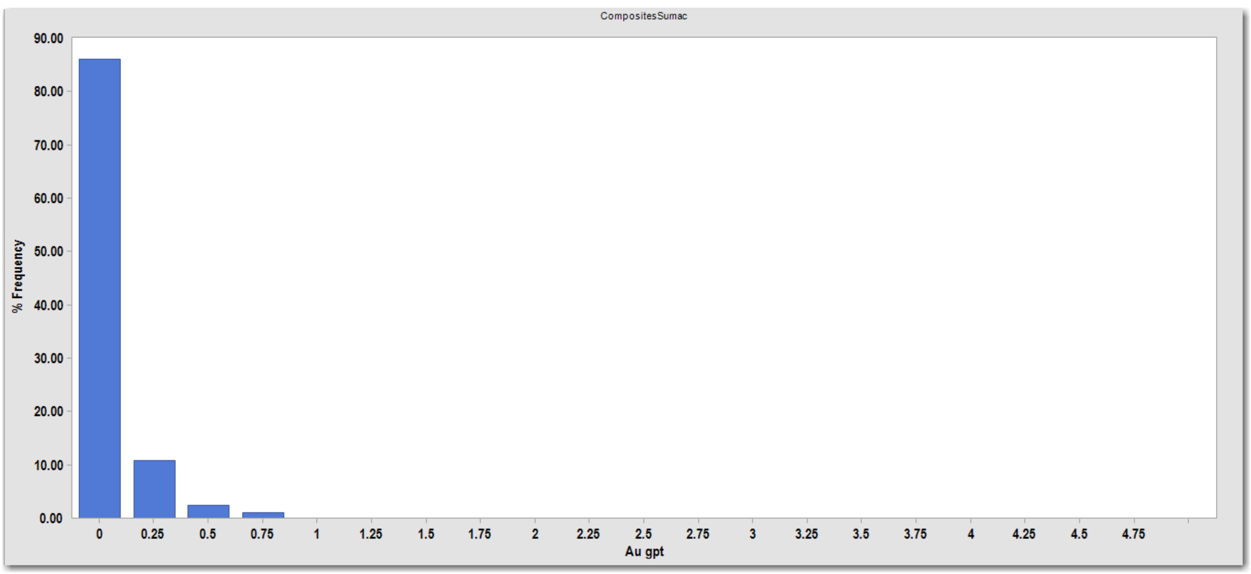

Source: JDS (2017).

## Figure 14-24: Cumulative Distribution Plot for Ag, 2.5 m Composites for Sumac Zone



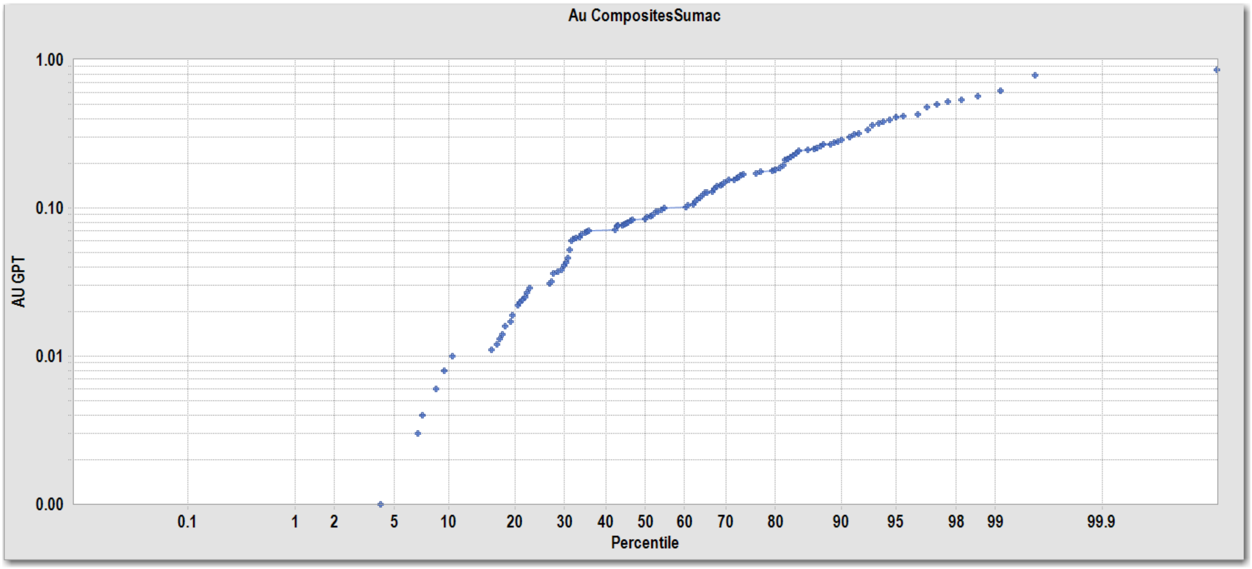

Source: JDS (2017).

## Figure 14-25: Histogram for Au, 2.5 m Composites for Sumac Zone




Source: JDS (2017).

**Figure 14-26: Cumulative Distribution Plot for Au, 2.5 m Composites for Sumac Zone**




Source: JDS (2017).

## 14.6 Outliers

Limiting the influence as opposed to capping of gold, copper, zinc, and silver assays was performed. Although the distribution of grades followed a normal distribution, the probability plots showed "breaks" which indicated multiple populations.

The outlier strategy utilized was to limit values greater than 15% Cu, 17.5% Zn, 100 g/t Ag, and 3 g/t Au for the Main and Sumac zones. However, it is important to note the method employed for this study is not to cut the high-grade outliers but to limit their influence. The range chosen at which to limit grades greater than the outlier cut-off was chosen to be 12 meters. In other words, composite grades greater than the threshold amounts would not be used in the estimation of blocks if those high-grade composites are outside the respective radius from that block.

The outlier strategy utilized was to cut values greater than 11% Cu, 27% Zn, 300 g/t Ag, and 2.2 g/t Au for the Esso deposit. In the case of the Esso deposit, it was determined that the best approach would be to utilize cutting for the purpose of grade limiting therefore the composite grades were cut to the threshold limits as shown above.

## 14.7 Specific Gravity Determinations

The specific gravities were supplied by Kutcho Copper (subsidiary of Capstone) as a field within the drillhole database. A total of 8,399 measurements were included with 2,877, 313, and 167 being within the Main, Esso, and Sumac deposits, respectively. Composites of the SGs from the drillholes were created and then interpolated into the blocks using the inverse distance.

## 14.8 Variography

The grade estimation methodology used for the Main and Esso zones involved ordinary kriging so geostatistics was of relevance. Inverse distance was used for the Sumac zone. The author carried out geostatistical analysis on the composites to evaluate the search parameters to be used in the grade estimate.

Downhole correlograms were generated to make an estimate of the nugget effect as that is the direction in which there is the most abundant data. Geostatistical analyses were performed on the assays and composites using no constraints in addition to the coded intervals within the zone solid.

For the Main zone, the ellipsoid direction for the estimation process was chosen to be 10º azimuth and -45º dip for the major axis, 100º and 0º for the minor axis and 10º and 45º for the vertical axis. This direction follows the orientation of the Main zone solid which is the mineralized structure used for the interpolation of grade.

For the Esso and Sumac zones, the ellipsoid direction chosen for the estimation process was 0º azimuth and -50º dip for the major axis, 90º and 0º for the minor axis and 0º and 40º for the vertical axis. This direction follows the orientation of the zone solids.

The spatial continuity estimator chosen for this study was the correlogram, which has been shown in previous work to be more robust with respect to drift and data variability, allowing for a better estimation of the observed continuity (Srivastava and Parker 1989). Note that the sill of the variograms has been standardized to one, and therefore they are in fact relative variograms.

Tables 14-4 and 14-5 shows the summaries of the correlogram models used to guide the estimation process for the Main zone and Esso zone resources, respectively. Note that ranges are in meters and azimuth and dip are in degrees. In this table, the rotations of the angles are given according to the convention used by GSLIB in MineSight Compass.

**Table 14-4: Main Zone Correlogram Model**

| | Cu | | | Zn | | | Au | | | Ag | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Nugget (C0) | 0.181 | | | 0.244 | | | 0.646 | | | 0.255 | | |
| C1 | 0.781 | | | 0.61 | | | 0.133 | | | 0.55 | | |
| C2 | 0.039 | | | 0.146 | | | 0.222 | | | 0.195 | | |
| **First Structure** | | | | | | | | | | | | |
| | **Range** | **Azim** | **Dip** | **Range** | **Azim** | **Dip** | **Range** | **Azim** | **Dip** | **Range** | **Azim** | **Dip** |
| Maximum | 54.8 | 106 | -8 | 15.8 | 358 | -4 | 29.8 | 91 | 34 | 22.4 | 267 | 12 |
| Minimum | 6.9 | 204 | -46 | 4.5 | 289 | 79 | 2.8 | 10 | -12 | 3.9 | 14 | 55 |
| **Second Structure** | | | | | | | | | | | | |
| Maximum | 311.9 | 250 | 24 | 368 | 289 | 8 | 202.6 | 109 | -4 | 162 | 85 | 13 |
| Intermediate | 221.8 | 147 | 27 | 140.5 | 190 | 50 | 28.9 | 19 | 0 | 147.9 | 319 | 69 |
| Minimum | 94.9 | 16 | 52 | 44.6 | 26 | 39 | 13.9 | 113 | 86 | 12.4 | 359 | -16 |

Source: JDS (2017).

**Table 14-5: Esso Zone Correlogram Model**

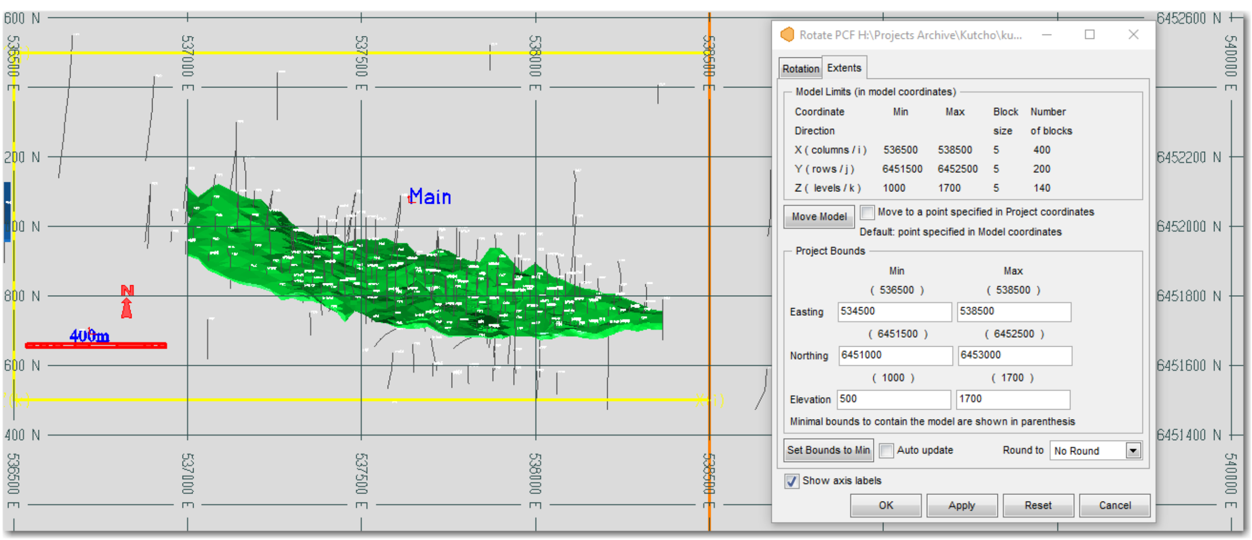
| | Cu | | | Zn | | | Au | | | Ag | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Nugget (C0) | 0.37 | | | 0.4 | | | 0.3 | | | 0.102 | | |
| C1 | 0.373 | | | 0.43 | | | 0.53 | | | 0.62 | | |
| C2 | 0.256 | | | 0.17 | | | 0.17 | | | 0.28 | | |
| **First Structure** | | | | | | | | | | | | |
| | **Range** | **Azim** | **Dip** | **Range** | **Azim** | **Dip** | **Range** | **Azim** | **Dip** | **Range** | **Azim** | **Dip** |
| Maximum | 33.3 | 30 | 33 | 16.6 | 292 | -11 | 24.7 | 285 | -25 | 6.2 | 14 | 50 |
| Intermediate | 38.6 | 156 | 42 | 34.6 | 18 | 15 | 40.3 | 36 | -37 | 49.7 | 71 | -24 |
| Minimum | 13.2 | 278 | 30 | 34.6 | 18 | 15 | 11.4 | 350 | 42 | 27.4 | 146 | 29 |
| **Second Structure** | | | | | | | | | | | | |
| Maximum | 21.8 | 6 | 31 | 24.5 | 12 | 32 | 287.2 | 276 | -5 | 17.8 | 340 | 34 |
| Intermediate | 21.8 | 6 | 31 | 100.2 | 49 | -52 | 12 | 4 | 28 | 23.1 | 52 | -24 |
| Minimum | 347 | 106 | 16 | 295.5 | 114 | 18 | 131.8 | 195 | 61 | 116.3 | 115 | 46 |

Source: JDS (2017).

## 14.9 Block Model Definition

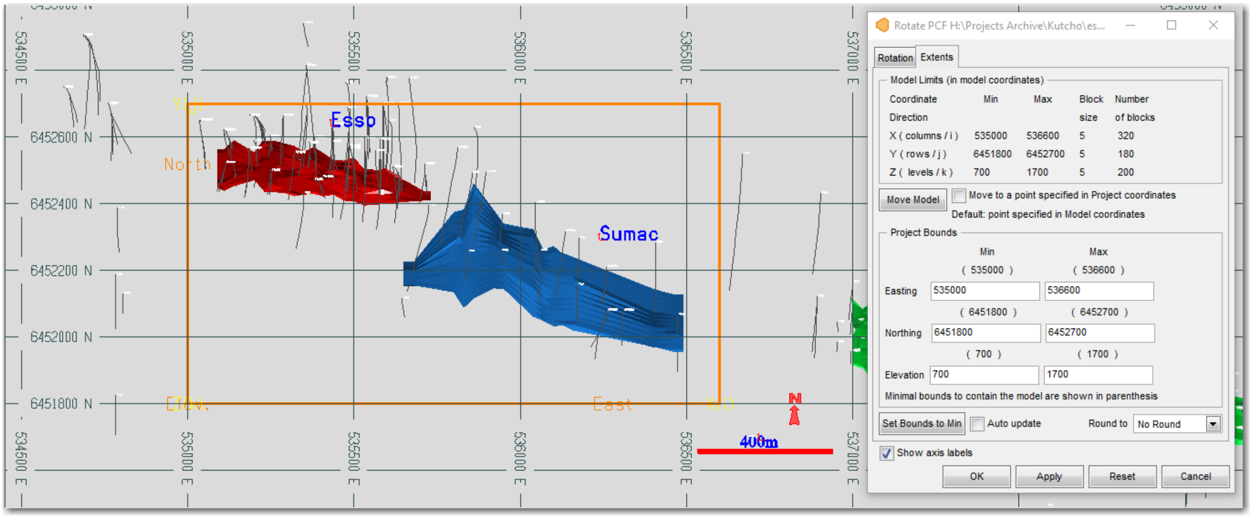
Two separate block models with differing origin and extents were created for the estimation of resources at Kutcho: 1) Main and 2) Esso and Sumac.

**Figure 14-27: Block Model Bounds for Main Zone**




Source: JDS (2017).

**Figure 14-28: Block Model Bounds for Esso and Sumac Zones**




Source: JDS (2017).

The block models are orthogonal and non-rotated reflecting the orientation of each deposit. Figures 14-27 to 14-29 show the position and orientation of the block model used for the Main zone and the Esso and Sumac zones, respectively. The block size chosen was 5 m x 5 m x 5 m to roughly reflect drillhole spacing available and to adequately discretize the deposit.

## 14.10 Resource Interpolation

The estimation plan includes the following items:

- Storage of the mineralized zone code and percentage of mineralization.
- Application of density based on inverse distance cubed estimate of SG.

Estimation of the grades for each of the metals using ordinary kriging using a three-pass strategy for the Main and Esso zones. Inverse distance was used for the Sumac zone. The three estimation passes were used to estimate the Resource Model because a more realistic block-by-block estimation can be achieved by using more restrictions on those blocks that are closer to drillholes, and thus better informed.

Tables 14-6 through 14-8 summarize the search ellipse dimensions for the estimation passes for the Main, Esso, and Sumac zones, respectively.

### Table 14-6: Search Ellipse Parameters for Main

| Pass | Major Axis (m) | Semi-Major Axis (m) | Minor Axis (m) | 1st Rotation Angle Azimuth (degree) | 2nd Rotation Angle Dip (degree) | 3rd Rotation Angle (degree) | Min. No. Of Comps (#) | Max. No. Of Comps (#) | Max. Samples per Drillhole (#) |
|---|---|---|---|---|---|---|---|---|---|
| 1 | 200 | 200 | 80 | 10 | -45 | 0 | 3 | 12 | 2 |
| 2 | 100 | 100 | 40 | 10 | -45 | 0 | 4 | 12 | 2 |
| 3 | 50 | 50 | 10 | 10 | -45 | 0 | 4 | 12 | 2 |

Source: JDS (2017).

### Table 14-7: Search Ellipse Parameters for Esso

| Pass | Major Axis (m) | Semi-Major Axis (m) | Minor Axis (m) | 1st Rotation Angle Azimuth (degree) | 2nd Rotation Angle Dip (degree) | 3rd Rotation Angle (degree) | Min. No. Of Comps (#) | Max. No. Of Comps (#) | Max. Samples per Drillhole (#) |
|---|---|---|---|---|---|---|---|---|---|
| 1 | 100 | 100 | 30 | 0 | -50 | 0 | 4 | 16 | 3 |
| 2 | 50 | 50 | 20 | 0 | -50 | 0 | 4 | 16 | 3 |
| 3 | 25 | 25 | 15 | 0 | -50 | 0 | 4 | 16 | 3 |

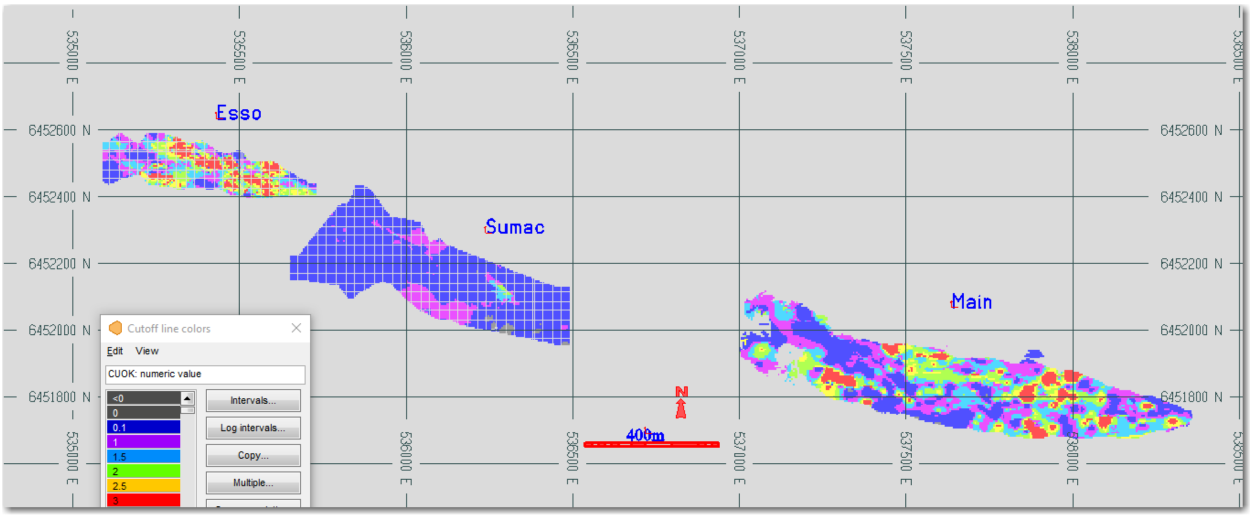
Source: JDS (2017).

### Table 14-8: Search Ellipse Parameters for Sumac

| Pass | Major Axis (m) | Semi-Major Axis (m) | Minor Axis (m) | 1st Rotation Angle Azimuth (degree) | 2nd Rotation Angle Dip (degree) | 3rd Rotation Angle (degree) | Min. No. Of Comps (#) | Max. No. Of Comps (#) | Max. Samples per Drillhole (#) |
|---|---|---|---|---|---|---|---|---|---|
| 1 | 200 | 200 | 50 | 0 | -50 | 0 | 3 | 10 | 2 |
| 2 | 100 | 100 | 25 | 0 | -50 | 0 | 3 | 10 | 2 |
| 3 | 50 | 50 | 10 | 0 | -50 | 0 | 3 | 10 | 2 |

Source: JDS (2017).

**Figure 14-29: Plan View of Grade Models for Main, Sumac, and Esso**




Source: JDS (2017).

## 14.11 Mineral Resource Estimate

The measured, indicated and inferred resources as defined by the parameters are listed in Tables 14-9 through 14-15 for Cu %, Zn %, Ag g/t, and Au g/t. These resources are listed at a cut-off grade of 1.0% copper for all zones. Classification has been done adhering to CIM standards as defined below.

Mineral Resource

Mineral Resources are sub-divided, in order of increasing geological confidence, into Inferred, Indicated and Measured categories. An Inferred Mineral Resource has a lower level of confidence than that applied to an Indicated Mineral Resource. An Indicated Mineral Resource has a higher level of confidence than an Inferred Mineral Resource but has a lower level of confidence than a Measured Mineral Resource.

A Mineral Resource is a concentration or occurrence of diamonds, natural solid inorganic material, or natural solid fossilized organic material including base and precious metals, coal, and industrial minerals in or on the Earth's crust in such form and quantity and of such a grade or quality that it has reasonable prospects for eventual economic extraction. The location, quantity, grade, geological characteristics and continuity of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge.

The term Mineral Resource covers mineralization and natural material of intrinsic economic interest which has been identified and estimated through exploration and sampling and within which Mineral Reserves may subsequently be defined by the consideration and application of technical, economic, legal, environmental, socio-economic and governmental factors. The phrase 'reasonable prospects for eventual economic extraction' implies a judgement by the QP in respect of the technical and economic factors likely to influence the prospect of economic extraction. A Mineral Resource is an inventory of mineralization that under realistically assumed and justifiable technical and economic conditions might become economically extractable. These assumptions must be presented explicitly in both public and technical reports.

Inferred Mineral Resource

An 'Inferred Mineral Resource' is that part of a Mineral Resource for which quantity and grade or quality can be estimated on the basis of geological evidence and limited sampling and reasonably assumed, but not verified, geological and grade continuity. The estimate is based on limited information and sampling gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drillholes.

Due to the uncertainty that may be attached to Inferred Mineral Resources, it cannot be assumed that all or any part of an Inferred Mineral Resource will be upgraded to an Indicated or Measured Mineral Resource as a result of continued exploration. Confidence in the estimate is insufficient to allow the meaningful application of technical and economic parameters or to enable an evaluation of economic viability worthy of public disclosure. Inferred Mineral Resources must be excluded from estimates forming the basis of feasibility or other economic studies.

Indicated Mineral Resource

An 'Indicated Mineral Resource' is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics can be estimated with a level of confidence sufficient to allow the appropriate application of technical and economic parameters, to support mine planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drillholes that are spaced closely enough for geological and grade continuity to be reasonably assumed.

Mineralization may be classified as an Indicated Mineral Resource by the Qualified Person when the nature, quality, quantity and distribution of data are such as to allow confident interpretation of the geological framework and to reasonably assume the continuity of mineralization. The Qualified Person must recognize the importance of the Indicated Mineral Resource category to the advancement of the feasibility of the Project. An Indicated Mineral Resource estimate is of sufficient quality to support a PFS which can serve as the basis for major development decisions.

Measured Mineral Resource

A 'Measured Mineral Resource' is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, and physical characteristics are so well established that they can be estimated with confidence sufficient to allow the appropriate application of technical and economic parameters, to support production planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration, sampling and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drillholes that are spaced closely enough to confirm both geological and grade continuity.

Mineralization or other natural material of economic interest may be classified as a Measured Mineral Resource by the Qualified Person when the nature, quality, quantity, and distribution of data are such that the tonnage and grade of the mineralization can be estimated to within close limits and that variation from the estimate would not significantly affect potential economic viability. This category requires a high level of confidence in, and understanding of, the geology and controls of the mineral deposit.

Classification of mineral resources is based on a number of criteria namely; distance to first composite, average distance of all composites used in a block, number of composites and the number of drillholes used to estimate a block. For measured resources, 30 m was used for the distance to the nearest composite, 30 m for average distance, a minimum of 4 composites and a minimum of 3 drillholes. For indicated resources, 30 to 60 m was used for the distance to nearest composite, 30 to 60 m for average distance, a minimum of 4 composites and a minimum of 2 drillholes. For inferred resources, greater than 60 m was used for the distance to nearest composite, greater than 60 m for average distance, a minimum of 4 composites and a minimum of 1 drillholes.

### Table 14-9: Main Zone Resources (Inclusive of Reserves)

| Cut-off Grade (% Cu) | Tonnes (kt) | Cu (%) | Zn (%) | Ag (g/t) | Au (g/t) |
|---|---|---|---|---|---|
| **Main Measured** | | | | | |
| 0.25 | 8,594 | 1.77 | 2.46 | 27.3 | 0.30 |
| 0.5 | 8,476 | 1.79 | 2.48 | 27.5 | 0.30 |
| 0.75 | 8,216 | 1.83 | 2.53 | 28.0 | 0.31 |
| ***1*** | ***7,695*** | ***1.89*** | ***2.61*** | ***28.7*** | ***0.31*** |
| 1.25 | 6,756 | 2.00 | 2.72 | 29.9 | 0.32 |
| 1.5 | 5,421 | 2.15 | 2.86 | 31.4 | 0.34 |
| 1.75 | 4,072 | 2.32 | 3.01 | 33.2 | 0.36 |
| 2 | 2,734 | 2.54 | 3.14 | 35.6 | 0.38 |
| 2.25 | 1,713 | 2.80 | 3.24 | 38.1 | 0.42 |
| 2.5 | 1,034 | 3.08 | 3.32 | 40.8 | 0.47 |
| 2.75 | 659 | 3.35 | 3.37 | 43.0 | 0.52 |
| 3 | 428 | 3.61 | 3.39 | 45.0 | 0.57 |
| **Main Indicated** | | | | | |
| 0.25 | 8,298 | 1.54 | 2.01 | 26.5 | 0.31 |
| 0.5 | 8,000 | 1.58 | 2.05 | 27.0 | 0.31 |
| 0.75 | 7,587 | 1.64 | 2.10 | 27.6 | 0.31 |
| ***1*** | ***6,777*** | ***1.73*** | ***2.20*** | ***28.6*** | ***0.32*** |
| 1.25 | 5,584 | 1.85 | 2.33 | 29.9 | 0.33 |
| 1.5 | 4,043 | 2.04 | 2.54 | 31.1 | 0.35 |
| 1.75 | 2,744 | 2.23 | 2.72 | 32.3 | 0.36 |
| 2 | 1,696 | 2.46 | 2.85 | 33.5 | 0.37 |
| 2.25 | 982 | 2.71 | 2.94 | 34.6 | 0.39 |
| 2.5 | 542 | 3.00 | 2.96 | 35.2 | 0.43 |
| 2.75 | 298 | 3.31 | 2.96 | 36.8 | 0.48 |

### Table 14-9: Main Zone Resources (Inclusive of Reserves) (continued)

| Cut-off Grade (% Cu) | Tonnes (kt) | Cu (%) | Zn (%) | Ag (g/t) | Au (g/t) |
|---|---|---|---|---|---|
| 3 | 192 | 3.56 | 2.91 | 37.7 | 0.54 |
| **Main Measured + Indicated** | | | | | |
| 0.25 | 16,892 | 1.66 | 2.24 | 27.0 | 0.30 |
| 0.5 | 16,476 | 1.69 | 2.27 | 27.3 | 0.31 |
| 0.75 | 15,803 | 1.74 | 2.32 | 27.8 | 0.31 |
| ***1*** | ***14,472*** | ***1.81*** | ***2.42*** | ***28.7*** | ***0.32*** |
| 1.25 | 12,340 | 1.93 | 2.55 | 29.9 | 0.33 |
| 1.5 | 9,464 | 2.10 | 2.72 | 31.3 | 0.34 |
| 1.75 | 6,817 | 2.29 | 2.89 | 32.8 | 0.36 |
| 2 | 4,430 | 2.51 | 3.03 | 34. 8 | 0.38 |
| 2.25 | 2,695 | 2.77 | 3.13 | 36.8 | 0.41 |
| 2.5 | 1,576 | 3.05 | 3.19 | 38.8 | 0.46 |
| 2.75 | 957 | 3.34 | 3.24 | 41.0 | 0.51 |
| 3 | 620 | 3.60 | 3.24 | 42.7 | 0.56 |
| **Main Inferred** | | | | | |
| 0.25 | 1,244 | 1.37 | 2.20 | 28.7 | 0.38 |
| 0.5 | 1,234 | 1.38 | 2.22 | 28.8 | 0.38 |
| 0.75 | 1,198 | 1.40 | 2.24 | 29.2 | 0.38 |
| ***1*** | ***1,019*** | ***1.48*** | ***2.36*** | ***30.2*** | ***0.39*** |
| 1.25 | 664 | 1.69 | 2.60 | 31.4 | 0.42 |
| 1.5 | 464 | 1.84 | 2.83 | 31.5 | 0.43 |
| 1.75 | 260 | 2.02 | 2.94 | 32.0 | 0.42 |
| 2 | 94 | 2.32 | 2.98 | 31.7 | 0.40 |
| 2.25 | 42 | 2.54 | 2.88 | 34.8 | 0.41 |
| 2.5 | 19 | 2.75 | 2.48 | 33.3 | 0.41 |
| 2.75 | 7 | 3.08 | 2.75 | 33.7 | 0.48 |
| 3 | 5 | 3.16 | 2.90 | 33.0 | 0.49 |

Source: JDS (2017).

### Table 14-10: Esso Zone Resources (Inclusive of Reserves)

| Cut-off Grade (% Cu) | Tonnes (kt) | Cu (%) | Zn (%) | Ag (g/t) | Au (g/t) |
|---|---|---|---|---|---|
| **Esso Indicated** | | | | | |
| 0.25 | 2,879 | 2.07 | 4.98 | 51.2 | 0.53 |
| 0.5 | 2,823 | 2.10 | 5.07 | 52.0 | 0.54 |
| 0.75 | 2,604 | 2.22 | 5.35 | 55.0 | 0.57 |
| ***1*** | ***2,381*** | ***2.35*** | ***5.63*** | ***58.0*** | ***0.60*** |
| 1.25 | 2,081 | 2.52 | 5.91 | 61.6 | 0.63 |
| 1.5 | 1,816 | 2.69 | 6.18 | 64.8 | 0.66 |
| 1.75 | 1,556 | 2.87 | 6.42 | 68.1 | 0.70 |
| 2 | 1,329 | 3.04 | 6.70 | 71.4 | 0.72 |
| 2.25 | 1,088 | 3.25 | 7.07 | 74.6 | 0.76 |
| 2.5 | 853 | 3.48 | 7.27 | 79.3 | 0.80 |
| 2.75 | 673 | 3.72 | 7.62 | 83.5 | 0.84 |
| 3 | 519 | 3.97 | 8.14 | 87.2 | 0.87 |

Source: JDS (2017).

### Table 14-11: Sumac Zone Resources

| Cut-off Grade (% Cu) | Tonnes (kt) | Cu (%) | Zn (%) | Ag (g/t) | Au (g/t) |
|---|---|---|---|---|---|
| **Sumac Inferred** | | | | | |
| 0.25 | 13,718 | 0.89 | 1.25 | 13.7 | 0.14 |
| 0.5 | 11,511 | 0.98 | 1.34 | 15.2 | 0.15 |
| 0.75 | 8,412 | 1.11 | 1.40 | 17.7 | 0.17 |
| ***1*** | ***4,779*** | ***1.30*** | ***1.48*** | ***21.7*** | ***0.21*** |
| 1.25 | 2,246 | 1.50 | 1.58 | 27.6 | 0.26 |
| 1.5 | 863 | 1.72 | 1.64 | 32.8 | 0.31 |
| 1.75 | 279 | 1.96 | 1.89 | 39.8 | 0.38 |
| 2 | 97 | 2.16 | 2.31 | 47.9 | 0.42 |
| 2.25 | 17 | 2.36 | 2.53 | 56.2 | 0.47 |
| 2.5 | 1 | 2.63 | 2.42 | 68.4 | 0.51 |

Source: JDS (2017).

### Table 14-12: Main Resource Summary (Inclusive of Reserves)

| Class | Main Deposit - Mineral Resource Estimate at a 1.0% Copper Cut-Off [1] | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|
| | Tonnes (kt) | Grade | | | | CuEq (%)[2] | Contained Metal | | | |
| | | Cu (%) | Zn (%) | Au (g/t) | Ag (g/t) | | Cu (M lb) | Zn (M lb) | Au (koz) | Ag (koz) |
| Measured (M) | 7,695 | 1.89 | 2.61 | 0.31 | 28.7 | 2.60 | 320.6 | 442.8 | 77 | 7,093 |
| Indicated (I) | 6,777 | 1.73 | 2.2 | 0.32 | 28.6 | 2.34 | 258.5 | 328.7 | 70 | 6,236 |
| **M&I** | **14,472** | **1.81** | **2.42** | **0.32** | **28.7** | **2.48** | **577.5** | **772.1** | **149** | **13,330** |
| Inferred | 1,019 | 1.48 | 2.36 | 0.39 | 30.2 | 2.21 | 33.5 | 53 | 13 | 989 |

Notes: 1 Numbers may not total due to rounding.
2 Copper Equivalent (CuEq%) is calculated as copper equivalent recovered and based on metal price assumptions of US$2.75 per pound of copper, US$1.10 per pound of zinc, US$17 per ounce of silver and US$1,250 per ounce of gold. Recoveries are 84.7%, 75.7%, 48.0%, and 41.2% for copper, zinc, silver, and gold, respectively.

Source: JDS (2017).

### Table 14-13: Esso Resource Summary (Inclusive of Reserves)

| Class | Esso Deposit - Mineral Resource Estimate at a 1.0% Copper Cut-Off [1] | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|
| | Tonnes (kt) | Grade | | | | CuEq (%)[2] | Contained Metal | | | |
| | | Cu (%) | Zn (%) | Au (g/t) | Ag (g/t) | | Cu (M lb) | Zn (M lb) | Au (koz) | Ag (koz) |
| Measured (M) | - | - | - | - | - | | - | - | - | - |
| Indicated (I) | 2,381 | 2.35 | 5.63 | 0.60 | 58.0 | 4.11 | 123.4 | 295.5 | 46 | 4,438 |
| **M&I** | **2,381** | **2.35** | **5.63** | **0.60** | **58.0** | **4.11** | **123.4** | **295.5** | **46** | **4,438** |
| Inferred | - | - | - | - | - | | - | - | - | - |

Notes: 1 Numbers may not total due to rounding.
2 Copper Equivalent (CuEq%) is calculated as copper equivalent recovered and based on metal price assumptions of US$2.75 per pound of copper, US$1.10 per pound of zinc, US$17 per ounce of silver and US$1,250 per ounce of gold. Recoveries are 84.7%, 75.7%, 48.0%, and 41.2% for copper, zinc, silver, and gold, respectively.

Source: JDS (2017).

### Table 14-14: Sumac Resource Summary

| Class | Sumac Deposit - NI43-101 Mineral Resource Estimate at a 1.0% Copper Cut-Off (1) | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|
| | Tonnes (kt) | Grade | | | | CuEq (%)(2) | Contained Metal | | | |
| | | Cu (%) | Zn (%) | Au (g/t) | Ag (g/t) | | Cu (M lb) | Zn (M lb) | Au (koz) | Ag (koz) |
| Measured (M) | - | - | - | - | - | | - | - | - | - |
| Indicated (I) | - | - | - | - | - | | - | - | - | - |
| **M&I** | **-** | **-** | **-** | **-** | **-** | | **-** | **-** | **-** | **-** |
| Inferred | 4,779 | 1.30 | 1.48 | 0.21 | 21.7 | 1.70 | 136.5 | 156.2 | 32 | 3,337 |

Notes: [1] Numbers may not total due to rounding.
[2] Copper Equivalent (CuEq%) is calculated as copper equivalent recovered and based on metal price assumptions of US$2.75 per pound of copper, US$1.10 per pound of zinc, US$17 per ounce of silver and US$1,250 per ounce of gold. Recoveries are 84.7%, 75.7%, 48.0%, and 41.2% for copper, zinc, silver, and gold, respectively.

Source: JDS (2017).

### Table 14-15: Kutcho Project Resource Summary (Inclusive of Reserves)

| Class | Kutcho Project - Mineral Resource Estimate at a 1.0% Copper Cut-Off for All Deposits (1) | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|
| | Tonnes (kt) | Grade | | | | CuEq (%)(2) | Contained Metal | | | |
| | | Cu (%) | Zn (%) | Au (g/t) | Ag (g/t) | | Cu (M lb) | Zn (M lb) | Au (koz) | Ag (koz) |
| Measured (M) | 7,695 | 1.89 | 2.61 | 0.31 | 28.7 | 2.60 | 320.6 | 442.8 | 77 | 7,093 |
| Indicated (I) | 9,158 | 1.89 | 3.09 | 0.39 | 36.3 | 2.80 | 381.8 | 624.2 | 116 | 10,674 |
| **M&I** | **16,853** | **1.89** | **2.87** | **0.36** | **32.8** | **2.71** | **700.8** | **1067.6** | **195** | **17,768** |
| Inferred | 5,798 | 1.33 | 1.64 | 0.24 | 23.2 | 1.79 | 170.0 | 209.2 | 45 | 4,326 |

Notes: [1] Numbers may not total due to rounding.
[2] Copper Equivalent (CuEq%) is calculated as copper equivalent recovered and based on metal price assumptions of US$2.75 per pound of copper, US$1.10 per pound of zinc, US$17 per ounce of silver and US$1,250 per ounce of gold. Recoveries are 84.7%, 75.7%, 48.0%, and 41.2% for copper, zinc, silver, and gold, respectively.

Source: JDS (2017).

## 14.12 Model Validation

A graphical validation was done on the block model. This graphical validation serves several purposes:

- Checks the reasonableness of the estimated grades, based on the estimation plan and the nearby composites;
- Checks the general drift and the local grade trends
- Insures that all blocks that should be filled in, are in fact filled in;
- Checks that topography has been properly accounted for;

- Checks against manual “ballbark” estimates of tonnage to determine reasonableness; and
- Inspection and explanation for high grade blocks created as a result of outliers.

A full set of cross-sections, long sections and plans were used to check the block model on the computer screen, showing the block grades and the composites. No evidence of any block being wrongly estimated was found: it appears that every block grade can be explained as a function of the surrounding composites, the correlogram models used, and the estimation plan applied.

These validation techniques are as follows but limited to:

- Visual inspections on a section-by-section and plan-by-plan basis;
- The use of Grade Tonnage Curves;
- Histograms at varying cut-off grades demonstrating a relatively uniform, normal distribution;
- Swath Plots showing the comparison of the Ordinary Kriged blocks versus Inverse Distance and Nearest Neighbour estimates; and
- An inspection of histograms of distance of first composite to nearest block and average distance to blocks for all composites used.

# 15 Mineral Reserve Estimate

A Mineral Reserve is the economically mineable part of a Measured or Indicated Mineral Resource demonstrated by at least a PFS. This PFS includes adequate information on mining, processing, metallurgical, economic, and other relevant factors that demonstrate, at the time of reporting, that economic extraction is justified.

Mineral Reserves are those parts of Mineral Resources which, after the application of all mining factors, result in an estimated tonnage and grade which is the basis of an economically viable project. This project has accounted for relevant processing, metallurgical, economic, marketing, legal, environmental, socioeconomic and governmental factors. Mineral Reserves are inclusive of diluting material that will be mined in conjunction with the Mineral Reserves and delivered to the processing plant or equivalent facility. The term ‘Mineral Reserve’ need not necessarily signify that extraction facilities are in place or operative or that all governmental approvals have been received. It does signify that there are reasonable expectations of such approvals.

Mineral Reserves are subdivided in order of increasing confidence into Probable Mineral Reserves and Proven Mineral Reserves. A Probable Mineral Reserve has a lower level of confidence than a Proven Mineral Reserve.

The Proven and Probable reserve classifications used in this report conform to the Canadian Institute of Mining, Metallurgy and Petroleum classification of NI 43-l0l resource and reserve definitions and Companion Policy 43-101CP and are listed below.

A ‘Proven Mineral Reserve’ is the economically mineable part of a Measured Mineral Resource demonstrated by at least a Preliminary Feasibility Study. This Study must include adequate information on mining, processing, metallurgical, economic, and other relevant factors that demonstrate, at the time of reporting, that economic extraction is justified. Application of the Proven Mineral Reserve category implies that the Qualified Person has the highest degree of confidence in the estimate with the consequent expectation in the minds of the readers of the report. The term should be restricted to that part of the deposit where production planning is taking place and for which any variation in the estimate would not significantly affect potential economic viability.

A ‘Probable Mineral Reserve’ is the economically mineable part of an Indicated Mineral Resource, and in some circumstances a Measured Mineral Resource, demonstrated by at least a Preliminary Feasibility Study. The study must include adequate information on mining, processing, metallurgical, economic, and other relevant factors that demonstrate, at the time of reporting, that economic extraction can be justified.

## 15.1 Cut-off Value and Grade Criteria

Mining reserve values were calculated from block model tonnes and grades and defined by net smelter return (NSR) and an assumed cut-off grade (COG). The calculated COG from Table 15-3 is 0.8% Cu. COGs of 1.5% Cu and 1% Cu for the Main and Esso deposits were selected for the LOM plan. The selected Esso COG is lower than the Main COG to take into account the significantly higher (double) zinc, silver, and gold (by-product) grades in Esso.

The COGs used in the LOM plan and reserves may be considered conservative as they don't fully take into account by-product values. Because the deposits contain multiple payable metals a more appropriate measure of economic viability is NSR cut-off value (COV) as opposed to COG. JDS conducted a COV calculation based on expected US$:C$ exchange rate, operating cost, smelter terms, metal recoveries, mining dilution, and commodity price assumptions. The parameters used for the calculations were based on the data shown in Tables 15-1 and 15-2 and the COV and COG calculations are in Table 15-3.

### Table 15-1: Net Smelter Return and Cut-off Calculation Metal Prices

| Commodity | Unit | Value |
|---|---|---|
| Copper Price | US$/lb | 2.75 |
| Zinc Price | US$/lb | 1.10 |
| Gold Price | US$/oz | 1,250 |
| Silver Price | US$/oz | 17.00 |
| Exchange Rate | US$:C$ | 0.75 |

Source: JDS (2017).

### Table 15-2: Copper and Zinc Concentrate Net Smelter Return Parameters

| NSR Assumptions | Unit | Cu Concentrate | Zn Concentrate |
|---|---|---|---|
| **Recoveries** | | | |
| Cu | % | 84.7 | - |
| Zn | % | - | 75.7 |
| Au | % | 41.2 | - |
| Ag | % | 48.0 | - |
| Concentrate Grade | %Cu, %Zn | 27.6 | 55.1 |
| Moisture Content | % | 9.0 | 9.0 |
| **Smelter Payables** | | | |
| Cu Payable | % | 96.50 | - |
| Au Payable | % | 90.00 | 85.00 |
| Ag Payable | % | 90.00 | - |
| **TC/RCs** | | | |
| Treatment Charge | US$/dmt concentrate | 70.00 | 120.00 |
| Transport Cost to Smelter | US$/dmt concentrate | 106.37 | 106.37 |
| Cu Refining | US$/lb | 0.07 | - |
| Au Refining | US$/oz | 6.00 | - |
| Ag Refining | US$/oz | 0.35 | - |

Source: JDS (2017).

### Table 15-3: Cut-off Value and Grade Calculation

| Item | Unit | Value |
|---|---|---|
| Mining Cost | C$/ore t | 40.41 |
| Milling Cost | C$/ore t | 20.79 |
| General and Administrative + Power Plant Lease | C$/ore t | 12.52 |
| Total Site Costs (excluding sustaining CAPEX) | C$/ore t | 73.72 |
| Millhead NSR Cut-off Value(1) | C$/t | 74 |
| Millhead Copper Cut-off Grade(1) | % Cu | 0.8 |

Note: (1) Excludes sustaining CAPEX and royalties and assumes that by product ratios don't change from LOM the average.

Source: JDS (2017).

## 15.2 Dilution

External dilution of 10% at zero metal grades has been applied to the LH stope designs.

## 15.3 Mineral Reserve Estimates

The stope designs with external dilution applied determined the Mineral Reserve estimate shown in Table 15-4.

### Table 15-4: Mineral Reserve Estimate Summary

| Category | Diluted Tonnes (kt) | Cu Grade (%) | Cu (Mlbs) | Zn Grade (%) | Zn (Mlbs) | Ag Grade (g/t) | Ag (Moz) | Au Grade (g/t) | Au (Moz) |
|---|---|---|---|---|---|---|---|---|---|
| Probable | 10,441 | 2.01 | 463 | 3.19 | 734 | 34.6 | 11.6 | 0.37 | 0.1 |
| **Total/Average** | **10,441** | **2.01** | **463** | **3.19** | **734** | **34.6** | **11.6** | **0.37** | **0.1** |

Notes: The Qualified Person for the Mineral Reserve estimate is Michael Makarenko, P. Eng., of JDS Energy & Mining Inc.

Mineral Reserves were estimated using the following metal prices. Copper: $2.75/lb Cu, $1.10/lb Zn, $1,250/oz Au, $17/oz Ag, and selected cut-offs of 1.5% Cu and 1.0% Cu for the Main and Esso deposits respectively.

Other costs and factors used for copper cut-off grade determination were mining, processing and other costs of $73.72/t and recoveries of 84.7% Cu, 75.7% Zn, 48.0% Ag, and 41.2% Au.

Tonnages are rounded to the nearest 1,000 t and metal grades are rounded to two decimal places. Tonnage and grade measurements are in percentages and metric units.

Source: JDS (2017).

The Mineral Reserves identified in Table 15-4 comply with CIM definitions and standards for a NI 43-101 Technical Report. Detailed information on mining, processing, metallurgical, and other relevant factors are contained in the followings sections of this report and demonstrate, at the time of this report, that economic extraction is justified. This study did not identify any mining, metallurgical, infrastructure or other relevant factors that may materially affect the estimates of the Mineral Reserves or potential production. Reserves by deposit are shown in Table 15-5.

### Table 15-5: Mineral Reserve Estimate by Deposit

| Deposit | Diluted Tonnes (kt) | Cu Grade (%) | Cu (Mlbs) | Zn Grade (%) | Zn (Mlbs) | Ag Grade (g/t) | Ag (Moz) | Au Grade (g/t) | Au (Moz) |
|---|---|---|---|---|---|---|---|---|---|
| Main(1) | 8,106 | 1.92 | 344 | 2.51 | 449 | 28.0 | 7.3 | 0.31 | 0.1 |
| Esso(2) | 2,335 | 2.32 | 119 | 5.53 | 285 | 57.5 | 4.3 | 0.59 | 0.0 |
| **Total/Average** | **10,441** | **2.01** | **463** | **3.19** | **734** | **34.6** | **11.6** | **0.37** | **0.1** |

Notes: (1) 1.5% Cu cut-off grade for the Main deposit.
(2) 1.0% Cu cut-off grade for the Esso deposit.
Numbers may not total due to rounding.

Source: JDS (2017).

# 16 Mining Methods

## 16.1 Introduction

The mine design and planning for the Project is based on the resource model completed by G. Kirkham P. Geo, as detailed in Section 14 of this report. Two underground mining methods were selected for the Project; sub-level LH open stoping, C&F, or MCF.

## 16.2 Deposit Characteristics

### 16.2.1 Main

The Main deposit has an elliptical, lenticular shape with approximate dimensions of 1,500 m long, 260 m wide (down-dip), and 36 m maximum thickness. The long axis of the deposit plunges to the west at about 12°, just slightly less than the regional fold axes. The deposit is conformable with stratigraphy, dipping moderately to the north. There is a gentle warping of the deposit, such that the dip of the deposit changes from east to west and north to south. The shallowest dip (about 38°) occurs at the south-eastern edge and becomes progressively steeper (to about 63°) at the north-western edge. In general, the up-dip edge of the sulphide lens is narrow and pinches out, whereas the down-dip edge is thicker and interlayered with tuffaceous rock, giving the deposit an approximate flattened arrowhead shape.

### 16.2.2 Esso

The Esso deposit lies between 400 to 550 m below the surface. It was discovered by following down plunge, the westward trend of mineralization beyond the Main and Sumac deposit areas. The Esso deposit has an elongate lens shape with a strike length of approximately 640 m, a dip direction of 240 m and is up to 21 m thick but averages approximately 12.2 m thick.

## 16.3 Geotechnical Parameters

A prefeasibility level geotechnical rock mechanics evaluation was conducted for the proposed underground excavations at the Main and Esso deposits and the proposed starter pit at the Main deposit.

This section was extracted from the following two EBA reports; (*i) Kutcho Project - Main Deposit Pre-feasibility Level Geotechnical Evaluation, dated November 29, 2010, and (ii) Kutcho Project - Esso Deposit Pre-feasibility Level Geotechnical Evaluation, dated November 29, 2010.* These reports are based on information collected during two drilling programs carried out during 2008 and 2010, as well as from the review of existing historical reports dated back to 1981. These reports should be referenced for additional detail (EBA 2010a,b).

### 16.3.1 Geotechnical Data Collection - 2008 and 2010 Drilling Programs

In 2008 and 2010, a total of 100 geotechnical diamond boreholes were drilled at the Main and Esso deposits (81 at the Main deposit and 19 at the Esso deposit). The 81 boreholes at the Main deposit were used to establish geotechnical parameters for the pre-feasibility geotechnical evaluation of the Main deposit. The 19 boreholes at the Esso deposit did not contain sufficient geotechnical information to perform a geotechnical evaluation. Therefore, geotechnical parameters for the pre-feasibility geotechnical evaluation of the Esso deposit were obtained from seven diamond boreholes drilled during the 2010 exploration program. The drilling contractor was under the direct supervision of Kutcho Copper personnel for the duration of the drilling programs. Kutcho Copper personnel also logged (geologically and geotechnically) and photographed the rock cores.

As part of the 2010 drilling program, field and laboratory testing were conducted to evaluate the strength of intact rock by unconfined compressive strength tests, and the tensile strength by Brazilian tensile tests.

### 16.3.2 Geological Discontinuity Features - Rock Fabric and Major Geological Structures

Measurement of discontinuity orientations was not part of the 2008 and 2010 drilling programs. The following information regarding the rock fabric at the Main deposit was extracted from a general description of rocks of the Kutcho formation presented in the following two reports; (i) Pit Slope Design, Kutcho Creek Project, October 1982, by Golder Associates, and (ii) Kutcho Project Prefeasibility Study, 2007, by Wardrop. *The main structural set mimics the schistosity, which strikes east-west and varies in dip – steeper at surface (up to 75 - 80°), then flattening to between 50 and 70° at depth. Two orthogonal structural sets also exist, one striking north-south and vertical, and another striking east-west with a relative flat dip (0 - 20°).*

According to the 1982 Golder report the structural information in the paragraph above was obtained by mapping surface outcrops and by core logging. The 1982 Golder report states, fractures in the core were oriented relative to the schistosity assuming that the schistosity always strikes east-west, perpendicular to the hole direction, with allowance made for changes in dip of the hole. While variations in the strike of the schistosity can produce errors in the orientation of other fractures, we believe that the strike of the schistosity is reasonably consistent across the property and that this method of orientation should not produce errors.

The structural trends found at the Main deposit have also been applied to the Esso deposit. The presence or absence of large faults (intermediate or regional fault structures) crossing the Main and Esso deposits need to be identified during further stages (feasibility and design).

### 16.3.3 Rock Mass Assessment and Geotechnical Model

The geotechnical parameters for the Main deposit and Esso deposits for the pre-feasibility assessment are interpreted from the 81 geotechnical diamond boreholes (WK08-088 through WK08-168) drilled at the Main deposit in 2008, and seven diamond boreholes (KC10-196 through KC10-202) drilled at the Esso deposit in 2010.

Simplified geotechnical models were established for the Main and Esso deposits based on the review of the borehole logs and core photographs of the boreholes referred to above as well as from the review of historical geotechnical information and laboratory testing results. The geotechnical models consist of domains which are rock masses with similar lithology, engineering characteristics and quality.

A total of four geotechnical domains were established for the Main deposit. Table 16-1 presents the four domains with the average rating value for three rock mass classification systems; Bieniawski's Rock Mass Rating ($RMR_{76}$), Laubscher's In-situ Rock Mass Rating ($IRMR_{90}$), and the Norwegian Geotechnical Institute's NGI Q-System.

**Table 16-1: Main Deposit – Domains and Summary of Estimated IRMR, RMR, and Q Ratings**

| Rock Group | Laubscher's IRMR (1990) | | Bieniawski's RMR (1976) | | Q-System[1] (Q=Q'/SRF) | |
|---|---|---|---|---|---|---|
| | Average Rating | Quality | Average Rating | Quality | Average Rating | Quality |
| **Domain 1 – Breccias** | | | | | | |
| Breccias | 48 | Fair | 61 | Good | 46/2.5=19 | Good |
| **Domain 2 – Hangingwall (HW) Tuff Rock** | | | | | | |
| Tuff-Ash | 43 | Fair | 52 | Fair | 31/2.5=12 | Good |
| Tuff-Fspar | 45 | Fair | 61 | Good | 31/2.5=12 | Good |
| Tuff-Lapilli | 40 | Poor | 49 | Fair | 23/2.5=9 | Fair |
| Tuff-Litchi | 40 | Poor | 47 | Fair | 20/2.5=8 | Fair |
| Tuff Qtz | 49 | Fair | 61 | Good | NA | NA |
| Average Hangingwall (HW) Tuff rock | 47 | Fair | 59 | Fair | 31.7/2.5=13 | Good |
| **Domain 3 – Orebody** | | | | | | |
| Sulphides | 43 | Fair | 55 | Fair | 32/2.5=13 | Good |
| **Domain 4 – Footwall (FW) Lapilli Tuff – Pyritic** | | | | | | |
| Footwall (FW) Lapilli Tuff – Pyritic | 44 | Fair | 53 | Fair | 53/2.5=21 | Good |

Note: [1] A stress reduction factor (SRF) of 2.5 was adopted for low stress, near surface.

Source: EBA (2010a).

Table 16-1 shows that in the Main deposit the rock mass quality of the hangingwall rock and mineralized material are very similar.

A total of four geotechnical domains were established for the Esso deposit. Table 16-2 presents the four domains with the average rating value for two rock mass classification systems; $RMR_{76}$ and the Norwegian Geotechnical Institute's NGI Q-System.

Table 16-2 shows that in the Esso deposit area, the rock mass quality of the hangingwall rock and mineralized material are similar.

**Table 16-2: Esso Deposit – Domains and Summary of Estimated IRMR, RMR, and Q Ratings**

| Rock Group | Bieniawski's RMR (1976) | | Q-System[1] (Q=Q'/SRF) | |
|---|---|---|---|---|
| | Average Rating | Quality | Average Rating | Quality |
| **Domain 1 – 5 metres and above the Orebody** | | | | |
| Tuff - Quartz Crystal (QXTF) | 60 | Fair | 63 / 2.5 = 25.2 | Good |
| Tuff – Quartz Feldspar Crystal (QFXT) | 60 | Fair | 30 / 2.5 = 12 | Good |
| Average Tuff rock | 60 | Fair | 55 / 2.5 = 22 | Good |
| **Domain 2 – Hangingwall (HW) Tuff Rock (5 m above Orebody)** | | | | |
| Tuff (LLAT, LLXT, FLTZ, PBLT, CQEX, QXTF) | 53 | Fair | 35 / 2.5 = 14 | Good |
| **Domain 3 – Orebody** | | | | |
| Sulphides[1] | 56 | Fair | 44/2.5 = 17.6 | Good |
| **Domain 4 – Footwall (FW) Lapilli Tuff – Ash Tuff** | | | | |
| Footwall (FW) Lapilli Tuff – Ash Tuff[1] | 54 | Fair | 43/2.5 = 17 | Good |

Note: [1] A SRF of 2.5 was adopted for single shear zones in competent rock / depth of excavation > 50 m.
Source: EBA (2010b).

### 16.3.4 Geotechnical Design Methods

Evaluation of the stability of the proposed stope spans for both excavation methods (LH and MCF) for the Main and Esso deposits was carried out using the following two empirical design methods; the Laubscher's stability diagram based on Laubscher's RMR, and the Modified Stability Graph Method based on the Q-system.

The 1990 Laubscher's stability diagram estimates the stability of an excavation in terms of mining rock mass rating (MRMR) and hydraulic ratios (HR), where HR is equal to the area of the stope divided by the perimeter of the stope.

In this method, the IRMR values are adjusted for mining environment parameters (weathering, mining-induced stresses, joint orientation, and blasting effects) to obtain the MRMR. The Modified Stability Graph Method uses factors A (rock strength factor), B (joint orientation adjustment factor) and C (gravity adjustment factor) to determine the modified stability number N' (where, N'=Q'xAxBxC). Geometry of unsupported stope-backs and hangingwalls can be estimated from an empirical modified stability chart based upon the calculated stability number N' and the calculated HR.

Tables 16-3 and 16-4 present the values of the parameters used for the design of LH and MCF stopes with the above empirical design methods.

### Table 16-3: Laubscher's Stability Diagram

| IRMR and Adjustment Values | | Deposit | |
|---|---|---|---|
| | | Main | Esso |
| IRMR (Hangingwall) | | 47 | -- |
| IRMR (Back) | | 43 | -- |
| Adjustment Values | Weathering | 1.0 | -- |
| | Mining Induced Stress | 0.9 | -- |
| | Joint Orientation | 0.8 | -- |
| | Blasting | 0.94 | -- |
| MRMR (Hangingwall) | | 32 | -- |
| MRMR (Back) | | 30 | -- |

Source: EBA (2010a,b).

### Table 16-4: Modified Stability Graph Method

| Parameters | Deposit | |
|---|---|---|
| | Main | Esso |
| Q (Hangingwall) | 9 | 14 |
| Q (Back) | 13 | 18 |
| Stress Reduction Factor | 2.5 | 2.5 |
| Q' (Hangingwall) | 23 | 35 |
| Q' (Back) | 32 | 44 |
| A | 0.8 | 0.3 |
| B | 0.2 | 0.2 |
| C | 5 | 5.5 |
| N' (Hangingwall) | 18 | 11.5 |
| N' (Back) | 10 | 5.3 |

Source: EBA (2010a,b).

An empirical design method based on the NGI Q-System was used for the design of support for the access drifts. Parameters used for this empirical design are the Q value of the waste rock and the equivalent stress support ratio (ESR).

#### 16.3.4.1 Longhole Stoping Design

For 10 m wide and 20 m high stopes (average stope dimensions), the span length was estimated for stope backs and hangingwalls consisting of mineralized material and tuffaceous rock, respectively. As described in this section, the rock mass quality of the hangingwall rock is similar to the mineralized material.

The hangingwall is considered to be inclined at 65° to 75° with the horizontal, and the back is considered to be flat.

Based on the empirical design methods described above, the maximum unsupported hangingwall span distance is to be approximately 30 to 40 m on the Main deposit and approximately 30 m on the Esso deposit.

No support is required on the stope back for the Main and Esso deposits.

Longer panels can be achieved by installing cable bolting on the hangingwall. At this pre-feasibility stage, an estimated cable bolting density of 0.15 bolts/m² (which corresponds to a 2.6 m by 2.6 m pattern) was considered for both deposits to provide hangingwall support, with cable bolting consisting of 6 to 9 m long, plain seven strand cable bolts.

#### 16.3.4.2 Mechanized Cut and Fill Design

The stability of 5 m wide, 5 m high lifts was assessed for the proposed 25 m high stopping blocks. Based on the empirical method described in this section, the stability of the proposed temporary MCF excavations are to be stable for unsupported spans.

Since the Laubscher's stability diagram and the modified stability chart are empirical design methods, they serve only as guides in terms of the assessment of excavation stability and support requirements. Within the temporary drift it would be possible that loose and potentially unstable blocks will be present possibly as a result of blast damage or relaxation of the rock due to adjacent blasting. The majority of large potentially unstable blocks could be stabilized by scaling and spot rock bolting (Swellex or equivalent). For cost estimation purposes at this prefeasibility stage, one 25 mm diameter, 3 to 4 m long bolt (or equivalent) should be considered every 1.5 to 2.0 m along the alignment of the excavation.

#### 16.3.4.3 Access Drifts

Cross-sections of proposed access drifts through waste rock are 5 m wide and 5 m high. The proposed locations of the drifts are above the hangingwall, where tuffs are the predominant type of rocks.

The assessment of the stability and estimation of support required for the proposed access drifts were carried out, as mentioned before in this section, with an empirical design method based on the tunnelling NGI Q-System. For this purpose, the Q-value of the waste rock and ESR are required for the stability assessment. The adopted Q values were 10 and 22 for the Main and Esso deposits, respectively. An ESR of 3 was used on this assessment (ESR suitable for temporary mini openings). Given the Q-values and ESR value listed above, the drift may not require support.

Since the NGI Q-system is an empirical rock mass classification, it is only a guide in terms of the assessment of excavation stability and support requirements. Within the drift it would be possible that several loose and potentially unstable blocks will be present possibly as a result of blast damage or loosening/relaxation of the rock due to adjacent blasting. The majority of larger potentially unstable blocks could be stabilized by scaling with spot rock bolting (Swellex or equivalent). For cost estimation purposes at this prefeasibility stage only, one 25 mm diameter, 3 m to 4 m long bolt should be considered every 1.5 to 2 m along the centerline of the drift.

#### 16.3.4.4 Safety Precautions for MCF Stopes and Access Drifts

In addition to the recommended spot bolting for the MCF stopes and access drifts as described above, safety support systems are recommended at these excavation developments to provide protection for personnel and equipment. This support is not called upon to support very heavy loads due to large wedge failures or due to stress induced instability, but its function is to provide protection from small rock falls. A safety support system may consist of a welded wire mesh extending over the roof and upper sidewalls of the drift. The mesh should be restrained with 2 m long anchored rock bolts (or 0.5 to 1 m long split sets) on a 2 m by 2 m grid.

#### 16.3.4.5 Main Deposit Starter Pit Slopes

The proposed starter pit at the Main deposit consists of 10 m high double benches with 70° bench face angles and 8 m wide catch benches. From its highest point, the pit is 60 m deep. Based on the 2008 borehole information and from review of historical reports, the starter pit would consist mainly of tuffaceous rocks (Domain 2 rock type). Bieniawski's RMR rating indicates the rock within this domain is classified at the upper end of fair. The intact tuffaceous rock is classified as medium strong to strong rock.

Given the quality of the rock mass, the strength of the rock, the generalized fabric description and the relative shallow depth of the proposed pit, the stability of the proposed benches at the starter pit will tend to be governed by the structural fabric of the rock mass. Therefore, stability of the slopes was assessed using kinematic analysis.

Preliminary kinematic stability assessment of the stability of the benches based on the generalized fabric information described in this section indicates no kinematic potential stability modes (planar, wedge or toppling) are to be formed on the proposed bench slopes. Therefore, the proposed configuration of the benches of the starter pit is considered to be adequate. However, the stability of the benches should be assessed at the feasibility stage once additional geotechnical information becomes available (primarily with oriented core).

The Inter-ramp angle (IRA) for the proposed bench geometry (70° slope, 10 m high benches) corresponds to 40°. The overall pit slope is to be the same as the IRA slope or approximately 5° to 10° flatter on slope sectors crossed by ramps. Given the known rock quality, it is considered that the moderate proposed IRA angle and overall angle of the pit are adequate. In addition to the rock mass quality, the stability of the IRA and overall pit slope may be affected by major faulting in the immediate vicinity of the proposed open pit. This aspect is to be assessed at the feasibility stage once more information about major faulting becomes available.

The slope angles recommended above assume that controlled blasting methods will be used for excavation of the slope faces.

#### 16.3.4.6 Recommendations for Feasibility Stage

Additional geotechnical characterization and analyses should be conducted at the feasibility and design levels for each of the proposed excavations.

Therefore, the feasibility stage study requires a geotechnical program for data collection from surface and oriented core drillholes to further characterize the quality of the rock mass with the RMR and Q-System mass classification systems. The geotechnical program should include geotechnical laboratory testing (Unconfined Compression Strength, Direct Shear, Brazilian, Tri-axial), field point-load testing, permeability assessment by packer testing, and installation of selected piezometers for evaluating water heads and a thermistor cable to evaluate the presence of permafrost. In addition, the feasibility study requires investigating the presence of large faults and assessment of in-situ stresses.

The proposed excavations discussed in this section (underground or surface excavations) should be re-assessed with the newly obtained geotechnical data.

## 16.4 Mining Methods

The Main and Esso deposits vary in dip from 30 to 70º and in width from 3 to 20 m. The Main deposit essentially outcrops on surface and extends to depth of approximately 250 m below surface, while the Esso deposit extends to a depth of 420 to 600 m below surface. The Esso deposit is approximately 1,500 m to west of the main deposit.

Two underground mining methods are proposed: MCF and sublevel LH stoping with paste backfill. The MCF will be utilized in the shallow dipping areas (less than 50º of both ore deposits, while LH is proposed for areas where the dip is greater than 50º.

Approximately 61% of the total underground mine ore tonnes will be mined with MCF stopes and the remaining 39% with LH stopes. The majority of the stopes will be mined longitudinally (along strike) with both methods.

Approximately 4% (446 kt) of the total ore tonnes mined will be from the Main open pit.

### 16.4.1 Longhole Stoping

Longhole stoping provides high productivity at low mining costs from a small number of working faces. All stopes will be filled with a mixture of paste fill and/or development waste.

Sublevels will be developed at intervals of 15 to 20 m depending on ore body geometry. Sublevels will be developed in waste to provide access for ore drifting and slashing. The sublevels developed in ore will be 5 m high and initially 5 m wide and then slashed to the ore boundaries. The ore sublevels will provide access for drilling, blasting, ground support and ore mucking.

Average LH stope dimensions of 10 m wide, 20 m high with 3.5 m ring burdens have been assumed for production and cost estimation. Cycle times for stoping operations are based on best practice productivities for personnel and equipment productivities. Blasthole drilling will use top hammer drills to drill 15 to 19.5 m long down holes from the upper sill to the lower extraction level. Blastholes will be 89 mm diameter, drilled on a 3.5 m burden by 1.5 m toe spacing pattern and will be charged with ANFO and high explosive boosters and initiated with NONEL caps. An approximate 0.37 kilograms per tonne (kg/t) powder factor has been assumed for LH blasting.

The broken ore will be mucked from the bottom of the stope by remote control load haul dump units (LHDs) and loaded into trucks and hauled to surface. The mined out stopes will then be backfilled.

### 16.4.2 Mechanized Cut and Fill Stoping

Mechanized cut and fill mining will be utilized in shallower dipping areas of both the Main and Esso deposits. MCF is a lower productivity, higher cost mining method than LH stoping, but provides highly selective mining with minimal dilution. Stopes can be sized with irregular backs and walls to match the ore boundaries.

Each 25 m high stoping block is accessed by a -15% access ramp and mined in five, 5 m high MCF stopes. Stopes are developed on the lowest level first, and each subsequent stope or 5 m lift is developed above the depleted and backfilled stope.

Production and cost estimation have been based on 5 m high and 5 m wide stopes.

Two-boom electric-hydraulic jumbos will drill 4 m long rounds on a standard development heading pattern with 45 mm diameter blastholes which will be charged with high explosives primers and ANFO and initiated with NONEL caps. After blasting, the heading will be washed and scaled and then bolted with a mechanized bolter as required.

The broken ore will then be mucked with LHD's into trucks and hauled to surface. The completed 5 m high stope is then filled with hydraulic backfill and/or development waste. The next 5 m lift will then commence on top of the hardened fill of the previous lift.

### 16.4.3 Mine Production Criteria

A daily production rate of 2,500 t/d over 365 operating days per year has been selected to mine 912,500 ore tonnes annually. The Main pit, at 1,223 t/d supplements underground mining in year 1. Underground mining starts in the Main deposit, and then proceeds to the Esso deposit and has been sequenced to deliver higher grades in the early years of the Project. The underground production rates by deposit are outlined below:

- Year 1: Main 1,277 t/d;
- Year 2: Main 2,500 t/d;
- Year 3: Main 1,846 t/d, Esso 654 t/d;
- Years 4 to 7: Main 1,000 t/d, Esso 1,500 t/d;
- Year 8: Main 1,000 t/d, Esso 1,500 t/d until complete, then Main 2,500 t/d; and
- Years: 9 to 12: Main 2,500 t/d.

## 16.5 Backfill

### 16.5.1 Backfill Summary

Backfill is an integral part of the underground mine plan and will become important in the operations phase as well. The backfill serves several purposes:

- Underground support and working platform – MCF accounts for more than half of scheduled production and will require the placement of backfill as a working surface, as well as wall support. The remainder of the production will come from LH stopes, some of which will be mined in adjacent panels and thus will also require backfill for support.

- Storage – Several waste products are generated in the mine plan including process plant tailings, waste rock (both PAG and non-PAG). The underground stopes are desirable locations for permanent storage of these materials.
- Mine development waste rock will preferentially be stored underground in MCF stopes as part of the regular backfill cycle. Some waste rock will be stored on the surface when there are no stopes in a backfill cycle, or there is a need for non-PAG construction material such as during the site construction phase.

Waste rock will be scheduled so that material mined early in the underground development effort and is more likely to be classified as non-PAG will be hauled and used on surface. As the stoping reaches a steady state underground, development rock will preferentially be stored underground. The backfill plan calls for all waste rock generated after production year 2 to be stored underground. This plan takes advantage of the location and timing of the mine development and allows for placing predominantly PAG waste rock underground and non-PAG on surface. After year 2, the only non-PAG waste rock that is hauled is that material required for on-going construction and closure requirements.

Un-cemented waste rock has its limitations for backfill, and an insufficient volume of waste rock is available for the backfill requirement. Unconsolidated fill can be used in MCF stopes or in LH stopes that will not be disturbed in a later phase of mining. However, panels of LH stopes that are adjacent to minable ore must be filled with a cemented material to provide some support and minimize dilution. Therefore, the use of paste fill has been incorporated into the mine plan.

Paste fill consists of process tailings partially dewatered and mixed with Portland cement. This material is of a consistency that can be positively directed to specific locations by positive displacement pump and pipeline. The paste fill plant will be operated such that tailings required for backfill will be converted into fill. In general, 50% of the tailings can be accommodated in underground stoping areas.

The remainder of the tailings will be deposited as paste on surface in a lined facility.

### 16.5.2 Backfill Plan

Conceptual backfill planning and design was completed by Mine Paste Engineering Ltd. Complete details of their study can be found in their *Backfill Concept Study* report dated February 14, 2011 (Mine Paste Engineering Ltd. 2011).

The preferred option for paste preparation for surface and backfill applications based on an assessment of capital and operating costs as well as minimizing environmental footprint is the use of a single deep tank paste thickener mechanism to densify tailings to high density paste-like product. The system will provide the benefits of paste in terms of high backfill rates and a good engineered fill though at lower capital costs and lower pumping pressures. It will also provide a non-segregating paste-like high-density slurry for surface disposal at lower pumping pressures yet with the ability to reduce dam containment and improve the environmental impact relative to conventional slurry system.

From a central paste operation located next to the mill, paste underflow will be directed to one of two locations, being:

1. Cemented (blended binder) backfill preparation system. Paste thickener underflow will be discharged into a mixer along with cement (blended with ground iron blast furnace slag or fly ash to form the binder) and pumped using large positive displacement pumps to transport the paste backfill to either to the Main or Esso deposits; or
2. By-pass the mixer and binder (cement) system, to be transported by large positive displacement pumps to the surface tailings management facility (TMF).

The flowsheet and mass balance is based on an assessment of the tailings through a review of various reports, as well as on a broad range of experience with similar projects. While at the time of writing, there is insufficient test data to support the assumptions, it's believed that the values used herein are reasonable for selecting a preferred option based on costs estimated.

Testing to complete this assessment and move forward is required for the following:

- Dewatering assessment including flocculation, thickening and paste thickening (paste thickening is a specialized test, and may require pilot plant testing pending laboratory results);
- Rheology from slurry to ultra-high densities; and
- Unconfined compressive strength test to determine potential strength gain, strength loss due to sulphation, and economics of binder versus slump (required to assess capital and operating costs of plant location, pumping versus gravity distribution, and major binder operating costs).

The paste plant and associated equipment has been sized for the production rate of 912,500 dry tonnes per year (dt/y) of mine tailings at an operating rate of 2,500 dry tonnes per day (dt/d) or 102 dry tonnes per hour (dt/h). The equipment within the paste plant has been sized for intermittent surges of 15% or 117 dt/h. It is anticipated that the tailings will be received from the flotation circuits at approximately 26 percent dry weight (wt%) solids, and dewatered to approximately 73 to 76 wt% solids. The equipment selected will provide cemented paste backfill, and if required, will be well suited for surface tailings disposal as a high density paste-like slurry pending any pertinent disposal studies.

Other flowsheets evaluated consisted of high-rate thickeners with either vacuum or pressure filtration. While these flowsheets would ultimately provide thick paste and the operation would benefit from lower day-to-day binder consumption, the assessment indicates that the payback would not be sufficient to warrant the increased upfront capital for filters and larger pumps and higher cost pipelines.

An assessment of two possible paste plant locations was also considered to minimize capital costs, operating costs and power consumption. These are:

- Paste Plant located alongside the Mill to share services (PLC/control room communications, sub-station and motor control room, gland water, compressed and instrument air, process water return systems, sumps, etc., as well as washroom and lunch facilities). In addition, sharing of mill personnel to run the paste plant would reduce operating costs; and

- Paste Plant located above the Main deposit. While those items addressed above would be needed, the option to reduce binder consumption has significant payback through discharging by very high paste solids concentration. If binder approaches $400/t, and if more resources are found, it may be economically important to review this assessment.

Each distribution system (1) from the paste plant to underground as backfill (Main, Esso), and (2) TMF will require protection in terms of water/air flushing system to prevent blockage, as well as freezing during sustained cold periods. Start-up and shutdown of either system should normally be over the course of days or at minimum several shifts. Pouring backfill can normally continue between shifts aided by pipeline monitoring systems and remote camera systems.

The Process Flowsheet illustrates tailing streams will be received via a splitter box and diverted to the Agitated Thickener Pump Feed Box. The slurry is then pumped using centrifugal pumps to the top of the paste thickener, roughly 20 m above ground level. Flocculation of solids will occur within a designed feedwell within the paste thickener to achieve flash thickening to enhance paste production and high underflow solids. The underflow is discharged into the high intensity mixer along with binder (cement/slag) and slump control water, if required.

While it is likely that a continuous mixing system is reasonable, a batch mixing system is shown, pending future test results. Close attention to solids concentration will be required within the plant to safeguard the pumps and pipelines distributions systems. The back-up pump may be used in case of emergency to assist in unplugging pipelines.

Hydraulically driven piston pumps are recommended for both paste backfill and surface tailings disposal, based on costs, power and duty. One operating and one back-up pump will be required to ensure overall system availability. Each pump will be powered by a 250 horse power (hp) electric power pack. Maximum pressures are expected when pumping to the surface access (by borehole) to the upper-most top sill of the Main deposit (elevation 1,595 m), in the order of 77 Bar for a paste-like product at friction losses ≤ 4 kilopascals per metre (kPa/m). Typically paste is defined as having a measureable slump, which is over a range of 50 mm (2-inch) for thick concrete to a maximum 250 mm (10-inch) for less viscous paste. As more water is added to a 250 mm slump paste, it becomes increasing difficult to reliably generate the same measureable slump value, since at that point, even slurry will measure approximately 250 to 270 mm slump. However at this point, slight water additions dramatically reduce friction losses and pumping becomes substantially easier.

Pumping to the Esso deposit will require less pressure due to down gradient of 100 m elevation. A key objective is to manage the paste viscosity through solids concentration to not exceed 4 kPa/m.

The paste distribution system will consist of nominal 6-inch (150 mm) I.D. pipe for both surface and underground distribution system. While a nominal pipeline velocity of 1 metres per second (m/s) is sought, pipe diameter selected is nominal 6-inch Schedule 80 and Schedule 40, with short sections of HDPE DR-11 (160 psi) up to within 250 m from the discharge points. It is believed that the very fine grind of the tailings will allow stable transport as a paste-like product even slightly below 1 m/s.

Surface pipelines will require burial, insulation and possibly heat-tracing to protect the installation during sustained cold periods. The underground distribution system will consist of pipelines traveling along surface above the deposit, with cased boreholes installed from level to level. Each main level will require a primary distribution pipe line, and branched off into the secondary short runs near the stopes.

Following completion of a backfill pour or upon emergency stoppages, flushing of the pipelines will be conducted through the use of a pipeline pig insert followed with water and compressed air to ensure removal of cemented products and to mitigate the potential of freezing.

Barricades constructed largely of development waste rock berms will be required at the stope entrances to contain the paste fill. Since high-slump paste is to be used, there may be a need to use some shotcrete to minimize some seepage through the rock barricade. Depending on costs and effectiveness, use of fabrene type cloth on the inside of the rock berm may be attempted as a lower cost option. Shotcrete application may be required over full face of the rock berm, while shotcrete barricades may be considered where necessary.

## 16.6 Mine Ventilation

Ventilation design has been completed in order to layout primary and secondary ventilation circuits and estimate conceptual level capital and operating costs. JDS completed an initial design which was refined and modelled by Stantec Mining (Stantec).

Analysis of the total air requirements is based on the following two main factors:

- The underground yearly production rates for each deposit; and
- The air requirements of diesel equipment that will be utilized in the Project.

Ventilation requirements for the initial fleet of diesel equipment utilized underground is summarized in Table 16-5.

**Table 16-5: Diesel Equipment Ventilation Requirements**

| Equipment | Units | Total (hp) | Availability (%) | Utilization (%) | Utilized (hp) | Air Volume (cfm) |
|---|---|---|---|---|---|---|
| Jumbo | 2 | 296 | 85 | 20 | 50 | 5,000 |
| LH Drill | 1 | 99 | 85 | 20 | 17 | 1,700 |
| Mechanized Bolter | 2 | 198 | 85 | 30 | 42 | 4,200 |
| 50 t Truck | 3 | 1,575 | 85 | 90 | 1,205 | 120,500 |
| 30 t Truck | 1 | 400 | 85 | 85 | 289 | 28,900 |
| 5.0 $m^3$ LHD | 3 | 885 | 85 | 90 | 675 | 67,700 |
| Grader | 1 | 110 | 90 | 50 | 50 | 5,000 |
| Scissor Lift | 2 | 294 | 90 | 30 | 79 | 7,900 |
| ANFO Loader | 1 | 147 | 90 | 30 | 40 | 4,000 |
| Fuel/Lube Truck | 1 | 147 | 90 | 35 | 46 | 4,600 |
| Utility Truck | 2 | 294 | 90 | 25 | 66 | 6,600 |
| Personnel Carrier | 3 | 441 | 90 | 20 | 79 | 7,900 |
| Utility Tractors/Trucks | 3 | 396 | 90 | 50 | 178 | 17,800 |
| Contingency (20%) | | | | | | 56,400 |
| **Total** | | | | | | **338,200** |

Source: JDS (2017).

With the maximum production rate of 2,500 t/d, the total ventilation required is 281,800 cubic feet per metre (cfm) (133 cubic metres per second [m³/s]). When considering a ventilation contingency of 20%, the total ventilation required will be 338,200 cfm (160 m³/s). Stantec's ventilation plans allows 350,000 cfm (165 m³/s) for the Main mine and 60% of this (210,000 cfm or 99 m³/s) for the Esso mine, which is at 1,500 t/d during peak production in years 4 through 8. Therefore, in summary Stantec's modeling uses 350,000 cfm (165 m³/s) and 210,000 cfm (99 m³/s) for the Main and Esso mines, respectively.

#### 16.6.1.1 Main Mine Fans

The fans will be located at the Portal Ramp and at the West Ramp Vent Raise extended up to the surface. The fans recommended for each of these locations are listed below.

Two fans at the New Portal Ramp with the following specifications:

- Series 152-091-1200-A-2 with 110 kW drive, 47.2 m³/s at 1.26 kPa (100,000 cfm at 5.06 in w.g.)

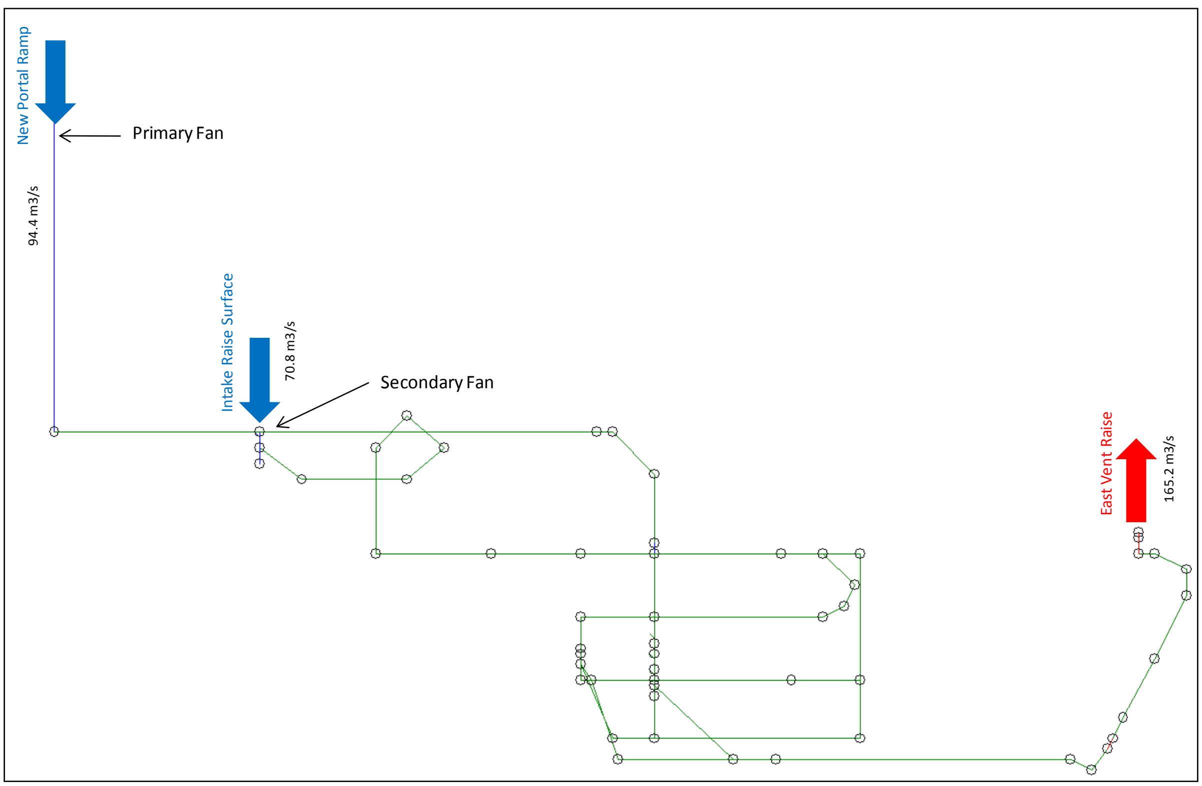
Two fans at the West Ramp Vent Raise with the following specifications:

- Series 152-091-1200-A-2 with 110 kW drive, 35.4 m³/s at 0.84 kPa (75,000 cfm at 3.36 in w.g.)

### 16.6.2 Main Mine

Figure 16-1 illustrates the overall network model of the Main mine ventilation design.

**Figure 16-1: Main Mine Ventilation Simulation Network**



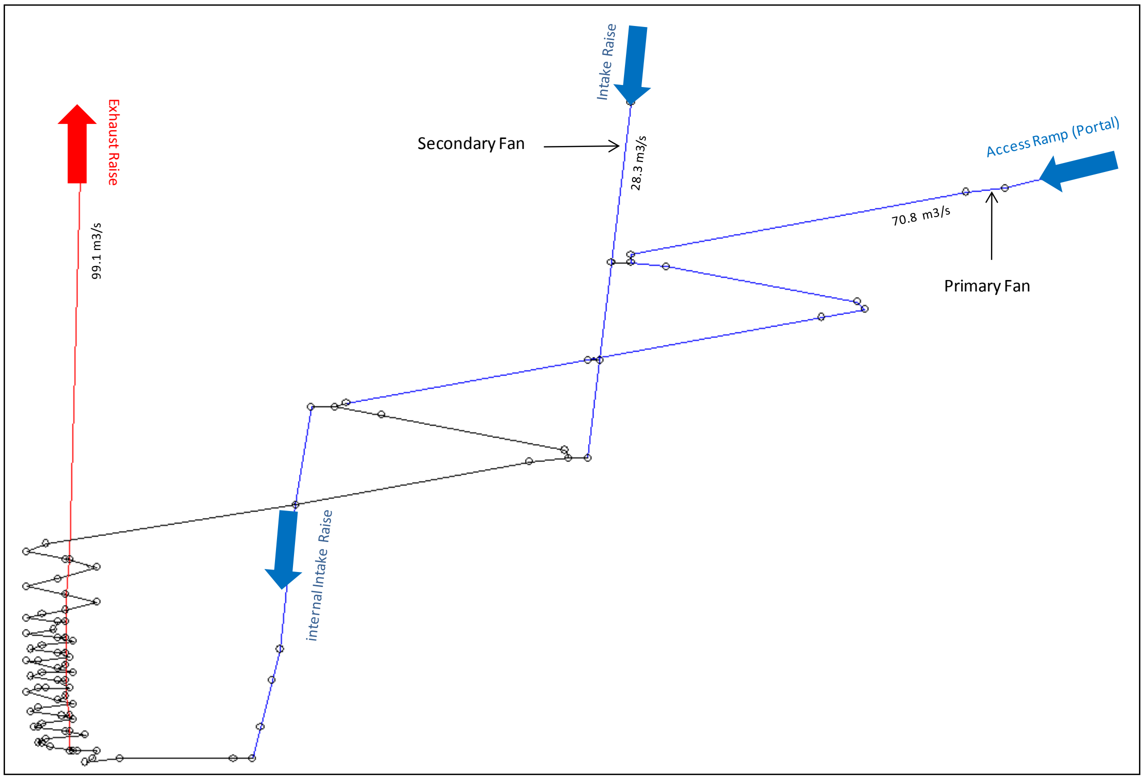

Source: Stantec (2011).

During full production, fresh air will enter the mine through the primary decline and the west fresh air raise. Air requirements are 200,000 cfm (94.4 m³/s) and 150,000 cfm (70.8 m³/s) for the portal ramp and west intake raise respectively. Fresh air exits the mine through the east exhaust raise. Ventilation will be distributed to the working areas via auxiliary ventilation fans and ducting. The development headings have been sized to accommodate the large ducting to reduce head losses.

### 16.6.3 Esso Mine

Figure 16-2 illustrates the overall network model of the Esso mine ventilation design.

**Figure 16-2: Esso Mine Ventilation Simulation Network**




Source: Stantec (2011).

A fresh air requirement of 210,000 cfm (99 m³/s) is required for full production at the Esso Mine. This requirement will be delivered by primary intake air fans with a capacity of 150,000 cfm (71 m³/s) at the portal and a secondary intake air fans with a capacity of 60,000 cfm (28 m³/s) at the access ramp vent raise.

In the this model, fresh air can flow from the top access ramp to the bottom with a flow of 50.6 m³/s, which allows adequate fresh air flow to accommodate the truck traffic hauling ore out to the surface. The exhaust flows through the exhaust raise to surface to complete the circuit. The air flow reaching the Main ramp (spiral) is adequate to provide ventilation for the truck traffic hauling ore out to the surface. Secondary ventilation in production areas can be provided from the Main ramp and using auxiliary fans to distribute fresh air in working areas.

The following fans are recommended for the ventilation circuits.

#### 16.6.3.1 Esso Mine Fans

The fans will be located at the Access Ramp (Portal) and at Access Vent Raise. The fans recommended for each of these locations are listed below.

Two fans at the Access Ramp Portal with the following specifications:

- Series 125-080-1800-A-1 with 110 kW drive, 35.4 $m^3/s$ at 2.24 kPa (75,000 cfm at 9.0 in w.g.)

Two fans at the Access Vent Raise with the following specifications:

- Series 100-060-1800-C-1 with 75 kW drive, 14.2 $m^3/s$ at 2.24 kPa (30,000 cfm at 9.0 in w.g.)

## 16.7 Underground Mine Development and Layout

The primary access for the Main mine will be a single straight incline from a starting floor elevation of 1,522 metres above sea level (masl). The cross-sectional area will be 5 m high by 5 m wide to provide clearance for equipment, ventilation and services.

Two ramp systems will be driven off the primary access ramp, one to the east and the other to the west to provide access to the other Main deposit ore zones. The east incline ramp will be driven at a maximum grade of +15%. The west ramp will split into upper and lower ramps driven at grades of +/-15%.

Access to the Esso deposit will be a 2,600 m long decline ramp from surface to the 1,090 masl elevation at the top of the Esso ore body. This ramp will also be 5 m by 5 m and will have an average grade of -15%. A central ramp will then be developed to the bottom of the Esso deposit, with sublevels and accesses driven east and west to the Esso mining zones. Although not designed for exploration purposes, the Esso access ramp could be used for future exploration drilling of the Sumac deposit.

Development has been classified as capital and sustaining. Capital waste development is the mine's permanent infrastructure and includes primary and secondary ramps, ventilation raise accesses, primary sumps, ore pass accesses and permanent explosive storage cut-outs as well as main ventilation raises and ore passes. Sustaining lateral waste development includes ore stope accesses and sublevels, temporary sumps and remucks.

The initial pre-production development period is estimated to be approximately 18 months. All lateral capital development is assumed to be completed by Desert Star.

Capital lateral development totals 8,339 m and sustaining waste development totals 16,835 m.

The mine will require an extensive system of raises for ventilation as well as ore handling. Each deposit will have a central ore pass raise system with dump points at various levels to minimize internal mine haulage. A truck loading chute will be installed at the bottom of each ore pass. The ore pass lengths are 87 m and 104 m for the Main and Esso deposits, respectively.

There will be one exhaust raise to surface at the east of the Main mine. An internal system of fresh air raises will also be required to complete Main's primary and secondary ventilation circuits. Ventilation raising for the Main mine totals 374 m.

The ramp to Esso mine will have one ventilation raise to surface, and three internal raises that total 500 m, which will be used to ventilate the ramp during development and will be part of Esso's primary ventilation circuit. One additional raise to surface will be required for exhaust air as well as a series of internal ventilation raises. Total ventilation raise development for the Esso mine is 1,279 m.

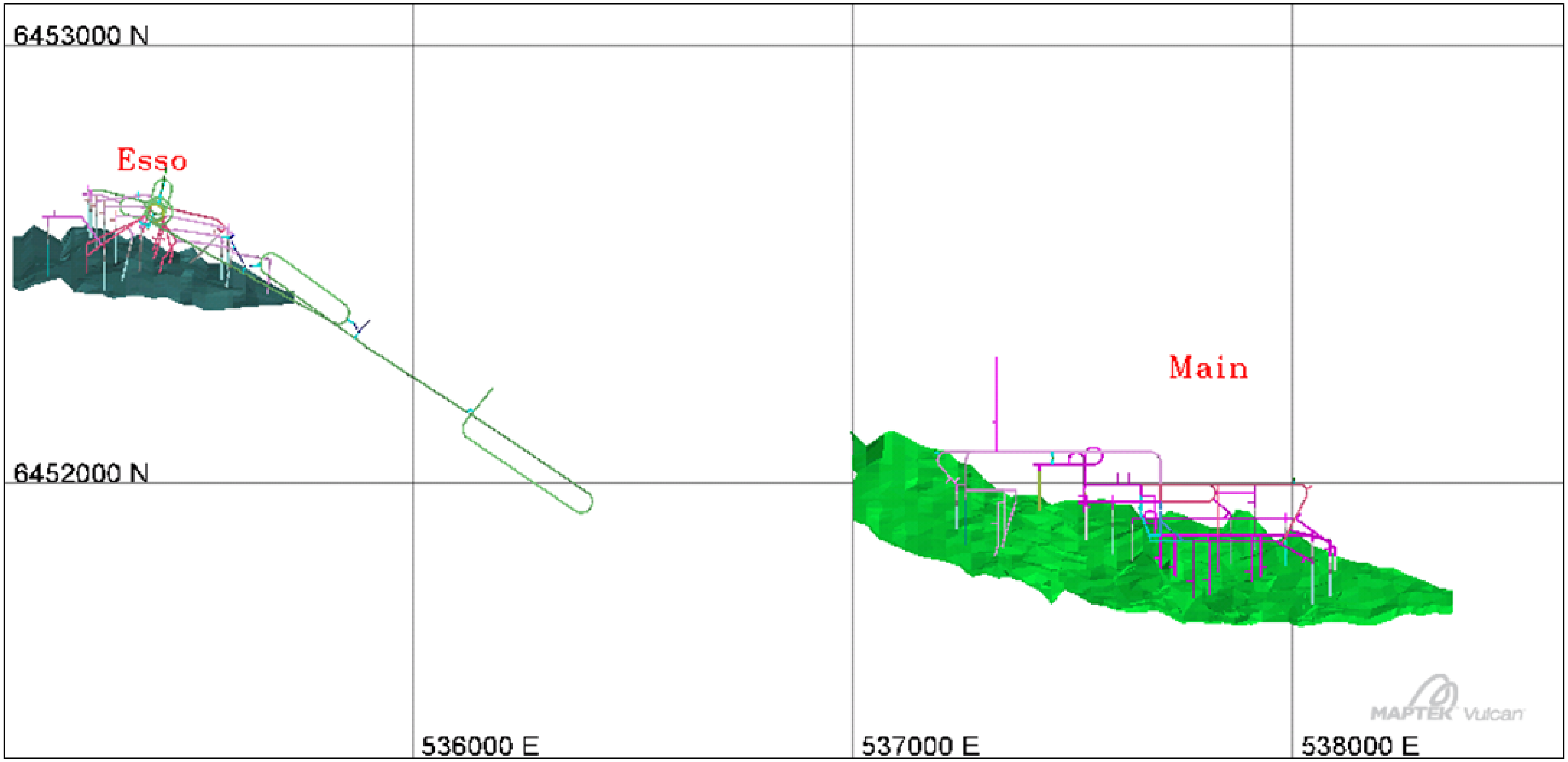
Ore pass and ventilation raise development for both mining areas is considered capital development and will be completed by a contractor and totals 1,757 m.

The following colours are used to depict the various development types and stope outlines in the figures that follow:

- Development ramps are shown are purple/pink and green for Esso;
- Ventilation raises are blue and green;
- Ore passes are yellow;
- MCF stope accesses are white/grey;
- MCF stopes are tan (Main) and light purple (Esso); and
- LH stopes and ore drifts are dark and light blue.

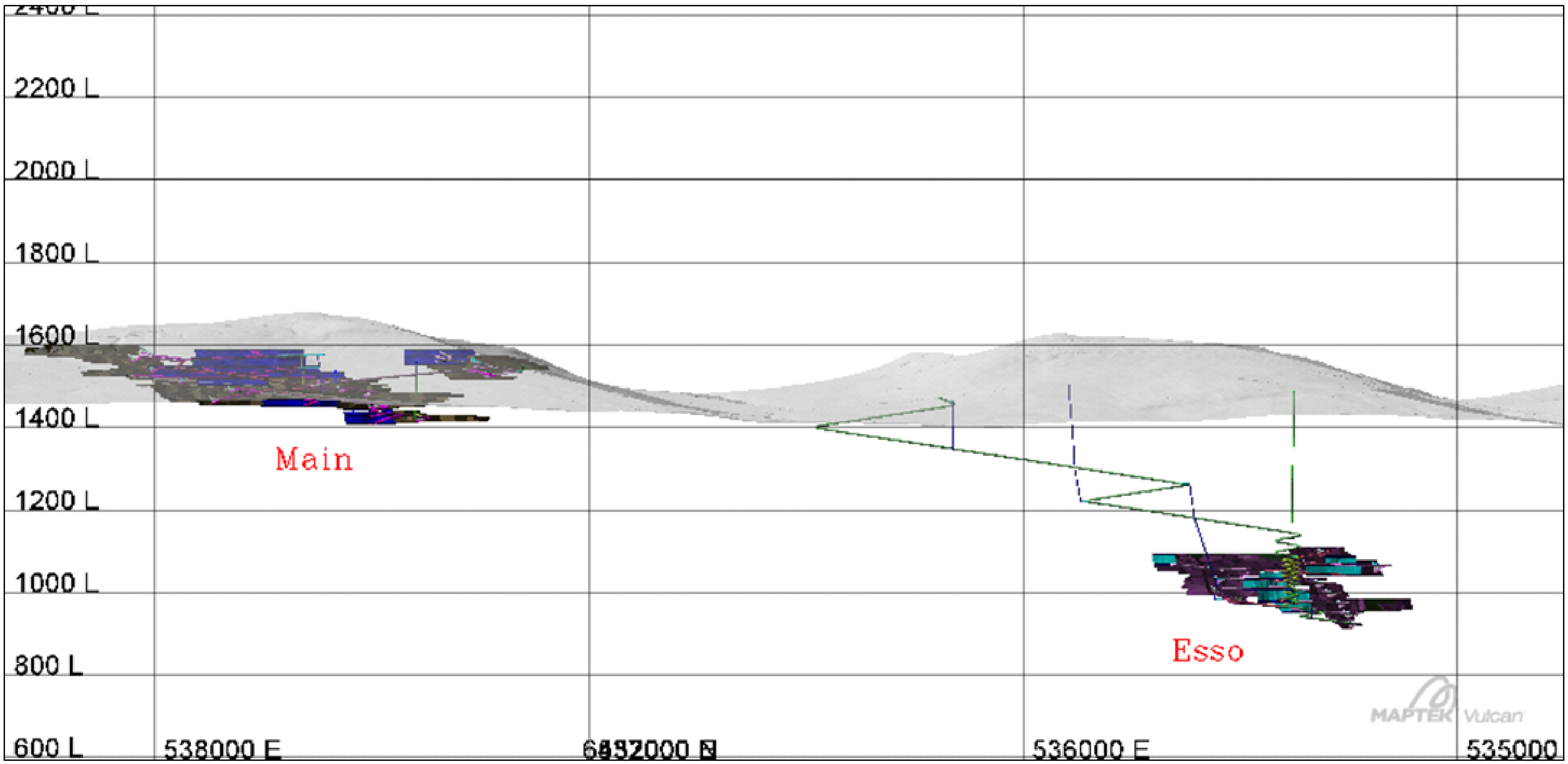
The mine development layout and Kutcho ore deposits is shown in Figure 16-3. Plan views, long sections, and 3D views of the Main and Esso mine layouts and stope designs are shown in Figures 16-4 to 16-10.

**Figure 16-3: Mine Layout and Kutcho Ore Deposits Plan View**



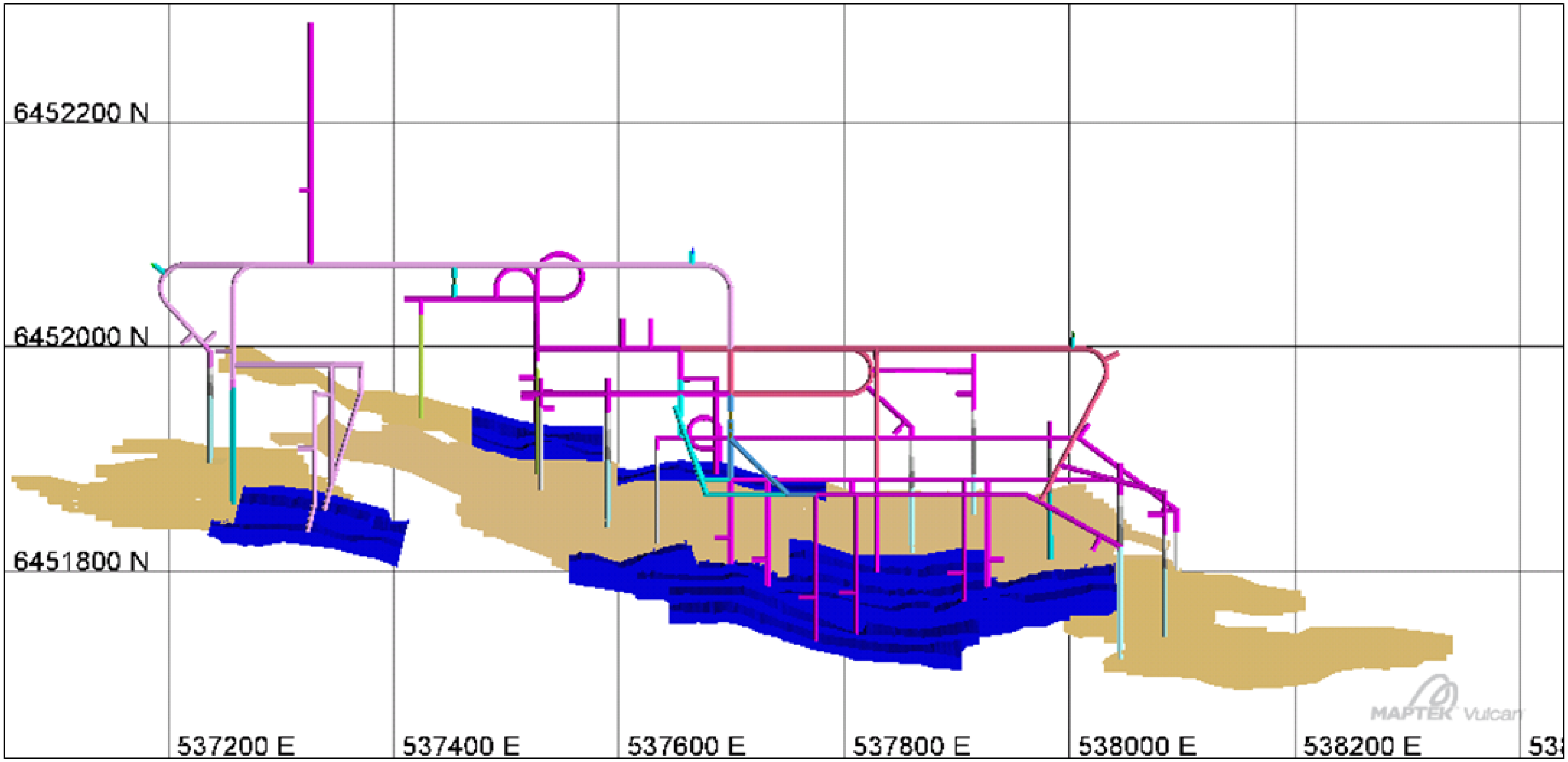

Source: JDS (2017).

**Figure 16-4: Mine Layout Long Section**



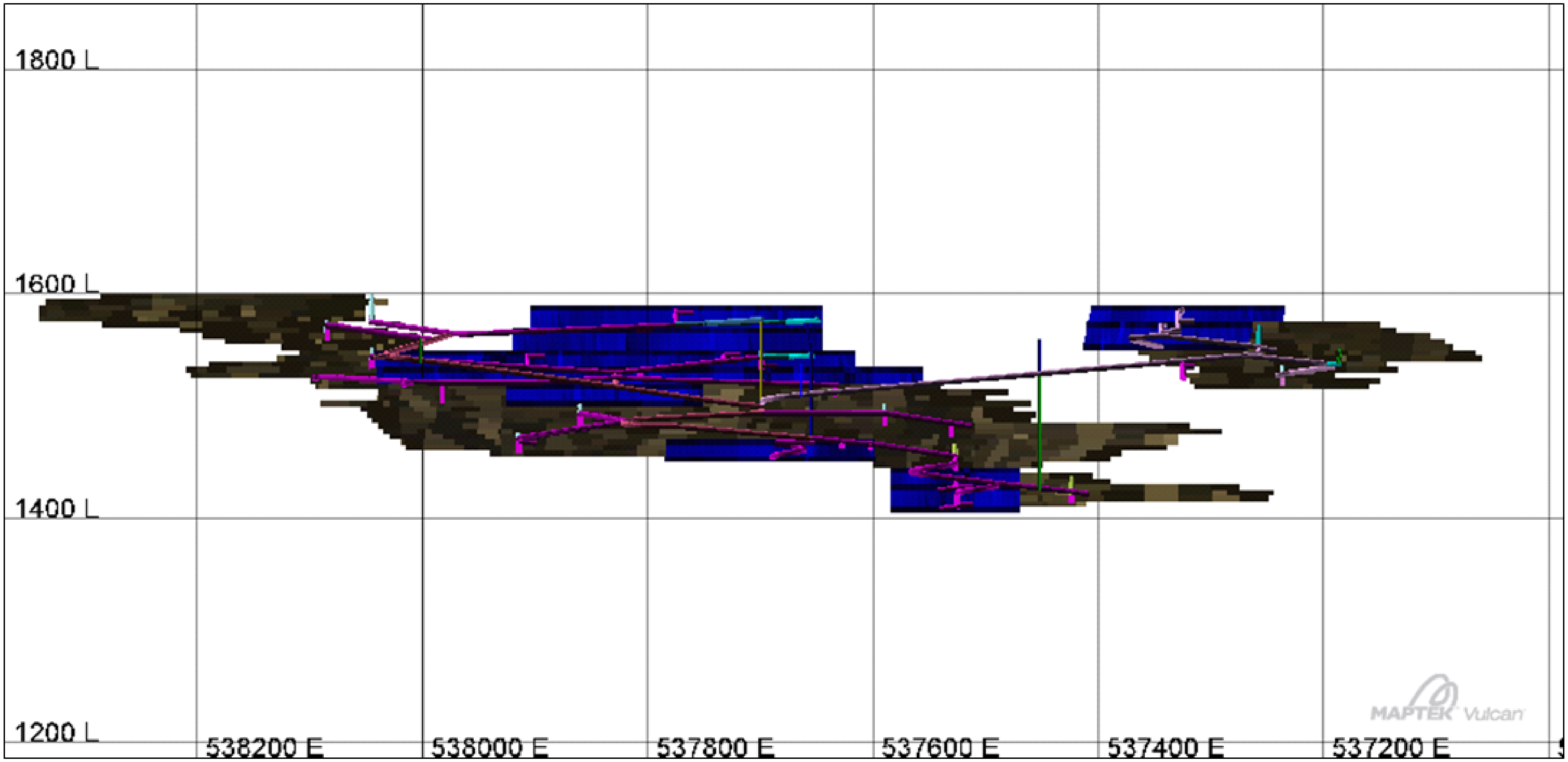

Source: JDS (2017).

**Figure 16-5: Main Mine Layout Plan View**



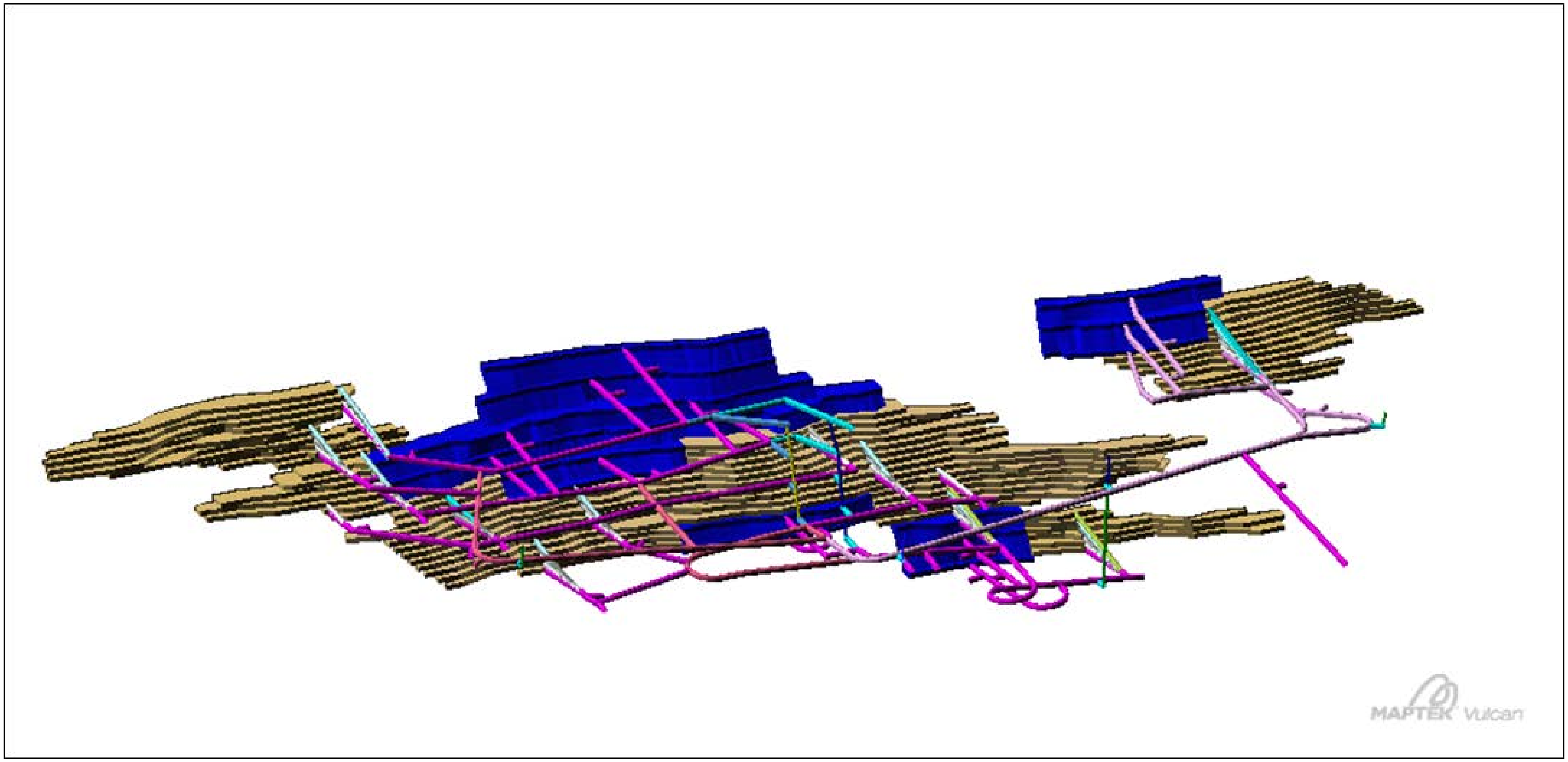

Source: JDS (2017).

**Figure 16-6: Main Mine Layout Long Section**



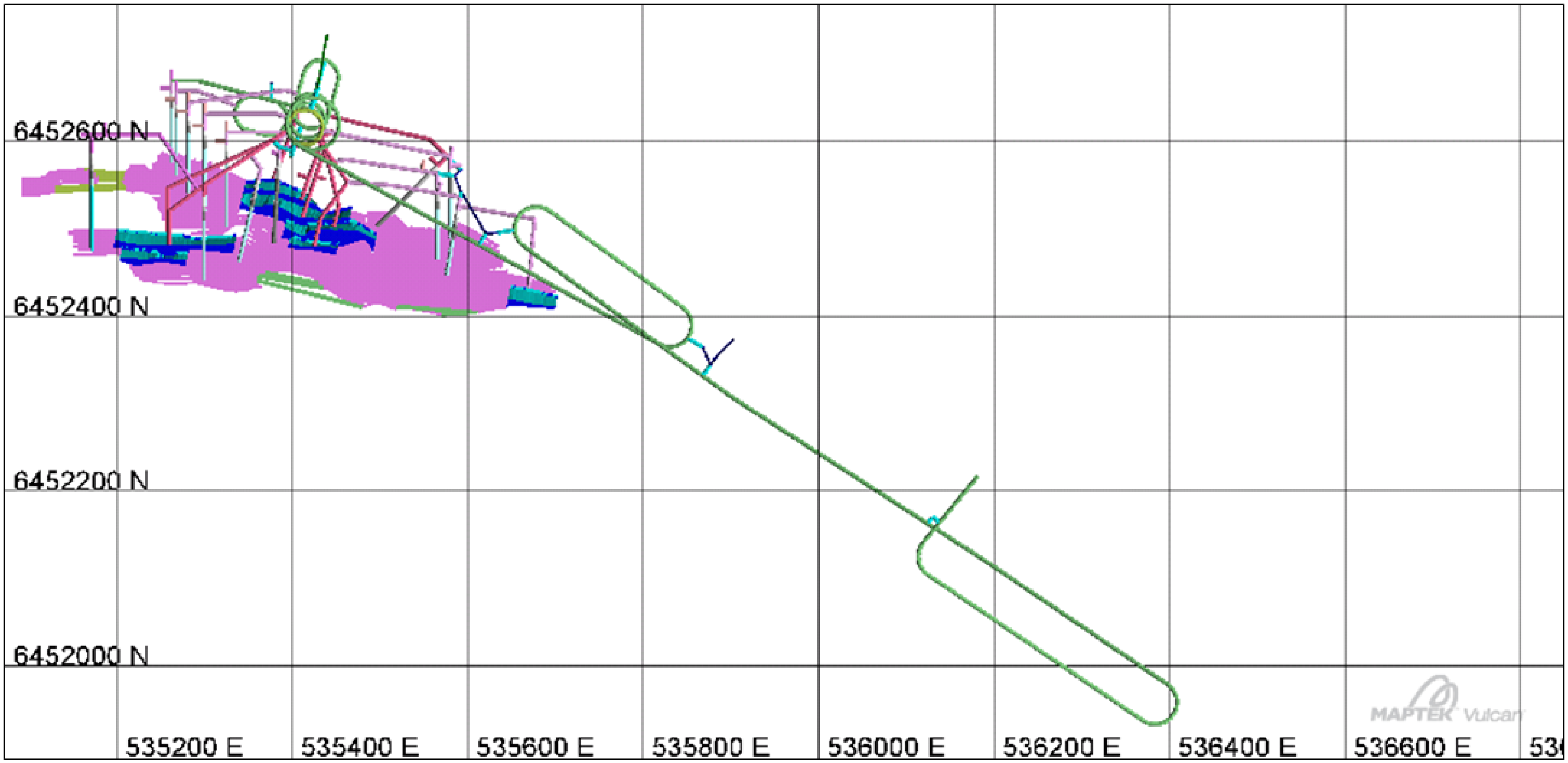

Source: JDS (2017).

**Figure 16-7: Main Mine Layout 3D View**



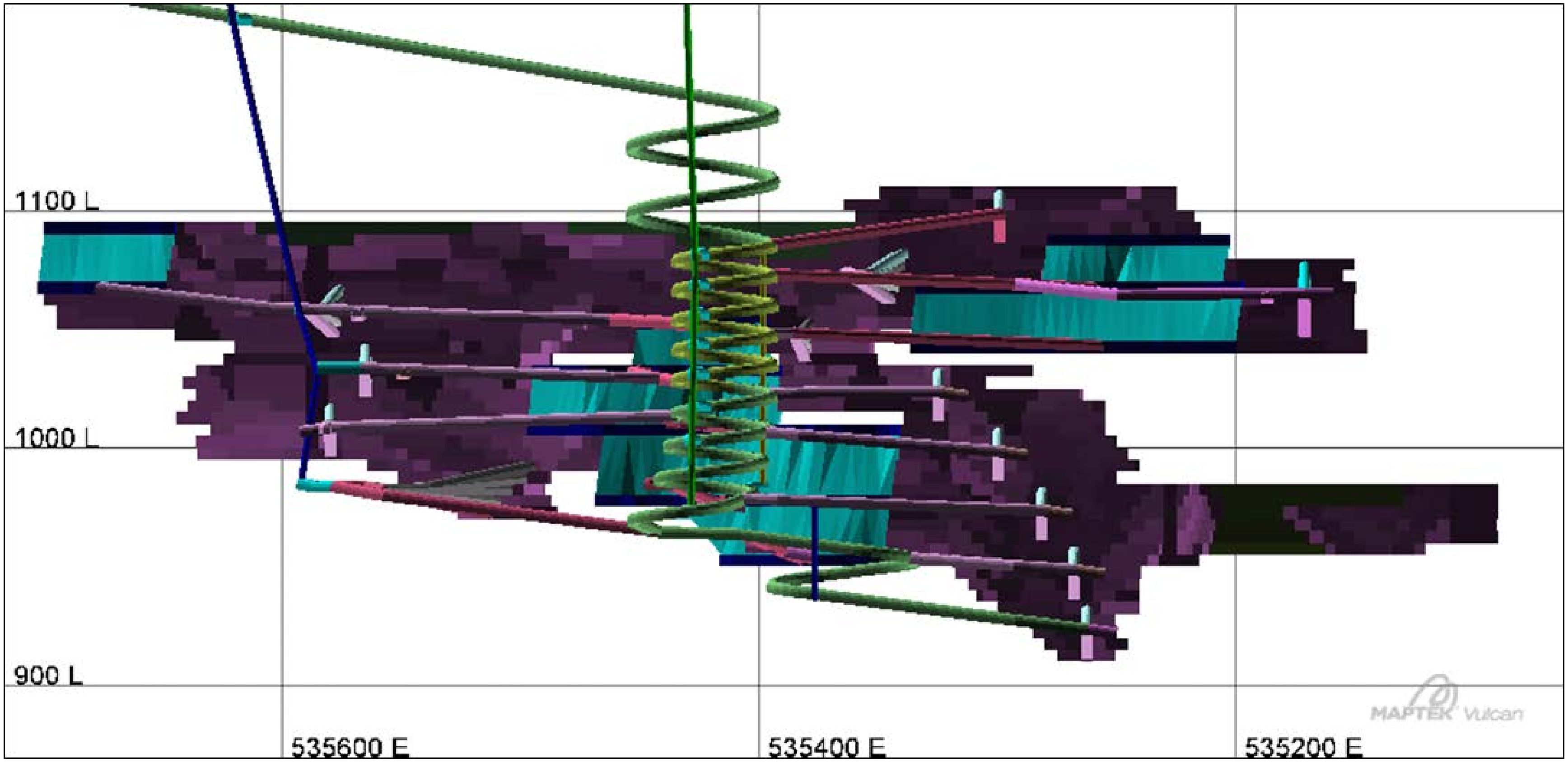
Source: JDS (2017).

**Figure 16-8: Esso Mine Layout Plan View**



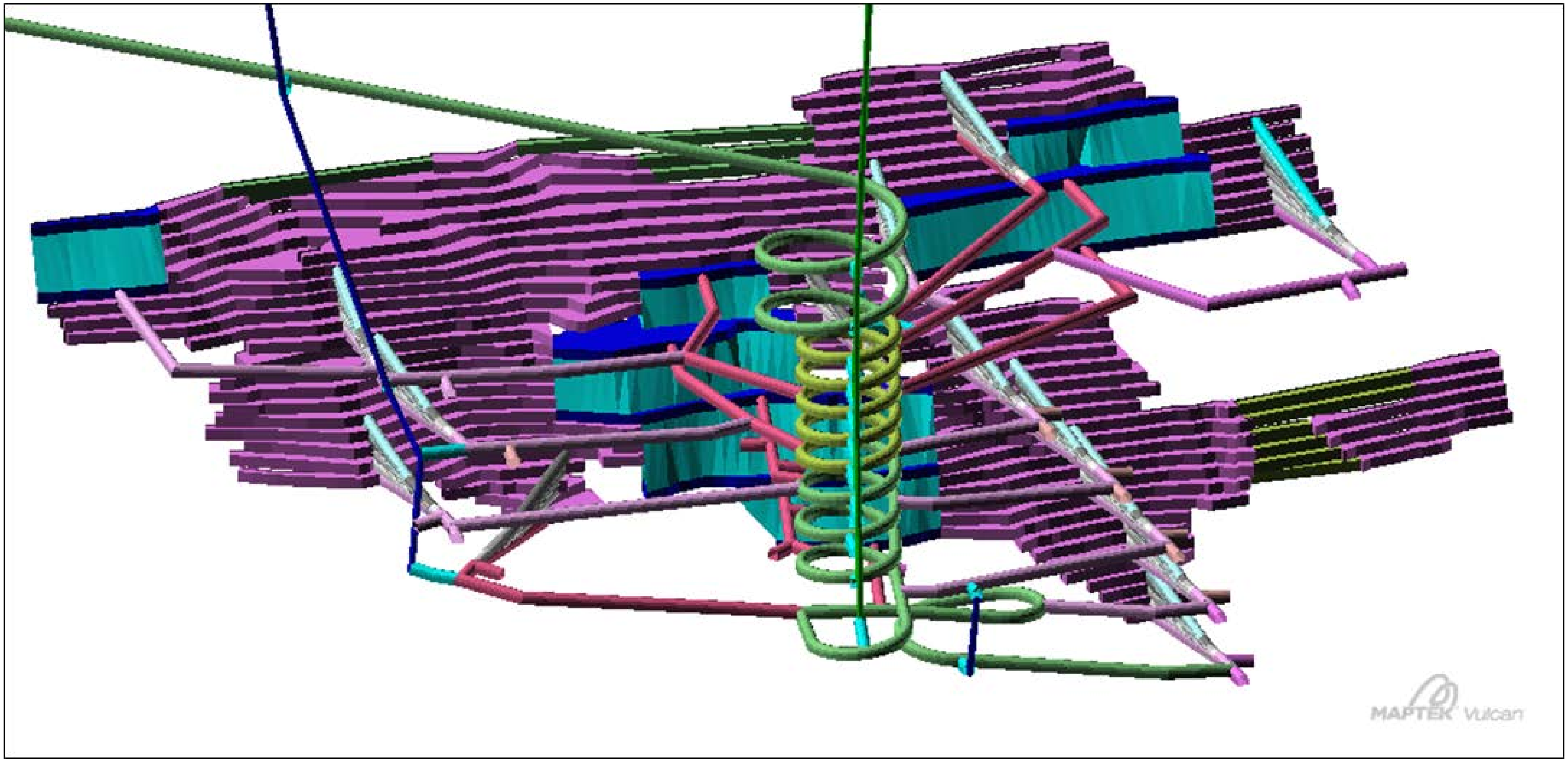

Source: JDS (2017).

**Figure 16-9: Esso Mine Layout Long Section**




Source: JDS (2017).

**Figure 16-10: Esso Mine Layout 3D View**




Source: JDS (2017).

## 16.8 Starter Pit

A small starter pit will be developed and mined by open pit mining methods to provide initial ore feed while the underground mine is being developed. Non-PAG waste rock from the pit will be used for surface construction. PAG pit waste rock will be stored temporarily in a surface storage facility adjacent to the pit. After completing the pit, PAG material will be stored permanently in the mined out pit and/or the tailings storage facility. The mined out pit will also be used for permanent tailings storage.

All starter pit ore and waste mining will be done by a contractor. Mobile equipment is provided by the Owner and will be used post open pit mining to maintain roads and miscellaneous earthworks.

### 16.8.1 Pit Design Criteria

The starter pit was designed with following criteria:

- Bench Height: 10 m (double benching);
- Bench Face Angle: 70º;
- Catch Bench Width: 8 m;
- Ramp Width: 16.5 m (Single Lane Traffic, CAT 777 class trucks);
- Ramp Grade: 10%;
- Dump Lifts: 10 m, with 10 m berms; and
- Dump Slope: 2 Horizontal: 1 Vertical.

### 16.8.2 Pit Production

The pit will provide 446,215 ore t at 1.84% Cu, 1.62% Zn, 23.2 g/t Ag, and 0.31 g/t Au. Waste mining will total 2,748,021 t for a waste to ore ratio of 6.1:1. Approximately, 1,524,390 t of waste is expected to be PAG, all of which will be stored permanently in the mined pit or TMF. The starter pit is summarized in Table 16-6.

**Table 16-6: Starter Pit Production Summary**

| Bench | Ore (kt) | Waste (t) | Cu (%) | Zn (%) | Ag (g/t) | Au (g/t) |
|---|---|---|---|---|---|---|
| 1,650 masl | - | 156 | - | - | - | - |
| 1,640 masl | - | 450 | - | - | - | - |
| 1,630 masl | 50 | 927 | 1.56 | 1.49 | 20.2 | 0.22 |
| 1,620 masl | 124 | 715 | 1.73 | 1.44 | 21.6 | 0.26 |
| 1,610 masl | 272 | 500 | 1.95 | 1.72 | 24.5 | 0.35 |
| **Total** | **446** | **2,748** | **1.84** | **1.62** | **23.2** | **0.31** |

Source: JDS (2017).

## 16.9 Mine Production Plan

The starter pit will be pre-stripped in year -1 and will provide ore in year 1, while the underground mines are being developed.

The access ramp to Main begins in year -1. Full production from underground is achieved in year 2 after an 18 month pre-production development period. Ore will be available after 12 months, but the Main mine won't be up to 2,500 t/d until mid-year 1. During pre-production, the primary ramp in Main will be established as well as secondary access ramps to the west, centre and east mining zones. Production is exclusively from the Main mine in years 1 to 2, while Esso is being developed.

The access ramp to Esso begins in year -1 and is complete in year 1. Esso's pre-production period is approximately 40 months. Ore production from Esso begins in year 3 and continues at 1,500 t/d until the deposit is exhausted in year 8. While Esso is in production, Main's rate is reduced to 1,000 t/d for a total rate of 2,500 t/d from both mines. Once Esso is exhausted, Main production returns to 2,500 t/d until the end of the mine in year 12.

Although the Esso access ramp passes adjacent to the Sumac resource, no development or production is included from the Sumac deposit in this conceptual mine plan. Exploration drilling on Sumac will be completed during the LOM and there's an opportunity to incorporate production from Sumac if the program is successful.

The annual mine production schedule is provided in Table 16-7 and shows annual summaries of ore tonnage mined by deposit, ore grades and development quantities.

### 16.9.1 Ore Mining and Grade

The average mined grades for the 12 year mine life are 2.01% Cu, 3.19% Zn, 34.6 g/t Ag, and 0.37 g/t Au. Annual production by mine and metal grades are shown in Figures 16-11 to 16-13.

### 16.9.2 Underground Waste Development

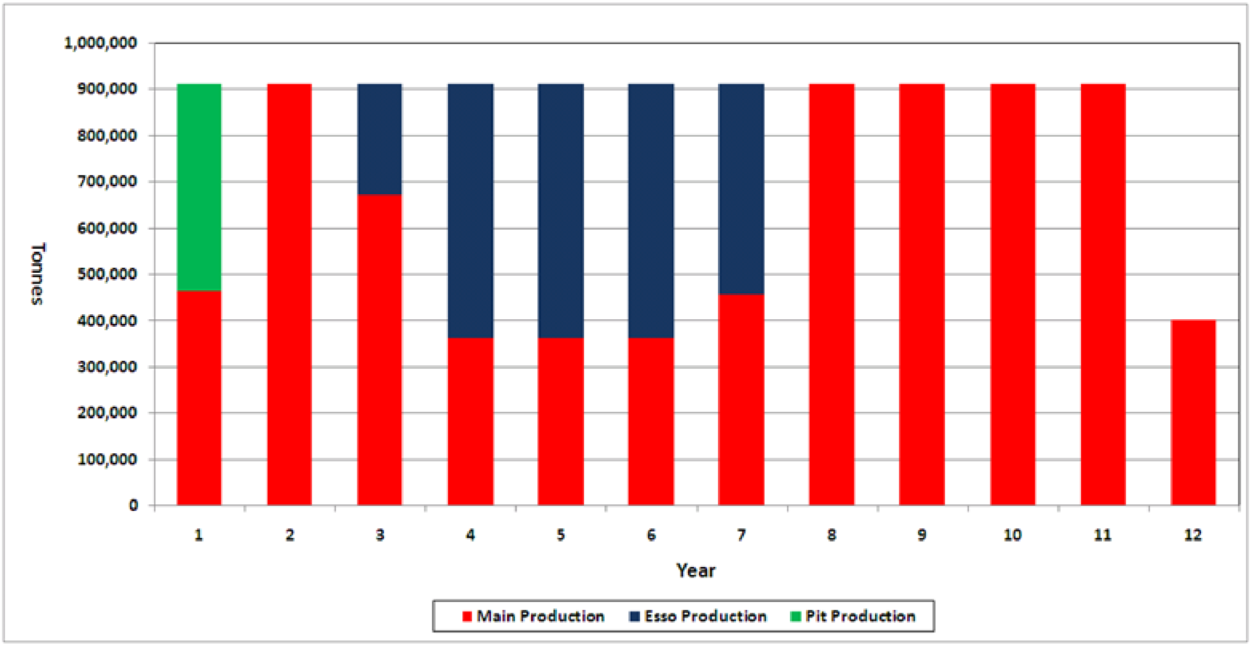
Total underground capital and sustaining lateral waste development is 25,174 m and averages 1,936 m per year or 5.5 m/day over the 12 year mine life. Annual waste development is shown in Figure 16-14.

**Table 16-7: Mine Production Schedule**

| Parameter | Unit | Production Year | | | | | | | | | | | | | Totals |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | -1 | 1 | 2 | 3 | 4 | 5 | 6 | 7 | 8 | 9 | 10 | 11 | 12 | |
| Starter Pit Production | kt | - | 446 | - | - | - | - | - | - | - | - | - | - | - | **446** |
| Main Production | kt | - | 466 | 913 | 674 | 365 | 365 | 365 | 459 | 674 | 913 | 913 | 913 | 404 | **7,660** |
| Esso Production | kt | - | - | - | 239 | 548 | 548 | 548 | 454 | 239 | - | - | - | - | **2,335** |
| Total Mine Production | kt | - | 913 | 913 | 913 | 913 | 913 | 913 | 913 | 913 | 913 | 913 | 913 | 404 | **10,441** |
| Daily Production Rate | t/d | - | 2,500 | 2,500 | 2,500 | 2,500 | 2,500 | 2,500 | 2,500 | 2,500 | 2,500 | 2,500 | 2,500 | 2,500 | **2,500** |
| Starter Pit Waste | kt | 1,533 | 1,215 | - | - | - | - | - | - | - | - | - | - | - | **2,748** |
| Copper Grade | % | - | 1.94 | 2.13 | 2.01 | 2.02 | 2.26 | 2.12 | 2.06 | 1.88 | 1.91 | 2.07 | 1.81 | 1.87 | **2.01** |
| Zinc Grade | % | - | 1.92 | 2.62 | 2.91 | 3.71 | 5.30 | 4.41 | 3.76 | 2.64 | 3.06 | 2.78 | 2.43 | 2.27 | **3.19** |
| Silver Grade | g/t | - | 26.4 | 31.0 | 42.3 | 47.2 | 41.5 | 46.9 | 35.2 | 27.6 | 29.2 | 28.0 | 27.3 | 30.1 | **34.6** |
| Gold Grade | g/t | - | 0.30 | 0.32 | 0.44 | 0.47 | 0.45 | 0.47 | 0.35 | 0.36 | 0.31 | 0.32 | 0.27 | 0.39 | **0.37** |
| Capital Development | m | 3,875 | 2,637 | 1,465 | 362 | - | - | - | - | - | - | - | - | - | **8,339** |
| Sustaining Development | m | 360 | 1,400 | 2,350 | 1,952 | 2,034 | 2,267 | 973 | 1,772 | 1,184 | 1,234 | 789 | 351 | 170 | **16,835** |
| Total Lateral Development | m | 4,235 | 4,037 | 3,815 | 2,314 | 2,034 | 2,267 | 973 | 1,772 | 1,184 | 1,234 | 789 | 351 | 170 | **24,174** |
| | m/day | 11.6 | 11.1 | 10.5 | 6.3 | 5.6 | 6.2 | 2.7 | 4.9 | 3.2 | 3.4 | 2.2 | 1.0 | 1.0 | **5.5** |
| Capital Raise Development | m | 467 | 809 | 441 | 40 | - | - | - | - | - | - | - | - | - | **1,757** |
| Mined Underground Waste | kt | 297 | 274 | 262 | 156 | 137 | 153 | 66 | 120 | 80 | 83 | 53 | 24 | 12 | **1,716** |
| Paste Backfill Placed | kt | - | 228 | 408 | 408 | 408 | 384 | 408 | 408 | 408 | 408 | 408 | 408 | 181 | **4,468** |

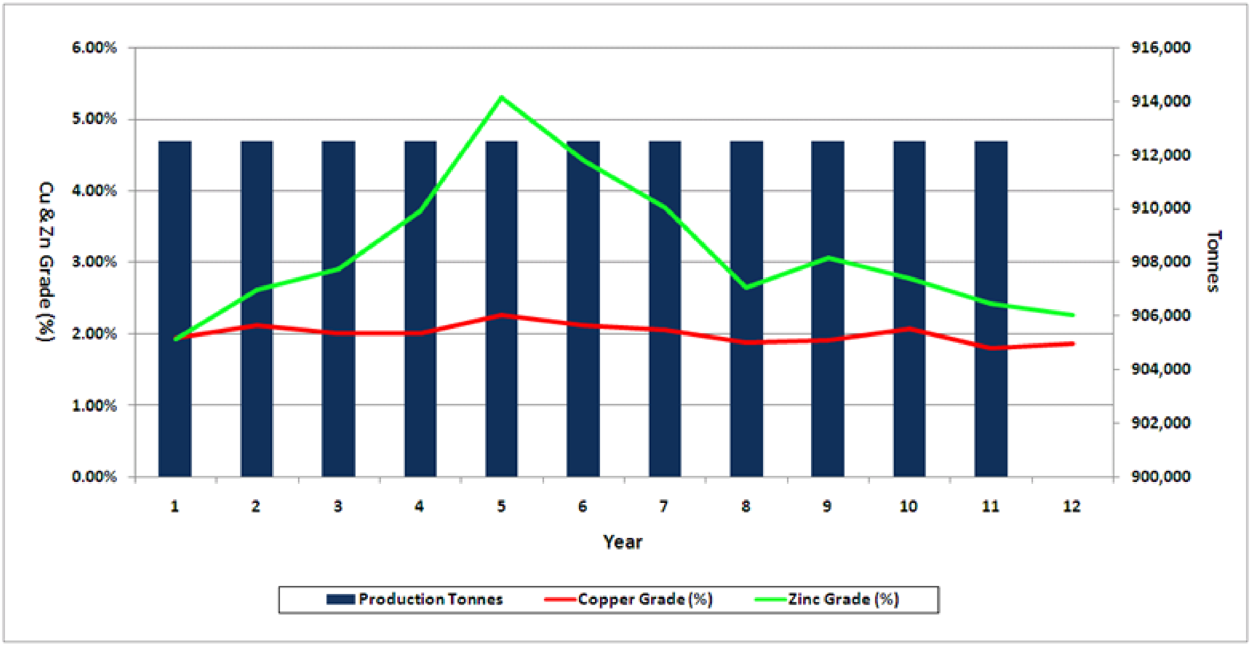
Note: numbers may not total due to rounding.
Source: JDS (2017).

## Figure 16-11: Annual Production by Deposit



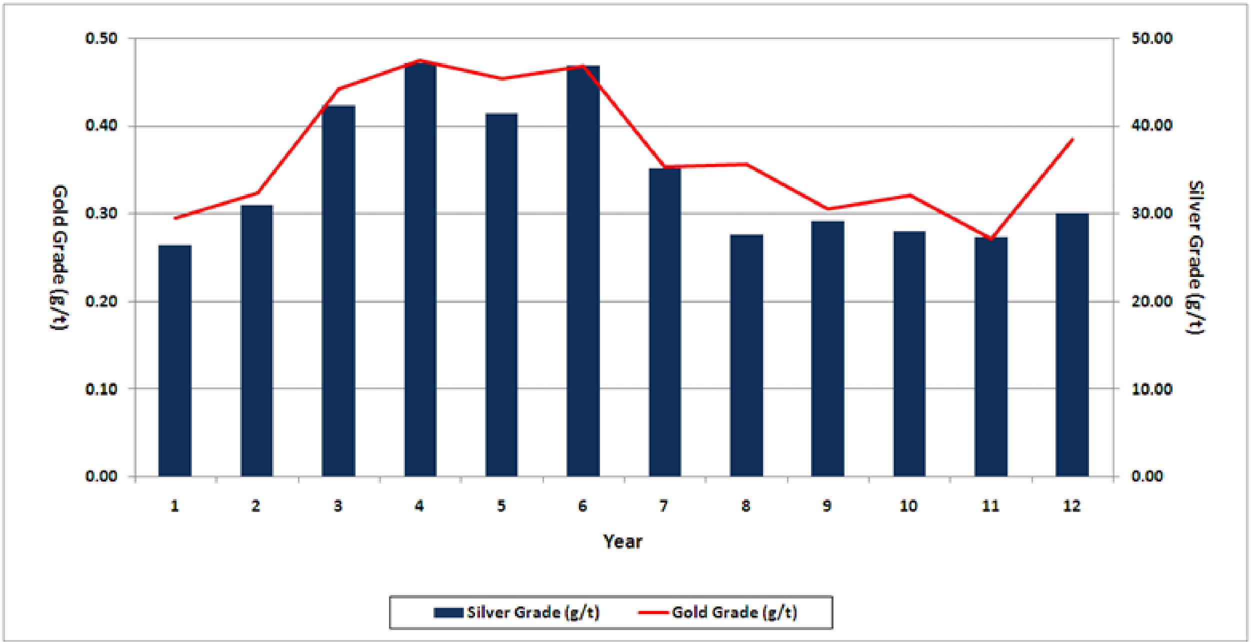

Source: JDS (2017).

## Figure 16-12: Annual Production and Millhead Copper and Zinc Grades



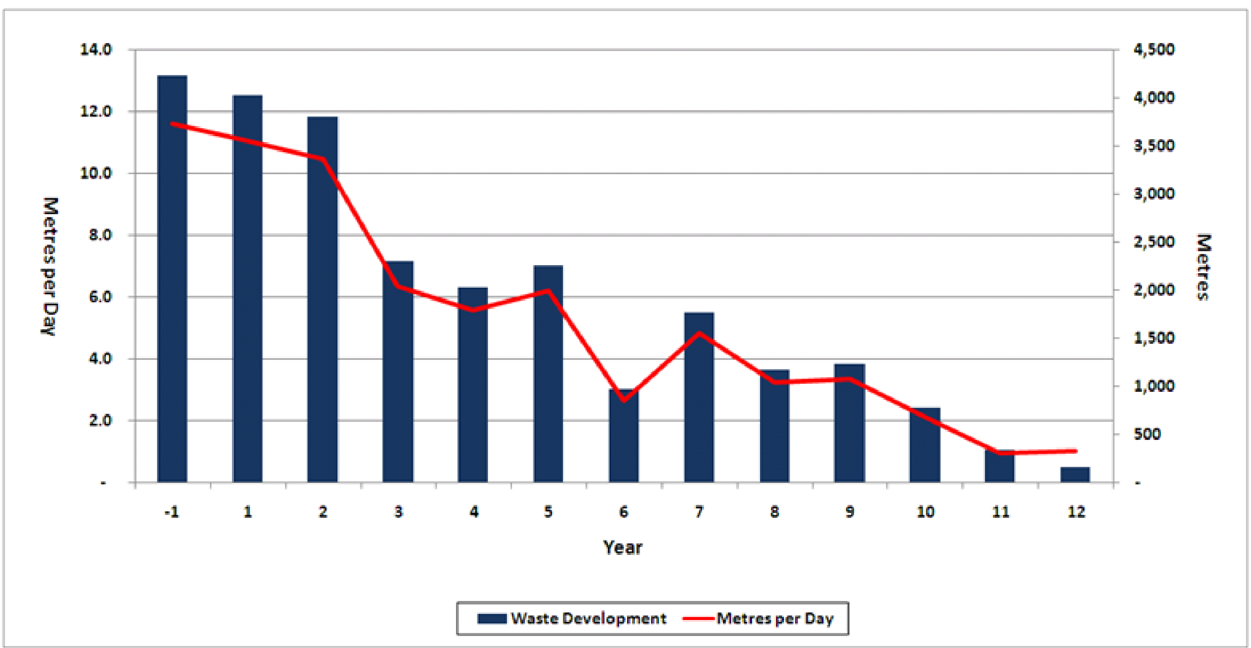

Source: JDS (2017).

## Figure 16-13: Annual Millhead Silver and Gold Grades




Source: JDS (2017).

## Figure 16-14: Annual Underground Waste Development




Source: JDS (2017).

## 16.10 Underground Mine Equipment

The selection of underground mining equipment is based on mine plan requirements, mining methods, operating drift and stope dimensions. No work was undertaken in this pre-feasibility study to evaluate alternates or new technology. All mobile equipment is new since the LOM plan is 12 years and major refurbishment expenditures are avoided.

Two boom diesel/electric jumbos will be used for lateral development and MCF stoping, while production drilling will be completed by a diesel/electric LH drill. Mucking will be with 5 cubic metres ($m^3$) LHDs with remote operating capabilities and will be used for development and stope mucking. Waste and some ore haulage will be with 50 and 30-tonne trucks, while ore from the ore pass systems will be hauled with larger 50-tonne trucks. Ore from the Esso deposit will also be hauled to surface by 50-tonne trucks.

The underground equipment fleet is summarized in Table 16-8.

**Table 16-8: Mine Equipment Summary**

| Equipment Type | Quantity |
|---|---|
| Two Boom Jumbo | 2 |
| LH Drill | 1 |
| 5 $m^3$ LHD with Remote | 3 |
| 30 Tonne Truck | 1 |
| 50 Tonne Truck | 3 |
| Mechanized Bolter | 2 |
| Fuel/ Lube Truck | 1 |
| Grader | 1 |
| Deck and Boom Truck | 2 |
| Scissor Lift | 2 |
| ANFO Loader | 1 |
| Personnel Carrier | 3 |
| Utility Tractor | 3 |

Source: JDS (2017).

### 16.10.1 Esso Ore Handling Alternatives

The following ore handling alternatives were considered for the Esso deposit to determine the most economical option:

- Option 1 - Ore collected at the bottom of the ore pass and hauled to surface with 30-tonne trucks;
- Option 2 - Ore collected at the bottom of the ore pass and hauled to surface with 50-tonne trucks;
- Option 3 - Internal Winze, parallel to the ore pass that would hoist ore approximately 135 m to a truck load-out at the 1,085 m elevation. Ore hauled to surface with 30-tonne trucks; and

- Option 4 - Internal Winze, parallel to the ore pass system that would hoist ore approximately 135 m to a truck load-out at the 1,085 m elevation. Ore hauled to surface with 50-tonne trucks.

Based on total capital and operating costs Option 2 and was selected for the pre-feasibility plan.

## 16.11 Underground Mine Personnel

The mine will operate on two 12-hour shifts, 365 days per year with four mining and maintenance crews. Two crews will be on-site at any one time, one on dayshift and one on night shift, with the other two crews' offsite on break. The majority of the mining and maintenance personnel will work a two weeks on two weeks off (2x2) rotation, while technical staff and management will work a 4 days on 3 days off (4x3) schedule.

Twelve hour shifts exceed the hours allowed underground by regulation and a variance will be required from the BC Labour Board. Given the nature and location of the mine, and referencing other northern BC operations where similar variances have been given, it's expected that this variance will be granted.

The underground mine personnel requirement peaks at 125 personnel during full production, with 69 on-site at one time. Excluded from this total are personnel required to operate the processing and paste fill plants, site services and site general administration as well as mining contractors.

Mining personnel requirements are summarized in Tables 16-9 to 16-12.

### Table 16-9: Mine Operations Personnel Summary

| Position | Quantity | Schedule | Hourly/Salary |
|---|---|---|---|
| Mine Superintendent | 1 | 4x3 | Salary |
| Mine Captain | 1 | 4x3 | Salary |
| Mine Shift Supervisors | 4 | 2x2 | Hourly |
| Mine Clerk | 1 | 4x3 | Salary |
| Jumbo and LH Drillers | 12 | 2x2 | Hourly |
| LHD Operators | 12 | 2x2 | Hourly |
| Truck Drivers | 16 | 2x2 | Hourly |
| Blasters | 8 | 2x2 | Hourly |
| Bolters | 8 | 2x2 | Hourly |
| General and Backfill Labourers | 12 | 2x2 | Hourly |
| Paste Plant Operators | 4 | 2x2 | Hourly |
| **Mine Operations Total** | **79** | **-** | **-** |

Source: JDS (2017).

### Table 16-10: Mine Maintenance Personnel Summary

| Position | Quantity | Schedule | Hourly/Salary |
|---|---|---|---|
| Maintenance and Electrical Superintendent | 1 | 4x3 | Salary |
| Maintenance Shift Supervisors | 4 | 2x2 | Hourly |
| Maintenance and Electrical Foreman | 1 | 4x3 | Salary |
| Maintenance Planner | 1 | 4x3 | Salary |
| Mechanics and Welders | 16 | 2x2 | Hourly |
| Servicemen | 4 | 2x2 | Hourly |
| Electrician | 4 | 2x2 | Hourly |
| Labourers | 4 | 2x2 | Hourly |
| **Mine Maintenance Total** | **35** | **-** | **-** |

Source: JDS (2017).

### Table 16-11: Technical Services Personnel Summary

| Position | Quantity | Schedule | Hourly/Salary |
|---|---|---|---|
| Chief Mine Engineer | 1 | 4x3 | Salary |
| Senior Mine Engineer | 1 | 4x3 | Salary |
| Mine Engineers | 2 | 4x3 | Salary |
| Senior Surveyor | 2 | 2x2 | Salary |
| Mine Technician | 2 | 2x2 | Salary |
| Mine Geologists | 2 | 4x3 | Salary |
| Ground Control Engineer | 1 | 4x3 | Salary |
| **Technical Services Total** | **11** | **-** | **-** |

Source: JDS (2017).

### Table 16-12: Total Mine Personnel Summary

| Position | Quantity |
|---|---|
| Mine Operations | 79 |
| Mine Maintenance | 35 |
| Technical Services | 11 |
| **Total Mine Personnel** | **125** |

Source: JDS (2017).

# 17 Recovery Methods

The results of the metallurgical test work described in Section 13 were used to select the recovery method for the Project. The resulting design criteria was used to design the process facility described in this section. The recovery method has not been changed from the 2011 PFS (JDS 2011) only the equipment costs were updated.

The plant will process material at a rate of 2,500 t/d with an average LOM head grade of 2.01% copper and 3.19% zinc. Based on test work, the overall LOM metal recoveries are expected to be approximately 84.7% for copper and 75.7% for zinc. The two stage grinding circuit will target a product size of 80% passing ($P_{80}$) 75 µm, followed by sequential flotation to produce copper and zinc concentrates. The tailings will be pumped to a TMF. The crushing circuit will operate at an availability of 70%, while the milling and flotation circuits will operate 24-hours per day, 365 days per year at an availability of 92%.

## 17.1 Introduction

The plant will consist of the following unit operations:

- Primary Crushing – A vibrating grizzly feeder and jaw crusher in open circuit, producing a final product $P_{80}$ of 125 mm;
- Crushed Material Storage and Reclaim – A 2,500 t live stockpile with two reclaim belt feeders feeding the Ball Mill Feed Conveyor;
- Primary Grinding – A SAG mill in open circuit, producing a transfer size $T_{80}$ of 1,000 µm;
- Secondary Grinding – A ball mill in open circuit, producing a final product $P_{80}$ of 75 µm;
- Copper Rougher and Cleaner Flotation – Rougher flotation cells, rougher concentrate regrind and cleaner flotation cells;
- Zinc Rougher and Cleaner Flotation – Rougher flotation cells and cleaner flotation cells;
- Concentrate Dewatering and Filtration – Copper and Zinc concentrate thickeners, stock tanks and filters; and
- Final Tailings Disposal – Centrifugal pumps to send slurry to the TMF and a barge reclaim system to pump reclaim water back to the process plant or to the paste plant for deposition underground.

## 17.2 Process Design Criteria

Conceptual design criteria for a 2,500 t/d process plant are listed in Table 17-1.

### Table 17-1: Process Design Criteria

| Description | Unit | Value | Source |
|---|---|---|---|
| **Mill Feed Characteristics** | | | |
| Specific Gravity of in situ mineralized rock | g/cm$^3$ | 3.6 | JDS PFS 2011 |
| Bulk Density | t/m$^3$ | 2.2 | JDS PFS 2011 |
| Moisture Content (Average) | % | 3.0 | JDS PFS 2011 |
| Abrasion Index (Average) | g | 0.16 | Met. Research Dec. 2010 |
| **Operating Schedule** | | | |
| Shift/Day | | 2 | JDS PFS 2011 |
| Hours/Shift | H | 12 | JDS PFS 2011 |
| Hours/Day | H | 24 | |
| Days/Year | D | 365 | |
| **Plant Availability/Utilization** | | | |
| Overall Plant Feed | t/a | 912,500 | Calculated |
| Overall Plant Feed | t/d | 2,500 | Mine Plan |
| Crusher Availability | % | 70 | JDS PFS 2011 |
| Grinding and Flotation Plant Availability | % | 92 | JDS PFS 2011 |
| Crushing Feed Rate | t/h | 150 | Engineering Calculation |
| Process Plant Feed Rate | t/h | 113 | Engineering Calculation |
| Average Head Grades | Cu% | 2.01 | LOM Average Mine Plan |
| | Zn% | 3.19 | LOM Average Mine Plan |
| Recovery: To Copper Concentrate | Cu% | 84.7 | JDS 2017 |
| To Zinc Concentrate | Zn% | 75.7 | JDS 2017 |
| Average Cu Concentrate Grade | Cu% | 27.6 | JDS 2017 |
| | Zn% | 7.3 | JDS 2017 |
| Average Zn Concentrate Grade | Zn% | 55.1 | JDS 2017 |
| | Cu% | 1.2 | JDS 2017 |
| Cu Concentrate Mass Recovery | % | 5.66 | Engineering Calculation |
| Zn Concentrate Mass Recovery | % | 3.96 | Engineering Calculation |
| **Crushing** | | | |
| Primary Crushing Parameters | | | |
| Feed Particle Size | Mm | 450 | JDS PFS 2011 |
| Crusher Type | | Jaw | JDS PFS 2011 |
| Crushers | # | 1 | JDS PFS 2011 |
| Crushing Processing Rate | t/h | 150 | Engineering Calculation |
| Product Size, $P_{80}$ | mm | 125 | JDS PFS 2011 |
| Crushed Mill Feed Stockpile Parameters | | | |
| Crushed Mill Feed Stockpile (Live capacity) | T | 2,500 | JDS 2017 |
| Crushed Mill Feed Bulk Density | t/m$^3$ | 2.2 | JDS PFS 2011 |

## Table 17-1: Process Design Criteria (continued)

| Description | Unit | Value | Source |
|---|---|---|---|
| Feeders | # | 2 | JDS PFS 2011 |
| Tonnage Rate (Each), Operating | t/h | 113 | Engineering Calculation |
| Type of Feeder | | Belt | JDS 2017 |
| **Grinding** | | | |
| Production Rate | t/h | 113 | Engineering Calculation |
| Bond Rod Mill WI, Global Composite | kWh/t | 8.9 | Met. Research Dec. 2010 |
| Bond Ball Mill WI, Global Composite | kWh/t | 12.2 | Met. Research Dec. 2010 |
| Abrasion Index | g | 0.16 | Met. Research Dec. 2010 |
| Primary Grinding | | | |
| Mill Type | | SAG Mill | JDS PFS 2011 |
| Mills | # | 1 | JDS PFS 2011 |
| Feed Solids | %w/w | 70 | Design |
| Feed Size, $P_{80}$ | mm | 125 | Design |
| Product Size, $P_{80}$ | µ | 1,000 | Design |
| Classification Type | | Screen | JDS 2017 |
| **Secondary Grinding** | | | |
| Mill Type | | Ball Mill | JDS PFS 2011 |
| Mills | # | 1 | JDS PFS 2011 |
| Feed Solids | %w/w | 70 | Design |
| Feed Size, $P_{80}$ | µ | 1,000 | Design |
| Product Size, $P_{80}$ | µ | 75 | Met. Research Dec. 2010 |
| Recirculation Load | % | 300 | Design |
| Classification Type | | Cyclones | Design |
| **Copper Flotation Circuit** | | | |
| Copper Rougher | | | |
| Solids Flow Rate | t/h | 113 | Engineering Calculation |
| Flotation Time | min | 20 | JDS PFS 2011 |
| Flotation pH | | 7 | JDS PFS 2011 |
| Mass Pull | % | 16 | JDS PFS 2011 |
| **Copper Regrind Circuit** | | | |
| Mill Type | | Ball mill | JDS PFS 2011 |
| Throughput | t/h | 18 | Engineering Calculation |
| Ball Mill Work Index | metric | 15.3 | JDS PFS 2011 |
| Feed size, $F_{80}$ | µ | 75 | JDS PFS 2011 |
| Product size, $P_{80}$ | µ | 35 | JDS PFS 2011 |

## Table 17-1: Process Design Criteria (continued)

| Description | Unit | Value | Source |
|---|---|---|---|
| **Copper First Cleaner and Scavenger** | | | |
| Flotation Time | min | 12 | JDS PFS 2011 |
| Mass Pull | % | 24 | JDS PFS 2011 |
| **Copper Second Cleaner** | | | |
| Flotation Time | min | 12 | JDS PFS 2011 |
| Mass Pull | % | 8 | JDS PFS 2011 |
| **Copper Third Cleaner** | | | |
| Flotation Time | min | 8 | JDS PFS 2011 |
| Mass Pull | % | 5.66 | Met. Research Dec. 2010 |
| Concentrate Solids | t/h | 6.41 | Engineering Calculation |
| **Zinc Flotation Circuit** | | | |
| Zinc Rougher | | | |
| Solids Flow Rate | t/h | 106 | Engineering Calculation |
| Flotation Time | min | 20 | JDS PFS 2011 |
| Flotation pH | | 11.4 | JDS PFS 2011 |
| Mass Pull | % | 9.96 | JDS PFS 2011 |
| **Zinc First Cleaner** | | | |
| Flotation Time | min | 10 | JDS PFS 2011 |
| Mass Pull | % | 7.5 | JDS PFS 2011 |
| **Zinc Second Cleaner** | | | |
| Flotation Time | min | 8 | JDS PFS 2011 |
| Mass Pull | % | 5.5 | JDS PFS 2011 |
| **Zinc Third Cleaner** | | | |
| Flotation Time | min | 6 | JDS PFS 2011 |
| Mass Pull | % | 3.96 | Met. Research Dec. 2010 |
| Solids Flow Rate | t/h | 4.48 | Engineering Calculation |
| **Concentrate Dewatering** | | | |
| Copper Concentrate | | | |
| Thickening | | | |
| Thickener Type | | High Rate | Design |
| Thickener Underflow Density | % solids | 55 | JDS PFS 2011 |
| Thickener Unit Area Rate | $t/m^2/h$ | 0.11 | JDS PFS 2011 |
| Thickener Underflow Slurry Storage Tank Capacity | h | 8 | Design |
| Filtration | | | |
| Filter Type | | Pressure | Design |
| Filtration Unit Area Rate | $kg/m^2/h$ | 110 | JDS PFS 2011 |
| Filter Operating Availability | % | 75 | Design |
| Filter Cake Moisture | % | 8.0 to 10.0 | JDS PFS 2011 |

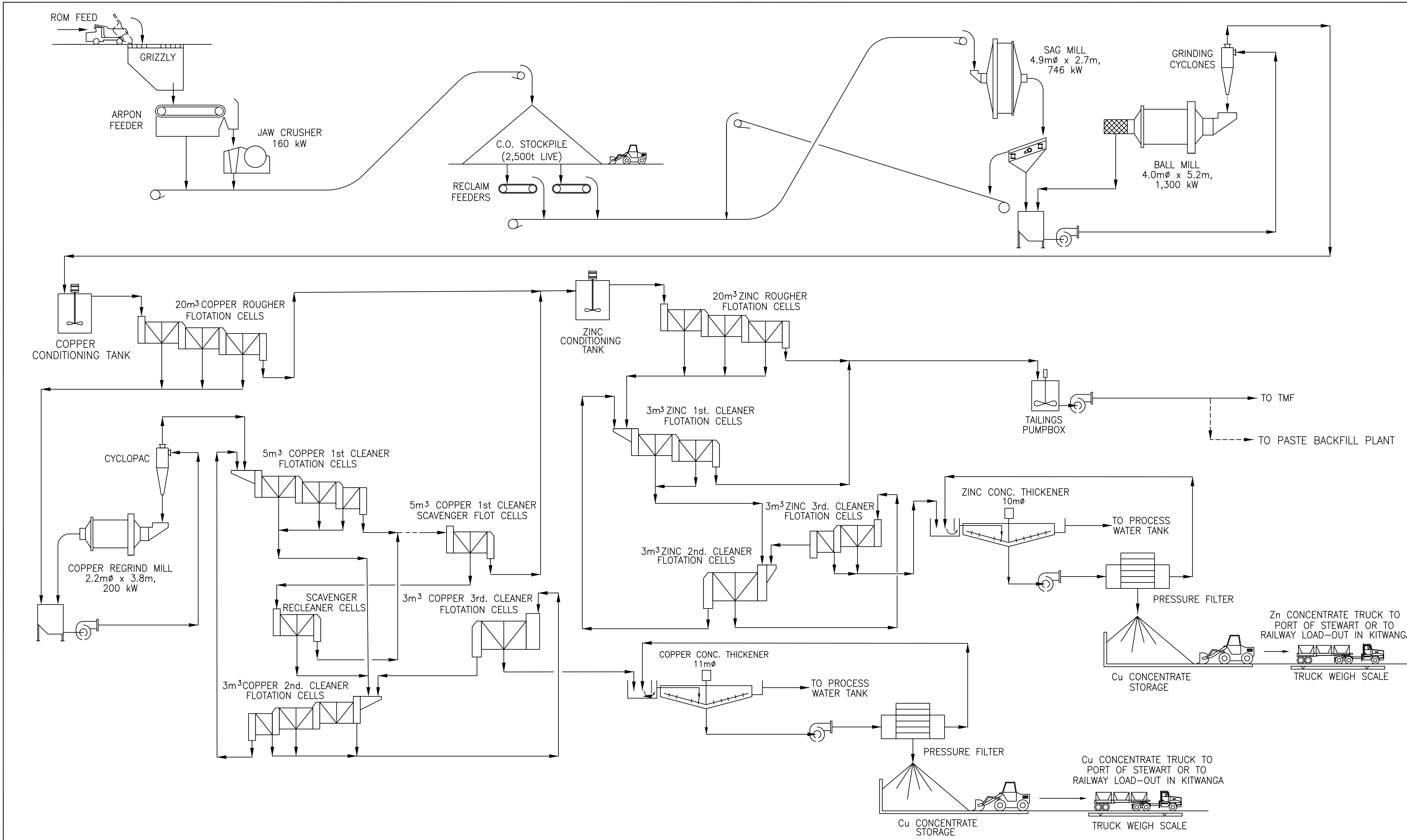
**Table 17-1: Process Design Criteria (continued)**

| Description | Unit | Value | Source |
|---|---|---|---|
| Concentrate Bulk Density | $t/m^3$ | 3.3 | JDS PFS 2011 |
| Zinc Concentrate | | | |
| Thickening | | | |
| Thickener Type | | High Rate | Design |
| Thickener Underflow Density | % solids | 55 | JDS PFS 2011 |
| Thickener Unit Area Rate | $t/m^2/h$ | 0.12 | JDS PFS 2011 |
| Thickener Underflow Slurry Storage Tank Capacity | h | 8 | Design |
| Filtration | | | |
| Filter Type | | Pressure | Design |
| Filtration Unit Area Rate | $kg/m^2/h$ | 130 | JDS PFS 2011 |
| Filter Operating Availability | % | 75 | Design |
| Filter Cake Moisture | % | 8.0 to 10.0 | JDS PFS 2011 |
| Concentrate Bulk Density | $t/m^3$ | 3.2 | JDS PFS 2011 |
| Reagents | | | |
| Lime (as $Ca(OH)_2$) (Includes water treat. lime requirements) | g/t | 2,276 | JDS PFS 2011 |
| 7583 | g/t | 50 | JDS PFS 2011 |
| 3477 | g/t | 10 | JDS PFS 2011 |
| Copper Sulphate ($CuSO_4$) | g/t | 550 | JDS PFS 2011 |
| Sodium Sulphite ($Na_2SO_3$) | g/t | 1,250 | JDS PFS 2011 |
| Frother 1064 | g/t | 30 | JDS PFS 2011 |
| Sodium Silicate | g/t | 450 | JDS PFS 2011 |
| Ammonium Bisulphate | g/t | 500 | JDS PFS 2011 |

Source: JDS (2017).

## 17.3 Plant Design

A summary of the process flowsheet is presented in Figure 17-1. The crushing and process facilities are displayed in Figure 17-2.

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| DESIGN. BY J.D.S. | June 26 17 |
| DRAWN BY: B. Wong | June 26 17 |
| CHECK. BY | |
| APP. BY: | |

JDS Energy & Mining Inc.

2,500tpd KUTCHO COPPER PROJECT
SIMPLIFIED FLOWSHEET

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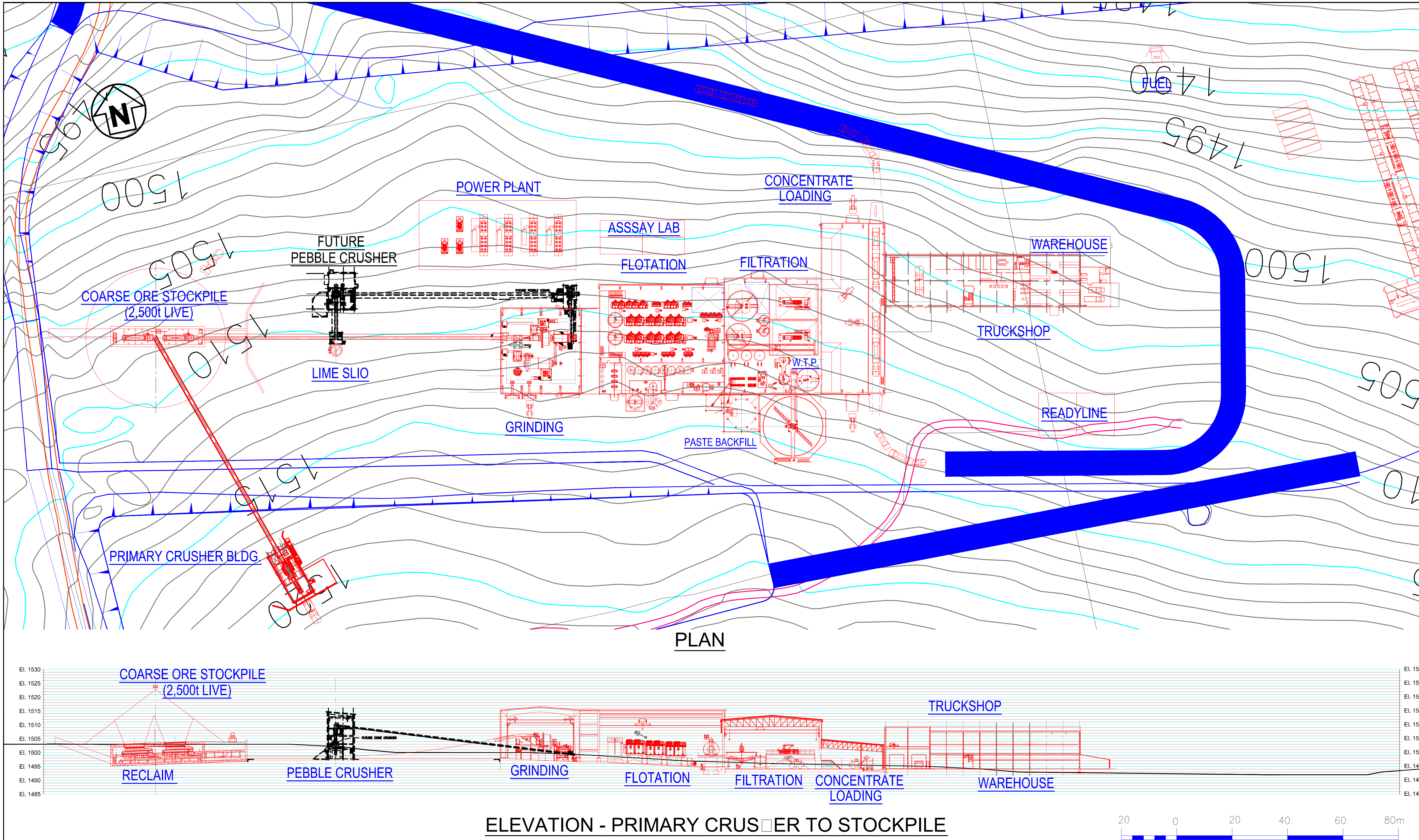


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DRAWN BY: B.W. Dec 15 10
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APP. BY:

JDS Energy □ Mining Inc.

KUTCHO PROJECT
STOCKPILE TO FILTRATION
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## 17.4 General Description

A 2,500 t/d plant is proposed for the Project to process the VMS mineralization. The metals contained are copper, zinc, gold, and silver. The process plant is assumed to operate 365 days per year at 92% availability. The mill feed will be crushed, ground and then sequentially subjected to copper and zinc flotation.

The zinc tailings will be sent to a paste backfill plant to produce a cemented paste; half of the paste will be sent to backfill for mine operations, while the other half will be sent to the surface tailings disposal. Copper and zinc concentrates are thickened and then dewatered before load-out. Precious metals will be contained in and shipped with the copper concentrate.

### 17.4.1 Process Description

#### 17.4.1.1 Crushing Operations

Run-of-mine (ROM) ore will be delivered to the surface primary crusher by haul trucks from the underground mine. The ROM material will feed a dump pocket via a stationary grizzly at an average rate of 150 tonnes per hour (t/h). Grizzly oversize will be broken by a rock breaker. Grizzly undersize will be discharged to a dump hopper and fed via feeder to the primary jaw crusher and crushed to a target $P_{80}$ 125 mm. Crusher product will be transported to a 2,500 t (live) stockpile via belt conveyor.

#### 17.4.1.2 Grinding Circuit Operations

The crushed mill feed will be reclaimed from the stockpile by two belt feeders at a controlled rate of 113 t/h, and fed via a conveyor to a 4.9 m diameter x 2.7 m long, 750 kW SAG mill.

The SAG discharge will feed onto a vibrating screen. Any screen oversize will be returned to the circuit.

Secondary grinding will take place in a 4.0 m diameter x 5.2 m long, 1,300 kW ball mill. SAG mill screen undersize will discharge into a cyclone feed pump box together with the ball mill discharge. The combined slurry will be pumped to cyclones. The cyclone underflow will report back to the ball mill grinding circuit. The cyclone overflow will be directed to the copper flotation circuit.

Steel balls will be used as the grinding media. Lime will be added to maintain design pH parameters.

#### 17.4.1.3 Copper Flotation Circuit

The cyclone overflow will feed a conditioner which feeds to six 20 $m^3$ flotation cells. Rougher flotation will produce a concentrate which will be advanced to the copper regrind mill, a 2.2 m diameter x 3.8 m long 200 kW ball mill, complete with hydrocyclone classification. The reground copper rougher concentrate will be cleaned in five 5.0 $m^3$ flotation cells. The first copper cleaner tailings will be scavenged before being sent to the zinc rougher circuit. The scavenger concentrate is cleaned once and sent to the second cleaning stage. The first copper cleaner concentrate will be cleaned again in five 3.0 $m^3$ flotation cells. The second copper cleaner tailings will return to the first cleaner flotation. The second copper cleaner concentrate is further cleaned in two 3.0 $m^3$ cells in the third copper cleaner circuit. The third cleaner concentrate will be pumped to the copper concentrate thickener. The third copper cleaner tailings will return to the second copper cleaner.

Reagents used in the circuit will include $Na_2SO_3$, P7583, Sodium Silicate, Frother CC-1064 and Ammonium Bisulphate. The copper rougher tailings will advance to the zinc circuit.

##### 17.4.1.4 Zinc Flotation Circuit

The copper tailings will be advanced to a conditioner and then to six 20 $m^3$ zinc rougher flotation cells to produce a zinc rougher concentrate. The zinc rougher flotation tailings will report to the tailings discharge pump. The zinc rougher concentrate will be cleaned in four 3.0 $m^3$ flotation cells. Tailings will join the final tails while the concentrate will advance to the second zinc cleaner consisting of three 3.0 $m^3$ flotation cells. The tailings will be re-circulated to the first cleaner and the concentrate advanced to the third zinc cleaner. This consists of two 3.0 $m^3$ flotation cells. The tailings will be re-circulated to the second cleaner while the concentrate will be pumped to the zinc concentrate thickener as the final zinc concentrate.

Reagents used in the zinc circuit include Lime, $CuSO_4$, and Aero 3477.

##### 17.4.1.5 Concentrate Dewatering

Copper and zinc concentrates will be separately thickened and further dewatered in pressure filters to a moisture content of 8%. The dewatered concentrates will discharge to separate storage areas. Dedicated front-end loaders will load the concentrates into road tractor trailers and weighed prior to transportation to the Port of Stewart or to a railway load-out in Kitwanga.

##### 17.4.1.6 Tailings Disposal and Hydraulic Backfill Plant

Tailings will be either pumped to the tailing management facility or to the paste backfill plant for deposition in the underground workings.

##### 17.4.1.7 Fresh Water Supply System

Fresh water will be supplied to the property from wells drilled on the mine site and from a water treatment plant for the process make up, fire, fresh, pump gland seal, and potable water requirements.

Potable water will be treated and stored separately.

##### 17.4.1.8 Reclaim Water

Reclaim water will be pumped from mine dewatering, concentrates and paste thickeners to the water treatment plant. Process water will be treated in a conventional high density lime plant for the removal of dissolved heavy metals prior to being recycled or discharged to the environment.

##### 17.4.1.9 Potentially Acid Generating Water Treatment

The PAG water treatment plant is located in the process plant.

##### 17.4.1.10 Services

Compressed air will be generated for filter, instrument and maintenance purposes. Blowers will produce low pressure air for the flotation process.

##### 17.4.1.11 Quality Control

The final concentrate and intermediate streams will be monitored by on-line analyzers. The assay data will be fed back to the central control room and used to optimize the process.

### 17.4.2 Process Control Philosophy

The process control system will be a Process Control System (PLC) based system. The PLCs will be used to control and monitor all the operations of the plant. The plant is broken into different process areas. Each process area is controlled by a single PLC system. The PLCs will be tied together to form a plant wide control system by the use of an Ethernet communication system.

Process control and monitoring for the facility will be performed in two centralized control rooms. The control rooms will be located in the main process plant and in the primary crusher area. Human Machine Interface (HMI) operator stations will be located in the control rooms. These HMIs will contain the graphical representation of the process equipment. The PLC in conjunction with the HMI will perform all equipment and process interlocks, level control, alarms, trends and report generation.

The motor starters and variable frequency drives (VFDs) will be controlled by the PLC via a device net communication system.

# 18 Project Infrastructure

The proposed layout of the process plant, administration buildings, accommodation camp and maintenance facilities are shown in the site layout in Figure 18-1. The facilities are located near the portal entrance to the underground mine. This location and arrangement takes advantage of favourable topography and minimizes paste fill pumping requirements.

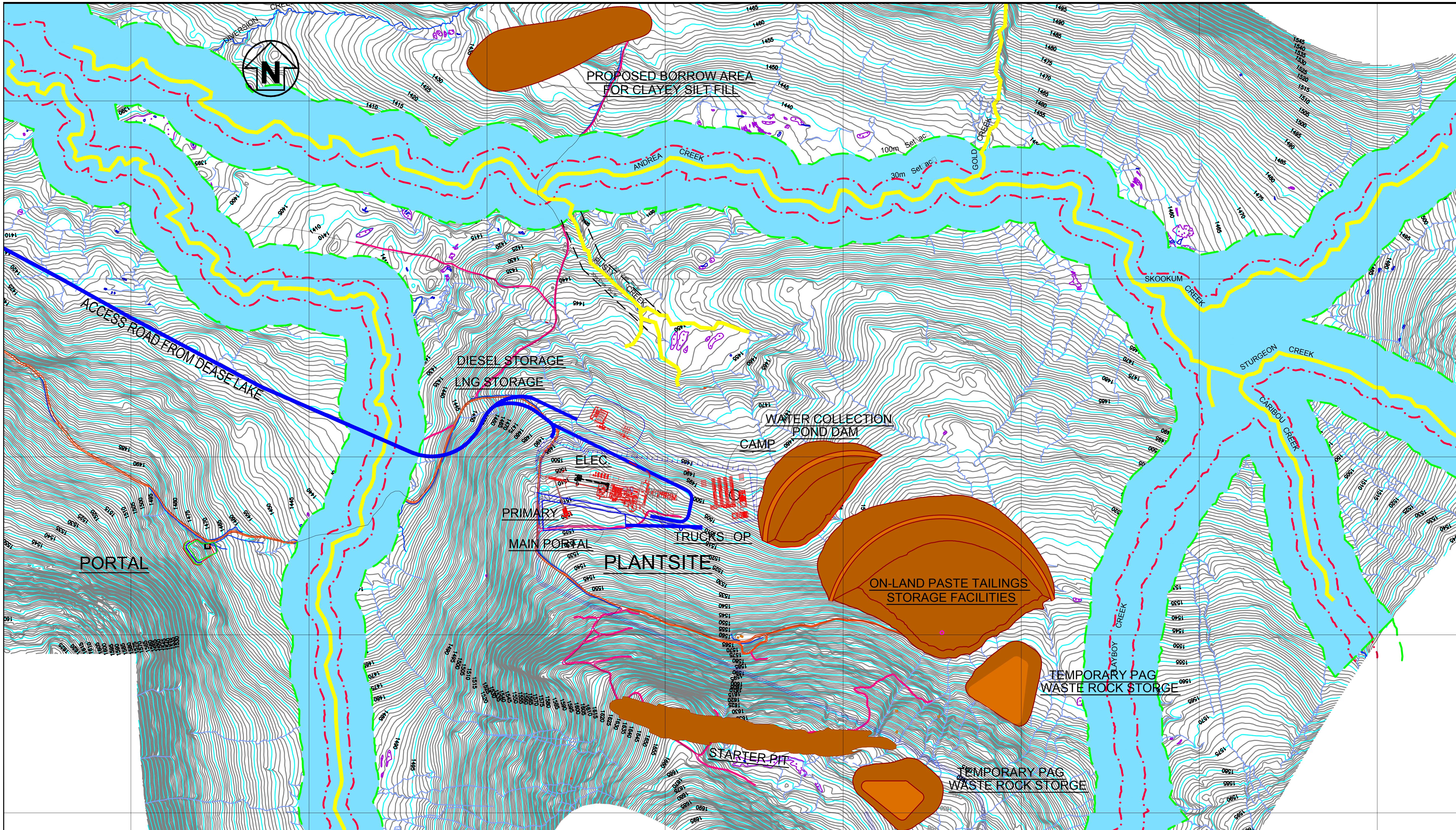
## 18.1 Access Road

JDS recommends a re-alignment and upgrade of the existing Kutcho access road to provide safe, efficient, year round access to the proposed Kutcho mine site.

The purpose of this section is to summarize the 119 km of road design completed by Onsite Engineering Ltd. (Onsite). Complete details can be found in Onsite's *Kutcho Access Road, Geometric Road Design Report* dated December 17, 2010 (Onsite 2010).

The following is a list of the information used by Onsite to complete the draft designs:

- Survey level global positioning system (GPS) survey of the existing roadway and adjacent ground;
- Hand traversing of ground unable to be GPS surveyed;
- High order GPS points for maintaining horizontal and vertical controls completed by Allnorth Consultants Ltd.;
- Review of crossing designs and hydrology previously completed by Allnorth Consultants Ltd.;
- Fisheries Assessments completed by Rescan Environmental Services Ltd.;
- Geology and Surficial Materials and Soils baseline report completed by Rescan Environmental Services Ltd.; and
- Gravel Source and Terrain Hazard Mapping completed by Madrone Environmental Services Ltd.




# OVERALL SITE PLAN

100 0 100 200 300 400m

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JDS Energy & Mining Inc.

KUTCHO PROJECT
OVERALL SITE
PLAN

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### 18.1.1 Layout and Survey

Sections of the original road location were re-aligned (re-engineered) to a lower design speed with the intent of reducing the road construction costs. Field location of the new road included pre-field and field reconnaissance works by a Professional Engineer followed by field layout and traverse of the revised road centerline taking into account the road design specifications provided by JDS. Sections where slope stability was assessed to be of concern were also reviewed in the field by a Professional Geoscientist specialized in the assessment of slope with respect to road construction. Onsite completed detailed hand traversing of the revised road centerline locations tied to Allnorth control points for coordinate system continuity. The level of detail required from the road survey is tied to the complexity and end results required. As the end result will be a gravel surfaced mining haul road, a level 2 survey was deemed to be appropriate for this project. If required, field layout of the designed centerline can be completed to aid in construction through any difficult sections.

### 18.1.2 Road Design

The purpose of a road design is to produce specifications for road construction by determining the optimum road geometry that will accommodate the design vehicle configuration for load and alignment, traffic volume, and provide for user safety, while minimizing the cost of construction and future road maintenance. The optimum road design reduces impacts on other resources by minimizing clearing widths and excavations. Considerations are made for anticipated construction equipment to be used to optimize material movement distances, construction techniques such as rolling grades or end-hauling, road widths, C&F angles, and horizontal and vertical control angles.

The geometric road design will include plans, profiles, cross-sections, and mass haul diagrams showing the optimum balance of waste, borrow, and end-haul volumes. An L-Line is designed for vertical and horizontal alignment and to calculate earthwork quantities. The design parameters used are discussed below.

### 18.1.3 Design Parameters

The parameters in Table 18-1 were used to design the horizontal and vertical alignment, construct design templates, and determine earthworks volumes. These parameters were specified by JDS (JDS 2011) with additional references from the Forest Road Engineering Guidebook (BC Ministry of Forests 2002) and the Ministry of Forests Engineering Manual (BC Ministry of Forests, Lands and Natural Resource 2009).

In some cases, Onsite has deviated from the two aforementioned references. For example, the road is being designed as a single lane private mine access road with visible turnouts and radio control. As such, a 5.0 m road width is more than adequate to accommodate users and road safety. The MOF Engineering Manual specifies an 8.0 m road width to be used for a 50 kilometre per hour (km/h) design speed as it assumes the road would be designed for full 2-way traffic.

The road design has been completed based on the determination of the most economical route using construction cost estimates, surficial and subsurface material types, and construction difficulty. Additional details can be found in Onsite's *Kutcho Access Road, Geometric Road Design Report* dated December 17, 2010.

### Table 18-1: Road Design Parameters

<table>
<tr><th>Design Parameter</th><th colspan="5">Comments</th></tr>
<tr><td>Finished road width</td><td colspan="5">5.0 m</td></tr>
<tr><td>Design speed</td><td colspan="5">50 km/h</td></tr>
<tr><td>Vehicle Loading</td><td colspan="5">BCL-625 Highway Legal Loading as specified by the Canadian Highway Bridge Design Code</td></tr>
<tr><td>Road surface</td><td colspan="5">20 to 60 cm of surfacing (depth is dependent on subsurface materials)</td></tr>
<tr><td>Road fill</td><td colspan="5">All large fills to be well drained granular materials, placed and compacted in maximum 0.3 m thick lifts where stability or settlement are critical. Other fills may be constructed with local materials and compacted with the excavation equipment</td></tr>
<tr><td>Ditch</td><td colspan="5">1.0 m depth, 0.3 m bottom width</td></tr>
<tr><td>Crown</td><td colspan="5">2 % crown</td></tr>
<tr><td>Turnouts</td><td colspan="5">7.5 m long entry and exit tapers, 15 m long turnout, 7.5 m width from centerline</td></tr>
<tr><td>Clearing Width</td><td colspan="5">20 m or 3 m from the limits of excavation, whichever is larger</td></tr>
<tr><td rowspan="4">Horizontal Alignment Criteria</td><td>Design Speed (km/h)</td><td>Minimum stopping sight distance (m)</td><td colspan="2">Minimum Curve Radius (m)</td><td>Maximum Road Gradient (short pitch)</td></tr>
<tr><td>50</td><td>135</td><td colspan="2">100</td><td>8% (10%)</td></tr>
<tr><td>40</td><td>95</td><td colspan="2">65</td><td>12% (14%)</td></tr>
<tr><td>30</td><td>65</td><td colspan="2">35</td><td>12% (14%)</td></tr>
<tr><td rowspan="5">Vertical Alignment Criteria</td><td rowspan="2">Design Speed (km/h)</td><td rowspan="2">Minimum stopping sight distance (m)</td><td colspan="2">Minimum K-Value</td><td rowspan="2">Maximum Road Gradient</td></tr>
<tr><td>Sag</td><td>Crest</td></tr>
<tr><td>50</td><td>135</td><td>12</td><td>11</td><td>10% (12)%</td></tr>
<tr><td>40</td><td>95</td><td>7</td><td>5</td><td>12% (14%)</td></tr>
<tr><td>30</td><td>65</td><td>4</td><td>3</td><td>12% (14%)</td></tr>
</table>

Source: Onsite (2010).

## 18.2 Airstrip

The existing airstrip approximately 900 m long, located approximately 10 km west of the Project site will be extended 1,525 m to accommodate regular passenger and freight services by a Beech 1900 class turbo prop aircraft. Access to the air strip from the camp will be via the main site access road leading to Dease Lake. With the exception of equipment to power landing lights, no permanent power or facilities will be available at the airstrip.

## 18.3 Accommodation Camp and Office Complex

The accommodation camp and office complex will be built with modular units manufactured offsite and placed on prepared foundations adjacent to the process plant. The camp will initially accommodate construction activities. The camp and office complex will consist of the following components:

- Kitchen and Dining Complex with capacity to serve up to 144 personnel at a time;

- Recreational facilities including games, television and fitness areas;
- Sleeping accommodations for permanent employees in single rooms;
- Washroom facilities (shower, basins, toilets, and laundry);
- Waste water treatment facility and solid waste incinerator;
- Potable water storage and treatment facility;
- Mine Offices for construction and operations personnel;
- Mine Dry for construction and operations;
- Emergency response and first aid facilities including covered emergency vehicle storage;
- Cold Storage and Seacan laydown area; and
- Parking area.

## 18.4 Warehouse and Maintenance Facility

Warehouse and cold storage facilities will be connected to the maintenance facility to serve both the mine and process plant.

The maintenance facility will accommodate maintenance of underground mining equipment as well as surface site service vehicles.

## 18.5 Liquefied Natural Gas Power

The LNG powered generators has been assumed for the Project.

The electrical power requirement for the mine, mill and camp is expected to vary over the LOM, reflecting the various stages of mine development. The peak average energy requirement is estimated to be 52.6 gigawatt hours per year (GWh/year). The peak capacity requirement is expected to be approximately 7 megawatts (MW). In addition the mine, mill and camp will require heat energy, which is expected to require approximately 6 MW, under peak winter conditions. The heat is expected to be supplied from the electrical power generation equipment. The power planning is being done to meet N-1 reliability criteria (ability to operate with an outage of 1 generator). Power will be distributed around the plant site at 13.8 kilovolt (kV) through a shared service corridor and underground service. For more remote areas, such as the mine, pump locations and other out buildings, overhead lines will provide service at 13.8 kV.

The mill power consumption is estimated to be 35.6 kWh/t.

Three potential suppliers of natural gas power generation equipment, CAT, GE Jenbacher, and Wartsila have been identified. Each supplier can supply a feasible power generation solution. The prefeasibility base case is based upon 4 of the GE Jenbacher J-616, each with a capacity of 2,665 kilowatts (KW). The expected operating cost for the power and heat generated is approximately $0.15/kilowatt hour (KWh). Further potential cost reduction opportunities have been identified for generation of additional electrical energy from waste heat recovery, using organic Rankine cycle equipment. This would be examined in the full feasibility study.

LNG would be supplied from Terasen Gas Inc.'s (TGI) facilities at Tilbury Island in Delta located in the Lower Mainland area of BC. This supply would require TGI to apply to the BC Utilities Commission for amendments to its tariff rates to accommodate the supply. TGI would require additional storage at its site to provide the service and that TGI would be able to obtain the necessary approvals.

The LNG would then be delivered to the mine site in cryogenic tanks by truck with tractors powered with diesel fuel. The LNG trucks are expected to be B-train configurations in order to provide for the most efficient transportation of the LNG fuel. Approximately, 1,200 Gigajoules (GJ) of natural gas energy in the form of LNG would need to be delivered per day. Each B-train would be expected to be able to deliver about 1,500 GJ. Therefore approximately 3 B-trains would be able to provide the LNG delivery.

LNG transportation options including barging from the lower mainland to Stewart, Prince Rupert or Kitimat were examined and found not to be cost effective versus the trucking options. Rail transportation was found to be not feasible.

The potential opportunity for LNG supply also exists with the planned natural gas export terminal in northern BC. Apache Canada Ltd. and EOG Resources Canada Inc. have agreed to pay $50 M to purchase the remaining 50% of Pacific Trail, a proposed $1.2-billion natural gas pipeline, from its developer, Pacific Northern Gas Ltd. The 463 km pipeline would deliver gas from north-eastern BC to Kitimat, where it could be compressed and then transported to the Kutcho mine site. With LNG available at Kitimat – this would reduce haulage distance by approximately 1,000 km with a subsequent reduction in operating costs of approximately ($0.05 to $0.10/lb production).

JDS examined the potential for LNG fuelled trucking to haul the concentrate from the mine site to Stewart and recommends that this option be examined in the full feasibility study as another opportunity for cost reduction.

## 18.6 Power Distribution

The power plant will feed power to a 13.8 kV metal clad switchgear located inside a prefabricated building. This building will be located near the power plant. The 13.8 kV switch gear will contain 7 breakers (1 spare) that will distribute the power throughout the site. Table 18-2 identifies the main power transformers that are located on-site. A single line diagram is shown in Figure 18-2.

**Table 18-2: Power Transformers**

| Process Area | Size and Type | Location |
|---|---|---|
| Crusher | 500 kVA, 13.8 kV/600V Oil filled Type, Outdoor Transformer | Electrical Room 1 |
| Grinding 4160 V Motors/Loads | 2500 kVA, 13.8 kV/4160V Dry Type, indoor Transformer | Electrical Room 3 |
| Grinding 600 V loads | 1500 kVA, 13.8 kV/600V Dry Type, indoor Transformer | Electrical Room 3 |
| Flotation | 2500 kVA, 13.8 kV/600V Oil filled, Outdoor Transformer | Electrical Room 4 |
| Filtration | 1500 kVA, 13.8 kV/600V Oil filled, Outdoor Transformer | Electrical Room 4 |
| Camp | 1000 kVA, 13.8 kV/600V Oil filled, Outdoor Transformer | Camp |
| Main Mine Portal | 3- 750 kVA, 13.8 kV/600V Dry Type portable Transformers<br>1 – 500 kVA 13.8 kV/ 600 V Oil filled, Outdoor transformer | Main Mine Portal |
| Esso Mine Portal | 3- 750 kVA, 13.8 kV/600V Dry Type portable Transformers<br>1 – 500 kVA 13.8 kV/ 600 V Oil filled, Outdoor transformer | Esso Mine Portal |

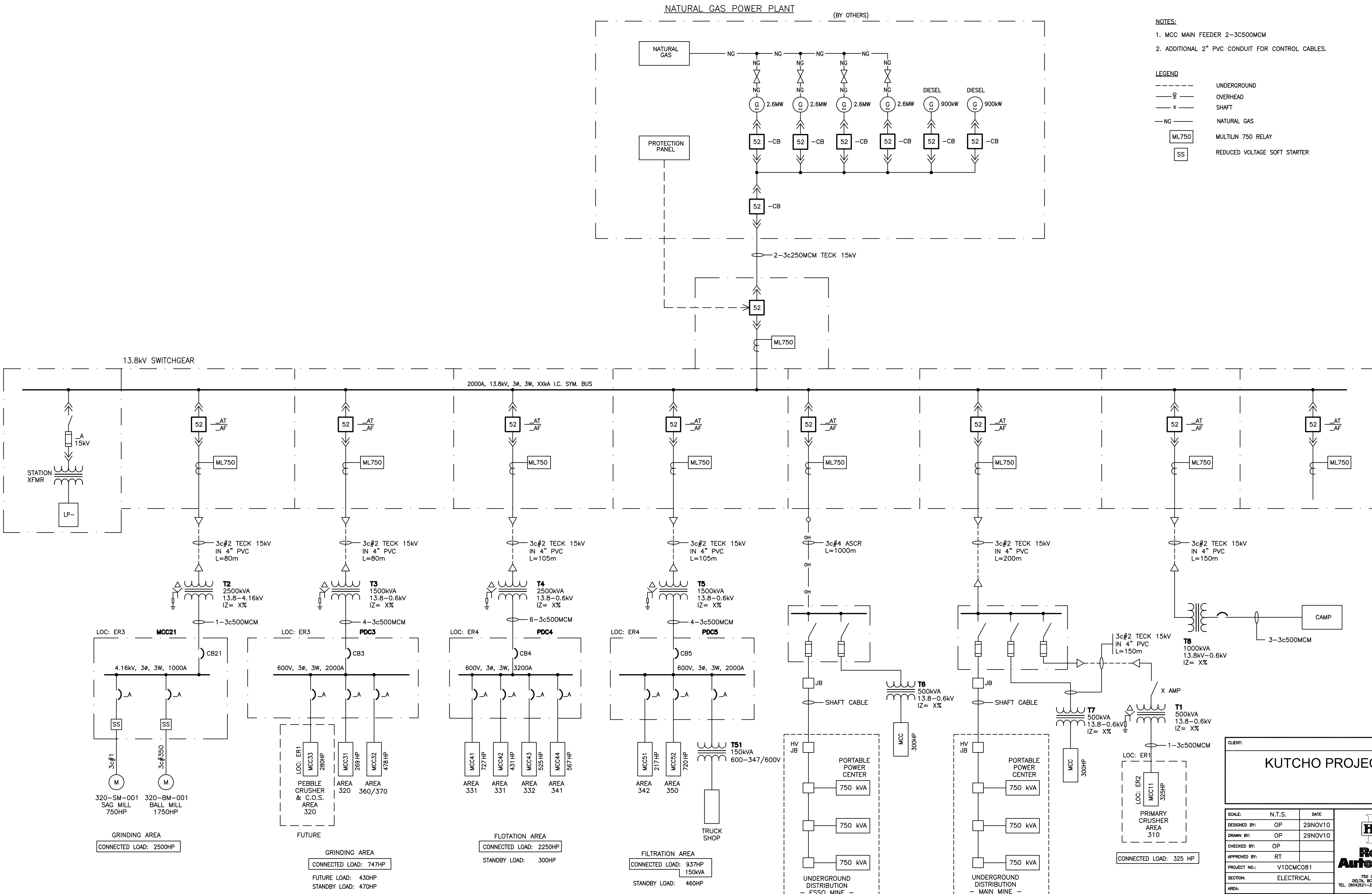
Source: Hinz (2011).

NOTES:

1. MCC MAIN FEEDER 2–3C500MCM
2. ADDITIONAL 2" PVC CONDUIT FOR CONTROL CABLES.

LEGEND

| | |
|---|---|
| – – – – – | UNDERGROUND |
| —OH— | OVERHEAD |
| —x— | SHAFT |
| —NG— | NATURAL GAS |
| ML750 | MULTILIN 750 RELAY |
| SS | REDUCED VOLTAGE SOFT STARTER |




TOTAL CONNECTED LOAD: 6758HP + 1150kVA
MINE LOAD: 1720HP
STANDBY LOAD: 930HP

REFERENCE

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| B | PRELIMINARY | 02FEB11 | OP | OP | RT |
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CLIENT:

KUTCHO PROJECT

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| DRAWN BY: | OP | 29NOV10 |
| CHECKED BY: | OP | |
| APPROVED BY: | RT | |
| PROJECT NO.: | V10CMC081 | |
| SECTION: | ELECTRICAL | |
| AREA: | | |

HINZ
Rockwell Automation
750 CHESTER ROAD
DELTA, BC, CANADA V3M 6J1
TEL. (604)521–2233 FAX. (604)521–4466

TITLE:

2500TPD KUTCHO COPPER PROJECT
POWER DISTRIBUTION
SINGLE LINE DIAGRAM

DRAWING NO.: D10081–30–001 REV. NO.: B

### 18.6.1 Remote Loads

Remote loads will be served by a 13.8 kV overhead power line. The power line will be a single pole structure and will feed power to the Esso mine portal.

### 18.6.2 Plant Voltage Level

The following is a summary of the plant voltage levels:

- Plant Distribution Voltage – 13.8 kV, 60 Hz, three phase;
- Plant Operating Voltage - 600 V, 60 Hz, three phase;
- Motor Voltage - under 300 hp - 600 V, 60 Hz, three phase
    - 300 hp and above – 4,160 V, 60 Hz, three phase;
- Control Voltage - 120 V, 60 Hz, single phase;
- Device Net – 24 VDC;
- Lighting Voltage - Indoor and Outdoor – 347 V, 60 Hz, single phase; and
- Offices, MCC and Control Rooms, 120 V, 60 Hz, Single phase.

## 18.7 Fuel Storage

The main components of the LNG gasification facility are the storage tanks, the hot water vaporizer, the containment system, and the control system. The recommended capacity of the LNG storage tanks is a minimum of 454,000 litres (L).This will provide 5.7 days of capacity after the tanks are completely filled. This is considered to be a minimum to allow for disruption in the fuel supply due to weather or equipment breakdown. The total footprint of the proposed facility is approximately 50 m x 50 m.

The diesel facility is similar in overall design but smaller than the LNG facility. The proposed facility with seven days of supply, approximately 120,000 L would have a footprint of 26 m x 35 m and would use double walled storage vessels which do not require additional berms or other secondary containment measures.

## 18.8 Explosives Storage

Packaged explosives and blasting agents will be utilized for all mining and will be stored at permitted locations as per the British Columbia Mines Act and Regulations. There are no plans to utilize bulk explosives.

## 18.9 Water Treatment Plant

Surface, underground and process plant water will be treated by a single water treatment plant with a capacity of 163.6 litres per second (L/s) (600 $m^3$/hr). Lime is the industry standard for precipitating heavy metals from water. There are no special contaminants that require any additional treatment or chemicals. Should inlet contaminant concentrations vary, as is expected with varying surface runoff, the operator would adjust lime addition to suit. A comprehensive monitoring program is required to ensure the plant operates efficiently.

The water treatment plant will consist of a rapid mix tank which receives the incoming water to be treated, lime slurry and recycled sludge from the clarifier. The slurry gravitates from the rapid mix tank to the lime reactor tank which allows for additional agitation and the injection of compressed air to accelerate the chemical reaction. From the lime reactor tank, the slurry gravitates to a clarifier. Flocculant is added to improve the sludge settling rate and density, as well as producing a clear overflow product.

A small quantity of the clarifier underflow is discharged to a disposal area while the majority is recycled to the agitated sludge mixing tank where the lime slurry is added.

## 18.10 Stewart Concentrate Storage Facility

Concentrates from the operation will be stored in two compartment concentrate storage and load-out facility in Stewart. The facility can store 20 t of copper concentrate and 10 t of zinc concentrate. The facility will be located in a planned expansion area of the existing Stewart concentrate storage and load-out area. The proposed facility for the Project will tie into the existing ship loading facilities.

## 18.11 Mine Waste and Water Management

### 18.11.1 Overview

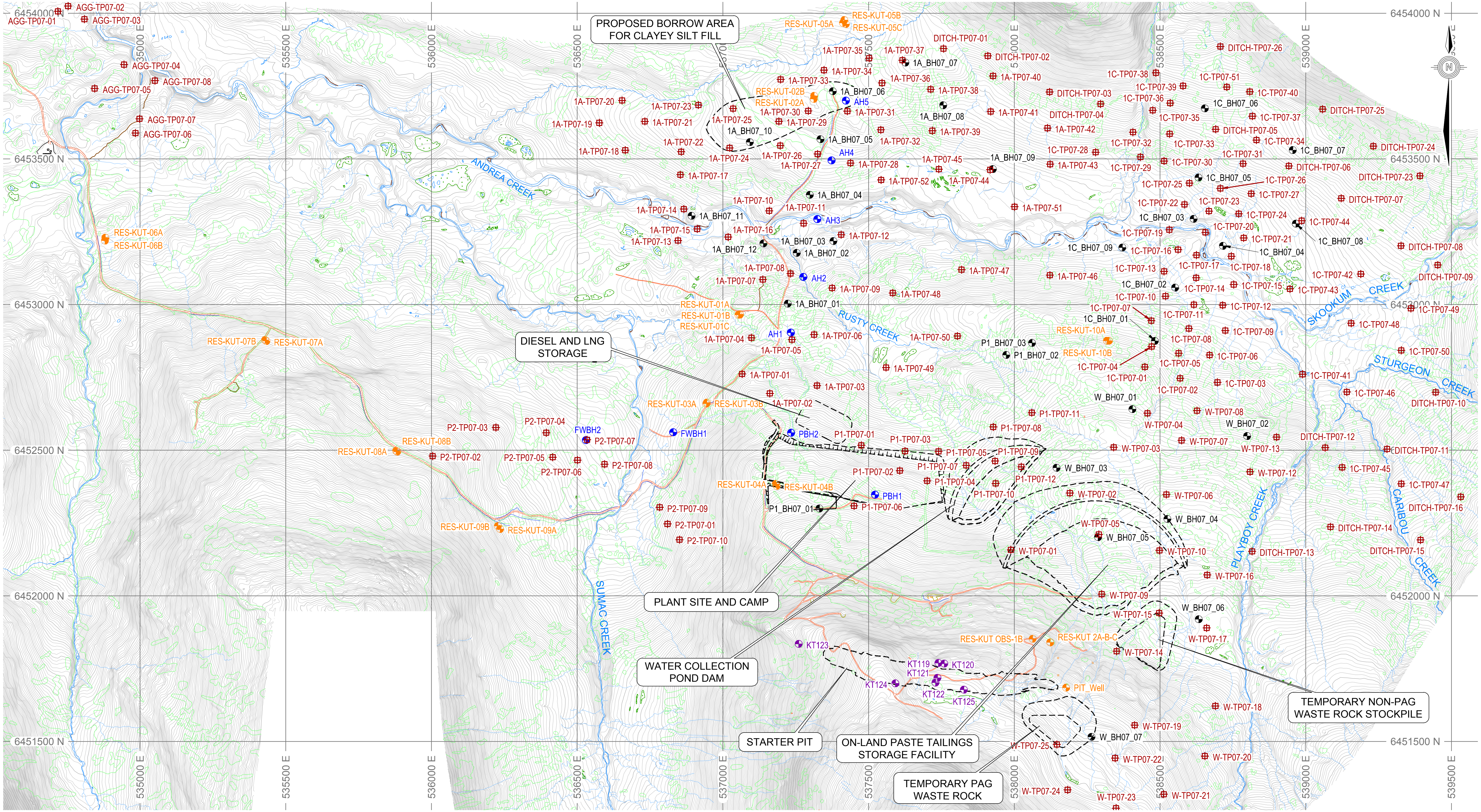
The waste management plan has been developed based on the mine plan that includes an initial small starter pit in the Main deposit and underground mines in the Main and Esso deposits. The starter pit will be pre-stripped in year -1 and will provide ore in year 1 while the underground mine is being developed.

The tailings from the mill will be disposed of in three locations: underground mines as paste backfill material, an on-land paste tailings storage facility, and the mined-out starter pit.

Waste rock from the mine operation includes PAG and non-PAG waste rock. A portion of the on-surface non-PAG waste rock will be used as site construction materials during the early site construction stage. The remaining portion of the non-PAG waste rock will be temporarily stored in a stockpile and later used as site construction materials and as underground mine backfill. Theon-surface PAG waste rock disposal will consist of three methods: 1) co-disposal with paste tailings in the on-land paste tailings storage facility, 2) co-disposal with paste tailings as backfill in the mined-out starter pit, and 3) hauled back to underground mines as backfill. A temporary PAG waste rock storage facility is required to store the PAG waste rock during early mine operation before the waste rock is permanently disposed.

The mine site water management plan includes diverting the clean surface water from undisturbed areas using ditches and berms, collecting the mine site contact water in a water collection pond and sumps, treating the site contact water in a water treatment plant, reclaiming and reusing water for ore processing, and discharging the excess treated water to the receiving environment when the water quality meets the site discharge criteria.

Figure 18-3 presents an overall site plan for the mine waste and water management facilities.




LEGEND

- 2007 INVESTIGATION BOREHOLES (AMEC 2008)
- 1982 INVESTIGATION BOREHOLES (GOLDER 1982)
- 1984 INVESTIGATION BOREHOLES (GOLDER 1984)
- 2006 AND 2007 RESCAN MONITORING WELLS (RESCAN 2010a)
- 2007 INVESTIGATION TEST PITS (AMEC 2008)
- EXISTING ROAD
- CREEK
- TREED AREA
- PROPOSED SITE FACILITIES

NOTES:

1. AS REQUESTED BY CLIENT. INFORMATION AND DATA UNCHANGED FROM FEBRUARY 9, 2011. UPDATED TO CURRENT CLIENT LOGO AND TETRA TECH FORMAT ON JULY 14, 2017.
2. BASE DRAWING PROVIDED BY JDS.

0 500 m

Scale: 1: 12 500

STATUS
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CLIENT

DESERT STAR RESOURCES

TETRA TECH

KUTCHO PREFEASIBILITY STUDY
MINE WASTE AND WATER MANAGEMENT

SITE PLAN

| PROJECT NO. | DWN | CKD | REV |
|---|---|---|---|
| E14101116 | EL | GZ | 0 |
| OFFICE | DATE | | |
| EDMONTON | FEBRUARY 9, 2011 | | |

Q:\Edmonton\Engineering\E141\Projects\Kutcho Temp\E14101116_FIG D1_R0.dwg [FIGURE D1] July 14, 2017 - 10:40:01 am (BY: LEE, ELVIN)

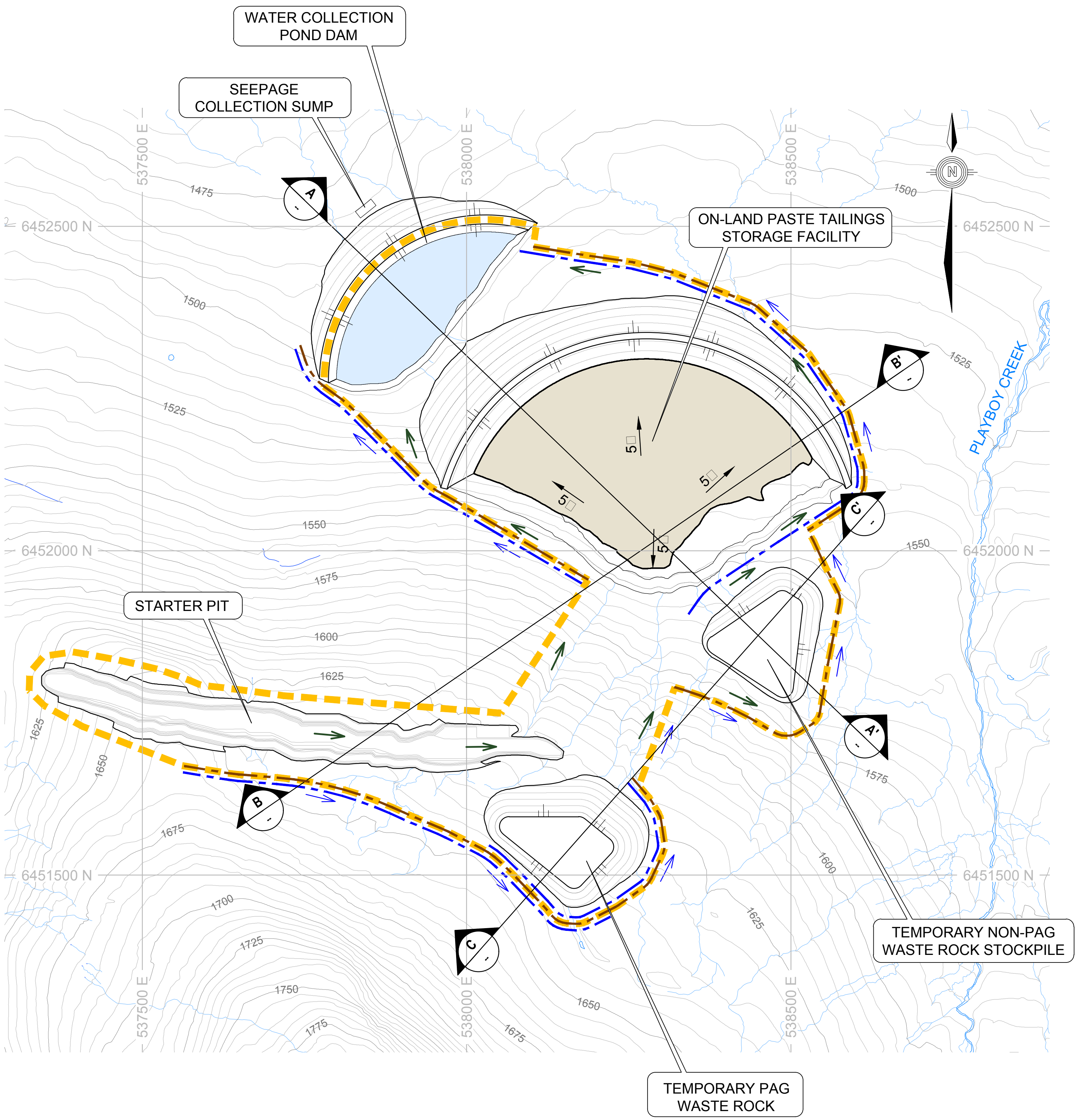
### 18.11.2 Location of Mine Waste and Water Management Facilities

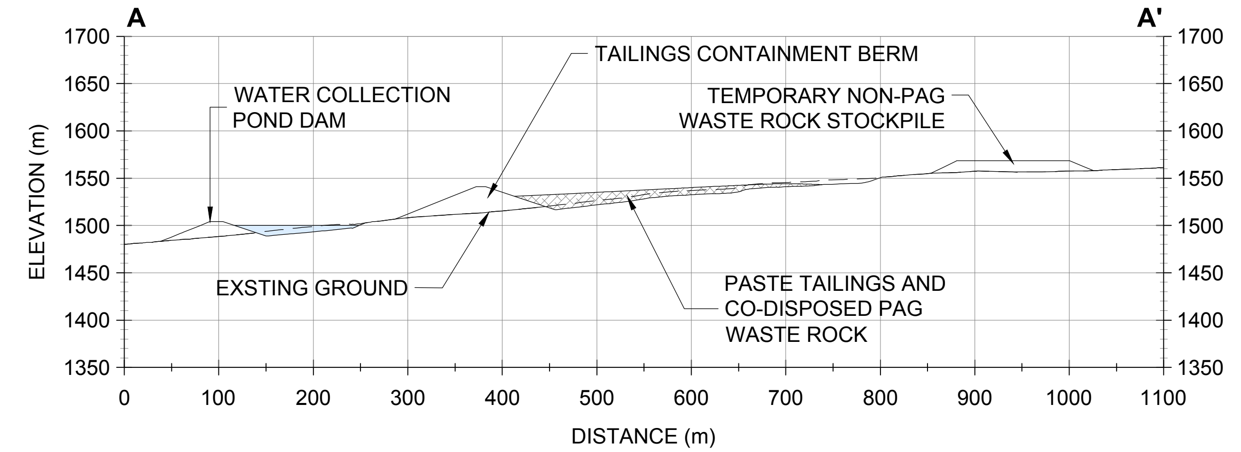
The proposed site for the on-land paste tailings storage facility is located northeast of the starter pit and southeast of the proposed process plant area, as shown in Figure 18-3. The proposed location has the following advantages:

- Close to the process plant area and starter pit, therefore, a short pumping distance for paste tailings disposal and a short haul distance from the starter pit to the facility for the PAG waste rock to be co-disposed with the paste tailings in the facility;
- Contact water from the facility will be easily collected in the downstream water collection pond immediately northwest of the facility; and
- Greater than 180 m away from major creeks (such as Andrea Creek, Sumac Creek, Playboy Creek, Rusty Creek, etc) and no fish bearing streams and creeks covered by the facility.

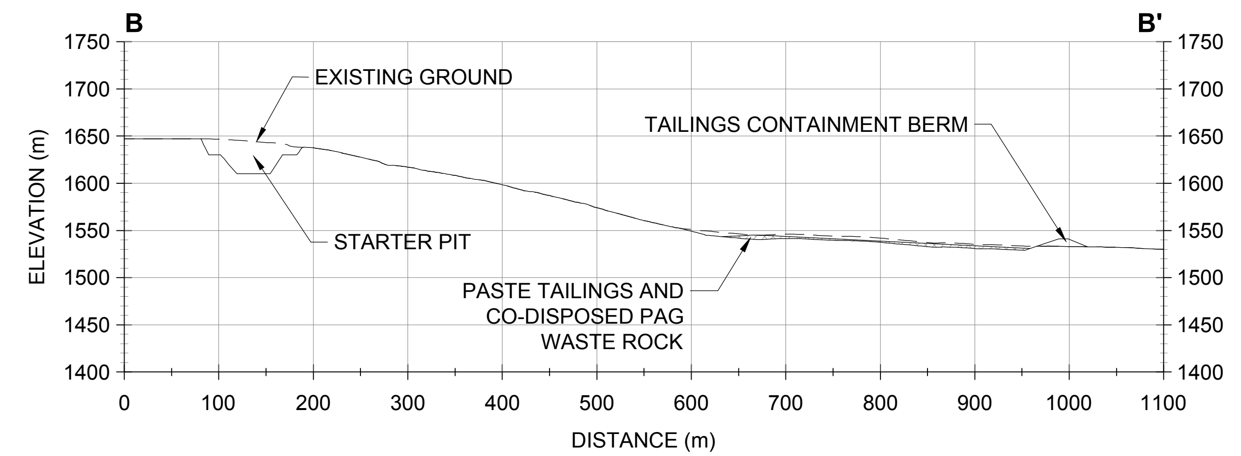
The proposed temporary PAG waste rock storage facility is located immediately southeast of the east end of the starter pit. A portion of the PAG waste rock from the starter pit will be placed in the dump in years -1 and 1 and re-handled back to the mined-out starter pit in years 2 and 3. The short haul distance between the starter pit and the temporary PAG waste rock storage facility will facilitate the mine operation and reduce the operation cost.

This waste rock storage facility location also facilitates the site water management since the contact water from the waste rock storage facility will naturally flow down the slope into the water collection pond, as shown in Figure 18-4.

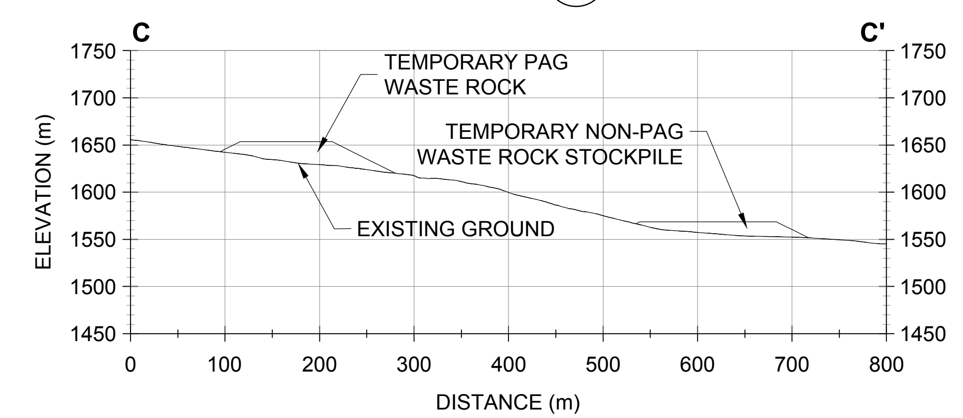



**SECTION** A
1:8000




**SECTION** B
1:8000




**SECTION** C
1:8000

LEGEND

- MAJOR CONTOUR (5 m INTERVAL)
- MINOR CONTOUR (1 m INTERVAL)
- DITC□
- BERM
- CATC□MENT BOUNDARY
- CLEAN SURFACE WATER
- CONTACT WATER
- EXISTING CREEKS AND STREAMS

0 500 m
Scale: 1: 8 000

NOTES:
1. AS REQUESTED BY CLIENT. INFORMATION AND DATA UNCHANGED FROM FEBRUARY 9, 2011. UPDATED TO CURRENT CLIENT LOGO AND TETRA TECH FORMAT ON JULY 14, 2017.

STATUS
ISSUED FOR REVIEW

CLIENT
DESERT STAR
RESOURCES

TETRA TECH

**KUTCHO PREFEASIBILITY STUDY**
**MINE WASTE AND WATER MANAGEMENT**

**MINE OPERATION (YEAR 1)**

| PROJECT NO. | DWN | CKD | REV |
|---|---|---|---|
| E14101116 | EL | GZ | 0 |
| OFFICE | DATE | | |
| EDMONTON | FEBRUARY 9, 2011 | | |

Q:\Edmonton\Engineering\E141\Projects\Kutcho Temp\E14101116_FIG D2_R0.dwg [FIGURE D2] July 14, 2017 - 10:54:24 am (BY: LEE, ELVIN)

A temporary non-PAG waste rock stockpile is required to store the non-PAG waste rock generated during early years of mine operation. Eventually this non-PAG waste rock will be used for construction of site facilities during mine operation and at mine closure and as underground backfill during late years of mine operation. The proposed temporary non-PAG waste rock stockpile is located immediately southeast of the paste tailings facility and northeast of the starter pit to reduce waste rock haul distances and facilitate water management.

The mined-out starter pit will becomes one of the mine waste management facilities.

The water collection pond and dam are located east of the process plant and camp area and northwest of the on-land paste tailings storage facility. This location facilitates overall site water management since the contact water from almost all mine waste management facilities will naturally flow down into the pond. The pond is also near the process plant area where the water treatment plant will be located so the water pumping distance from the pond to the plant is short. The proposed dam is located greater than 330 m away from major creeks (such as Andrea Creek, Sumac Creek, Playboy Creek, Rusty Creek, etc.) and does not cover any fish bearing streams and/or creeks.

### 18.11.3 Surface Conditions

The proposed on-land paste tailings storage facility, temporary non-PAG waste rock stockpile, and water collection pond dam are situated in the areas immediately below the tree-line boundary. The areas are generally covered with peat, grasses, bushes, and groups of subalpine forest (mainly stunted alpine spruce) (Figure 18-3). The proposed starter pit and temporary PAG waste rock storage facility are just above the tree-line boundary. These areas are covered with peat, grasses, and some bushes. Bedrock outcrops can be seen over a strip of zone slightly north of the north side rim of the proposed starter pit.

The existing ground in the mine waste management area has natural slopes ranging from 5% to 30% with a typical slope around 10%. The existing streams and channels are presented in Figure 18-3.

### 18.11.4 Geotechnical Conditions

The interpretation of the geotechnical conditions in the proposed mine waste and water management facility areas is based on the investigations carried out in 2007 and reported in AMEC (2008), and a site reconnaissance by EBA and JDS in September 2010. The interpretation of the geotechnical conditions in the starter pit area were based on a site investigation conducted in 1982 by Golder (1982). The locations of the boreholes and test pits are shown in Figure 18-3. Groundwater conditions were based on the site investigations and the hydrogeology baseline study reported by Rescan (2010a). Table 18-3 summarizes the geotechnical conditions in the proposed mine waste and water management facility areas.

## Table 18-3: Mine Waste and Water Management Facility Area Geotechnical Conditions

| Mine Waste and Water Management Facility | On-land Paste Tailings Storage Facility | Temporary non-PAG Waste Rock Stockpile | Temporary PAG Waste Rock Storage Facility | Starter Pit | Water Collection Pond Dam |
|---|---|---|---|---|---|
| Boreholes | W-BH07-04 and W-BH07-05 | None | W-BH07-07 | KT119 to KT125 | Close to W-BH07-03 |
| Test Pits | W-TP07-01, W-TP07-02, W-TP07-05, W-TP07-09, and W-TP07-10 | W-TP07-14 and W-TP07-15 | W-TP07-25 | None | P1-TP07-07, P1-TP07-09, P1-TP07-10, and P1-TP07-12 |
| Peat/organic Soil Thickness | 0.1 to 0.3 m | 0.1 m | 0.1 to 0.2 m | Not logged | 0.1 to 0.2 m |
| Total Overburden Thickness | 2.4 m and 2.9 m in the two boreholes; 2.4 to 2.7 m in three test pits and refusal at 2.9 m and 3.4 m on hard soils for the other two test pits | Two test pits refusal at 2.6 m and 4.0 m on hard soils | 3.1 m in the borehole and 2.3 m in the test pit | 4.0 m to 8.2 m | 0.9 m to greater than 3.0 m (end of test pits) |
| Types of Overburden Soils below Peat/Organic Soil | Till (sandy silt to silt, local sand and gravel zone) | Till (sand to silt and sand) | Till (sand and gravel, silty sand, sand, silt and sand) | No description | Sand, gravelly sand, sand and gravel, and sandy clay |
| Shallow Bedrock Conditions | Extremely weathered (exhibits soil properties) to depths of 4.0 to 7.3 m from the ground surface; RQD of 0 to 20% from 7.3 to 10 m at W_BH07_04 and 32% from 4 m to 5 m at W-BH07_05 | unknown | Extremely weathered (exhibits soil properties) to a depth of 6.7 m from the ground surface; RQD of 0 to 58% from 6.7 m to 10.6 m at W_BH07_07 | Closely spaced joints/fractures with low RQD to depths from 5 m to 15 m from the ground surface | RQD of less than 37% to depth of 4.8 m in W-BH07-03; RQD greater than 74% from 4.8 to 9.3 m |
| Groundwater Conditions | Groundwater observed at 1.8 m and 2.9 m from the ground surface in the two boreholes; seepage was observed at 0.9 m depth in one of the five test pits and no seepage observed in the remaining four test pits | No seepage was observed in the two test pits | Groundwater observed at 2.1 m depth in the borehole and seepage observed at 0.9 m depth in the test pit | Below the bottom (at elevation of 1,610 m) of the proposed starter pit in all boreholes, except for one borehole (KT122) where the groundwater was observed at 20 m depth (or around 1627 m elevation); Estimated groundwater surface elevation around 1600 m based on a groundwater elevation contour map in Rescan (2010a) | Groundwater at 1.4 m dept h in the borehole W-BH07-03;<br>No seepage was observed in the four test pits in this area |

Source: EBA (2010a,b).

### 18.11.5 Meteorology and Hydrology Conditions

The meteorology and hydrology conditions at the mine site are documented in various baseline reports by Rescan (2007, 2008, 2009, and 2010b). There were two meteorology stations in the proposed Project area. The Kutcho Airstrip Station was located on the west side of Kutcho Creek, approximately 7.5 km west of the proposed mine site.

The Deposit Station was located near the proposed mine site. The Kutcho Airstrip Station (1,247 masl) has been operational since September 2005 and the Deposit Station (1,435 m) since September 2006.

Both meteorological stations had been removed by the end of 2015.

At the Deposit Station, measured monthly average air temperatures were as low as -18.6°C in December 2008 and as high as 14.0°C in August 2008. The extreme minimum hourly temperature of -41.0°C was recorded in January 2008, while the extreme maximum temperature of 28.2°C was recorded in July 2009 (Rescan 2008, 2010b). The average annual temperature at the Deposit Station was -2.3°C from November 2007 to October 2008 and -2.2°C from October 2008 to September 2009. Permafrost ground is not expected under these climatic conditions and was not observed during site investigations in the mine site area.

The average annual precipitation estimates for the Andrea Creek watershed, which includes the proposed mine site, were between 771 to 806 mm.

The observed annual runoff at four locations in Andrea Creek during 2006 to 2008 ranged from 571 to 976 mm. Rescan (2009) reported that the annual runoff estimates for the Project area were 470 mm, 738 mm, and 1,010 mm for return periods of 1 in 100 dry, mean, and 1 in 100 wet years. These values were used for the water management in this study.

The estimated average monthly runoff for the Andrea Creek watershed was documented in the 2008 hydrology baseline report (Rescan 2009). The monthly percentage runoff values (Table 18-4) were used in the water management in this study. Rescan (2009) reported that observed flow data in Andrea Creek indicted that freshet runoff began in late-May 2006, with high flow conditions ending in late June. The peak flow for the year occurred on June 15, 2006.

The monthly lake evaporation values were estimated for various elevations in the Project area in Rescan (2010b). The values at an approximate elevation of 1,515 m were presented in Table 18-4.

**Table 18-4: Estimated Monthly Runoff and Lake Evaporation in Andrea Creek Watershed**

| Month | 1 | 2 | 3 | 4 | 5 | 6 | 7 | 8 | 9 | 10 | 11 | 12 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Estimated Average Monthly Runoff (mm) | 10 | 7 | 7 | 9 | 87 | 219 | 157 | 88 | 69 | 51 | 21 | 13 |
| Monthly Percentage of Runoff (%) | 1.4 | 0.9 | 0.9 | 1.2 | 11.8 | 29.7 | 21.3 | 11.9 | 9.4 | 6.9 | 2.8 | 1.8 |
| Estimated Lake Evaporation at Elevation of 1,515 m (mm) | N/A | N/A | N/A | N/A | 71 | 74 | 67 | 83 | 36 | N/A | N/A | N/A |

Source: Rescan (2010b).

Flood flow estimates for the small watersheds in the Andrea Creek watershed area were documented in the 2008 hydrology baseline report (Rescan 2009). The estimated equivalent runoff for a flood event (estimated time of concentration of 5.36 hours) was 13.2 mm, 19.5 mm, and 27.6 mm for a return period of 1 in 2, 1 in 10, and 1 in 100, respectively, for the smallest watershed (Skookum Creek) in the Andrea Creek watershed area evaluated in Rescan (2009). The Skookum Creek watershed has a catchment area of 5.6 $km^2$.

Between October 2008 and September 2009, average wind speed observed at the Deposit Station was 2.1 m/s. Wind mainly blew from the south at the Deposit Station. A peak snow depth recorded at the Deposit Station was 161 cm on February 6, 2009 and 160 cm on April 11, 2008.

### 18.11.6 Tailings Characteristics

The Kutcho tailings will be derived from the massive sulphide ore deposit. The specific gravity of tailings was determined to be 3.54. A particle-size distribution curve for a tailings sample indicated that the tailings have a maximum particle size of 0.42 mm with 42.4% passing a size of 0.038 mm.

Morin and Hutt (2007c) reported the metallurgical analysis results of the Kutcho tailings solids and supernatants and summarized the following findings:

- The tailings solids were dominated by iron and sulphur, which was likely mostly pyrite. Silicon and aluminum were also sometimes high, reflecting the presence of quartz and aluminosilicate minerals. Copper and zinc were also elevated in the tailings from some cycles;
- The tailings solids contained some calcium and magnesium, likely representing some neutralization potential (NP), so there would likely be some lag time before acidic water would be generated from any aerially exposed tailings; and
- The tailings supernatants were alkaline, from pH 8.65 to 12.1, which is typical of this metallurgical process. Despite the alkaline pH values, some samples contained more acidity than alkalinity. This was likely due, as least in part, to detectable total thiosalts, which are partially oxidized species in water and generate acidity as they oxidize fully to sulphate.

The acid-base accounting (ABA) data of the Kutcho tailings samples were documented in Morin and Hutt (2007b). The tailings had a total sulphur content of 27.2%, a NP value of 258 kg $CaCO_3$ equivalent /tonne, and an acid generation potential (AP) value of 850 kg $CaCO_3$, which indicated net acid generating.

Based on mineralogy statistics of the core samples from boreholes drilled in the Main and Esso deposits, the ore deposits contain up to 100% of pyrite. Tailings are approximately 80% pyrite and are strongly net acid generating.

### 18.11.7 Waste Rock Characteristics

The Project has a long track record of geochemical studies, reaching back decades to the early 1980's. Morin and Hutt (2007a) summarized and re-interpreted the historical information, which included more than 100 ABAs, mineralogical studies, 25 laboratory-based humidity cells, three on-site leach cribs, and periodic monitoring of on-site water chemistry from the buried Main deposit adit and natural accelerated metal leaching and acid rock drainage (ML/ARD) locations. Morin and Hutt (2007a) also compiled and interpreted the newer (2006) information of ABAs and total-element contents of core samples from the footwall and hangingwall relatively close to the ore zone in the Main deposit. The key findings from Morin and Hutt's study are summarized below:

- Total-element analyses showed that the footwall and hanging-wall rock in the Main deposit consisted mostly of silica, aluminum, iron, magnesium, sodium, sulphur, potassium, and calcium;
- Mineralogy analyses of the samples indicated that for samples with less than 1% pyrite, the dominant mineral was plagioclase. Other major minerals included quartz, sericite, biotite, epidote, amphibole, chlorite, and carbonate. For samples with 5% or more of pyrite, the dominant minerals were pyrite, sericite, quartz, and carbonate;
- ABA results indicated that total sulphur in the historical and recent rock samples ranged from below detection (<0.01%S) to 47.1%S. In many samples, most total sulphur consisted of iron-based, PAG sulphide. Bulk NP in the recent rock samples ranged from zero kg $CaCO_3$ equivalent/t to 600 kg/t. Samples from the footwall showed they had the least amount of neutralizing capacity and thus the lowest time to net acid generation. The NP in many hangingwall samples was dolomite with some calcite;
- The results of laboratory humidity cells of the rock samples indicated that only 2 of the 25 humidity cells containing rock from the Main deposit became acidic during 20 weeks of testing in 1989. The rate of sulphate production spanned roughly three orders of magnitude in the cells, and correlated well with solid-phase total sulphur in the samples. Compared to the International Kinetic Database with 75 mine sites around the world, the Kutcho sulphate rates were very low to high;
- The three on-site, 20-tonne test cribs with layered rock from the Main deposit adit have become more acidic (pH 2.2 to 2.5) since 1990. Aqueous concentrations of some dissolved elements generally increased or decreased as pH fell, including aluminum, copper, and molybdenum; a few generally correlated with sulphate fluctuations, including dissolved cadmium and manganese, and several appeared independent of changes in pH and sulphate; and
- Monitoring data from on-site locations like the buried Main deposit adit, rusty creek with natural accelerated ML/ARD, and flowing (artesian) drillholes were consistently at or above pH 7.0, with relatively low sulphate (recently around 40 milligrams per litre [mg/L]) and relatively high alkalinity (recently around 80 mg/L, except for the natural ML/ARD around 20 mg/L). Dissolved concentrations of many elements at these locations were often near or below detection, but a few were sometimes elevated like dissolved cadmium in the natural ML/ARD, copper, and zinc.

### 18.11.8 Tailings Management Plan

The tailings from the mill will be disposed of in three areas: 1) underground mines as backfill material, 2) on-land paste tailings storage facility, and 3) mined-out starter pit as backfill. Table 18-5 summarizes the overall tailings management plan.

### Table 18-5: Tailings Management Plan

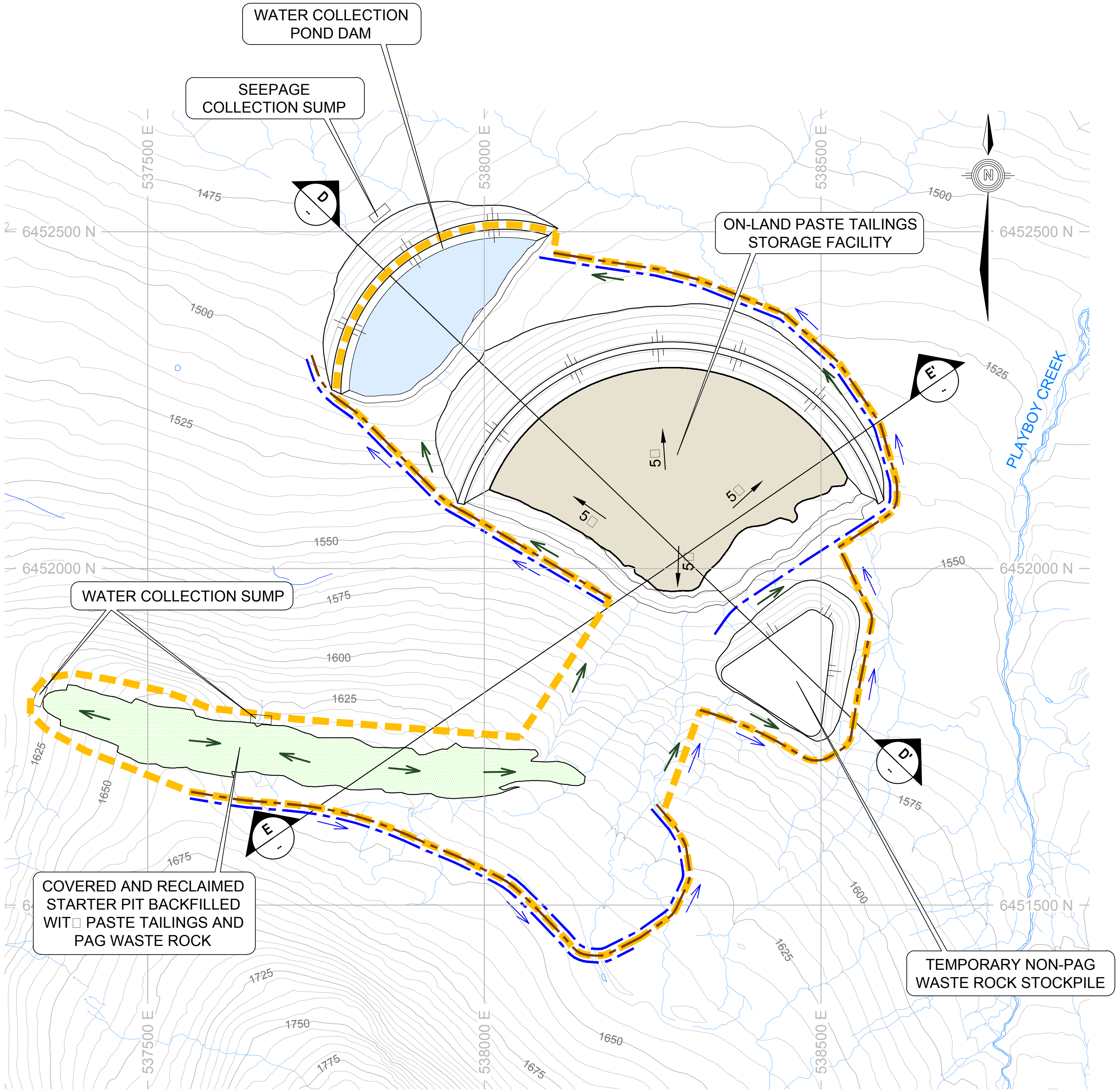
| Production Year | Total Tailings (kt, dry solids) | Tailings Used as Underground Mine Backfill (kt, dry solids) | Tailings Placed in On-land Paste Tailings Storage Facility (kt dry solids) | Tailings Placed in Mined-out Starter Pit (kt dry solids) |
|---|---|---|---|---|
| -1 | Pre-production | | | |
| 1 | 817 | 228 | 589 | |
| 2 | 817 | 408 | | 408 |
| 3 | 817 | 408 | | 408 |
| 4 | 817 | 408 | 225 | 183 |
| 5 | 817 | 384 | 433 | |
| 6 | 817 | 408 | 408 | |
| 7 | 817 | 408 | 408 | |
| 8 | 817 | 817 | | |
| 9 | 817 | 429 | 388 | |
| 10 | 817 | 408 | 408 | |
| 11 | 817 | 408 | 408 | |
| 12 | 361 | 181 | 181 | |
| **Total** | **9,345** | **4,897** | **3,448** | **1,000** |

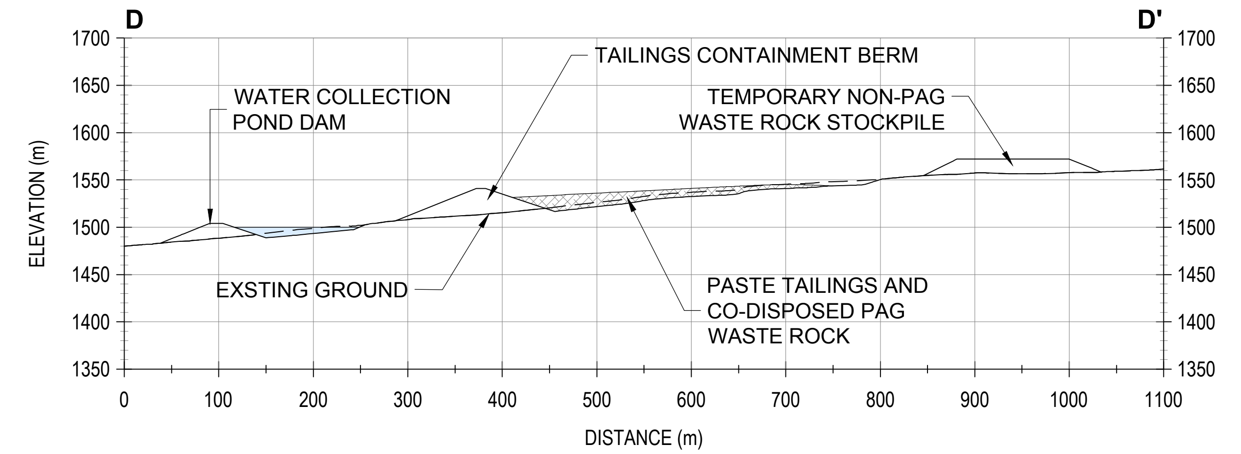
Source: JDS (2017).

A portion of the tailings will be directed to a backfill plant to be conditioned and sent to the underground mine as paste backfill. The portion of the tailings that can be used as paste backfill is the coarse fraction of the tailings. Some fine fraction of the tailings will also be used to backfill mined Esso drifts in year 8 and early year 9. The remaining portion of the tailings will be dewatered to a consistency of paste and pumped into the on-land paste tailings storage facility and the mined-out starter pit for disposal.

Paste tailings properties were not available at this stage of study. The dry density of the in-place tailings was assumed to be 2.0 tonnes per cubic metres ($t/m^3$) based on the tailings specific gravity of 3.54 and an assumed void ratio of 0.77. This dry density is equivalent to a solids content of 82.3%. An average of surface slope of the paste tailings after deposition was assumed to be 5% in this study. These parameters should be re-evaluated in the next stage of study when site specific testing on the paste tailings is available.

Figures 18-5 to 18-6 present plan views and several cross-sections through the mine waste and water management facilities during mine operation and early years after mine closure.

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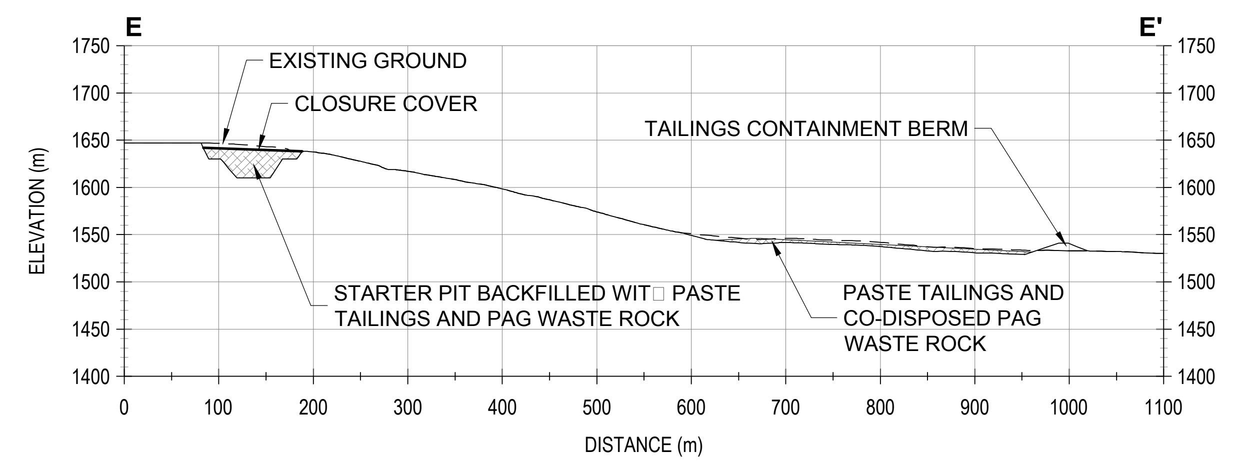
1:8000




**SECTION** E
1:8000

LEGEND

- MAJOR CONTOUR (5 m INTERVAL)
- MINOR CONTOUR (1 m INTERVAL)
- DITC□
- BERM
- CATC□MENT BOUNDARY
- CLEAN SURFACE WATER
- CONTACT WATER
- EXISTING CREEKS AND STREAMS

0 500 m
Scale: 1: 8 000

NOTES:
1. AS REQUESTED BY CLIENT. INFORMATION AND DATA UNCHANGED FROM FEBRUARY 9, 2011. UPDATED TO CURRENT CLIENT LOGO AND TETRA TECH FORMAT ON JULY 14, 2017.

STATUS
ISSUED FOR REVIEW

CLIENT
DESERT STAR RESOURCES

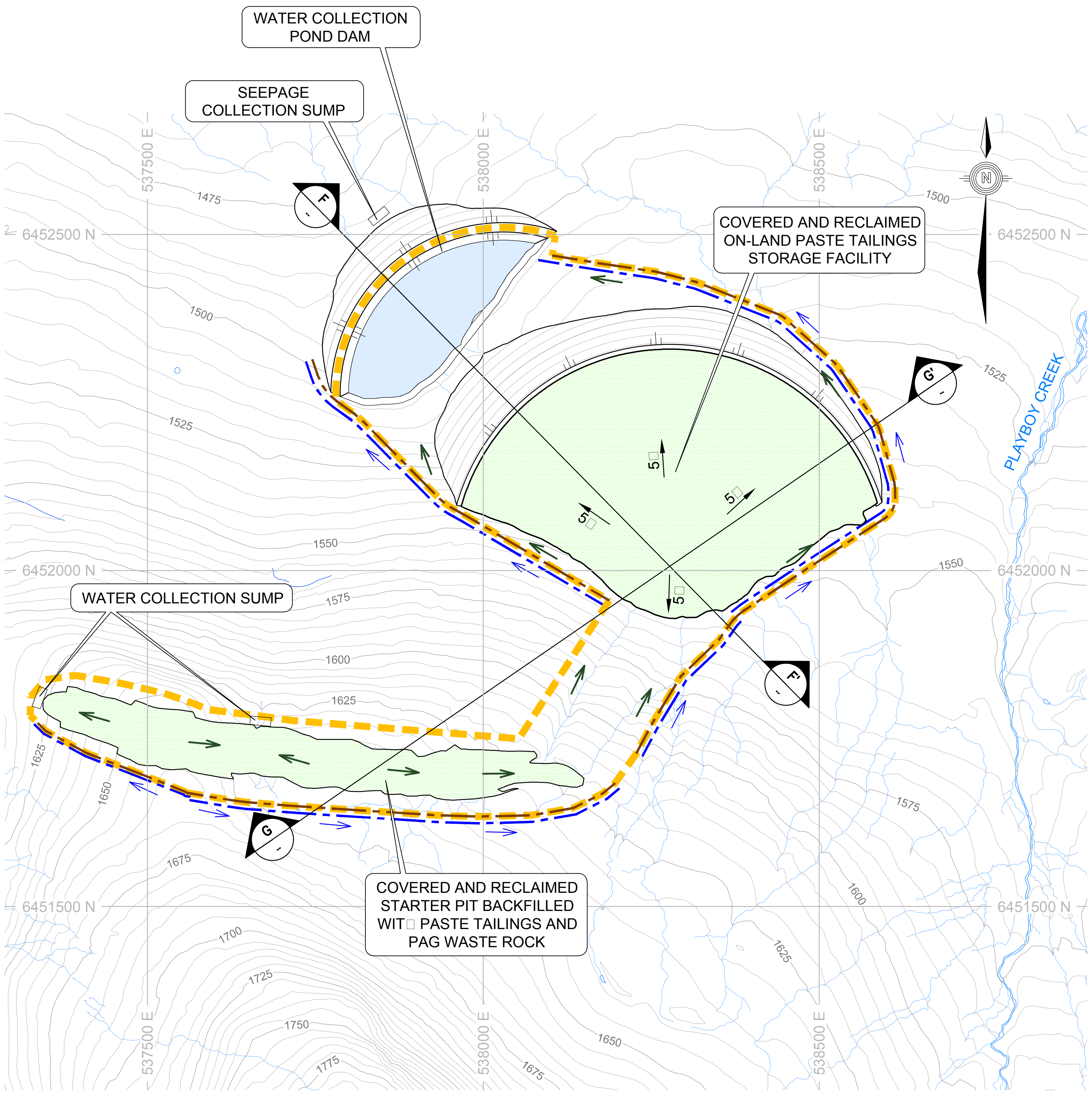
TETRA TECH

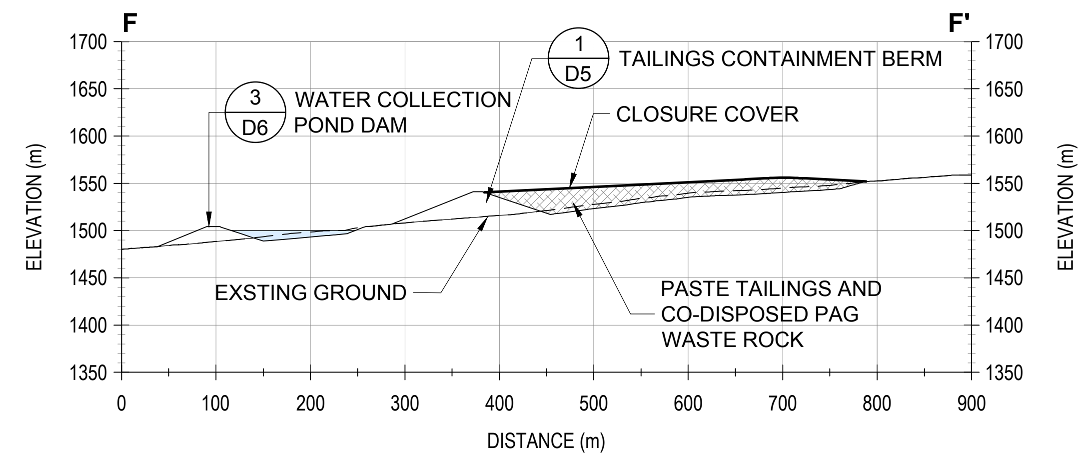
**KUTCHO PREFEASIBILITY STUDY**
**MINE WASTE AND WATER MANAGEMENT**

**MINE OPERATION (YEAR 4)**

| PROJECT NO. | DWN | CKD | REV |
|---|---|---|---|
| E14101116 | EL | GZ | 0 |

| OFFICE | DATE |
|---|---|
| EDMONTON | FEBRUARY 9, 2011 |

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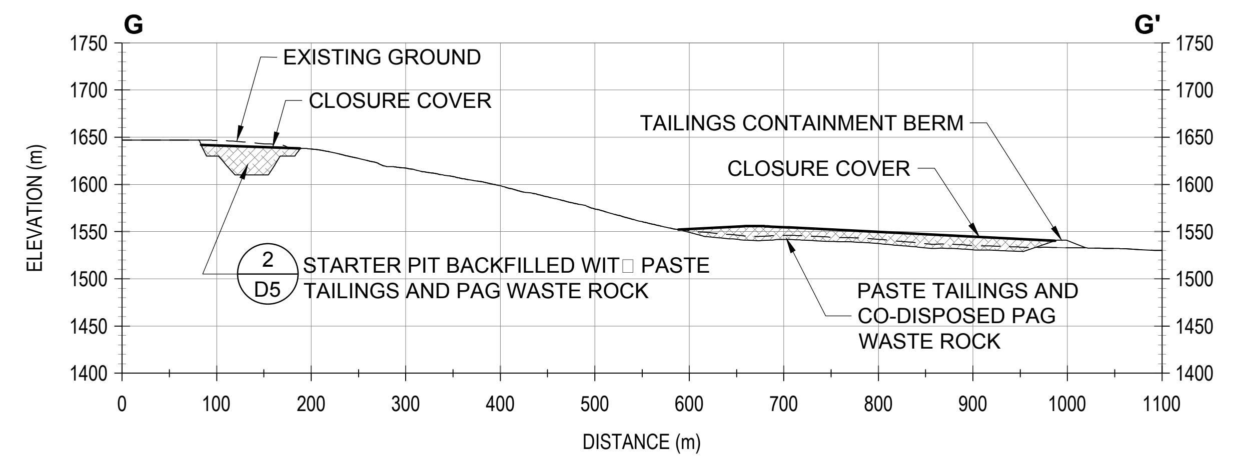



**SECTION** G / --
1:8000

LEGEND

- MAJOR CONTOUR (5 m INTERVAL)
- MINOR CONTOUR (1 m INTERVAL)
- DITC□
- BERM
- CATC□MENT BOUNDARY
- CLEAN SURFACE WATER
- CONTACT WATER
- EXISTING CREEKS AND STREAMS

0 500 m
Scale: 1: 8 000

NOTES:

1. AS REQUESTED BY CLIENT. INFORMATION AND DATA UNCHANGED FROM FEBRUARY 9, 2011. UPDATED TO CURRENT CLIENT LOGO AND TETRA TECH FORMAT ON JULY 14, 2017.

STATUS
ISSUED FOR REVIEW

CLIENT
DESERT STAR RESOURCES

TETRA TECH

KUTCHO PREFEASIBILITY STUDY
MINE WASTE AND WATER MANAGEMENT

INITIAL YEARS AFTER MINE CLOSURE

| PROJECT NO. | DWN | CKD | REV |
|---|---|---|---|
| E14101116 | EL | GZ | 0 |
| OFFICE | DATE | | |
| EDMONTON | FEBRUARY 9, 2011 | | |

Q:\Edmonton\Engineering\E141\Projects\Kutcho Temp\E14101116_FIG D4_R0.dwg [FIGURE D4] July 14, 2017 - 10:39:21 am (BY: LEE, ELVIN)

### 18.11.9 Waste Rock Management Plan

Waste rock from the mine operation includes both PAG and non-PAG waste rock. The non-PAG waste rock herein also includes a small portion of the overburden materials from the starter pit excavation. The total tonnage of the overburden materials from the starter pit is not implicitly estimated from the block model at this stage. The PAG and non-PAG waste rock will be separated at the sources during mine operation. A portion of waste rock generated in the underground mines will remain in underground as backfill. The remaining waste rock will be hauled to the surface. Table 18-6 summarizes the waste rock tonnage from the starter pit and underground mines over the LOM.

**Table 18-6: Mine Operations Waste Rock Production**

| Production Year | Starter Pit (kt) | | Underground Mines (kt) | | | |
|---|---|---|---|---|---|---|
| | | | Remaining in Underground | | To Surface | |
| | non-PAG | PAG | non-PAG | PAG | non-PAG | PAG |
| -1 | 837 | 695 | | | 296 | 1 |
| 1 | 386 | 829 | 49 | | 242 | 1 |
| 2 | | | 70 | 5 | 187 | 6 |
| 3 | | | 108 | 14 | 36 | |
| 4 | | | 106 | 31 | | |
| 5 | | | 135 | 18 | | |
| 6 | | | 63 | 3 | | |
| 7 | | | 87 | 32 | | |
| 8 | | | 71 | 9 | | |
| 9 | | | 75 | 8 | | |
| 10 | | | 45 | 8 | | |
| 11 | | | 24 | | | |
| 12 | | | 11 | | | |
| **Total** | **1,224** | **1,524** | **845** | **128** | **761** | **8** |

Source: JDS (2017).

A portion of the on-surface non-PAG waste rock will be directly used as site construction materials during the initial site construction stage. The remaining non-PAG waste rock will be temporarily stored in a stockpile and used as site construction materials during the late stage of mine operation and at mine closure and as underground mine backfill during the late stage of mine operation.

The on-surface PAG waste rock disposal will consist of three methods: 1) co-disposal with paste tailings in the on-land paste tailings storage facility, 2) co-disposal with paste tailings as backfill in the mined-out starter pit, and 3) hauled back to underground mines as backfill.

The PAG waste rock generated from the starter pit in year -1 and early year 1 will be temporarily stored in the PAG waste rock storage facility. The waste rock in the PAG waste rock storage facility will be later moved to the mined-out starter pit and hauled down to underground mines as backfill. Table 18-7 summarizes the overall waste rock management plan.

### Table 18-7: Waste Rock Management Plan

| Production Year | On-Surface non-PAG Waste Rock (kt) | | | On-Surface PAG Waste Rock (kt) | | | |
|---|---|---|---|---|---|---|---|
| | To Temporary non-PAG Waste Rock Stockpile | As Site Construction Materials | Hauled Underground as Backfill | To Temporary PAG Waste Rock Storage Facility | To On-Land Tailings Storage Facility | To Mined-out Starter Pit | Hauled Undergournd as Backfill |
| -1 | 215 | 919 | | 696 | | | |
| 1 | 232 | 396 | | 207 | 623 | | |
| 2 | 187 | | | -566 | | 500 | 73 |
| 3 | 36 | | | -319 | | 300 | 19 |
| 4 | -110 | 110 | | -18 | | | 18 |
| 5 | | | | | | | |
| 6 | -59 | | 59 | | | | |
| 7 | -42 | | 42 | | | | |
| 8 | -57 | | 57 | | | | |
| 9 | -53 | | 53 | | | | |
| 10 | -68 | | 68 | | | | |
| 11 | -67 | | 67 | | | | |
| 12 | -214 | 185 | 29 | | | | |
| **Total** | **-** | **1,610** | **375** | **-** | **623** | **800** | **110** |

Source: JDS (2017).

The dry density of the placed waste rock was assumed to be 2.0 $t/m^3$ for the mine waste management in this study. The overall dry density could be higher when partial or entire voids in the waste rock are filled with co-disposed paste tailings, which introduces some conservatism into this study.

### 18.11.10 Water Management Plan

The water management plan during the mine operation includes the following components:

- Constructing diversion ditches and berms around the proposed mine waste management facilities to divert the clean surface runoff water from the undisturbed ground above the mine facilities to minimize the overall quantity of the contact water;
- Constructing a water collection pond dam to store contact water from the mine waste facility areas;
- Regularly pumping the contact water from the water collection pond to a water treatment plant;
- Directly pumping contact water from underground mine and other mine site areas to the water treatment plant for treatment;
- Reclaiming a portion of the treated or untreated site contact water and process water for ore processing; and
- Discharging the treated water to the receiving environment after the water quality meets the discharge criteria.

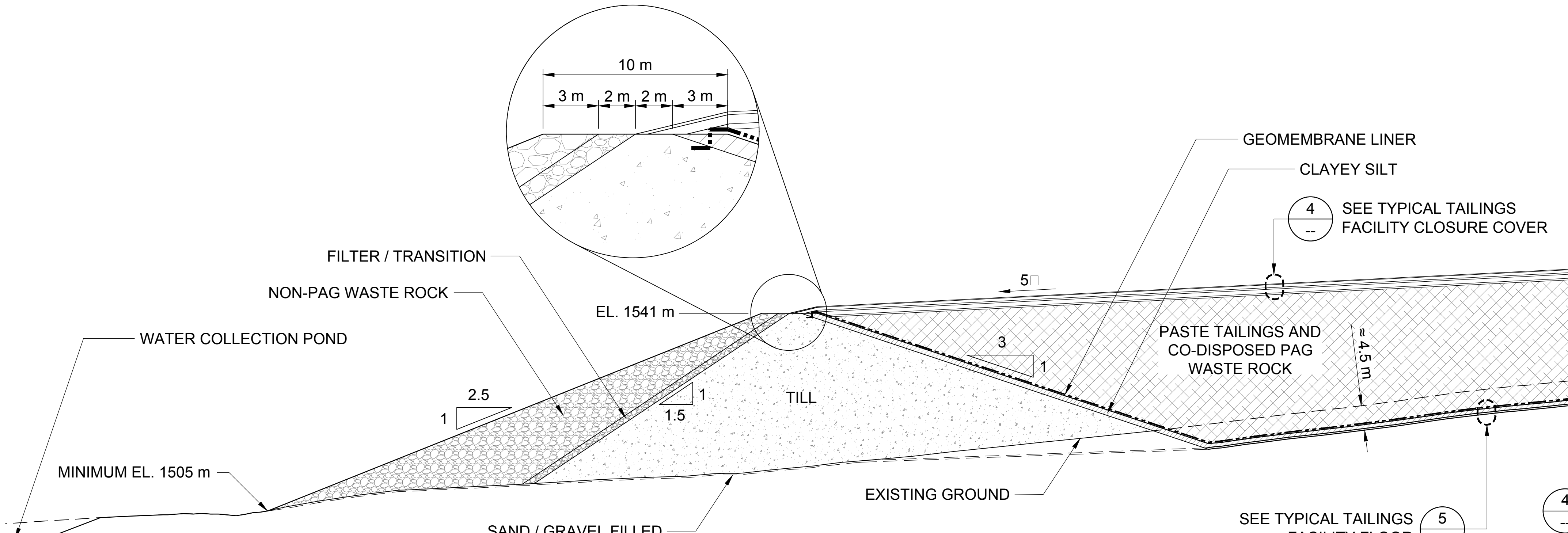
After mine closure, the water from the covered on-land paste tailings storage facility and starter pit areas will be collected in the water collection pond and then pumped to the water treatment plant for treatment. The water can be discharged directly to the environment when the water quality meets the discharge criteria. The water collection pond can be decommissioned and the dam be breached after a monitoring period specified in the water licence.

### 18.11.11 Design Overview of Mine Waste and Water Management Facilities

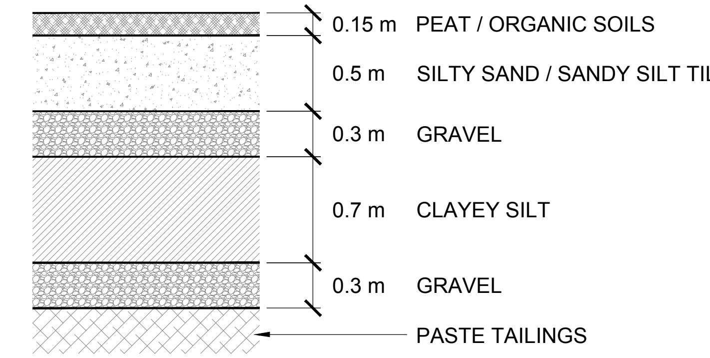
The previous geochemical characterization of the sulphur-containing tailings indicated that the tailings would be net acid generating when oxidized. The drainage from the oxidized tailings would have a low pH value and elevated concentrations of heavy metals.

A consideration of the tailings storage facility design is to control the tailings oxidation by minimizing the influx of oxygen and water influx into the tailings and contain the tailings in a near water-tight containment system. All of these aspects prevent the acid drainage from percolating into the receiving environment. The on-land PAG waste rock will be permanently co-disposed with the paste tailings in the on-land paste tailings storage facility and mined-out starter pit.

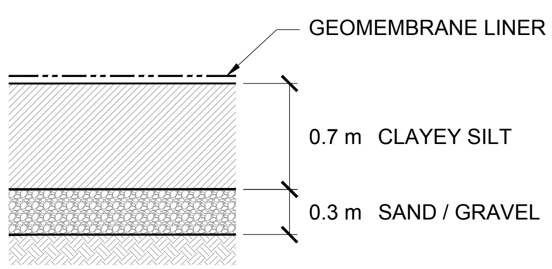
Typical design sections are shown in Figure 18-7 for the on-land paste tailings storage facility and the closure cover for the mined-out starter pit. Typical design sections for the water collection pond dam and diversion ditches/berms are shown in Figure 18-8. The sections were developed in consideration of a number of factors such as control and prevention of mine waste oxidation and acid drainage, structure integrity and stability, constructability, construction material availability, and maximizing the usage of local materials. The main features and considerations of the designs are summarized in the following sections.



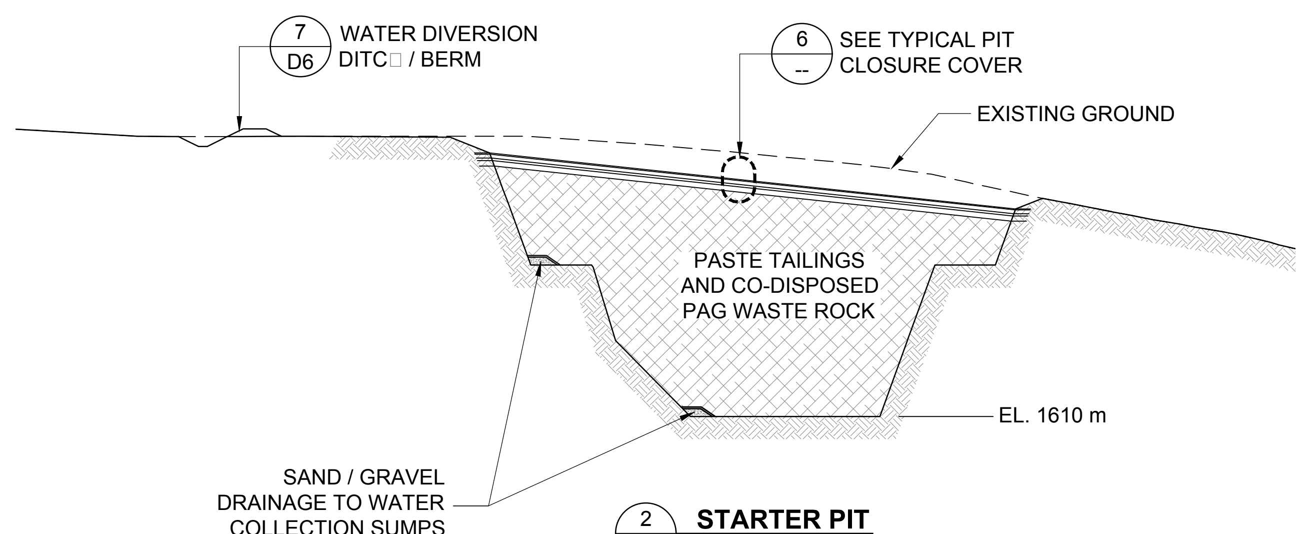

1 / D4 **TYPICAL TAILINGS BERM**
N.T.S.



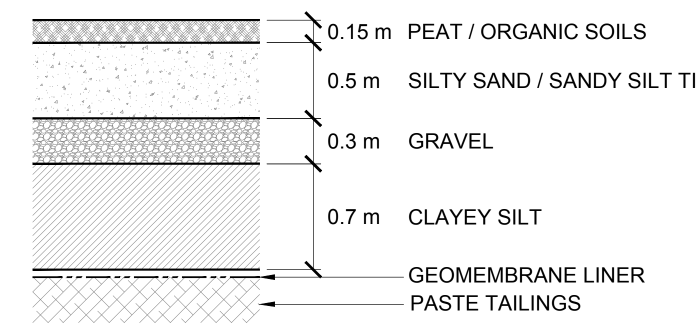

4 / -- **TYPICAL TAILINGS FACILITY CLOSURE COVER**
N.T.S.




5 / -- **TYPICAL TAILINGS FACILITY FLOOR**
N.T.S.




2 / D4 **STARTER PIT**
N.T.S.




6 / -- **TYPICAL PIT CLOSURE COVER**
N.T.S.

NOTES:
1. AS REQUESTED BY CLIENT. INFORMATION AND DATA UNCHANGED FROM FEBRUARY 9, 2011. UPDATED TO CURRENT CLIENT LOGO AND TETRA TECH FORMAT ON JULY 14, 2017.

STATUS
ISSUED FOR REVIEW

CLIENT
DESERT STAR RESOURCES

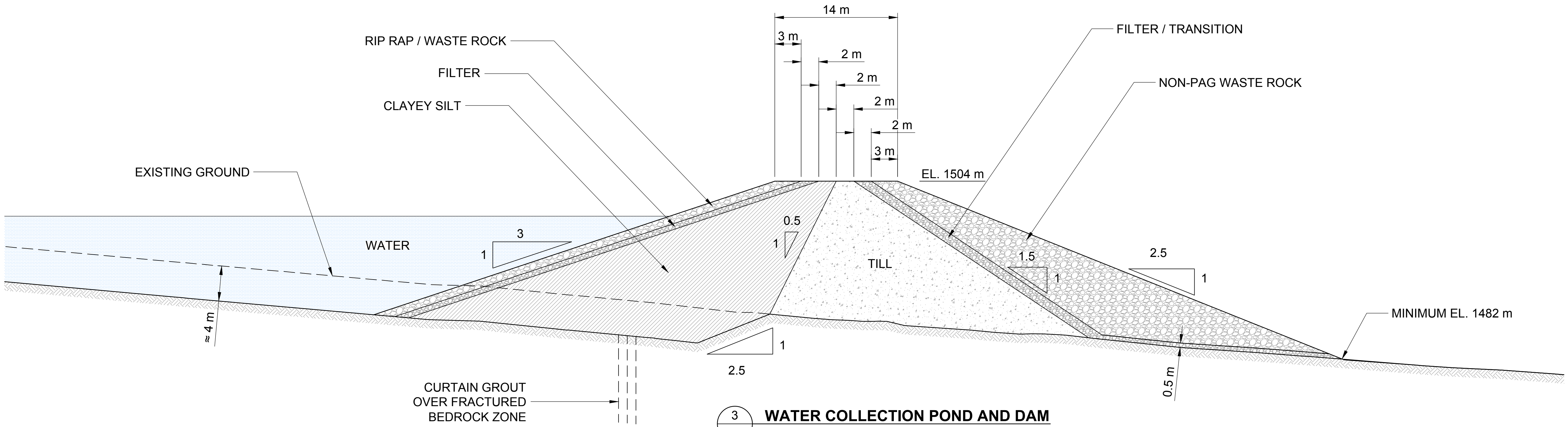
TETRA TECH

KUTCHO PREFEASIBILITY STUDY
MINE WASTE AND WATER MANAGEMENT

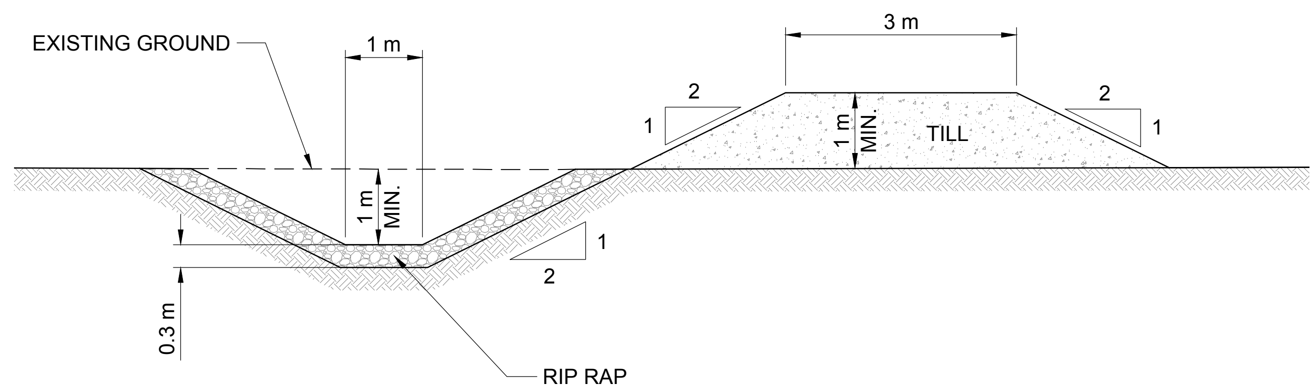
MINE WASTE MANAGEMENT FACILITY
TYPICAL SECTIONS AND DETAILS

| PROJECT NO. | DWN | CKD | REV |
|---|---|---|---|
| E14101116 | EL | GZ | 0 |
| OFFICE | DATE | | |
| EDMONTON | FEBRUARY 9, 2011 | | |

Q:\Edmonton\Engineering\E141\Projects\Kutcho Temp\E14101116_FIG D5-D6_R0.dwg [FIGURE D5] July 14, 2017 - 10:53:21 am (BY: LEE, ELVIN)




3
D4
**WATER COLLECTION POND AND DAM**
N.T.S.




7
D5
**TYPICAL DITCH AND BERM**
N.T.S.

NOTES:
1. AS REQUESTED BY CLIENT. INFORMATION AND DATA UNCHANGED FROM FEBRUARY 9, 2011. UPDATED TO CURRENT CLIENT LOGO AND TETRA TECH FORMAT ON JULY 14, 2017.

STATUS
ISSUED FOR REVIEW

CLIENT
DESERT STAR RESOURCES

TETRA TECH

KUTCHO PREFEASIBILITY STUDY
MINE WASTE AND WATER MANAGEMENT

WATER MANAGEMENT STRUCTURE
TYPICAL SECTIONS

| PROJECT NO. | DWN | CKD | REV |
|---|---|---|---|
| E14101116 | EL | GZ | 0 |

| OFFICE | DATE |
|---|---|
| EDMONTON | FEBRUARY 9, 2011 |

Q:\Edmonton\Engineering\E141\Projects\Kutcho Temp\E14101116_FIG D5-D6_R0.dwg [FIGURE D6] July 14, 2017 - 10:38:49 am (BY: LEE, ELVIN)

## 18.11.12 Design Overview of On-land Paste Tailings Storage Facility

The proposed on-land paste tailings storage facility consists of a containment berm, a bottom liner system, and a top closure cover system. The containment berm would be a zoned earth and rock fill structure with an upstream low-permeability clay silt zone covered with a geomembrane liner to contain the paste tailings. The clay silt zone serves as a bedding layer for the liner and also provides a secondary containment. The central till fill provides support of the upstream clay silt zone and liner. The non-PAG waste rock downstream shell provides slope erosion protection and overall stability of the berm. A transition/filter zone between the waste rock and the till fill and natural till zones protects the till zones against potential internal instability. The final containment berm has a crest elevation of 1541 m, a maximum berm height of approximately 36 m, a crest width of 10 m, a 2.5H:1V downstream slope and a 3H:1V upstream slope.

The ground below the footprint of the bottom paste tailings in the facility will be excavated to a depth of 4.5 m to increase the storage capacity of the tailings facility and obtain till fill for construction. The bottom liner system over the excavated surface consists of a geomembrane liner over a low-permeability clay silt layer (0.7 m thick) and a basal drainage layer (0.3 m thick). The geomembrane liner and clay silt layer in the bottom liner system will tie into the upstream liner system of the containment berm. The basal drainage layer will drain any groundwater accumulated beneath the liner system to prevent potential pore water pressure built-up beneath the liner system. The high pore water pressure beneath the liner would result in a floating liner during early mine operation and potential upward seepage into the tailings. The basal drainage layer will connect to several sand/gravel-filled drainage channels excavated beneath the containment berm to drain the groundwater into the downstream water collection pond.

A multiple-layer soil cover will be placed over the top of the final paste tailings at mine closure to minimize the water infiltration into the facility and provide an oxygen diffusion barrier to minimize the influx of oxygen. The cover system consists of a top native soil layer (0.65 m thick), a top capillary barrier layer (0.3 m thick), a compacted low-permeability clay silt layer (0.7 m thick), and a bottom capillary barrier layer (0.3 m thick) over the paste tailings. The top native soil layer includes a thin veneer of peat or organic soils (0.15 m thick) and a native silty sand or sandy silt till layer (0.5 m thick). These soils provide a physical protection of the underlying layers and facilitate establishment of a sustainable vegetation cover. The native soil cover also provides water storage capacity to prevent drying of the low-permeability soil layer during dry seasons. The top capillary barrier layer provides a hydraulic discontinuity between the top native soil cover and the underlying low-permeability layer to minimize upward movement of moisture in the low-permeability layer during summer drying and early winter freezing seasons. In addition, this coarse-grained soil layer provides physical protection to the low-permeability layer against potential damage from root penetration and burrowing animals. This layer also serves as a drainage layer to drain any infiltrated water away from the top surface of the low-permeability layer.

The low-permeability layer is a key component in the cover system to control and prevent tailings oxidation after mine closure. Research and case histories have indicated that an effective barrier to oxygen diffusion will result if the degree of saturation of a soil layer can be maintained greater than approximately 85 to 90% (MEND 2001). Therefore, the key design objective of the low-permeability layer is to maintain a high degree of saturation under all conditions. This objective appears achievable for the current cover design under the meteorological, hydrological, hydrogeological, and ground conditions of this project site.

The bottom capillary barrier layer provides a hydraulic discontinuity between the low-permeability layer and the underlying paste tailings to minimize loss of moisture in the low-permeability layer by downward moisture movement and to control upward movement of process water and oxidation products in the tailings.

It is expected that the moisture content in the majority of the paste tailings placed in the on-land paste tailings storage facility will be maintained in a nearly saturated condition over the long-term because of the lined sides and bottom of the facility, the fine-grained nature and intrinsic low permeability of the paste tailings, the cover design that limits moisture loss of the tailings, and a gentle surface slope.

The PAG waste rock co-disposed with the paste tailings in the facility will be placed in the lined basin in year 1 and completely encapsulated with saturated paste tailings placed during the following years of mine operation. It is expected that the voids in the waste rock will be filled either the saturated paste tailings or the process water from the tailings to result in a saturated condition.

The design storage capacity of the tailing facility is approximately 2.04 M $m^3$ or 4.08 Mt of dry mine waste (including 3.45 Mt of tailings and 0.63 Mt of PAG waste rock). It was assumed that the paste tailings would be discharged at a single point that is approximately 300 m away from the upstream crest of the final containment berm. The final elevation of the discharge point is 1,556 m, which is 15 m higher than the final crest elevation of the tailings containment berm. The paste tailings would form a cone shape sloping down from the proposed discharge point.

### 18.11.13 Design Overview of Mined Out Starter Pit Backfilled with Mine Waste

The focus of the closure cover design for the mined-out starter pit was to apply best engineering measures to minimize or prevent surface infiltration and ingress of oxygen into the mine waste to reduce the risk of mine waste oxidation and generation of acid drainage. It would be difficult, if not impractical, to place a lined system over the rugged, steep side walls and bottom of the mined-out starter pit to contain the co-disposed paste tailings and PAG waste rock.

A permanent multiple-layer lined cover is proposed to be placed over the top of the final paste tailings disposed in the mined-out starter pit. This cover will minimize the water infiltration into the mine waste and provide an oxygen diffusion barrier to minimize the influx of oxygen. The cover system consists of a top native soil layer (0.65 m) followed by a capillary barrier layer of 0.3 m, a low-permeability clay silt layer of 0.7 m, and a geomembrane liner installed over the final paste tailings. The top native soil layer includes a thin veneer of peat or organic soils (0.15 m) and a native silty sand or sandy silt till layer of 0.5 m. These soils provide physical protection of the underlying layers and facilitate establishment of a sustainable vegetation cover. The native soil cover also provides some water storage capacity to prevent drying of the low-permeability soil layer during dry seasons. The capillary barrier layer provides a hydraulic discontinuity between the top native soil cover and the underlying low-permeability layer to minimize upward movement of moisture in the low-permeability layer during summer drying and early winter freezing seasons. In addition, this coarse-grained soil layer provides physical protection to the low-permeability layer against potential damage from root penetration and burrowing animals. This layer can also serve as a drainage layer to drain any infiltrated water away from the top surface of the low-permeability layer. The geomembrane liner together with the low-permeability layer, which serves as a secondary defence in potential areas where locally damaged liner patches exist, will greatly minimize and even prevent tailings oxidation after mine closure.

Site hydrogeological investigations indicated that the sustained groundwater table in the starter pit area would be below the pit bottom. This means that the mine waste placed in the pit could be partially drained over the long term. On the other hand, this suggests that the risk of constant groundwater flushing through the mine waste would be low; therefore, the risk of potential release of acid drainage into the groundwater system is also low in case some of the mine waste is oxidized.

There is a possibility that an interim perched shallow groundwater table may be present during wet seasons in the overburden and shallow bedrock zones. The groundwater may flow through the south (up-slope side) pit walls to contact the mine waste and potentially produce acid drainage. To reduce this risk, a water collection and drainage material is proposed to be placed over the south pit wall bench and bottom corners to collect any groundwater and convey the contact water to an outside water collection system for treatment before releasing to the environment.

Previous site investigations indicated that the overburden thickness around the pit areas, especially areas close to the south wall of the pit, were more than 4 m. To limit the risk of the interim groundwater in the shallow overburden soils flowing into the mine waste, it is designed that the final pit closure cover be several metres lower than the original ground surface to intercept any seepage water through the shallow overburden zone.

The total volume of the starter pit below the original ground surface is approximately 1.1 M $m^3$. The required storage capacity for the mine waste is 0.9 M $m^3$ or 1.8 M tonnes of dry mine waste (including 1.0 M tonnes of tailings and 0.8 M tonnes of PAG waste rock). The closure cover fill volume is estimated to be approximately 0.1 M $m^3$. Therefore, the overall final closure cover surface would be about two metres lower than the original ground surface.

### 18.11.14 Design Overview of Water Management Structures

The proposed water collection pond dam is a zoned earth and rock fill structure with an upstream low-permeability clay silt zone to control the seepage through the dam. The central till fill zone provides support of the upstream clay silt zone. The downstream non-PAG waste rock shell provides slope erosion protection and overall stability. The upstream riprap zone protects the upstream slope against erosion from wave actions. The transition/filter zones protect the till and clay silt zones against potential internal instability. The dam has a crest elevation of 1504 m, a maximum dam height of approximately 22 m, a crest with of 14 m, a 2.5H:1V downstream slope and a 3H:1V upstream slope.

The ground will be excavated under the portion of the dam and under the proposed water collection pond to a depth of approximately 4 m as shown in Figure 18-8. This will increase the water storage capacity of the water collection pond, reduce the seepage through the overburden zone in the dam foundation, and obtain sufficient till fill materials for construction. The shallow bedrock below the clay silt zone could be highly fractured. A zone of curtain grout is proposed in the bedrock below the clay silt zone to reduce the potential seepage through the bedrock foundation.

A seepage collection sump located immediately downstream of the water collection pond is proposed to collect minor seepage through the dam. The water in the seepage collection sump will be regularly pumped back to the water collection pond.

Water diversion ditches/berms are proposed to divert the clean runoff water from the undisturbed areas and to convey the contact water into the water collection pond. The conceptual design section for the ditches/berms is shown in Figure 18-8. A layer of riprap material will be placed over the excavated surface of the ditch to control erosion. A preliminary hydraulic calculation indicates that the maximum water level in the ditches would be below 0.5 m under an extreme rainfall event with a 1 in 100 return period because of relatively small catchment areas.

### 18.11.15 Design Overview of Other Mine Waste Facilities

The temporary PAG waste rock storage facility has design average side slopes of 2H: 1V, a crest elevation of 1,653.5 m, and a height ranging from 14 to 34 m. The storage capacity of the temporary PAG waste rock storage facility is approximately 0.46 $Mm^3$. The PAG waste rock will be placed in the dump in years -1 and 1 and re-handled back to the mined-out starter pit in years 2 and 3. During this relatively short period, a small fraction of the sulphur-containing PAG waste rock could be oxidized. Therefore, the contact water will be collected in the water collection pond for treatment before releasing to the receiving environment.

The temporary non-PAG waste rock stockpile has design average side slopes of 2H: 1V and a maximum crest elevation of 1,572 m. The stockpile has a maximum height of approximately 22 m and a storage capacity of 0.36 $Mm^3$ in year 3. The height and storage volume will be gradually reduced over the LOM when some non-PAG waste rock is used as construction materials and underground mine backfill. The contact water from the non-PAG waste rock stockpile will be collected in the water collection pond for treatment before releasing to the receiving environment.

### 18.11.16 Stability Analyses

The slope stability design criteria for earth structures are summarized in Table 18-8.

**Table 18-8 Slope Stability Design Criteria for Earth Structures**

| Loading Conditions | Minimum Factor of Safety | Slope |
|---|---|---|
| Static loading, end of construction (before reservoir filling, if present) | 1.3 | Downstream and Upstream |
| Static loading, normal operation or normal reservoir level | 1.5 | Downstream and Upstream |
| Full or partial rapid drawdown | 1.2 or 1.3 | Upstream |
| Earthquake, normal operation or normal reservoir level | 1.1 | Downstream and Upstream |

Source: EBA (2010a,b).

The Project site is situated in an area of low seismic risk. The peak ground acceleration (PGA) for the area was estimated from the 2005 National Building Code of Canada seismic hazard website (http://earthquakescanada.nrcan.gc.ca). The estimated PGA was 0.035 g for a 5% in 50 year probability of exceedance (0.001 per annum or 1 in 1,000 year return) and 0.059 g for a 2% in

50 year probability of exceedance (0.000404 per annum or 1 in 2,475 year return) for the Kutcho area. A design PGA of 0.06 g was adopted for the earth structure designs.

Limited laboratory shear strength tests on the native sandy silt, silty sand, and clay silt samples were conducted and reported in Golder (1984). This information together with past experience on similar soils was applied to adopt typical material parameters for slope stability evaluations in this study. The material parameters are presented in Table 18-9.

**Table 18-9 Material Properties Used in Stability Analyses**

| Material | Unit Weight ($kN/m^3$) | Strength Parameter | Pore Pressure Conditions |
|---|---|---|---|
| Waste Rock or Riprap | 19.6 | φ′=38°, c′=0 kPa | Phreatic surface |
| Filter or Transition | 20 | φ′=35°, c′=0 kPa | Phreatic surface |
| Clay Silt Fill | 20 | φ′=30°, c′=0 kPa | Short-term: $r_u$=0.3<br>Long-term: Phreatic surface |
| Till Fill | 20 | φ′=34°, c′=0 kPa | Short-term: $r_u$=0.1<br>Long-term: Phreatic surface |
| Overburden Till | 20 | φ′=32°, c′=0 kPa | Short-term: $B_{bar}$=0.2<br>Long-term: Phreatic surface |
| Bedrock | N/A | Assumed impenetrable | N/A |

Source: EBA (2010a,b).

Limit equilibrium slope stability analyses were conducted for the water collection pond dam using a commercial computer program, SLOPE/W, GeoStudio 2007. The design side slopes for other water and mine waste management facilities in this study were determined based on the findings from the stability analyses for the water collection pond dam and past engineering designs for similar structures. The design slope is 2.5H: 1V for the downstream slope and 3H: 1V for the upstream slope for both the water collection pond dam and the on-land paste tailings containment berm.

The design average side slope is 2H: 1V for both the temporary non-PAG waste rock stockpile and PAG waste rock storage facility. These design slopes are considered to be reasonably conservative and are expected to meet the design criteria.

Detailed slope stability analyses with known soil properties of both the construction materials and foundation soils will be required to finalize and optimize the geometries of the earth structures in the next stage of design.

## 18.11.17 Seepage Analyses

Preliminary, steady state finite element seepage analyses (SEEP/W; GeoStudio 2007) were carried out for the proposed water collection pond dam in support of this prefeasibility level design. The analyses provide an estimate of the seepage rate through the dam and foundation that would require interception, collection, and pump back to the water collection pond. Two typical sections through the dam were analyzed, one through the highest dam location and another through the intermediate dam height location.

Seepage analyses are sensitive to the assumed hydraulic conductivities of the various materials. These parameters are inherently variable and can be difficult to define accurately. At this prefeasibility stage there is limited site-specific data on which to evaluate these parameters. Engineering judgment was therefore required in selecting the parameters that are presented in Table 18-10.

**Table 18-10: Hydraulic Conductivity Values Adopted in Seepage Analyses**

| Material | Hydraulic Conductivity (m/s) |
|---|---|
| Waste Rock or Riprap | 1E-03 |
| Filter or Transition | 1E-03 |
| Clay Silt Fill | 8E-09 |
| Till Fill | 2E-07 |
| Overburden Till | 5E-07 |
| Fractured Bedrock | 2E-07 |
| Competent Bedrock | 1E-08 |
| Curtain Grouted Fractured Bedrock | 1E-08 |

Source: EBA (2010a,b).

The estimated flow rate through the dam and foundation based on the two sections evaluated was 15 $m^3$/day under the maximum design water level of 1,500.3 m for a mean precipitation year and 20 $m^3$/day under the maximum projected water level of 1,503 m for a 1 in 100 wet precipitation year. The pond water level after the spring freshet will be maintained at a low level during mine operation; therefore, the seepage rate during the post-freshet period would be much less. The majority of the seepage can be intercepted by the downstream seepage collection sump and pumped back to the water collection pond.

## 18.11.18 Other Design Evaluations and Considerations

The paste tailings will consolidate after placement. The consolidation properties of the tailings are unknown at this stage of study. Depending on the actual tailings properties, drainage boundary conditions, and overall tailings thickness, partial or even full consolidation may occur during the mine operation. It is expected that the degree of the tailings consolidation before mine closure in the on-land tailings facility will be high due to a relatively thin (2 to 3 m) layer of the paste tailings placed annually over a large area.

On the contrary, the average thickness of the paste tailings to be placed in the mined-out starter pit would be 10 to 15 m. Depending on the actual tailings properties, the average degree of consolidation of the tailings in the pit immediately after fully backfilled might be relatively low.

The closure cover design and construction should consider the tailings consolidation status before the final closure cover is placed over the tailings since consolidation-induced settlement may be detrimental to the integrity of the closure cover system. Further evaluations using site specific tailings parameters should be conducted in the next stage of study to optimize the tailings placement schedule, finalize the closure cover design, and determine the best cover construction schedule.

The conceptual designs developed in this study are considered reasonable at this stage of design. Additional engineering evaluations should be carried out to assess the long-term effectiveness of the mine waste containment structures and closure covers and optimize the designs using tested site-specific material properties in the next stage of design.

### 18.11.19 Water Balance and Collection Pond

A water balance of the waste disposal areas and catchment areas was carried out to determine required maximum design water levels in the water collection pond.

The maximum design water level in the water collection pond depends on the maximum design pumping rate from the water collection pond to the water treatment plant. The higher the pumping rate, the lower the water level in the pond would be; however, a higher water treatment capacity requires a higher initial capital cost for the water treatment plant. On the other hand, the lower pond water level means a lower water collection pond dam and therefore a lower dam construction cost.

The maximum pumping rate of the water from the water collection pond to the water treatment plant and the resulting dam height were adopted based on considerations of increasing the overall water management system reliability and flexibility while lowering the combined cost of the dam construction cost and the capital cost of the water treatment plant. SDE (2011) provided capital cost estimates for several water treatment plant options. The proposed maximum pumping rate from the water collection pond to the water treatment plant is 67 L/s (or 5,800 $m^3$/day) in this study.

The inflows and outflows of the water collection pond are as follows:

Inflows:

- Direct precipitation on the pond surface;
- Runoff water from the inside catchment area of the water collection pond;
- Water that leaks from the diversion ditches and berms into the catchment area;
- Groundwater intercepted by the water collection pond dam; and
- Seepage water pumped back to the pond from the seepage collection sump.

Outflows:

- Water pumped to the water treatment plant;
- Pond water surface evaporation; and
- Seepage through the dam and foundation.

Water that leaks out of the diversion ditches into the inside catchment was assumed to be 20% of the runoff from the outside diverted catchment area. Similarly, the groundwater that is sourced from the outside diverted catchment area and intercepted by the water collection pond dam was assumed to be 10% of the runoff in the outside diverted catchment area. It was assumed that the seepage water pumped back to the pond from the seepage collection sump would be equal to the seepage through the dam and foundation as determined from the seepage analysis.

It was assumed that pumping (at the maximum rate) from the water collection pond to the water treatment plant will start on May 16 each year and continue until the pond water elevation level is lowered down to approximately 1,492 m. After this water level is reached, continuous or intermitted pumping at the maximum pumping rate or lower rate may be required to maintain the pond level in a relatively low level. By October 15, the pond water elevation should be lower than 1,492 m. It was assumed that no pumping will occur during the period of October 16 to May 15 each year when the runoff is small and the air temperature is cold.

The water storage capacity of the water collection pond is 264,800, 125,800, 35,100, and 2,100 $m^3$ for a water elevation of 1,503, 1,499, 1,495, and 1,491 m, respectively.

Three scenarios were considered for the water balance model: 1) for a mean precipitation year, 2) for a mean precipitation year plus an extreme flood event with a wet 1 in 100 return period, and 3) for a wet precipitation year with a 1 in 100 return period. Table 18-11 summarizes the estimated maximum pond water levels under these conditions and the associated dam freeboards above the water levels.

**Table 18-11: Estimated Maximum Pond Water Levels and Dam Freeboards under Design Conditions**

| Scenario | Estimated Maximum Water Elevation in Water Collection Pond (m) | Dam Freeboard above the Estimated Maximum Water Elevation (m) |
|---|---|---|
| A mean precipitation year | 1,500.3 | 3.7 |
| A mean precipitation year plus an extreme flood event with a wet 1 in 100 return period | 1,501.0 | 3.0 |
| A wet precipitation year with a 1 in 100 return period | 1,503.0 | 1.0 |

Source: EBA (2010a,b).

### 18.11.20 Overall Mine Site Water Balance

The major inflows and outflows of the overall mine site include the following:

Inflows:

- Water pumped from the water collection pond to the water treatment plant, which is equal to the net water inflow from the mine waste and water management facility areas;
- Surface contact water collected in the process plant and camp areas when the water quality does not meet direct discharge criteria;
- Groundwater seepage into underground mines to be pumped to the water treatment plant for treatment;

Outflows:

- Net water locked-in the tailings and concentrates, which is equal to the make-up water required for ore processing after the majority of the process water is re-cycled and re-used;
- Other mine site water usage such as water for site construction, and
- Treated water discharge to receiving environment after water quality meets the site discharge criteria.

The groundwater seepage rates into the underground Main and Esso mines were not estimated in this study. In the 2007 Kutcho Project Prefeasibility Study, Wardrop (2007) estimated that the groundwater seepage rate was 12 L/s into the originally proposed open pit in the Main deposit and 25 L/s into the Esso underground mine. In the current prefeasibility study, these values were adopted as preliminary values of groundwater seepage rates into the underground Main and Esso mines. These values should be verified in the next stage of study.

The average annual water volumes of the mine site inflows and outflows under a mean precipitation year are summarized in Table 18-12.

**Table 18-12: Estimated Average Annual Water Volumes for Mine Site Inflows and Outflows**

| Water Sources | | Average Annual Water Volume ($Mm^3$) |
|---|---|---|
| **Inflows** | Water pumped from the water collection pond | 0.54 |
| | Surface contact water collected in the process plant and camp areas | 0.13 |
| | Groundwater seepage into underground mines | 1.17 |
| **Outflows** | Net water locked-in the tailings and concentrates (or mill make-up water for ore processing) | 0.16 |
| | Other mine site water usage | 0.05 |
| | Excess water discharge to receiving environment after treatment | 1.63 |

Source: EBA (2010a,b).

## 18.11.21 Construction Schedule and Sequence

The water collection pond dam and ditches/berms should be constructed during the initial stage of construction before the start of mine operation.

The on-land paste tailings storage facility can be constructed in stages to reduce initial construction requirements and associated costs. The construction requirements during the initial stage before mine operation are as follows:

- Excavate several drainage channels through the lower original ground area beneath the proposed containment berm to drain the water from the proposed excavation area below the bottom paste tailings footprint to the downstream water collection pond;
- Backfill the channels with sand/gravel drainage material;
- Excavate the original ground below the proposed bottom paste tailings footprint as designed;
- Stockpile the surficial peat and organic soils to be used as final surficial fill for the tailings facility closure cover;
- Place and compact the excavated till and extremely weathered bedrock materials in the till fill zone in the containment berm;
- Place and compact the compacted sand/gravel drainage layer over the excavated surface;
- Place and compact the bottom clay silt layer over the drainage layer;
- Place and compact the clay silt layer on the upstream side of the containment berm to an elevation of 1,532 m or higher;

- Install the geomembrane liner over the compacted clay silt surface;
- Place filter/transition material downstream of the till fill zone; and
- Place non-PAG waste rock in the waste rock zone.

The remaining portion of the containment berm can be constructed in Stage 2 in year 1 to year 3. The tasks mainly include construction of the remaining zones of the various fill and installation of the remaining geomembrane liner over the upstream face.

A water collection and drainage material is proposed to be placed over the south pit wall bench and bottom corners to collect any groundwater and convey the contact water to an outside water collection system for treatment before releasing to the environment. The cross-sectional area of the drainage materials at each location should be approximately 2 $m^2$. The material should be placed before the pit is backfilled with the mine waste. A layer of 1 m thick clay silt can be placed over the drainage material to separate the mine waste from the drainage material.

The closure cover over the mined-out starter pit will be placed after the pit is backfilled with paste tailings and PAG waste rock after the paste tailings are consolidated. Additional excavation along the pit rim is required to key the geomembrane liner in the ground at least 2 m beyond the backfilled mine waste.

The closure cover over the final paste tailings in the on-land tailings facility will be placed after the end of mine operation and paste tailings are consolidated.

### 18.11.22 Construction Materials

Five main construction material types are required for the construction of the mine waste and water management facilities, including:

- Till fill;
- Clay silt fill;
- Processed granular (sand and gravel) fill for filters, transition, riprap and drainage zones;
- non-PAG waste rock; and
- Geomembrane liners.

The till fill is general fill from the following sources:

- Excavation of the ground below the bottom tailings footprint in the on-land tailings facility;
- Excavation of the ground below the water collection pond and the upstream side of the water collection pond dam; and
- Overburden excavation in the starter pit.

The majority of the till fill will be sandy silt or silty sand till with various amounts of sand and gravel. The extremely weathered bedrock can also be treated as till fill. Processing of the till fill is not required.

It is anticipated that the required clay silt fill will be derived from the thick till deposits in a borrow area north of Andrea Creek. The proposed location of the borrow area is shown in Figure 18-3. Based on previous investigations carried out by Golder in 1984 and AMEC in 2007, it is anticipated that the till borrow material will consist of relatively shallow (0.3 to 1.1 m) sandy to gravelly silt till over the thick gravelly, clay silt or silty clay with a thickness of up to more than 30 m. Based on limited laboratory tests on samples from the clay silt or silty clay materials, the materials have a fines (<0.076 mm) content ranging from 45 to 64%, a clay (<0.0002 mm) content ranging from 18% to 28%, and a hydraulic conductivity of 8E-09 m/s. The silty clay was a low to medium plasticity clay with a plastic limit of 14% and a liquid limit of 30 to 42% based on two tests on the samples from the two test pits in the area.

Select clay silt or silty clay materials from the thick clay silt or silty clay layer in the proposed borrow area would be used as clay silt materials for construction of the clay silt zones in the water collection pond dam, on-land tailings facility including the closure cover, and the closure cover for the starter pit.

It is anticipated that the granular fills for filter, transition, riprap, and drainage zones would be processed from non-PAG waste rock. The crushed and processed non-PAG waste rock should be hard, durable rock waste rock.

The non-PAG waste rock would be a run-of-mine material from the starter pit and underground mines. The fill should be angular and shall be derived from hard, durable, non-acid generating rock.

A geomembrane liner is proposed as a seepage and oxygen barrier over the upstream surface of the tailings containment berm, the bottom of the on-land paste tailings storage facility, and the final paste tailings in the mined-out starter pit. Generally, three types of geomembrane liners are commercially available for this application. They are HDPE, polypropylene, or bituminous geomembrane liners; each has its advantages and disadvantages. A 60 mil HDPE liner is adopted in this study. The final selection of the liner type will be made during the final stage of design based on design/construction requirements, construction season, chemical resistivity, durability, and other considerations.

### 18.11.23 Construction Material and Burrow Excavation Quantities

Table 18-13 summarizes the estimated construction material and excavation quantities for major mine waste and water management facilities.

The till from the excavations in the on-land tailings facility and water collection pond dam areas and the overburden over the starter pit will meet the total till material quantity required for the construction of the facilities. The clay silt will be sourced from the proposed borrow area north of Andrea Creek. It is estimated that an average excavation depth of approximately 6 m in the area would provide the required total clay silt quantity. The non-PAG waste rock from mine operation will meet the required total quantities of the non-PAG waste rock and filter/transition/drainage materials.

### Table 18-13: Construction Material and Excavation Quantities

| Structure | Construction Material Quantity | | | | | Excavation ($m^3$) |
|---|---|---|---|---|---|---|
| | Waste Rock/Riprap ($m^3$) | Filter/ Transition/ Drainage Materials ($m^3$) | Till ($m^3$) | Clay Silt ($m^3$) | Geomembrane Liner ($m^2$) | |
| Water Collection Pond Dam | 76,500 | 25,800 | 77,300 | 102,000 | N/A | 150,700 |
| On-land Paste Tailings Containment Berm (Initial Stage Construction) | 80,400 | 48,200 | 607,200 | 99,500 | 136,200 | 483,400 |
| On-land Paste Tailings Containment Berm (Stage 2) | 176,000 | 21,900 | 0 | 22,800 | 24,000 | 0 |
| Closure Cover of On-Land Paste Tailings Storage Facility | N/A | 92,700 | 100,500 | 108,200 | N/A | N/A |
| Closure Cover of Mined-out Starter Pit | N/A | 16,500 | 35,800 | 38,500 | 58,200 | N/A |
| **Total** | **332,900** | **205,100** | **820,800** | **371,000** | **218,400** | **634,100** |

Source: JDS (2017).

## 18.11.24 Mine Waste and Water Management Alternative Assessment

Several alternatives of the mine waste and water management plan were assessed during the early stage of the current study. These included on-land dry-stacked tailings disposal and wet slurry tailings dam options. The alternatives evaluated included:

- Dry stack facility at the same location as the proposed paste facility;
- Slurry tailings retained by a dam across Andrea Creek;
- Slurry tailings retained by a dam across the upper portion of Sumac Creek; and
- Slurry tailings retained by a dam across the headwater channel of Playboy Creek.

The options for the on-land dry-stacked tailings disposal and the slurry tailings dam across the headwater channel of Playboy Creek were not adopted because of high structure construction/operation costs. The other two slurry tailings options were attractive in the long-term disposal of mine waste and cost perspectives; however, both options would destroy sections of potentially fishing-bearing creeks. The slurry tailings option with the dam location across the upper portion of Sumac Creek would destroy a relatively short section of potentially fishing-bearing creek and could be considered in further studies.

# 19 Markets and Contracts

## 19.1 Markets

The anticipated long-term demand for copper and zinc concentrates is not easily determined. For the purpose of this study, it has been assumed that concentrate demand will remain flat over time at rates approximating current spot.

Gold and silver demand is also assumed to remain stable over the Project period, at current spot rates.

The metal price drivers in the world economy are extremely complex and there is not any assurance that expected results will be met.

No direct marketing has been done for the potential Kutcho concentrates and therefore no off-take agreements exist. Based on current industry demands it is envisioned that both the copper and zinc concentrates would be best suited for smelters in Asia, namely, Japan, Korea, or China. There is the potential for the sale of concentrate to Canadian smelters such as Trail, BC and Thompson, Manitoba; however, these options will be reviewed in detail should the Project proceed to the feasibility stage.

## 19.2 Metal Prices and Exchange Rates

Table 19-1 shows metal prices and exchange rates used in the PFS economic model. The base case prices are based on recent prices and exchange rates used in technical reports. The alternate cases represent the upper and lower bounds of recent pricing.

**Table 19-1: Assumed Metal Prices and Exchange Rate**

| Metal | Unit | Base Case | Case 2 | Case 3 |
|---|---|---|---|---|
| Copper | US$/lb Cu | 2.75 | 2.50 | 3.00 |
| Zinc | US$/lb Zn | 1.10 | 1.00 | 1.20 |
| Gold | US$/oz Au | 1,250 | 1,125 | 1,375 |
| Silver | US$/oz Ag | 17.00 | 15.30 | 18.70 |
| Exchange Rate | C$/US$ | 1.33 | 1.38 | 1.29 |

Source: JDS (2017).

## 19.3 Contracts

There are currently no established contracts relating to mining, concentrating, smelting, refining, transportation, handling, sales, hedging, or forward sales. The information in Table 19-2 is based on generally accepted industry terms as well as quotations from service providers and may vary upon actual negotiations.

### Table 19-2: Offsite Concentrate Contract Assumptions

| Potential Contract Item | Units | Assumed Terms |
|---|---|---|
| Concentrate Moisture Content | % | 9.0 |
| Land Transportation to Stewart Port Facility | US$/wmt | 60.99 |
| Port, Refereeing and Handling Costs | US$/dmt | 15.66 |
| Insurance | US$/dmt | 5.00 |
| Ocean Transport Costs (to Far East) | US$/wmt | 30.00 |
| Copper Concentrate Smelting Terms | US$/dmt | 70.00 |
| Copper Concentrate Payables | Cu % | 96.5 (1 unit deduct @ <30% con) |
| | Zn % | 0 |
| | Ag % | 90 (with no deduction) |
| | Au % | 90 (with no deduction) |
| Zinc in Copper Concentrate Penalty | US$/dmt | $2/dmt for each 1% Zinc content is above 4.0% |
| Zinc Concentrate Smelting Terms | US$/dmt | 120.00 |
| Zinc Concentrate Price Participation | | Add US$0.10 to the unit smelting costs for every dollar above $1,742/t Zn.<br>Subtract US$0.08 from the unit smelting cost for every dollar below $1,742/t Zn. |
| Zinc Concentrate Payables | Zn % | 85 (with no deductions) |
| | Cu % | 0 |
| | Ag % | 0 |
| | Au % | 0 |
| Refining Terms | US$/lb Cu | 0.07 |
| | US$/lb Zn | 0.00 |
| | US$/oz Ag | 0.35 |
| | US$/oz Au | 6.00 |
| Forward Sales Pricing | | See Economics Section |

Source: JDS (2017).

# 20 Environmental Considerations

## 20.1 Regulatory Approval Process

### 20.1.1 Overview

The Kutcho Project is subject to the British Columbia Environmental Assessment Act and the Canadian *Environmental Assessment Act*. The former requires that large projects undergo an environmental assessment and obtain an Environmental Assessment Certificate before they can proceed. The latter applies when a federal department or agency is required to make a decision on a proposed project such as issuing a permit, licence or authorization. Projects in BC are subject to cooperation protocols between the BC EAO and the CEAA which results in a “one-window” process for environmental assessment with both organizations participating in a coordinated joint review of the Project.

The environmental assessment process for the Project commenced with the submission of a Project Description Report to the BC EAO in July 2005. The Project was in the environmental review process between 2005 and 2016. Through this period, proponents and regulators worked to adjust the Project design, develop environmental and social baseline information, engage with the public and First Nations, and prepare a draft set of requirements for the ultimate Project Application.

Advancement of the Kutcho Project ceased in 2013 when it was deemed to be non-core to Capstone’s portfolio.

In 2016 Kutcho Copper Corp voluntarily terminated the environmental assessment process, citing no activity or planned activity at the Project in the future. The Project will be required to re-enter the process however much of the information developed during previous reviews can presumably be applied to future project permitting activities.

### 20.1.2 British Columbia Authorizations, Licences, and Permits

In addition to approval under BC EAO and CEAA, the Project will require a number of other provincial and federal authorizations, licences and permits to operate.

Table 20-1 shows a list of the potential BC authorizations, licences, permits, and guidelines that may apply to the Project. This list requirements and guidelines will be reviewed and updated as the Project advances through the environmental assessment and permitting process.

### 20.1.3 Federal Authorizations, Licences, and Permits

In addition, Table 20-2 shows a list of the potential federal authorizations, licences, permits and guidelines that may be required to operate the Project.

The Project will be subject the Metal Mining Effluent Regulations (MMER) enabled by the Fisheries Act. The regulations require the Project to achieve the specified effluent discharge standards, to implement a comprehensive Environmental Effects Monitoring program, and to provide compensation for the harmful alteration of fish habitat.

### 20.1.4 Community Engagement and Consultation Requirements

Previous community engagement programs for the Project have adhered to the guidelines derived from the Environmental Assessment Act and the Act's Public Consultation Policy Regulation (BC 2002).

The Public Consultation Policy Regulation (BC 2002) was used with respect to public consultation during the previous Environmental Assessment process. This sets out guidelines related to the proponent's consultation program, public notice, public comment periods and documents to be available through the BC EAO's Project Information Centre (e-PIC).

Moving forward into a future Environmental Assessment process, the Project will be subject to the same Public Consultation Policy Regulation (BC 2002) which was in place for previous Project iterations in 2005 to 2016.

**Table 20-1: Potential BC Authorizations, Licences and Permits for the Kutcho Project**

| BC Government Permits and Licences | Enabling Legislation |
|---|---|
| Environmental Assessment Certificate | BC Environmental Assessment Act |
| Permit Approving Work System and Reclamation Program (Mine Site – Initial Development) | Mines Act |
| Amendment to Permit Approving Work System and Reclamation Program (Pre-production) | Mines Act |
| Amendment to Permit Approving Work System and Reclamation Program (Bonding) | Mines Act |
| Amendment to Permit Approving Work System and Reclamation Program (Mine Plan - Production) | Mines Act |
| Permit Approving Work System and Reclamation Program (Gravel Pit/Wash Plant/Rock Borrow Pit) | Mines Act |
| Chief Inspector's Permit | Mines Act |
| Notice of Work | Mines Act |
| Water Licence – Notice of Intention (Application) | Water Act |
| Water Licence – Storage and Diversion | Water Act |
| Water Licence – Use | Water Act |
| Water Licence – Construction of fences, screens and fish or game guards across streams for conservation | Water Act |
| Water Licence – Alteration of Stream or Channel | Water Act |
| Authority to Make a Change In and About a Stream – Notification | Water Act / Water Regulation |
| Authority to Make a Change In and About a Stream – Approval to Make a Change | Water Act / Water Regulation |
| Authority to Make a Change In and About a Stream – Terms and Conditions of Habitat Officer | Water Act / Water Regulation |
| Occupant Licence to Cut – Access Road | Forest Act |
| Occupant Licence to Cut –Mine Site/Tailings Impoundment | Forest Act |
| Occupant Licence to Cut – Gravel Pits | Forest Act |
| Occupant Licence to Cut – Borrow Areas | Forest Act |

### Table 20-1: Potential BC Authorizations, Licences and Permits for the Kutcho Project (continued)

| BC Government Permits and Licences | Enabling Legislation |
|---|---|
| Road use Permit (existing Forest Service Road) | Forest Act |
| Special Use Permit – Access Road | Provincial Forest Use Regulation |
| Licence of Occupation – Staging Areas | Land Act |
| Licence of Occupation – Pump House/Water Discharge Line | Land Act |
| Licence of Occupation – Borrow/Gravel Pits | Land Act |
| Surface Lease – Mine site Facilities | Land Act |
| Waste Management Permit – Effluent (Sediment, Tailings and Sewage) | Environmental Management Act |
| Waste Management Permit – Air (Crushers, Ventilation, Dust) | Environmental Management Act |
| Waste Management Permit – Refuse | Environmental Management Act |
| Special Waste Generator Permit (Waste Oil) | Environmental Management Act (Special Waste Regulations) |
| Sewage Registration | Environmental Management Act |
| Camp Operation Permits (Drinking Water, Sewage Disposal, Sanitation and Food Handling) | Health Act / Environmental Management Act |
| Waterworks Permit | Drinking Water Protection Act |
| Fuel Storage Approval | Fire Services Act |
| Food Service Permits | Health Act |
| Highway Access Permit | Highway Act |

Source: JDS (2017).

### Table 20-2: Potential Federal Authorizations, Licences and Permits for the Kutcho Project

| Federal Government Approvals and Licences | Enabling Legislation |
|---|---|
| CEAA Approval | Canadian Environmental Assessment Act |
| Metal Mining Effluent Regulations (MMER) | Fisheries Act / Environment Canada |
| Fish Habitat Compensation Agreement | Fisheries Act |
| Section 35(2) Authorization for harmful alteration, disruption or destruction of fish habitat | Fisheries Act |
| Navigable Water: Stream Crossings Authorization | Navigable Waters Protection Act |
| Dangerous Occurrence Report | Transportation of Dangerous Goods Regulations |
| Explosives Magazine Licence | Explosives Act |
| Radio Licences | Radio Communications Act |
| Radioisotope Licence (Nuclear Density Gauges/X-ray Analyzer) | Atomic Energy Control Act |

Source: JDS (2017).

## 20.2 Environmental Baseline Studies

### 20.2.1 Overview

Baseline Studies for the Project have been undertaken to provide the information necessary to prepare an Environmental Assessment Application and to develop management and monitoring plans/programs. These studies have been intermittent over the past two decades.

### 20.2.2 Study Areas

Baseline studies were carried out at two general scales: broad (regional study area) and fine (local study area). The regional study area (RSA) covers approximately 305,000 ha and is located within the Stikine Ranges which form a part of the larger Cassiar Mountains. The local study area (LSA) includes the proposed mine site area and a 2 km wide corridor (1 km either side of the centre line) surrounding the proposed access road.

### 20.2.3 Meteorology and Air Quality

The meteorological monitoring program was based on data from: two automated weather stations, the Kutcho Airstrip and Deposit stations, dustfall monitoring stations and manual snow surveys. The automated weather stations were equipped with sensors for temperature, wind speed and direction, snow depth, total precipitation and evaporation. Long-term records from five government weather stations and some historical baseline data are also available.

The nearest automated meteorological station with a long-term period of record is Dease Lake (station no. 1192340 operated by Environment and Climate Change Canada, ECCC). The Dease Lake meteorology station is located approximately 100 km west of the Project. The Project site has broadly similar meteorological conditions to Dease Lake. The Project is in a climatic region characterized by short, cool summers and long, cold winters. Long-term climate data (1981 to 2010) from the Dease Lake monitoring station (ECCC 2017) indicates that the mean annual air temperature is -0.5ºC, ranging from an extreme maximum of 35.3ºC (May 30, 1983) to an extreme minimum of -51.2ºC (January 31, 1947). The minimum and maximum monthly mean temperatures during 1981 to 2010 were -20.4ºC for January and 19.5ºC for July. There is an annual average of 58 frost-free days. On average, there are 156 days with precipitation per year with an average annual precipitation of 445 mm (280 mm as rain and 165 mm as snow-water-equivalent). The Project site will have a higher precipitation rate than Dease Lake because the Project site is at a higher elevation. The Dease Lake meteorology station is located at 807 meters above sea level (masl) while the Kutcho Airstrip and Kutcho Deposit meteorology stations are located at 1,247 and 1,435 masl, respectively.

Results from the meteorological stations indicate that weather conditions in the Project area are highly variable, with large day-to-day temperature swings being common, particularly in winter. The monitoring results available to date indicate a temperature inversion at the mine site, which may have implications for the management of air quality during operations. The Project will generate air contaminants from mining activity and power generation. A temperature inversion could inhibit mixing and dilution of air contaminants.

### 20.2.4 Soils

A baseline soils inventory study was undertaken in 2006. The study included an assessment of the soils within the mine site area, covering approximately 4,600 ha, and the proposed access road corridor. Slope analyses were also carried out.

The proposed road corridor is an upgrade of an existing road route. There were 154 inspection sites in the mine site, 64 were located along the road corridor. The soils along the road corridor are developed on colluvial, morainal, glaciofluvial, fluvial, and organic deposits and the surface materials are mostly coarse textured along the road. As a result, many of the soils are classified as well-drained Orthic Humo-Ferric Podzols.

During 2007, additional soils mapping and sampling was conducted in the mine site area (LSA) to inform the Terrestrial Ecosystem Mapping and the development of Closure and Reclamation Plan.

Soil samples were collected for metal and fertility analyses at eight sites in 2006 and nine sites in 2007 at the proposed mine site area and 14 sites along the road corridor. Background metal levels in the soil were compared with federal and provincial guidelines. Initial findings showed exceedances for some metals in both the mine site area and the road alignment suggesting that these areas have naturally high background levels.

The proposed location of the mine site is in a wide valley which has a bottom that is level to very gently sloping. Approximately 35% of the proposed mine site occurs on gentle to moderate slopes, 35% on strong slopes and 20% on very strong slopes. Approximately 50% of the road corridor occurs on gentle to moderate slopes, with 25% on level to very gently sloping topography and 20% on strong slopes.

### 20.2.5 Hydrogeology

A total of 29 groundwater monitoring wells have been established for the Project. Fourteen groundwater monitoring wells were drilled during October 2006. Groundwater levels were measured and hydraulic tests on both bedrock and overburden were performed. Groundwater samples from the monitoring wells were analyzed for water quality. Eight new monitoring wells (at four different locations) were installed in 2007, two of which were down-gradient from Main deposit. Site location, testing and installation of a water supply well for the plant site camp and a potable water supply well for the airstrip camp also took place.

In August 2008, five additional ground water monitoring wells were installed up gradient and down gradient from the Main deposit. Hydraulic conductivity tests and ground water level measurements were completed on those monitoring wells to characterize the physical groundwater flow regime in the Main deposit area. Groundwater sampling was performed twice (spring and winter) in 2008 to characterize and assess temporal variations in groundwater quality in the Main deposit area.

### 20.2.6 Geology - Metal Leaching and Acid Rock Drainage Potential

Acid Base Accounting (ABA) and Metal Leaching/Acid Rock Drainage (ML/ARD) studies were initiated in the Project area in the mid to late 1980's, and further work commenced in 2005 after the Project was reinitiated by Kutcho Copper. Drill-hole sampling in 2005 focused on the footwall of the Main deposit, particularly along the up-dip edge, which would be exposed by mining. More extensive work was conducted in 2006 which included the collection of continuous samples through the hangingwall and footwalls, proximal to the mineralization, in order to define contacts between the PAG rock and non-PAG rock areas. Subsequent sampling in late 2006 included additional samples throughout the upper parts of the hangingwall rocks.

Additional ML/ARD studies were conducted between 2007 and 2015, to provide a better understanding of ARD potential in Project rock. The studies included:

- Ongoing on-site monitoring of the ARD cribs and ML/ARD seeps in Sumac and Andrea creek valleys;
- Additional ML/ARD studies, including a geochemical comparison of the Main Sumac and Esso deposits;
- ML/ARD survey of the access road to characterize naturally occurring ARD/ML potential;
- Completion of column tests of simulated tailings; and
- Geochemical analysis of fresh tailings solids and fresh and aged tailings supernatant.

Further testing and a more detailed baseline study may be required prior to road construction.

Tailings are approximately 80% pyrite and are strongly net acid generating. Tailings will be managed by surface paste deposition within a lined enclosure contained within a non-PAG waste rock berm. During mine operation, water that runs off the surface of the paste fill pad will be collected and piped to one or more retention ponds for potential use as make-up process water. If it has to be discharged, then it will be pumped to an effluent treatment plant and treated to meet the discharge standards stipulated under the MMER and provincial water quality criteria before being released into the receiving environment (i.e., Andrea Creek). The effluent treatment plant will remain operational for as long as necessary after mine closure until the quality of water from the mine site is suitable for direct discharge.

#### 20.2.6.1 Potentially Acid Generating Modelling for Main and Esso Deposits

To better define PAG and non-PAG material quantities, a complete solid model of the Lapilli Tuff unit for both the Main and Esso Deposits was created. Past modeling exercises that were focused on the PAG material were based on a hybrid of sulphur content (S%), geology and TNPR (Adjusted Total Sulphur based Net Potential Ratio). This work centered on the work performed on behalf of WKM which resulted in a PAG study authored by Peter Holbek and Kevin Morin. The study used the analyses for S%, NP (Neutralization Potential) and TNPR (Adjusted Total Sulphur based Net Potential Ratio) as the criteria for determining what material is PAG and non-PAG. This report also stated that the boundary between the Hangingwall Lapilli Tuff and the Quartz Feldspar Crystal Tuff is the hard boundary between non-PAG and PAG.

It was determined that the most complete and effective way to model and ensure that all PAG material is captured in a solid representation was to adhere to the hard Lapilli Tuff boundary on both the hangingwall and footwall for both the Main and Esso deposits.

As noted above in Section 19, the majority of non-PAG waste rock will be used for surface construction with the remainder placed underground in mined voids. PAG waste rock will be stored permanently in the mined starter pit as well as underground in mined voids. Approximately 50% of the tailings will be used underground as backfill, while the remainder will be placed on surface as a thickened paste for permanent storage in a lined facility and the mined starter pit.

#### 20.2.6.2 Reclamation Security Bond

Section 10 of the BC Mines Act stipulates that the Chief Inspector of Mines may, as a condition of issuing a permit, require that the mine owner provide monetary security for mine reclamation and to provide for protection of, and mitigation of damage to, watercourses and cultural heritage resources affected by the mine.

Performance bonds are an acceptable means of providing this security. In addition, enough hard security must be posted so that at any point in time, the amount will fully cover the next five-year period of expected post-closure costs related to water treatment and site management and monitoring (BC MEMPR 2006, cited in Wardrop 2007).

The amount of security required, and the form in which the security is to be provided, will be agreed between Kutcho Copper and the Chief Inspector of Mines as part of the permitting process. The predicted capital and long-term operating costs of the mine site water collection and treatment system will likely be taken into consideration when deciding the amount of security required (BC MEM and MELP 1998, cited in Wardrop 2007).

### 20.2.7 Hydrology

Hydrometric stations were established at a number of locations within the mine site study area during three periods of study. Data was collected from 2006 through to 2008. There is also historical data for the site from 1984 and 1985. Estimates of a number of key hydrological parameters were made for the study area, including average annual runoff, average monthly flows, flood flows and extreme low flows.

Flow rates in Andrea Creek, which flows into Kutcho Creek, are seasonally variable with low to negligible stream flow during the winter months and high flow rates during the spring as a result of snowmelt. Stream flows during summer months are generally low, with short-lived high flow events following rainfall. There are no glaciers or permanent ice fields within the mine site area and as a result snow melts by early to mid-summer.

### 20.2.8 Water Quality

Stream baseline studies were conducted in the Project area in 2007 and 2008, and from 2011 to 2015. Water samples were analyzed for general physical variables, anions, nutrients, total cyanide, total organic carbon, and total and dissolved metals. Data for each site were compared to the Canadian Council of Ministers of the Environment (CCME) and British Columbia water quality guidelines (BCWQG), and the percentage of samples that exceed each guideline was calculated for each water variable.

In 2007 water quality was measured at monthly, weekly and quarterly intervals at 16 receiving environment streams and five road streams. In 2008, water quality monitoring continued to further characterize spatial and temporal variation in surface waters under baseline conditions. Sampling was conducted at 18 receiving streams; 12 of these were sampled quarterly and 6 were sampled monthly. Five streams along the road corridor were also sampled quarterly. Six wetland/lake sites were sampled monthly from June-September. All water samples were analyzed for a full suite of physical/general, N and P nutrients, total and dissolved metals, total cyanides and total organic carbon.

In general, streams of the region contained relatively low suspended sediment loadings, and showed moderate hardness, dissolved ions, nutrients and conductivity. Most metals present were in dissolved form. Water pH, aluminum, cadmium, chromium, copper, iron, silver and zinc exceeded CCME and/or BCWQG at one or more sites; however, the Project is in the pre-development stage and therefore any exceedances that occur are not related to disturbance but reflect current natural values prior to any mining. Andrea Creek-Adit and Sumac tributary had the highest frequency and number of variables exceeding guidelines. These sites drain from the main deposit zone; therefore, they had the highest concentration of sulphate and many metals including copper, manganese, zinc, lead and nickel.

### 20.2.9 Aquatic Resources

Aquatic resources include water quality, sediment quality, and the spatial distribution, taxonomic composition and abundance of plants and animals other than fish. This includes periphyton (streams), phytoplankton (lakes), benthic invertebrates (streams and lakes) and zooplankton (lakes). There are 16 streams and rivers that receive water from the proposed mine footprint (based on the 2007 version of the mine plan). Five streams in different watersheds spanning the proposed road route were also surveyed. Two wetlands within the Andrea Creek Watershed were sampled, along with 3 wetlands and 3 lakes situated along the proposed road route.

Aquatic resource studies on receiving water bodies were carried out in the summers of 2006 and 2007. Studies were also conducted in streams, wetlands and lakes situated along the proposed road route. Monthly and quarterly sampling of water quality has been conducted at sites throughout the mine site area and will continue through the construction, operation, closure and post-closure phases of the Project.

Water quality was assessed four times from June to September on all five wetlands and three lakes. Waters at these sites were slightly alkaline and were of moderate hardness and conductivity, with generally low turbidity with some exceptions possibly related to storm events. Nitrogen and organic carbon were higher at several wetlands compared to the lakes, as is typical due to dilution of nutrient and organic inputs in larger water bodies. Phosphorus concentrations were less variable among sites. Cyanides exceeded BC and CCME guidelines only at RWL-3 (Three Kettle Pond). Several metals exceeded water quality guidelines for the protection of aquatic life at one or more sites, including total aluminum, chromium, copper, iron, nickel and zinc, similar to the variables that exceeded in stream sites. These exceedances reflect natural conditions, since the Project has not been initiated. Wolverine Lake (RWL-6) showed the highest concentrations of most of the metals, and WL-1 had very high copper and zinc concentrations, as expected since it is down-slope of the main deposit zone in Andrea Creek Watershed.

### 20.2.10 Fish and Fish Habitat

Studies have been conducted using a variety of methods such as, gillnets, electro-fishing, minnow traps and angling. Results of these studies indicate fish community consist mainly of bull trout (*Salvelinusconfluentes*), rainbow trout (*Oncorhynchusmykiss*), mountain whitefish (*Prosopiumwilliamsoni*), Arctic grayling (*Thymallusarcticus*), lake trout (*Salvelinusnamaycush*) and longnose sucker (*Catostomuscatostomus*).

The studies were designed to provide the information necessary to develop fish habitat compensation plans and monitoring programs to satisfy assessment and regulatory requirements. The objectives were to:

- Radio-track bull trout movements in the Project area, focusing on locating spawning areas;
- Identify the degree of genetic relatedness of bull trout in upper and lower Andrea Creek and Kutcho Creek;
- Confirm the fish-bearing status of streams crossed by the proposed access road where previous studies did not find fish, and to assess streams along the new proposed access road;
- Assess stream crossing designs at planned road crossings where Fisheries and Oceans Canada authorizations for harmful alterations, disruptions, or destruction of fish habitat will be required;
- Collect detailed fish and fish habitat data in streams throughout the Andrea Creek Watershed and elsewhere to address regulatory agency information requirements;
- Collect additional baseline information on fish health and habitat quality at receiving environment sites, wetlands, and lakes within the Project area;
- Develop compensation concepts and collect preliminary information on potential compensation areas; and
- Collect biological data on fish at proposed Aquatic Effects Monitoring Program sites, including length, weight, age, sex, fecundity and tissue metal concentrations.

Bull trout dominate the fish communities of streams in the proposed mine site and receiving environment, and accounted for half of all fish captured from streams, lakes and wetlands within the Project area. Bull trout are blue-listed in British Columbia, which means they are considered to be vulnerable to human activities and natural events. Genetic studies showed there were no Dolly Varden (*Salvelinusmalma*) in the study area; historical data from the 1980s also support these findings.

### 20.2.11 Ecosystem Mapping and Vegetation

Ecosystem and vegetation studies and mapping were conducted during the summers of 2006 and 2007. Ecosystem maps were developed using both Predictive Ecosystem Mapping (PEM) and Terrestrial Ecosystem Mapping (TEM) techniques. The PEM was used to describe the RSA. A combination of PEM and TEM was used to describe the Local Study Area. The Local Study Area covers 27,683 ha, of which 23,100 ha (83%) is described by PEM and 4,584 ha (17%) is described by TEM.

Five Biogeoclimatic Ecosystem Classification (BEC) units are present within the RSA and LSA.

No listed ecological communities tracked by the BC Conservation Data Centre (BC CDC) were identified while in the field. Additionally, no plants tracked by the BC CDC or the Committee on the Status of Endangered Wildlife in Canada (COSEWIC) were identified during field surveys.

Ten plant species identified as being of cultural and/or traditional significance to members of First Nations groups of the area were identified from field plots surveyed during the ecosystem mapping and vegetation field studies conducted in 2006 and 2007. The majority are berry producing species (e.g., blueberries, crowberry, stoneberry) and are found throughout the study area. A total of 35 plant tissue samples from six different species were collected for metals analysis.

### 20.2.12 Wildlife

Comprehensive wildlife studies for the Project have been conducted for the LSA, RSA, and the proposed access road corridor (Section 20.2.2 and Table 20-3). These studies and surveys will need to be reinitiated as many were conducted in 2007, although a moose and caribou late winter survey was conducted in winter 2012/2013.

From those studies and surveys, Mountain caribou appear to be the most abundant mountain ungulate in the Project area. Stone's sheep, mountain goats and moose were also observed, although, the majority of moose observations were at lower elevations and in areas of flatter topography. Mountain caribou, Stone's sheep and mountain goats are provincially blue-listed species of conservation concern.

The breeding songbird surveys conducted in the LSA identified four provincially blue-listed species of conservation concern: short-eared owl, barn swallow (*Hirundorustica*), red-necked phalarope (*Phalaropuslobatus*), and rusty blackbird (*Euphaguscarolinus*).

The mine site area appears to support very little habitat for waterfowl, but the wetlands and lakes along the proposed access corridor support large numbers of migrating waterfowl and provide important breeding habitat. One waterfowl species of concern was detected: the provincially blue-listed surf scoter (*Melanittaperspicillata*).

The eight species that had been selected for habitat suitability modelling were selected based on the Dease-Liard Sustainable Resource Management Plan (BC MSRM 2004), on input from regulatory agencies, and from consultation with First Nations and stakeholders.

**Table 20-3: Summary of Wildlife Baseline Studies**

| Species/Group | Survey Type | Study Area | Date |
|---|---|---|---|
| Breeding songbirds | Variable radius point counts | Local | June 2006 |
| Tree and cliff nesting raptors | Stand watches | Local | June 2006 |
| Waterfowl and riverine pairs | Aerial spring migration survey | Local | May 2006 |
| | Summer breeding survey | Local | July 2006 |
| | Brood survey | Local | July 2006 |
| Moose | Winter aerial survey | Regional | February/March 2007, Winter 2012/2013 |
| Mountain Ungulates | Winter aerial survey | Regional | February/April 2007, Winter 2012/2013 |
| | Summer aerial survey | Regional | July/August 2006 Summer 2007 |
| Wolverine and Fisher(1) | Snow track survey | Local | March 2007 |
| | Fur harvest database: 1985 - 2003 | Regional | Winter 2007 |
| Bats | Ultrasonic detection | Local | Summer 2007 |
| Habitat Suitability Mapping:<br>grizzly bear(1)<br>moose(2)<br>mountain goat(1)<br>Stone's sheep(1)<br>Mountain caribou(1)<br>American marten(2)<br>hoary marmot(2)<br>western toad(2) | In conjunction with Terrestrial Ecosystem Mapping and Predictive Ecosystem Mapping<br><br>Development of the wildlife habitat ratings adhered to the Resource Information Standards Committee standards (RIC 1999) | Regional | 2006 and 2007 |

Notes: (1) Provincially blue-listed species, Ministry of Environment, Government of British Columbia. Blue List species are at risk but are not extirpated, endangered or threatened.

(2) Provincially yellow-listed species, Ministry of Environment, Government of British Columbia. Yellow List species are at risk of being lost.

Source: JDS (2017)

### 20.2.13 Wetlands

Wetlands ecosystem surveys were undertaken in 2006 and 2007 within the LSA. Hydrology surveys and aquatic biological samples were collected at selected wetlands and used in conjunction with available land use information to identify wetland functions and values.

A total of 2,677 ha of wetlands 10% of the local study area have been identified and mapped. Approximately 65% of the wetlands were mapped in the access corridor and 35% in the mine site area. A total of 22 wetland ecosystems were identified. The majority of the wetland classes identified were fens and alpine seepage sites.

#### 20.2.13.1 Archaeology

An Archaeological Impact Assessment (AIA) of the LSA and road alignment was conducted in accordance with a Heritage Conservation Act Permit.

The objectives of the AIA were to; (1) identify and evaluate any archaeological sites located within and adjacent to the potential impact zone of the proposed developments; (2) identify and assess possible impacts of the proposed developments on any identified archaeological sites; (3) provide recommendations regarding the need and appropriate scope of further archaeological studies prior to the initiation of any proposed developments; and (4) recommend viable alternatives for managing adverse impacts.

Special effort was made to address archaeological issues identified in the region by local First Nations. These issues include: ancient continental movement of obsidian from present-day Mt. Edziza; cairns or “rock piles”; identification of tephra layers; examination of rock cliff or cave shelters for sites; ice patch archaeology; climate change and; Tanzilla Village.

A total of 1,828 shovel tests were conducted. Eleven pre-contact archaeological sites were recorded, two historic sites and five Topographic Survey of Canada cairns were located. Further, an Archaeological Chance Find Procedure was developed for the Project.

## 20.3 Environmental Management Plans

### 20.3.1 Overview

Development and operation of the mine and associated access road and airstrip will affect a range of aquatic and terrestrial habitat types and wildlife species. Operations will also affect air quality at the mine site and surrounding locations.

A number of management plans, including mitigation measures, will be developed for the Project prior to submission of the Environmental Assessment Application. At a minimum, the following management plans will need to be included:

- Fisheries Offsetting Measures Plan;
- Fish and Fish Habitat Management Plan;
- Access Road Management Plan;
- Aquatic Effects Monitoring and Management Plan;
- Waste Rock Management Plan;
- Tailings Management Plan;
- ML/ARD Prediction and Prevention Management Plan;
- Water Management Plan;
- Air Quality Management Plan;
- Noise Management Plan;
- Materials Handling and Management Plan;
- Soil Management Plan;

- Erosion Control and Sediment Control Plan;
- Vegetation Management Plan;
- Wildlife Management Plan;
- Spill Contingency and Emergency Response Plan;
- Waste Management Plan;
- Airport and Aircraft Management Plan;
- Archaeological and Heritage Site Protection Plan; and
- Closure and Reclamation Plan.

### 20.3.2 Closure, Decommissioning, and Reclamation

#### 20.3.2.1 Introduction

The Project will be developed, operated and closed with the objective of leaving the property in a condition that will mitigate potential environmental impacts and restore the land to its pre-mining land use and capability. Progressive reclamation activities will be carried out concurrent with mine operation wherever possible, and final closure and reclamation measures will be implemented at the time of mine closure.

#### 20.3.2.2 Reclamation Units

For the purposes of reclamation planning the Project has been broken down into the following key reclamation units:

- Underground mine;
- Starter pit;
- Waste rock storage;
- Tailings Storage Facility;
- Water treatment facility;
- Mine site facilities;
- Landfill;
- Landfarm;
- Access road; and
- Airstrip.

Under published guidance in support of the Application Requirements for a Permit Approving the Mine Plan and Reclamation Program Pursuant to the Mines Act R.S.B.C. 1996, C.293, the BC Ministry of Energy and Mines has established key information that is to be provided in the reclamation program component of the Environmental Assessment Application. These requirements are summarized as follows:

- Pre-mine land uses and proposed end land use objectives;

- Pre-mine land capability or productivity and proposed post-mine capability or productivity objectives for all significant land uses. This information is required to create the property reclamation program and is used as a measure of reclamation success;
- Plans for characterizing the soils and overburden resource for reclamation purposes;
- Plans for salvaging, stockpiling, and replacing soils and other suitable growth media;
- Consideration of future erosion and mass wasting for long-term stability;
- Treatment of structures and equipment;
- Reclamation of water courses;
- Sealing of underground workings;
- Tailings impoundment reclamation;
- Road reclamation;
- Pre- and post-mine trace element concentrations in soils and vegetation;
- The general composition, size, shape, and location of all consolidated and unconsolidated geological units disturbed by the Project;
- Prediction of the geochemical performance of the various geological units in the form which they will be exposed, and a determination of the potential for deleterious effects;
- Determination of disposal and remediation methods, their effectiveness, and quantities by area requirements;
- Determination of monitoring requirements for extraction, waste handling, and disposal operations;
- Determination of the time to onset of ML/ARD in materials where there is a delay in the application of remedial measures;
- Programs for prevention, treatment, and control of acid rock drainage and metal leaching;
- Toxic chemical disposal;
- Environmental monitoring;
- Preliminary characterization of surficial and bedrock materials for geotechnical assessments; and
- Preliminary design of:
  - Ore processing facilities;
  - Tailings management facility;
  - Pit and underground workings;
  - Access roads;
  - Water storage facilities; and
  - Other significant transportation or utilities infrastructure.

### 20.3.2.3 Reclamation Objectives

Under the BC Mines Act and the Health, Safety, and Reclamation Code for BC, the primary objective of the reclamation plan will be to return, where practical, all areas disturbed by mining operations to their pre-mining land use and capability. Before exploration began in the Project area, the principal land uses were wildlife habitat that supported hunting, guide outfitting, trapping, and some general outdoor recreation. The following goals are implicit in achieving this primary objective:

- The long-term preservation of water quality within and downstream of de-commissioned operations;
- The long-term stability of engineered structures;
- The removal and proper disposal of all access roads, structures, and equipment that will not be required after the end of the mine life;
- The long-term stabilization of all exposed erodible materials;
- The natural integration of disturbed areas into the surrounding landscape, and the restoration of a natural appearance to the disturbed areas after mining ceases, to the best practical extent;
- An integrated passive reclamation system to eliminate long term monitoring; and
- The establishment of a self-sustaining cover of vegetation that is consistent with existing forestry and wildlife needs.

### 20.3.2.4 Underground Mine

Once all material disposal underground during reclamation is complete, the portal entrance and all surface raise openings will be permanently sealed with concrete bulkheads.

### 20.3.2.5 Mine Waste Management Structures - Starter Pit and Tailings Storage Facility

A multiple-layer soil cover will be placed over the top of the paste tailings at mine closure to minimize the water infiltration into the facility and provide an oxygen diffusion barrier to minimize the influx of oxygen. The cover system consists of a top native soil layer, a top capillary barrier layer, a compacted low-permeability clay silt layer, and a bottom capillary barrier layer over the paste tailings. The key design objective for the low-permeability layer is to maintain a high degree of saturation under all conditions. This objective is achievable for the current cover design under the meteorological, hydrological, hydrogeological, and ground conditions of this project site.

The moisture content in the majority of the paste tailings placed in the on-land paste tailings storage facility will be maintained in a nearly saturated condition over the long-term because of the lined sides and bottom of the facility, the fine-grained nature and intrinsic low permeability of the paste tailings, the cover design that limits moisture loss of the tailings, and a gentle surface slope.

The closure cover design for the mine waste in the mined-out starter pit applies best engineering measures to minimize or prevent surface infiltration and ingress of oxygen to reduce the risk of mine waste oxidation and generation of acid drainage. The cover system consists of a top native soil layer followed by a capillary barrier layer, a low-permeability clay silt layer, and a geomembrane liner installed over the final paste tailings.

A temporary PAG waste rock storage facility is required to store the PAG waste rock during early mine operation before the waste rock is permanently disposed. The storage capacity of the temporary PAG waste rock storage facility is approximately 0.46 $Mm^3$. The PAG waste rock will be placed in the temporary PAG waste rock storage facility in years -1 and 1. The PAG waste rock will be re-handled and placed back into the mined-out starter pit in years 2 and 3.

A temporary non-PAG waste rock stockpile is required to store a portion of the non-PAG waste rock generated during early years of mine operation to be later used as site construction materials and as underground mine backfill. The stockpile has a maximum storage capacity of 0.36 $Mm^3$ in year 3. The storage volume will be gradually reduced over the rest of the mine life when some non-PAG waste rock is used as construction materials and underground mine backfill.

#### 20.3.2.6 Water Treatment

A high density sludge lime treatment plant will be constructed on-site during the operational phase of the mine to treat mine effluent. This plant will pump water from the retention pond and treat the water for discharge. The volume of water treated from the pond will be managed in order to prevent release of any contaminated water from the retention pond into Andrea Creek. Sludge produced by the treatment plant will be disposed in the landfill or underground.

Flows from the temporary PAG waste rock storage facility will be directed to the retention pond during operations, and this configuration may also be maintained after closure if required.

The effluent treatment plant may not be required post closure as all mined PAG waste rock will be permanently stored underground and within the starter pit.

During mine operation, alternative treatment methods will be assessed and considered for the post-closure phase. Water management strategies will also be assessed and optimized to minimize the volumes of water requiring treatment.

#### 20.3.2.7 Mine Site Facilities

Buildings and structures that compose the mine site facilities (mill, camp, administration, maintenance shop, laboratory, site roads, and fuel storage will be removed at closure. These facilities will be dismantled or demolished. Salvageable materials will be removed from site and sold. Hazardous wastes will be removed from site and disposed of in an approved facility.

The majority of the non-hazardous, inert building materials will be disposed of in the site landfill or placed underground. Concrete footings will be broken up and disposed of underground. Any metal-contaminated soils will be removed and disposed underground. Hydrocarbon-contaminated soils will be excavated and treated on-site in a land farm. Once successfully treated, these soils will be placed in the landfill or underground.

Following removal of the facilities and any associated contamination, the disturbed areas will be re-graded, capped with top soil where needed, and fertilized and seeded with native species. Mine site roads will be scarified and seeded, with all stream crossings returned to their pre-mining condition. The site landfill will be closed using best practice methods.

#### 20.3.2.8 Access Road

Road access to the site may be required during the post-closure monitoring phase.

#### 20.3.2.9 Airstrip

The airstrip does not belong to Kutcho Copper as such reclamation requirements will be determined jointly with owners, users and regulatory agencies.

#### 20.3.2.10 Post Closure Monitoring

Long-term monitoring requirements will be developed in detail during the operational phase of the mine life. During the active closure and reclamation phase, where the mine is being decommissioned and reclaimed, monitoring will continue at the same level as during the operational phase. However, once the major closure and reclamation activities are completed, and the mine moves into the post-closure phase, the monitoring requirements will decrease. Post-closure monitoring will likely consist of the following:

- Water quality monitoring of applicable sampling stations, including the effluent treatment plant discharge, retention pond, open pit discharge and downstream flows on Andrea and Sumac Creeks;
- Environmental effects monitoring including studies on water quality, sediment quality, benthos and fish to assess effects on the aquatic receiving environment; and
- Engineering inspections by qualified persons of the retention pond, and all engineered structures including the effluent treatment plant and landfill.

Water quality will be monitored on a regular basis by the on-site effluent treatment plant staff. It is assumed that daily measurements of the plant inflows and outflows will be required as part of the plant operation.

Monitoring requirements will decrease once water quality meets discharge criteria.

## 20.4 Socio-Economic Considerations

### 20.4.1 Regional Overview

Northwestern BC exhibits a larger dependence on primary resource industries, including mining, forestry, and fishing, than the rest of the province. The region is defined by a number of small, predominantly First Nations communities, which are generally scattered along the north-south corridor of Highway 37. The larger centres of Smithers and Terrace, located along the east-west corridor of Highway 16, provide services and supplies to much of the region.

The region is further characterized by its remoteness. Communities are dispersed and transportation and communication options are limited. In general, the population of northwestern BC has been in decline in recent years, particularly in the smaller and more remote settlements.

First Nations account for a relatively high proportion of the regional population. Much of the area is included in the Traditional Territory of the Tahltan Nation, with the Kaska Dena Traditional Territory to the north. The Project is located within an area of overlap between the traditional territories of the Tahltan and Kaska Dena Nations.

The regional mining industry currently constitutes a significant source of employment for the Highway 37 communities, supplying an estimated 30% of jobs and also employs a significant number of residents from Smithers and Terrace. At present, two mines are currently in operations in the region: Brucejack and Red Chris. Mineral exploration activity in the region is significant.

### 20.4.2 Previous Studies

Baseline socio-economic and cultural studies were conducted in consultation with both the Tahltan and Kaska Dena Nations, as well as individual communities and relevant government and planning organizations. The following summarizes the results of the 2007/2008 study. Data were collected through desk-based research and field interviews on population and demographics, governance, economy and employment, education, health, social issues, culture and community services, and infrastructure.

#### 20.4.2.1 Primary Study Communities

Study communities were identified based on their proximity to the proposed Project; their likelihood as a source of employment and/or services; and consideration of First Nations interests. Primary study communities include: Dease Lake, Telegraph Creek, Iskut, Good Hope Lake, Lower Post, and Stewart. A number of common issues are evident among the study communities named, including:

- Declining populations (with the exception of Iskut);
- Relatively low rates of employment;
- Relatively low levels of education and skills development;
- Lack of educational options within the northwest;
- Increasing capacity needs for the provision of health, social, utilities and emergency services;
- A lack of facilities and services for organized recreation and entertainment;
- Lack of elder-care facilities;
- Social and mental health issues often associated with isolation and boredom, including depression, substance abuse and anti-social behaviour; and
- Concerns regarding the lasting benefits of finite resource projects, including mines.

#### 20.4.2.2 Secondary Study Communities

Smithers and Terrace were identified as secondary study communities due to their roles as the primary service centres for the north-western BC region. Employment is also expected to be drawn from these communities.

Other settlements along Highway 37, including Tatogga Lake, Bob Quinn Lake, Bell II and Meziadin Junction, were also included as secondary study communities due to their location along the proposed haul route from the mine site to port facilities in Stewart.

### 20.4.3 Recommended Future Studies

Desert Star will update the regional socioeconomic baseline evaluation as a component of future project activity and permitting submissions. Since 2007/08 the region has seen some changes with the opening of two new mines (Red Chris and Brucejack) and concurrent updates to transportation and power infrastructure including highway upgrades, port facilities, power projects and transmission line projects. In addition, Statistics Canada has completed an updated census (2011, 2016) and updated data will be available through provincial sources and municipal governments which will need to be considered and incorporated.

An updated Input-Output model will also be developed based on updated Kutcho Project and regional economics.

## 20.5 First Nations

The Kutcho Project is located within the Territory of the Tahltan Nation and the Kaska Dena Nation. A portion of the northern most area of the claim block is also located within the asserted Territory of Treaty 8, as asserted by the BC Treaty 8 First Nations and the Government of Canada.

For projects in BC and Canada, Consultation with First Nations is required to ensure Indigenous interests are considered and attempts are made to address and/or accommodate First Nations issues and concerns. The Government of BC provides guidance to proponents through the Provincial First Nations Consultation Policy (2010). Desert Star will develop a First Nations engagement strategy and plan that meets the current policy and industry best practice for First Nations engagement. The BC Environmental Assessment Office currently maintains a flexible and adaptive process for First Nations involvement and seeks to work collaboratively with First Nations in identifying and accommodating for impacts to Indigenous interests, rights, and title.

### 20.5.1 Tahltan Nation

The Tahltan Traditional Territory covers approximately 93,500 km$^2$ in northwestern BC. The Tahltan Nation is comprised of the Tahltan Band (based in Telegraph Creek and Dease Lake) and the Iskut First Nation (based in Iskut). Issues of joint interest for the Tahltan Nation are represented by the elected Tahltan Central Government, which is a registered society under the BC Society Act.

The total Tahltan population is estimated to be approximately 5,000 persons (Tahltan First Nation and IISD 2004, cited in Wardrop 2007), although the number of registered band members in 2005 was 2,189 (INAC 2006, cited in Wardrop 2007). Of the estimated 1,300 people who live within the Tahltan traditional territory, approximately 1,000 are Tahltan.

Tahltan are involved in a reconciliation process with the BC Government and have established a Shared Decision-making Agreement which includes timelines for review of referred projects. The Tahltan have opted to not engage in Treaty negotiation with the BC Government and have opted instead to negotiate participation and revenue agreements on a project by project basis. The Tahltan government has successfully negotiated numerous agreements with mining projects proponents and several revenue sharing agreements on mining and hydro-electric developments with the BC Government.

Tahltan have developed a mining policy, the purpose of which is to identify Tahltan expectations and requirements to proponents and governments seeking to develop mining projects in Tahltan Territory. Goals include self-determination, economic self-sufficiency, environmental stewardship, and healthy communities. These goals require Tahltan involvement in the planning, management, and decision-making process regarding resources within Tahltan Territory, as well as the fair distribution of all benefits, impacts and risks.

#### 20.5.1.1 Historic Engagement

Previous project Proponents have carried out in depth consultation with Tahltan including community meetings, briefings for community leadership, Traditional Land Use studies, environmental assessment participation, and communications agreements. Capstone Resources has maintained a line of communications with Tahltan leadership up to and including the transfer of the Kutcho Project to Desert Star.

#### 20.5.1.2 Current and Future Engagement

Desert Star has reached out to Tahltan leadership through a letter of introduction and has met with the President and advisors of the Tahltan Central Government.

### 20.5.2 Kaska Dena Nation

The Traditional Territory of the Kaska Dena Nation covers approximately 240,000 km$^2$, including approximately 10% of BC, 25% of the Yukon, and adjacent areas of the Northwest Territories (NWT).

These communities are loosely represented by the Kaska Dena Council (KDC) although community and regional affairs are often managed through Chief and Council governance established for each community. Collectively the Kaska Dena communities have a registered population of 2,505 members. The largest community is Liard First Nation (1,212 members) and the smallest being Dease River First Nation (181 members). The Project is closest to the Dease River First Nation and to date, discussions regarding consultation, engagement, and agreement negotiations have been focussed primarily with them. Collectively the Kaska Dean Nation has been involved in numerous mining projects and developments.

The mining industry is a significant factor in the Kaska Dena economy and employment, reflecting the important role of resource development in northern BC and the Yukon. In “Working Together for Mutual Benefit” (KDC 2003, cited in Wardrop 2007), the Kaska Dena Council identifies how proponents can further their relationship and interaction with the Kaska Dena for mutual benefit. Kaska Dena have four steps for proponents to further their relationship and interaction with the Kaska Dena for mutual benefit; Communication, Consultation, Capacity and Commitment.

#### 20.5.2.1 Historic Engagement

Previous project Proponents have carried out in depth consultation with the Kaska Dena including community meetings, briefings for community leadership, Traditional Land Use studies, and environmental assessment participation. Capstone Resources has maintained a line of communications with leadership at Dease River First Nation, Lower Post, and Kwadacha up to and including the transfer of the Kutcho Project to Desert Star.

#### 20.5.2.2 Current and Future Engagement

Desert Star has reached out to the BC Kaska Dena communities leadership (Dease River First Nation, Kwadacha, and Daylu Dena Council) as well as the Kaska Dena Council through a letter of introduction.

### 20.5.3 Treaty 8

Treaty 8 is a historic treaty signed between the Government of Canada and several First Nation groups. The Treaty covers a significant area in north-eastern BC, northern Alberta, and the south-western corner of the Northwest Territories. The western border of the Treaty is disputed. First Nation adherents to the Treaty in BC and the Government of Canada contend that the border is the Arctic/Pacific watershed divide. BC First Nations within this area who are not a part of Treaty 8, as well as the Government of BC, contend that the western border is actually the height of land of the Rocky Mountains; significantly farther to the east. The issue is currently under litigation and pending judgement in BC Supreme Court.

A portion of the northern most area of the claim block is located within the asserted Territory of Treaty 8, as asserted by the BC Treaty 8 First Nations and the Government of Canada. (Arctic watershed divide).

The BC Treaty 8 Nations include:

- Doig River First Nation;
- Halfway River First Nation;
- Prophet River First Nation;
- Saulteau First Nations;
- West Moberly First Nations;
- Fort Nelson First Nation; and
- Macleod Lake Indian Band.

#### 20.5.3.1 Current and Future Engagement

Desert Star has reached out to the BC Treaty 8 communities leadership as well as the Treaty 8 Tribal Association through a letter of introduction.

### 20.5.4 Highway Access Corridor

The access road to the Project (Highway 37) passes through the Territories of the Nisga’a Lisims Government, The Gitxsan Hereditary Chiefs, and the Gitan’yow Hereditary Chiefs. Engagement regarding highway activity with these nations and the multiple highway users (mining, forestry, power project, and government interests) is carried out through the Highway 37 Road Users Group. This working group, comprised of representatives from road users and these First Nation groups, meet twice annually to discuss upcoming road maintenance and development activities as well as highlight road use developments over the preceding and upcoming months.

### 20.5.5 Traditional Ecological Knowledge

Although Traditional Use and Traditional Ecological Knowledge studies were previously conducted, these studies will require review, and where necessary updating, prior to entering into the EA process. This will be undertaken in collaboration with potentially affected First Nations.

#### 20.5.5.1 Country Foods

A country foods baseline assessment was conducted in 2006 and 2007. The country foods evaluated were moose (*Alces alces*), Woodland caribou (*Rangifer tarandus*), snowshoe hare (*Lepus americanus*), grouse (*Phasianidae*), caribou weed (*Artemesiatilesii*), and crowberry (*Empetrumnigrum*). These species are consumed by the country foods harvesters and are located within the Project area.

The baseline assessment evaluated metals in country foods. Moose and vegetation tissue sampling was conducted in 2006 and 2007 and fish tissue sampling was conducted in 2007.

The results of this assessment indicate no unacceptable risks to human receptors from the consumption of moose, caribou, grouse, snowshoe hare, caribou weed, and crowberry. Based on the measured and predicted levels of metals in these foods, the amounts currently consumed by country foods harvesters are within the recommended maximum weekly intakes.

# 21 Capital Cost Estimate

## 21.1 Summary and Assumptions

The capital cost estimate for the Project is summarized in Table 21-1.

### Table 21-1: Capital Cost Estimate

| CAPEX | Pre-Production (M$) | Sustaining (M$) | LOM Total (M$) |
|---|---|---|---|
| Underground Mine Equipment and Infrastructure | 5.5 | 15.7 | 21.2 |
| Underground Capital Development | 12.8 | 23.8 | 36.6 |
| Pre-Production | 11.5 | - | 11.5 |
| Pre-Stripping | 3.1 | - | 3.1 |
| Owner's Costs | 9.2 | - | 9.2 |
| Offsite (Road, Airstrip Ext.) | 15.7 | 4.0 | 19.7 |
| Backfill System | 8.0 | 1.3 | 9.3 |
| Waste and Water Management | 10.3 | 5.3 | 15.6 |
| Process Plant | 62.7 | - | 62.7 |
| Site Infrastructure (Camp, Roads, Fuel, Office) | 19.1 | - | 19.1 |
| EPCM | 6.3 | - | 6.3 |
| Indirects | 27.9 | - | 27.9 |
| Sustaining Capital (~0.5% of OPEX) | - | 2.4 | 2.4 |
| Closure | - | 6.8 | 6.8 |
| Subtotal | 191.9 | 59.2 | 251.1 |
| Contingency | 28.8 | 7.9 | 36.7 |
| **Total Capital** | **220.7** | **67.1** | **287.8** |

Source: JDS (2017).

The target accuracy of the capital cost estimate is in the range of +/-25%, which represents a JDS PFS estimate. The capital cost estimate was compiled based on the following parameters:

- All capital costs are in Canadian dollars;
- Firm or budget prices for major equipment were obtained from a number of proven suppliers;
- Commodity rates/unit were obtained from general contractor and reliable suppliers and subcontractors familiar with working in the region;
- Capitalized operating expenses were developed from first principles' operating costs; and
- Labour rates for all works for the operations workforce were based on recent JDS experience.

The following costs are not included in the capital cost estimate:

- PST and GST;
- Schedule acceleration costs;

- Schedule delays and associated costs, such as those caused by:
  - Unexpected site conditions;
  - Unidentified ground conditions;
  - Labour disputes;
  - Force majeure; and
  - Permit applications.
- Development fees and approval costs beyond those specifically identified;
- Cost of any disruption to normal operations;
- Commodity specific escalation rates;
- Economy factors/pressure on labour productivity (less skilled workforce);
- Financing costs;
- Cost associated with third party delays;
- Working capital;
- Sunk costs including but not limited to purchase cost, permitting, environmental monitoring, resource drilling, feasibility study field programs and testwork and further study work; and
- Escalation.

A contingency of 15% has been applied to the capital cost estimate.

## 21.2 Mining Capital Summary

### 21.2.1 Pre-Stripping

Capital pre-stripping of 1.5 Mt of waste in the open pit will cost $3.1 M in year -1, excluding equipment lease cost. Approximately 0.8 Mt of non-PAG rock will be produced in year -1 and will be used for construction material. Approximately 0.7 Mt of PAG rock will be mined in year -1 and will be stockpiled and returned into the mined out pit at the end of year 1.

Total pre-production open pit equipment CAPEX will be $1.6 M in lease payments for $4.7 M of equipment that includes two 40-t articulated trucks, a loader, dozer, grader, drill and other ancillary mining equipment.

### 21.2.2 Underground Mining

The CAPEX for the underground mine is shown in Table 21-2 and assumed an owner-operator workforce and leased equipment.

**Table 21-2: Underground Capital Cost Estimate (excluding contingency)**

| CAPEX | Pre-Production (M$) | Sustaining (M$) |
|---|---|---|
| Mobile Equipment | 1.5[1] | 9.4[2] |
| Stationary Equipment and Other | 4.1 | 5.2 |
| Capital Development (ramp and lateral) | 9.9 | 14.6 |
| Capital Raises | 2.8 | 7.8 |
| Waste Development | 1.0 | Included in OPEX |
| Mine General | 5.7 | Included in OPEX |
| Equipment Lease Payment | 1.9 | Included in OPEX |
| Mine Maintenance | 2.9 | Included in OPEX |
| **Total** | **29.8** | **39.5** |

Note: [1] Equipment lease down payment.
[2] Production equipment capital rebuilds and replacements.
Source: JDS (2017).

Pre-production underground stationary equipment and other costs include ventilation equipment, pumps, refuge chambers, etc.

Pre-production capital development will be comprised of 4,235 m of lateral development and 467 m of raise development for a total pre-production capital cost of $9.6 M and $2.8 M, respectively, excluding mine maintenance, equipment leases and mine general costs.

Pre-production waste development includes 360 m of development for stoping normally included in operating costs.

Mine general capital includes technical services personnel, support equipment and fuel, power costs, communications and other supplies. These costs are included in operating costs in year 1 and beyond.

Underground mine equipment lease payments of $1.9 M occur in year -1 and then are included in operating costs for year 1 and later.

## 21.3 Processing Plant

The capital cost estimate is based on processing 2,500 tonnes of ore per day at 92% mill availability to produce copper and zinc concentrates. Silver and gold is recovered in the copper concentrate. The capital cost for the processing plant was estimated based on the process design criteria, the conceptual site layout, and budget quotations from vendors for major capital equipment. The processing plant consists of primary crushing, grinding and classification, flotation and regrind, concentrate dewatering, tailings disposal, reagents mixing and distribution, the process control system and mill building. The costs are a mix of new and used equipment. Used equipment cost estimates are 'expected' used prices at the time of purchase and include a provision for refurbishment and shipping.

The total estimated initial capital cost for the process plant is summarized in Table 21-3.

**Table 21-3: Pre-production Process Plant Capital Summary (excluding contingency, EPCM and indirect costs)**

| Area | Total Pre-production Cost (M$) |
|---|---|
| Crushing and Ore Handling | 7.0 |
| Grinding and Gravity Concentration | 8.3 |
| Copper Rougher Flotation | 8.5 |
| Zinc Rougher Flotation | 6.0 |
| Copper Dewatering and Filtration | 2.4 |
| Zinc Dewatering and Filtration | 2.5 |
| Reagents | 2.5 |
| Plant Building and Services | 25.5 |
| **Total** | **62.7** |

Source: JDS (2017).

## 21.4 Power

Capital for the LNG generators totals $12.8 M, but these costs have been assumed to be a capital lease. Capital leases are discussed further in Section 21.12.

## 21.5 Capitalized General and Administration Operating Costs

Capitalized operating costs are general and administration (G&A) costs that occur in the year -1, or the construction year of the Project. G&A costs include camp operations, transportation, site services, G&A labour, and miscellaneous expenses. Capitalized G&A operating costs are listed in Section 22.1.3, Table 22-5.

## 21.6 Backfill Plant

Capital for the backfill plant and surface piping totals $9.3 M.

## 21.7 Waste and Water Management

Capital costs in this area include liner, ditching, earthworks and material crushing costs to build the surface tailings management facility, water collection pond dam and cover the mined and filled starter pit.

## 21.8 Site Infrastructure

To support construction, mine, and processing plant operations, the following facilities are listed in the sections below.

### 21.8.1 Construction and Operations Camp

The complete self-contained 144 room camp, including sewage disposal, potable water plant, kitchen, dining, recreation areas, and bunk houses is sized to provide accommodation for construction crews with two people per bed room for a total of 288 people. Single occupancy will be the norm during operations. The camp sewage and water treatment plants are sized for both construction and operations requirements.

### 21.8.2 Emergency Response Centre

The fully equipped first aid room will be part of the maintenance complex. Initially set up to support construction it will then be turned over to operations. A separate first aid/safety training room is located adjacent to the first aid room.

### 21.8.3 Administration

Office areas are included in the camp complex for this purpose. Initially the space will be utilized by the construction management group.

### 21.8.4 Maintenance Shop

This six bay facility will have routine and rebuild capability for all surface and underground mobile equipment, both tracked and rubber tired. Areas will be set aside for welding, bench repairs and electrical work. Limited machine tools will also be included in the building.

### 21.8.5 Warehouse

A heated building complete with shelving is provided for weather sensitive equipment and supplies. The facility is conveniently located in the maintenance shop. In addition a graded, fenced area is provided for larger items and bulk spares (e.g., grinding media). Reagents will be warehoused in designated areas in the process plant. Bulk cement will be stored in a weather proof silo.

### 21.8.6 Water Treatment Plant

Surface, underground and process plant water will be treated by a single water treatment plant located in the process plant building.

### 21.8.7 Liquefied Natural Gas and Diesel Storage Facilities

The LNG and diesel storage tanks are also included in site infrastructure capital.

## 21.9 Offsite Infrastructure

### 21.9.1 Access Road

Capital is included to upgrade of the existing road to forestry road standards.

### 21.9.2 Air Strip

An allowance is included to extend the runway of the existing air strip to 1,525 m.

## 21.10 Indirect Capital

An allowance of $6.3 M is included for Engineering, Procurement, and Construction Management (EPCM) services during construction.

A general EPCM is not required for the underground development and underground construction program. The Owner's team will source additional detailed engineering and QA/QC support for specific underground requirements on a case by case basis.

Total construction indirects are included at 17% of all direct costs and include contractor mob/demobilization, consumables, equipment rentals etc.

The location of the Project will require logistics support and hence higher freight costs. An allowance of 6% is considered practical for this item.

Commissioning spares are included at 1% of directs.

Allowances for the initial fills, commissioning and owners cost have also been included.

## 21.11 Sustaining Capital

The planned sustaining CAPEX is made up of:

- Underground equipment additions as the underground mine expands;
- Select underground mobile equipment rebuilds and replacements starting in year 6;
- Underground capital development as the mine expands;
- Road upgrades;
- Completion of backfill system;
- TMF expansions and water management;
- General sustaining CAPEX of 0.5% of operating costs; and
- 15% contingency.

## 21.12 Capital Leases

Underground and surface mobile equipment and LNG power generators have been classified as capital leases and appear as an operating cost. The total capital converted to capital leases is $34.77 M at an interest rate of 5.0%.

# 22 Operating Cost Estimate

## 22.1 Introduction and Estimate Results

Preparation of the operating cost estimate is based on the JDS philosophy that emphasizes *accuracy over contingency* and utilizes *defined and proven project execution strategies*. The estimate was developed using first principles and applying direct applicable project experience, and avoiding the use of general industry factors. The operating cost is based on Owner owned and operated mining/services fleets and minimal use of permanent contractors except where value is provided through expertise and/or packaged efficiencies/skills. Virtually all of the estimate inputs are derived from engineers, contractors, and suppliers who have provided similar services to existing operations and have demonstrated success in executing the plans set forth in the study.

The operating cost estimate takes into account how the Project will be executed during the operational phase. Site labour cost is considered a fixed cost and independent of mining rates; therefore, the mining workforce levels are sized to meet the peak year requirements and are not decreased until the requirements have diminished in the later years of the operation.

The operating cost estimate was compiled based on the following parameters:

- Labour rates for all works for the operations workforce were based on wage scales from JDS recent experience;
- Fuel prices are based on current quotes from Imperial Oil using the Prince George, BC 'rack price'. Imperial also provided oil and lube prices;
- Spare parts and consumables are based on engineering and vendor estimates of quantities and pricing from suppliers; and
- The target accuracy of the operating cost estimate is +/-25%, which represents a JDS Prefeasibility Study Estimate.

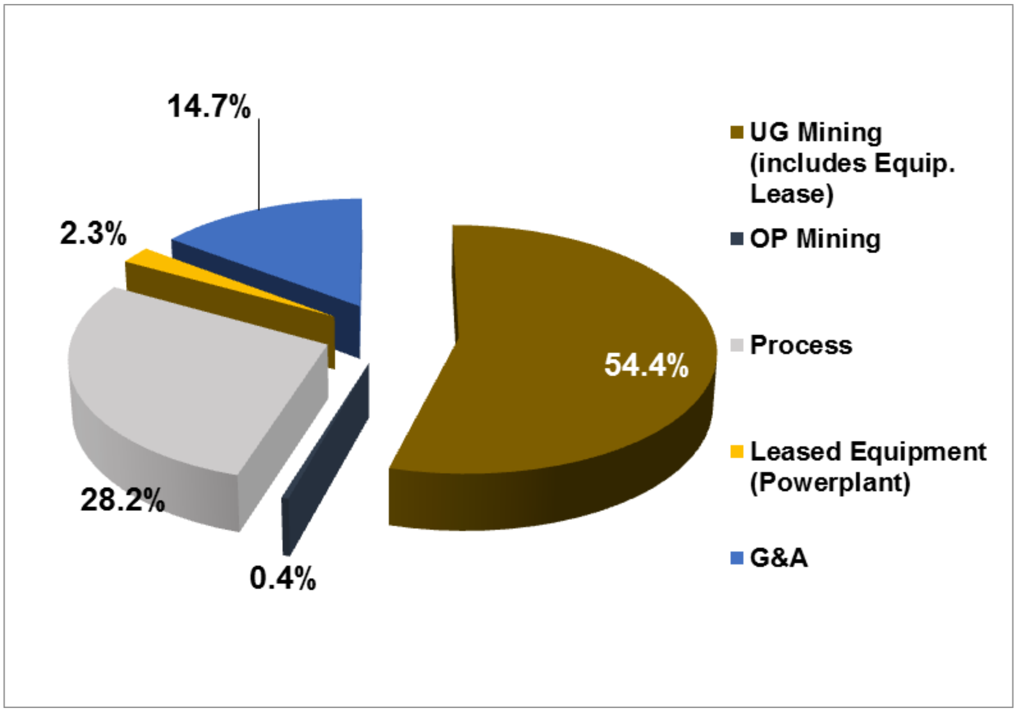
The total on-site unit operating cost estimate for the Project is $73.72 per ore tonne processed including capital leases is summarized in Table 22-1. Figure 22-1 illustrates the distribution of operating costs by area and the LOM annual cost profile is shown in Figure 22-2.

**Table 22-1: Total Project Operating Cost**

| Area | Average (M$/Year) | LOM (M$) | Processed ($/t) |
|---|---|---|---|
| Open Pit Mining (based on total ore tonnes) | 0.3 | 3.2 | 0.31 |
| Underground Mining (including Equipment Lease) | 42.2 | 418.7 | 40.10 |
| Processing | 18.1 | 217.1 | 20.79 |
| G&A | 9.5 | 113.4 | 10.86 |
| Capital Leases – Power Plant | 1.4 | 17.3 | 1.66 |
| **Total** | **64.1** | **769.7** | **73.72** |

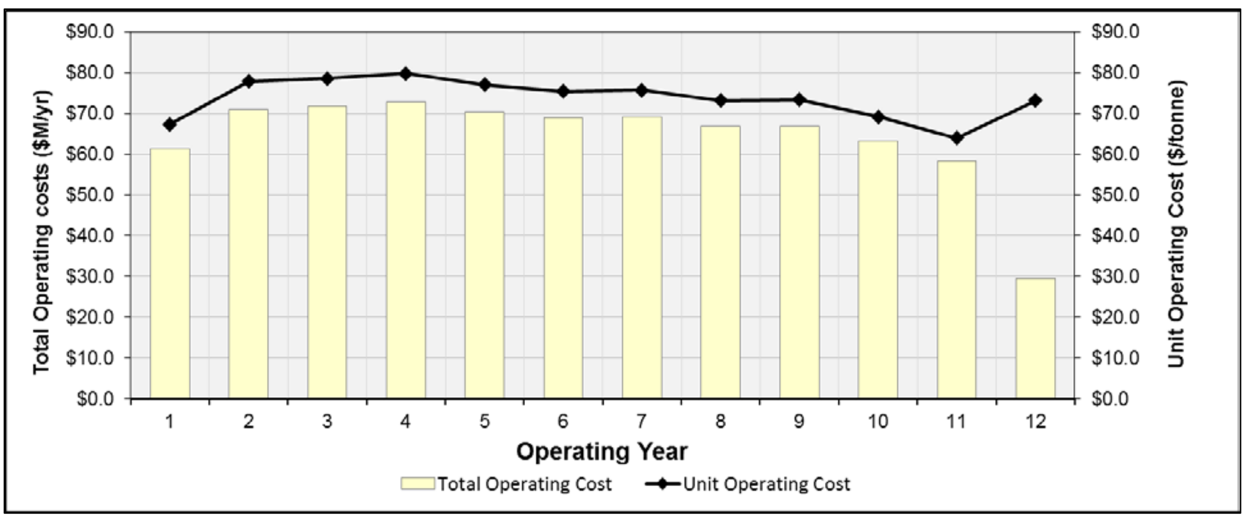
Source: JDS (2017).

## Figure 22-1: Operating Cost Distribution by Area




Source: JDS (2017).

## Figure 22-2: Life of Mine Annual Cost Profile




Source: JDS (2017).

### 22.1.1 Mining

The underground mining operating costs include the following functional areas:

- Waste development – costs of drilling, blasting, mucking, and hauling for development of ramps, raises, drifts and attack ramps;
- Production – costs related to the drilling, blasting, mucking, and hauling of ore;
- Backfill – costs related to backfill operations, including the paste plant;
- Mine General – mine support activities, such as technical services, shared infrastructure, support equipment;
- Equipment Lease Payments – equipment lease payments; and
- Mine Maintenance – maintenance labour costs that support all other sectors.

Open pit mining costs are $1.93/t mined, while underground mining costs are, including equipment lease, are $41.89/t mined. Total open pit and underground mining cost is $40.41/t processed. Power and fuel prices were assumed at $0.15/kWh and $0.97/L, respectively. Open pit and underground mining costs are summarized in Tables 22-2 and 22-3.

**Table 22-2: Open Pit Mining Operating Cost**

| Area | Average (M$/Year) | Mined ($/t) |
|---|---|---|
| Fuel | 0.5 | 0.29 |
| Equipment | 0.8 | 0.49 |
| Labour | 1.5 | 0.90 |
| Explosives | 0.4 | 0.23 |
| **Total** | **3.2** | **1.92** |

Source: JDS (2017).

**Table 22-3: Underground Mining Operating Cost**

| Area | Average (M$/Year) | Mined ($/t) |
|---|---|---|
| Waste Development | 7.3 | 8.77 |
| Production | 10.4 | 12.49 |
| Backfill | 5.4 | 6.53 |
| Mine General | 6.2 | 7.49 |
| Equipment Lease Payments | 1.4 | 1.72 |
| Mine Maintenance | 4.1 | 4.88 |
| **Total** | **34.9** | **41.89** |

Source: JDS (2017).

The average cost of owner waste development is approximately $4,500/m, whereas the LH and MCF unit cost per tonne are $30.27 and $55.82, respectively.

### 22.1.2 Processing Plant

Process operating costs were estimated to include all copper and zinc recovery steps required to produce concentrates on-site. The crushing and process plants, operating at availabilities of 70% and 92% respectively, are designed for a throughput of 2,500 t/d. Labour rates and benefit packages were based on industry information compiled by JDS. Power costs were estimated to be $0.15/kWh and applied to the annual power consumption. Reagent costs were developed using metallurgical test results summarized in Section 13 and pricing was supplied by vendors. Grinding media consumption rates were estimated using a bond abrasion index of 0.16 and liner requirements based on vendor recommendations. Maintenance costs were calculated using a benchmarked figure of 4% of equipment capital costs. The total unit operating cost estimate is $20.79 per ore tonne processed and is summarized in Table 22-4.

**Table 22-4: Processing Operating Cost**

| Area | Average (M$/year) | Unit Cost ($/t) |
|---|---|---|
| Labour | 6.1 | 6.73 |
| Power and Fuel | 4.2 | 4.64 |
| Maintenance and Operating Consumables | 8.6 | 9.42 |
| **Total** | **19.0** | **20.79** |

Source: JDS (2017).

### 22.1.3 General and Administration

General and Administration costs include G&A labour, surface support equipment, infrastructure, and other G&A items including environment, health and safety, insurance, legal freight, travel, road maintenance and camp operation expenses. The administration unit operating cost is $10.86 per total ore tonne processed and is summarized in Table 22-5.

**Table 22-5: General and Administration Operating Cost**

| Area | Average (M$/year) | Unit Cost ($/t) |
|---|---|---|
| G&A Labour | 2.7 | 3.10 |
| Surface Support Equipment | 1.2 | 1.32 |
| Infrastructure | 0.0 | 0.01 |
| Other G&A Items | 5.6 | 6.43 |
| **Total** | **9.5** | **10.86** |

Source: JDS (2017).

### 22.1.4 Power Plant Capital Leases

The average annual cost of the power plant capital leases is $1.4 M or $1.66/t processed.

# 23 Economic Analysis

The economic assessment in this pre-feasibility study considers measured and indicated resources. All results unless noted otherwise are in Canadian dollars.

The economic model was used to evaluate three cases, Base Case, Case 2, and Case 3. All net present values (NPV) reported are “Post-tax” values.

## 23.1 Assumptions

The three cases modeled three different metals price scenarios. Each was evaluated for this pre-feasibility study with prices designed to approximate various long term forecasts. All cases assume constant metals prices and exchange rates over the LOM.

The price assumptions for each economic case are shown in Table 23-1.

**Table 23-1: Metal Price Assumptions**

| Metal | Unit | Base Case | Case 2 | Case 3 |
|---|---|---|---|---|
| Cu | US$/lb Cu | 2.75 | 2.50 | 3.00 |
| Zn | US$/lb Zn | 1.10 | 1.00 | 1.20 |
| Au | US$/oz Au | 1,250 | 1,125 | 1,375 |
| Ag | US$/oz Ag | 17.00 | 15.30 | 18.70 |
| Exchange Rate | C$/US$ | 1.33 | 1.38 | 1.29 |

Source: JDS (2017).

The Canadian dollar to US dollar exchange rate utilized in each case is different. The Base Case model assumes C$1.33 is equal to one US$. In Case 2, the assumed exchange rate is C$1.38 is equal to one US$ while in Case 3, the exchange rate is C$1.29 to one US$. All cases relate behavior of the exchange rate to the copper price used (i.e., stronger metal price correlating historically to stronger Canadian dollar).

## 23.2 Economic Model Summary

A summary of the Economic analysis is shown in Table 23-2. The calculations in the cash flow model do not take into account the interest on capital. Payback is calculated using undiscounted cash flows.

### Table 23-2: Economic Analysis

| Item | Unit | Base Case | Case 2 | Case 3 |
|---|---|---|---|---|
| Unit Open Pit Mining Costs[1] | $/t milled | 0.31 | 0.31 | 0.31 |
| Unit Underground Mining Costs | $/t milled | 40.10 | 40.10 | 40.10 |
| Unit Milling Costs | $/t milled | 20.79 | 20.79 | 20.79 |
| Unit G&A and Site Services | $/t milled | 10.86 | 10.86 | 10.86 |
| Capital Leases | $/t milled | 1.66 | 1.66 | 1.66 |
| Royalties | $/t milled | 3.63 | 3.38 | 3.87 |
| Unit Total OPEX (with royalties) | $/t milled | 77.35 | 77.10 | 77.58 |
| Unit OPEX (net of trans, ref, credits) | US$/lb Cu | 0.59 | 0.66 | 0.67 |
| Total Initial Capital | $M | 220.7 | 220.7 | 220.7 |
| $NPV_{8\%}$ Pre Tax | $M | 423.5 | 340.7 | 501.0 |
| $NPV_{8\%}$ After Tax | $M | 265.2 | 211.1 | 315.7 |
| IRR Pre Tax | % | 34.6 | 29.9 | 38.8 |
| IRR After Tax | % | 27.6 | 23.9 | 31.0 |
| Payback Period (Post-tax) | years | 3.5 | 3.9 | 3.3 |

Note: [1] Open pit mining costs in year 1 are $1.93/t moved which averaged over the LOM milled tonnes is $0.31/t.
Source: JDS (2017).

Tables 23-3 and 23-4 show the output NPV of each case at various input parameters.

### Table 23-3: Pre-tax Net Present Values (8% Discount Rate) Results by Case

| Case | Cu (US$/lb) | Zn (US$/lb) | Au (US$/oz) | Ag (US$/oz) | Forex (US$:C$) | Pre-Tax NPV (C$M) | IRR (%) | Payback (Years) |
|---|---|---|---|---|---|---|---|---|
| Base Case | 2.75 | 1.10 | 1,250.00 | 17.00 | 0.750 | 423.5 | 35 | 3.3 |
| Case 2 | 2.50 | 1.00 | 1,125.00 | 15.30 | 0.725 | 340.7 | 30 | 3.6 |
| Case 3 | 3.00 | 1.20 | 1,375.00 | 18.70 | 0.775 | 501.0 | 39 | 3.0 |

Source: JDS (2017).

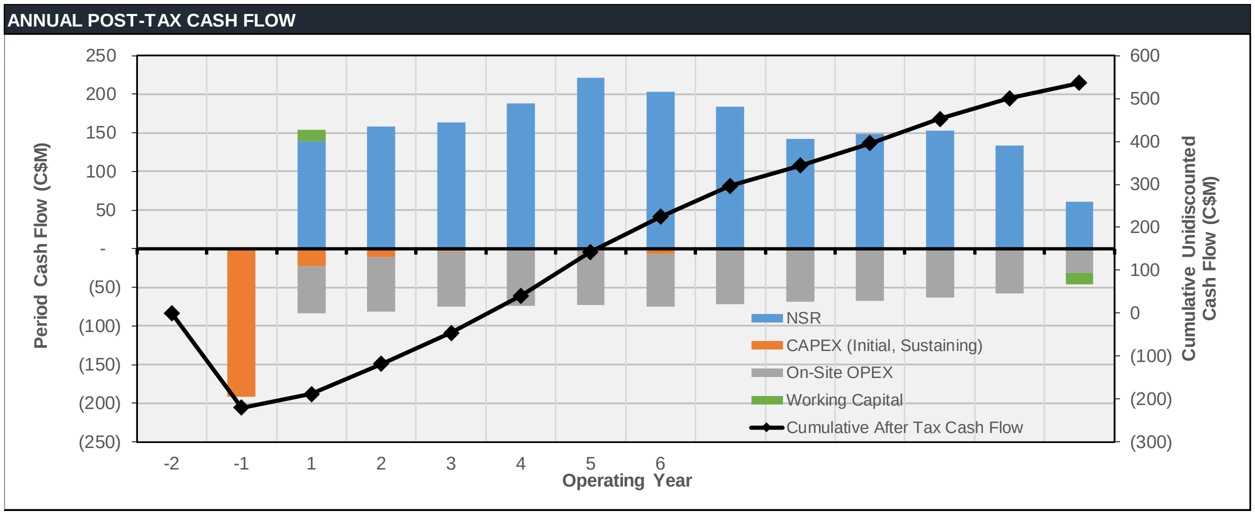
### Table 23-4: Post-tax Net Present Values (8% Discount Rate) Results by Case

| Case | Cu (US$/lb) | Zn (US$/lb) | Au (US$/oz) | Ag (US$/oz) | Forex (US$:C$) | After-Tax NPV (C$M) | IRR (%) | Payback (Years) |
|---|---|---|---|---|---|---|---|---|
| Base Case | 2.75 | 1.10 | 1,250.00 | 17.00 | 0.750 | 265.2 | 28 | 3.5 |
| Case 2 | 2.50 | 1.00 | 1,125.00 | 15.30 | 0.725 | 211.1 | 24 | 3.9 |
| Case 3 | 3.00 | 1.20 | 1,375.00 | 18.70 | 0.775 | 315.7 | 31 | 3.3 |

Source: JDS (2017).

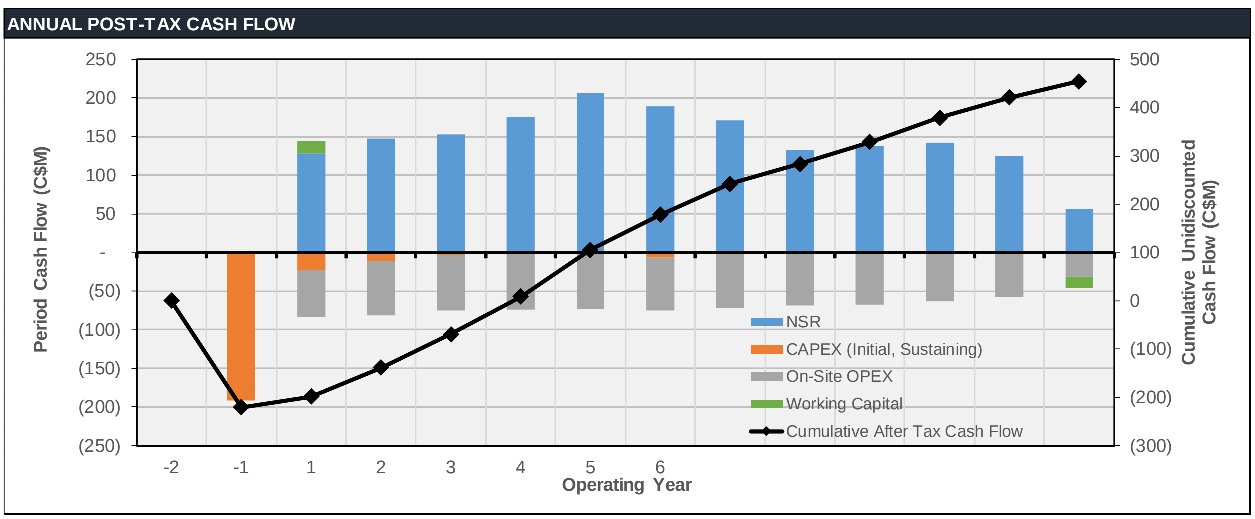
Figures 23-1 to 23-3 are graphical representations of the yearly and cumulative undiscounted cash flows for this project as modeled in each case. Table 23-5 shows the annual economic model results summary.

## Figure 23-1: Base Case: Annual and Cumulative Cash Flow



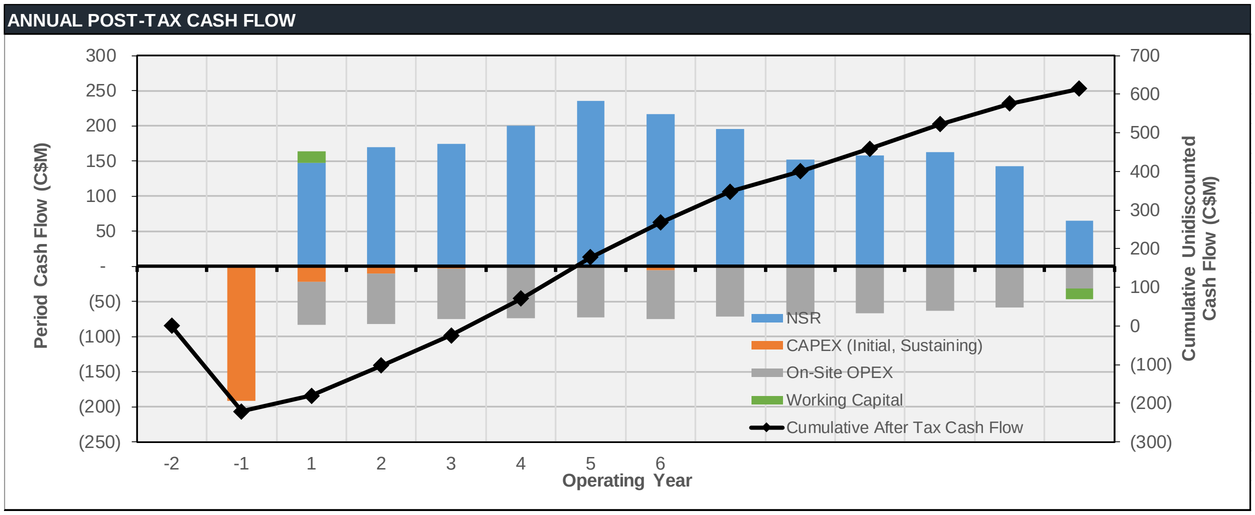

Source: JDS (2017).

## Figure 23-2: Case 2: Annual and Cumulative Cash Flow




Source: JDS (2017).

## Figure 23-3: Case 3: Annual and Cumulative Cash Flow




Source: JDS (2017).

### Table 23-5: Kutcho Project Annual Economic Model Summary

| | Pre-Tax | After-Tax |
|---|---|---|
| NPV | 423.5 | 265.2 |
| IRR | 34.6% | 27.6% |
| Payback | 3.3 | 3.5 |

| Parameter | Unit | LOM | Year | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | -1 | 1 | 2 | 3 | 4 | 5 | 6 | 7 | 8 | 9 | 10 | 11 | 12 |
| **Metal Price & F/X** | | | | | | | | | | | | | | | |
| Cu | US$/lb | **2.75** | 2.75 | 2.75 | 2.75 | 2.75 | 2.75 | 2.75 | 2.75 | 2.75 | 2.75 | 2.75 | 2.75 | 2.75 | 2.75 |
| Zn | US$/lb | **1.10** | 1.10 | 1.10 | 1.10 | 1.10 | 1.10 | 1.10 | 1.10 | 1.10 | 1.10 | 1.10 | 1.10 | 1.10 | 1.10 |
| Au | US$/oz | **1,250** | 1,250 | 1,250 | 1,250 | 1,250 | 1,250 | 1,250 | 1,250 | 1,250 | 1,250 | 1,250 | 1,250 | 1,250 | 1,250 |
| Ag | US$/oz | **17.00** | 17.00 | 17.00 | 17.00 | 17.00 | 17.00 | 17.00 | 17.00 | 17.00 | 17.00 | 17.00 | 17.00 | 17.00 | 17.00 |
| F/X Rate | US$:CA$ | **0.75** | 0.75 | 0.75 | 0.75 | 0.75 | 0.75 | 0.75 | 0.75 | 0.75 | 0.75 | 0.75 | 0.75 | 0.75 | 0.75 |
| **Mine Production** | | | | | | | | | | | | | | | |
| **Open Pit** | | | | | | | | | | | | | | | |
| Waste | Mt | **2.7** | 1.5 | 1.2 | - | - | - | - | - | - | - | - | - | - | - |
| Ore | Mt | **0.4** | - | 0.4 | - | - | - | - | - | - | - | - | - | - | - |
| **Total Open Pit** | **Mt** | **3.2** | **1.5** | **1.7** | **-** | **-** | **-** | **-** | **-** | **-** | **-** | **-** | **-** | **-** | **-** |
| Strip Ratio | w:o | 6.2 | - | 2.7 | - | - | - | - | - | - | - | - | - | - | - |
| **Underground** | | | | | | | | | | | | | | | |
| Main Ore | Mt | **7.7** | | 0.5 | 0.9 | 0.7 | 0.4 | 0.4 | 0.4 | 0.5 | 0.9 | 0.9 | 0.9 | 0.9 | 0.4 |
| Esso Ore | Mt | **2.3** | | - | - | 0.2 | 0.5 | 0.5 | 0.5 | 0.5 | - | - | - | - | - |
| **Total Underground Ore Mined** | **Mt** | **10.0** | - | **0.5** | **0.9** | **0.9** | **0.9** | **0.9** | **0.9** | **0.9** | **0.9** | **0.9** | **0.9** | **0.9** | **0.4** |
| Throughput | t/d | 2,500 | - | 2,500 | 2,500 | 2,500 | 2,500 | 2,500 | 2,500 | 2,500 | 2,500 | 2,500 | 2,500 | 2,500 | 2,492 |
| Waste (total) | Mt | **2.7** | 1.5 | 1.2 | - | - | - | - | - | - | - | - | - | - | - |
| Ore (total) | Mt | **10.4** | - | 0.9 | 0.9 | 0.9 | 0.9 | 0.9 | 0.9 | 0.9 | 0.9 | 0.9 | 0.9 | 0.9 | 0.4 |
| **Total Mined** | **Mt** | **13.2** | **1.5** | **2.1** | **0.9** | **0.9** | **0.9** | **0.9** | **0.9** | **0.9** | **0.9** | **0.9** | **0.9** | **0.9** | **0.4** |
| **Mill Feed** | | | | | | | | | | | | | | | |
| Ore | Mt | **10.4** | - | 0.9 | 0.9 | 0.9 | 0.9 | 0.9 | 0.9 | 0.9 | 0.9 | 0.9 | 0.9 | 0.9 | 0.4 |
| Copper Grade | Cu % | **2.01%** | | 1.94% | 2.13% | 2.01% | 2.02% | 2.26% | 2.12% | 2.06% | 1.88% | 1.91% | 2.07% | 1.81% | 1.87% |
| Zinc Grade | Zn % | **3.19%** | | 1.92% | 2.62% | 2.91% | 3.71% | 5.30% | 4.41% | 3.76% | 2.64% | 3.06% | 2.78% | 2.43% | 2.27% |
| Gold Grade | Au g/t | **0.37** | | 0.30 | 0.32 | 0.44 | 0.47 | 0.45 | 0.47 | 0.35 | 0.36 | 0.31 | 0.32 | 0.27 | 0.39 |
| Silver Grade | Ag g/t | **34.6** | | 26.4 | 31.0 | 42.3 | 47.2 | 41.5 | 46.9 | 35.2 | 27.6 | 29.2 | 28.0 | 27.3 | 30.2 |
| **Contained Metal** | | | | | | | | | | | | | | | |
| Cu | kt | **210** | - | 18 | 19 | 18 | 18 | 21 | 19 | 19 | 17 | 17 | 19 | 16 | 8 |
| | Mlbs | **464** | - | 39 | 43 | 41 | 41 | 45 | 43 | 41 | 38 | 38 | 42 | 36 | 17 |
| Zn | kt | **334** | - | 18 | 24 | 27 | 34 | 48 | 40 | 34 | 24 | 28 | 25 | 22 | 9 |
| | Mlbs | **735** | - | 39 | 53 | 59 | 75 | 107 | 89 | 76 | 53 | 62 | 56 | 49 | 20 |

**Table 23-5: Kutcho Project Annual Economic Model Summary (continued)**

| Parameter | Unit | LOM | Year | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | -1 | 1 | 2 | 3 | 4 | 5 | 6 | 7 | 8 | 9 | 10 | 11 | 12 |
| Au | kg | **3,867** | - | 270 | 296 | 403 | 433 | 414 | 427 | 323 | 325 | 279 | 293 | 248 | 156 |
| | koz | **124** | - | 9 | 10 | 13 | 14 | 13 | 14 | 10 | 10 | 9 | 9 | 8 | 5 |
| Ag | kg | **361,354** | - | 24,101 | 28,290 | 38,637 | 43,063 | 37,856 | 42,769 | 32,141 | 25,223 | 26,657 | 25,521 | 24,927 | 12,169 |
| | koz | **11,618** | - | 775 | 910 | 1,242 | 1,384 | 1,217 | 1,375 | 1,033 | 811 | 857 | 821 | 801 | 391 |
| **NSR - Cu Concentrate** | **US$M** | **1,036.8** | - | **86.3** | **95.0** | **95.1** | **99.3** | **102.3** | **100.8** | **93.9** | **81.2** | **81.8** | **88.0** | **77.1** | **36.1** |
| **NSR - Zn Concentrate** | **US$M** | **385.4** | - | **17.6** | **24.0** | **27.8** | **42.0** | **64.0** | **51.6** | **44.0** | **25.4** | **29.4** | **26.7** | **23.3** | **9.7** |
| **NSR** | **US$M** | **1,422.3** | - | **104.0** | **119.0** | **122.9** | **141.2** | **166.2** | **152.4** | **137.8** | **106.6** | **111.2** | **114.7** | **100.4** | **45.7** |
| | **C$M** | **1,896.3** | - | **138.6** | **158.7** | **163.8** | **188.3** | **221.6** | **203.2** | **183.8** | **142.1** | **148.3** | **152.9** | **133.9** | **61.0** |
| **Sumac Royalty** | **C$M** | **37.9** | **-** | **2.8** | **3.2** | **3.3** | **3.8** | **4.4** | **4.1** | **3.7** | **2.8** | **3.0** | **3.1** | **2.7** | **1.2** |
| **Total Royalties** | **C$M** | **37.9** | - | **2.8** | **3.2** | **3.3** | **3.8** | **4.4** | **4.1** | **3.7** | **2.8** | **3.0** | **3.1** | **2.7** | **1.2** |
| **NSR After Royalties** | **C$M** | **1,858.4** | - | **135.8** | **155.5** | **160.5** | **184.6** | **217.2** | **199.2** | **180.1** | **139.3** | **145.3** | **149.8** | **131.3** | **59.8** |
| | **C$/tonne** | **177.99** | - | **148.87** | **170.46** | **175.93** | **202.25** | **238.04** | **218.26** | **197.37** | **152.65** | **159.24** | **164.20** | **143.84** | **148.07** |
| **Operating Costs** | | | | | | | | | | | | | | | |
| Underground Mining (includes Equip. Lease) | C$/tonne mined | **41.89** | - | 61.26 | 44.81 | 45.51 | 46.74 | 43.98 | 42.38 | 42.62 | 40.25 | 40.30 | 36.15 | 30.97 | 31.23 |
| | C$M | **418.7** | | 28.6 | 40.9 | 41.5 | 42.7 | 40.1 | 38.7 | 38.9 | 36.7 | 36.8 | 33.0 | 28.3 | 12.6 |
| Open Pit Mining | C$/tonne mined | **1.93** | - | 1.93 | - | - | - | - | - | - | - | - | - | - | - |
| | C$M | **3.2** | | 3.2 | | | | | | | | | | | |
| Process | C$/tonne milled | **20.79** | - | 20.79 | 20.79 | 20.79 | 20.79 | 20.79 | 20.79 | 20.79 | 20.79 | 20.79 | 20.79 | 20.79 | 20.79 |
| | C$M | **217.1** | - | 19.0 | 19.0 | 19.0 | 19.0 | 19.0 | 19.0 | 19.0 | 19.0 | 19.0 | 19.0 | 19.0 | 8.4 |
| Leased Equipment (Power plant) | C$/tonne milled | **1.66** | - | 0.14 | 0.14 | 0.14 | 0.14 | 0.14 | 0.14 | 0.14 | 0.14 | 0.14 | 0.14 | 0.14 | 0.14 |
| | C$M | **17.3** | | 1.4 | 1.4 | 1.4 | 1.4 | 1.4 | 1.4 | 1.4 | 1.4 | 1.4 | 1.4 | 1.4 | 1.4 |
| G&A | C$/tonne milled | **10.86** | - | 10.13 | 10.66 | 10.66 | 10.66 | 10.66 | 10.66 | 10.66 | 10.57 | 10.64 | 10.61 | 10.59 | 17.52 |
| | C$M | **113.41** | | 9.2 | 9.7 | 9.7 | 9.7 | 9.7 | 9.7 | 9.7 | 9.6 | 9.7 | 9.7 | 9.7 | 7.1 |
| **Total** | **C$/tonne milled** | **73.72** | **-** | **67.32** | **77.85** | **78.55** | **79.78** | **77.01** | **75.42** | **75.66** | **73.19** | **73.32** | **69.14** | **63.93** | **73.11** |
| | **C$M** | **769.7** | **-** | **61.4** | **71.0** | **71.7** | **72.8** | **70.3** | **68.8** | **69.0** | **66.8** | **66.9** | **63.1** | **58.3** | **29.5** |
| **Net Operating Income** | **C$M** | **1,088.7** | - | **74.4** | **84.5** | **88.9** | **111.8** | **146.9** | **130.3** | **111.1** | **72.5** | **78.4** | **86.7** | **72.9** | **30.3** |
| | **C$/t** | **104.27** | - | **81.56** | **92.61** | **97.38** | **122.48** | **161.03** | **142.84** | **121.71** | **79.46** | **85.92** | **95.07** | **79.90** | **74.95** |
| **Capital Costs** | | | | | | | | | | | | | | | |
| Underground Mine Equipment and Infrastructure | C$M | **21.2** | 5.5 | 3.4 | 2.8 | 0.1 | - | - | 5.6 | 2.2 | 1.5 | - | - | - | - |
| Capital Development | C$M | **36.6** | 12.8 | 13.7 | 8.0 | 2.0 | - | - | - | - | - | - | - | - | - |
| Pre-Production | C$M | **11.5** | 11.5 | - | - | - | - | - | - | - | - | - | - | - | - |
| Site Services Equipment | C$M | **0.0** | | | | | | | | | | | | | |
| Pre-Stripping | C$M | **3.1** | 3.1 | | | | | | | | | | | | |
| Owner's Costs | C$M | **9.2** | 9.2 | | | | | | | | | | | | |
| Offsite (Road, Airstrip Ext.) | C$M | **19.7** | 15.7 | 4.0 | | | | | | | | | | | |
| Backfill System | C$M | **9.3** | 8.0 | 1.3 | | | | | | | | | | | |

### Table 23-5: Kutcho Project Annual Economic Model Summary (continued)

| Parameter | Unit | LOM | Year | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | -1 | 1 | 2 | 3 | 4 | 5 | 6 | 7 | 8 | 9 | 10 | 11 | 12 |
| Waste and Water Management | C$M | **15.6** | 10.3 | | | 0.8 | 0.8 | 2.0 | | | | | | | 1.7 |
| Power Plant (incl. in operating lease) | C$M | **0.0** | | | | | | | | | | | | | |
| Process Plant | C$M | **62.7** | 62.7 | | | | | | | | | | | | |
| Site Infrastructure (Camp, Roads, Fuel, Office) | C$M | **19.1** | 19.1 | | | | | | | | | | | | |
| EPCM | C$M | **6.3** | 6.3 | | | | | | | | | | | | |
| Indirects | C$M | **27.9** | 27.9 | | | | | | | | | | | | |
| Sustaining Capital (~0.5% of OPEX) | C$M | **2.4** | | | | | 0.4 | 0.4 | 0.3 | 0.3 | 0.3 | 0.3 | 0.3 | | |
| ***Subtotal*** | ***C$M*** | ***244.3*** | ***191.9*** | ***22.4*** | ***10.8*** | ***3.0*** | ***1.2*** | ***2.4*** | ***6.0*** | ***2.6*** | ***1.8*** | ***0.3*** | ***0.3*** | ***-*** | ***1.7*** |
| Contingency | C$M | 36.7 | 28.8 | 3.4 | 1.6 | 0.4 | 0.2 | 0.4 | 0.9 | 0.4 | 0.3 | 0.1 | 0.0 | - | 0.3 |
| **Closure Costs, Salvage** | **C$M** | **6.8** | | | | | | | | | | | | | **(3.0)** |
| **Total Capital** | **C$M** | **287.8** | **220.7** | **25.7** | **12.4** | **3.4** | **1.3** | **2.7** | **6.9** | **3.0** | **2.1** | **0.4** | **0.4** | **-** | **(1.0)** |
| Working Capital | C$M | **0.0** | | 15.4 | | | | | | | | | | | (15.4) |
| **Net Pre-Tax Cash Flow** | **C$M** | **800.9** | **(220.7)** | **33.3** | **72.1** | **85.4** | **110.4** | **144.2** | **123.5** | **108.1** | **70.4** | **78.0** | **86.4** | **72.9** | **46.7** |
| **Cumulative Pre-Tax CF** | **C$M** | | **(220.7)** | **(187.4)** | **(115.3)** | **(29.9)** | **80.5** | **224.8** | **348.2** | **456.3** | **526.7** | **604.8** | **691.2** | **764.1** | **810.7** |

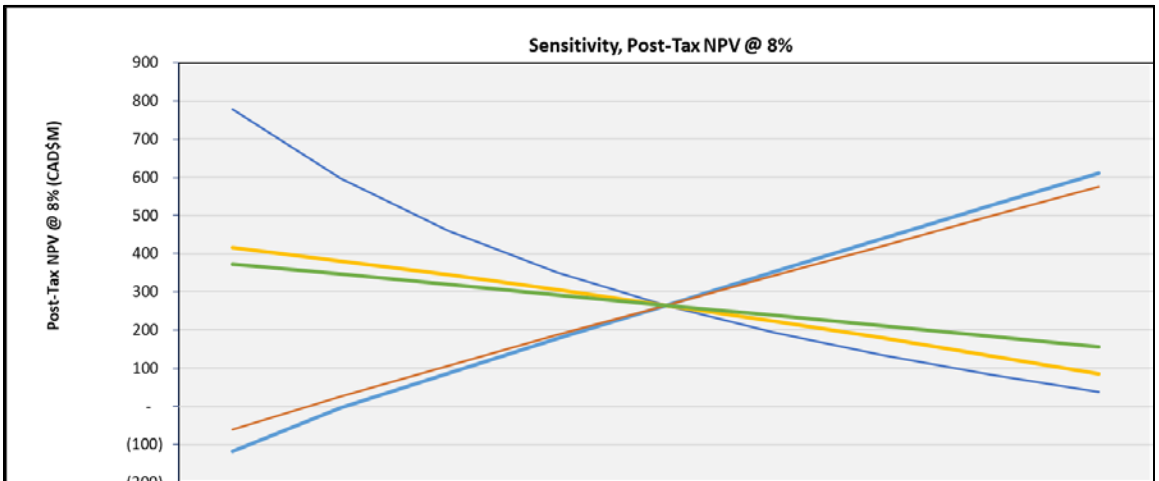
Source: JDS (2017).

## 23.3 Sensitivity Analysis

A sensitivity analysis was performed in order to evaluate the impact of changes in metal prices, mill head grade, capital costs and operating costs on the NPV of the Project, as modeled. Each variable was changed independently while all other variables were held constant at each case level. Sensitivity charts were generated using the Post-tax NPV at an 8% discount rate as the measure of project performance.

The results of the sensitivity analyses are shown on spider graphs in Figures 23-4 to 23-6.

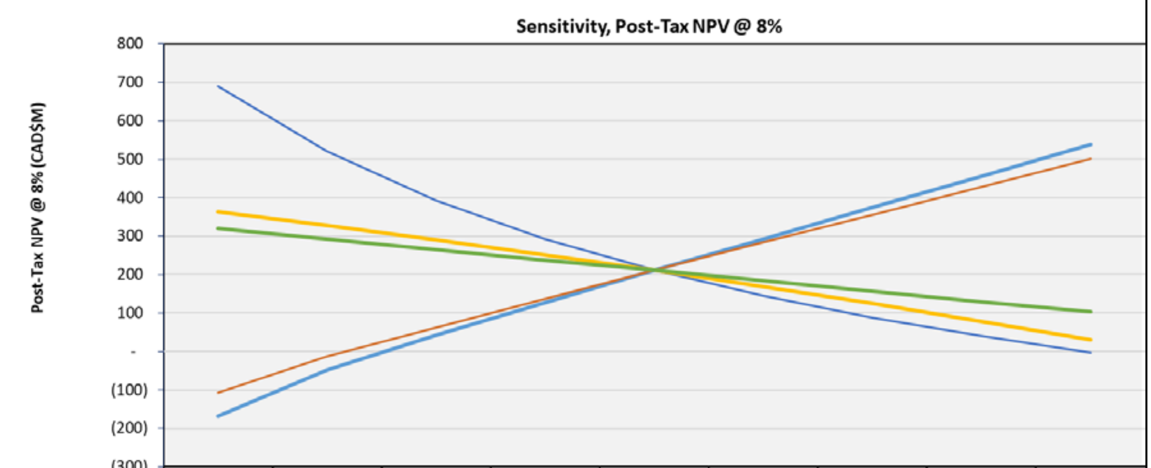
**Figure 23-4: Base Case Sensitivity Graph and Table**




| | -40% | -30% | -20% | -10% | Base | +10% | +20% | +30% | +40% |
|---|---|---|---|---|---|---|---|---|---|
| Price | (117) | (3) | 88 | 177 | 265 | 352 | 439 | 526 | 612 |
| CAD:US FX | 779 | 597 | 459 | 352 | 265 | 194 | 134 | 83 | 39 |
| Mill Feed Grade | (61) | 27 | 108 | 187 | 265 | 343 | 421 | 498 | 575 |
| OPEX | 417 | 381 | 344 | 306 | 265 | 223 | 179 | 133 | 86 |
| CAPEX | 374 | 347 | 319 | 292 | 265 | 238 | 211 | 184 | 157 |

Source: JDS (2017).

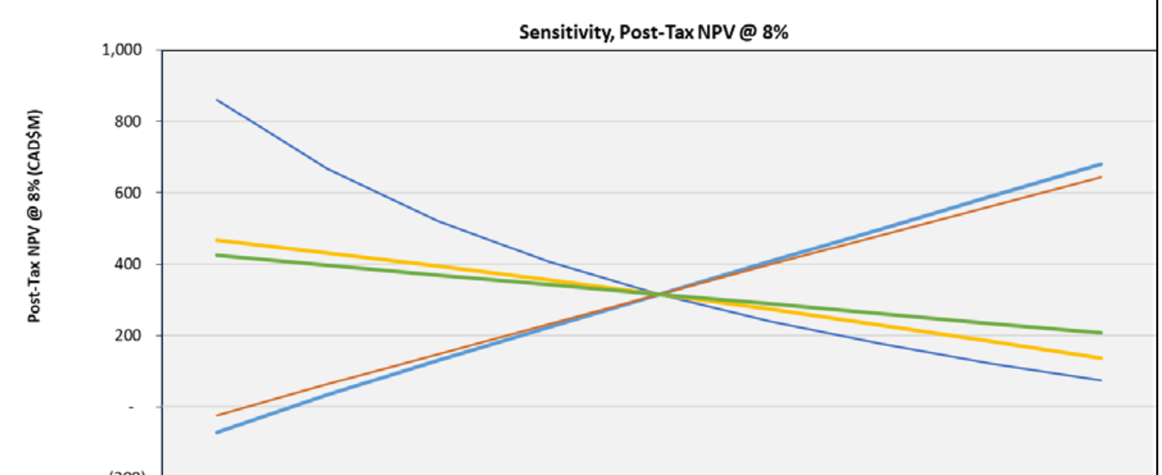
## Figure 23-5: Case 2 Sensitivity Graph and Table




| | -40% | -30% | -20% | -10% | Base | +10% | +20% | +30% | +40% |
|---|---|---|---|---|---|---|---|---|---|
| Price | (167) | (48) | 43 | 128 | 211 | 293 | 375 | 457 | 538 |
| CAD:US FX | 691 | 521 | 392 | 292 | 211 | 144 | 88 | 40 | (2) |
| Mill Feed Grade | (107) | (13) | 63 | 138 | 211 | 284 | 356 | 429 | 500 |
| OPEX | 363 | 328 | 291 | 252 | 211 | 169 | 124 | 78 | 30 |
| CAPEX | 320 | 292 | 265 | 238 | 211 | 184 | 157 | 130 | 103 |

Source: JDS (2017).

## Figure 23-6: Case 3 Sensitivity Graph and Table




| | -40% | -30% | -20% | -10% | Base | +10% | +20% | +30% | +40% |
|---|---|---|---|---|---|---|---|---|---|
| Price | (71) | 35 | 130 | 223 | 316 | 408 | 499 | 590 | 681 |
| CAD:US FX | 861 | 667 | 522 | 407 | 316 | 240 | 177 | 123 | 76 |
| Mill Feed Grade | (25) | 64 | 149 | 233 | 316 | 398 | 481 | 562 | 644 |
| OPEX | 467 | 432 | 394 | 356 | 316 | 274 | 230 | 185 | 137 |
| CAPEX | 424 | 397 | 370 | 343 | 316 | 289 | 261 | 234 | 207 |

Source: JDS (2017).

In all cases the NPV of the Project is primarily affected by the market factors such as exchange rate and metal price and mill feed grade. The Project is less sensitive to capital and operating costs though these are still significant factors to consider in project development and operations.

## 23.4 Life of Mine

The LOM for the Project, based on the assumptions made in this study, is 12 years. There are a number of potential factors that could extend the LOM and or justify an increase in production capacity that have not been included in this report.

## 23.5 Taxation

A preliminary taxation model was included in the cash flow analysis. The tax estimate takes into account investment allowances and new mine allowances for the BC Mineral Tax. Both the Federal (to a low of 15%) and the BC (to a low of 10%) taxation rates used are as per current legislation. The full tax burden is realized in year 4.

# 24 Adjacent Properties

The properties adjacent to the Project are primarily related to jade mining and are not significant to the Project. This PFS does not rely on any information from adjacent properties.

# 25 Interpretation and Conclusions

The project contains a substantial sulphide resource that can be selectively mined by underground mining methods. It has several potential advantages versus mining by large scale open pit methods including but not limited to:

- Selectivity in mining which would deliver a higher grade feed to the process plant;
- Less total material moved which translates into decreased surface disturbance and waste material stored;
- Significantly reducing the exposed PAG waste rock in the footwall of the deposit which, in the larger open pit scenario, could have resulted in greater ongoing acid generating potential; and
- The opportunity to permanently store a large portion of the tailings and significant quantities of PAG waste rock underground.

The environmental advantages for local stakeholders should increase the likelihood of receiving permits and approvals to proceed with the Project in a timelier manner because it offers an attractive alternative to open pit mining.

With these benefits come the higher costs associated with underground mining and production limitations. At the metal prices used for evaluation, the Project is economic and should proceed to the feasibility stage.

There is also good potential to improve economic performance by identifying additional ore within the development area, such as the Sumac deposit, that may justify increased underground production and/or extended LOM, as well as improvements from higher recoveries. Other opportunities are discussed in Section 25.2.

As with all mine development projects, there are a number of risks and opportunities that can affect the successful outcome of the Project. This section identifies the most significant potential risks and opportunities for the development of the Project at this preliminary evaluation stage.

Subsequent higher-level engineering studies will be required to further define the risks and opportunities and develop mitigation strategies.

## 25.1 Risks

### 25.1.1 Mineral Resource Estimate

- High grade zones may not be continuous enough for low cost underground bulk mining methods;
- Variation between the predicted and actual deposit shapes may lead to modifications in mining methods, potentially higher dilution or additional definition drilling; and
- Variations in downhole surveys may result in discrepancies in the location, extent and volumes of the zones particularly the Esso deposit.

### 25.1.2 Mining

- Ground water inflow may occur in greater quantities than predicted; and
- Weaker underground rock masses may increase ground support requirements.

### 25.1.3 Construction and Operations

- Shortages of qualified construction and operations personnel; and
- Long lead times for capital equipment.

### 25.1.4 Backfill

- Tailings may not be amenable for paste fill and surface paste disposal after further detailed testing; and
- Process tailings may require higher than assumed quantities of cement to produce an acceptable paste backfill working platform.

### 25.1.5 Metallurgy

- Reduction in metallurgical recoveries due to the complex nature of the mineralization; and
- Power consumption of major process equipment units may be higher than indicated by the test work completed to date.

### 25.1.6 Processing

- Suitability of process design for the complex Kutcho material; and
- Increased water treatment due to excessive water quantities.

### 25.1.7 Environment and Permitting

- Ability to get all necessary permits in a timely manner; and
- Sufficient process water to feed the mill.

### 25.1.8 Economics

- Decreased metal prices are the greatest potential risks to the Project’s economics;
- Detrimental exchange rates;
- Availability of suitable used process equipment at the prices assumed in this study;
- Variable transportation costs and smelting terms; and
- Capital and operating cost increases due to equipment and labour shortages and increased consumable prices.

## 25.2 Opportunities

The top five opportunities are further summarized in the following sub-sections.

### 25.2.1 Mining

- Further mine design to enhance haulage routes; and
- Used and rebuild equipment from OEM's to reduce capital requirements.

### 25.2.2 Processing

- Improved zinc separation from copper concentrate;
- Recoveries and reagent usage may be improved by further metallurgical test work, particularly zinc, which was not optimized during the latest round of metallurgical testing;
- Additional metallurgists and lab equipment has been added to optimize process plant performance; and
- Market opportunities for used plant equipment exist and may help to reduce capital costs.

### 25.2.3 Power Generation

- Alternate LNG sources other than Vancouver could have a significant impact on operating costs; and
- Potential connection to grid power.

### 25.2.4 Exploration Potential

- The Sumac deposit has a lower grade inferred resource of over 11 Mt, based on 11 holes, two of which have grades comparable to the Main deposit that could be upgraded with further exploration and delineation drilling, to define high grade zones within the overall Sumac resource; and
- The region has the potential to yield further mineral resources in zones parallel to or along strike of the known mineral resources, based on extensive exploration work completed to date.

### 25.2.5 Mineral Resources and Project Life

- There is a possibility that there may be sampling and grade opportunities related to tighter controls and more consistent methods;
- Tighter definition drilling may increase the continuity of isolated high-grade zones;
- Revise the cut-off grade calculation and usage to reflect current more favourable metal prices, resulting in a greater degree of mineral resource to reserve conversion;
- The Main and Esso deposits have a resource of over 19 million tonnes of which only 10.5 Mt are proposed to be mined in this study, leaving a significant portion that could be mined with favourable metal prices and operating costs;

- Additional infill drilling at Sumac is likely to show improvements in grade and for improved classification of resources;
- Areas between Main and Sumac appears to be relatively untested;
- Drilling at the Main deposit along strike and down dip could show extensions and high-grade trends that would increase resources; and
- Down dip drilling at Esso would resolve downhole survey questions along with potentially increasing the extent of the deposit.

# 26 Recommendations

It is recommended that the Project progress towards a feasibility study by conducting exploration, definition and expansion drilling in an attempt to convert inferred resources to measured or indicated. The resource drilling program is estimated to be $5,400,000. Additional work around metallurgy and geotechnical work should also be conducted. These programs are estimated to cost an additional $3,000,000.

# 27 Qualified Persons Certificates

JDS Energy & Mining Inc.
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Vancouver, BC V6C 2W2
t 604.558.6300
jdsmining.ca

**CONSENT OF QUALIFIED PERSON**

I, Michael Makarenko, P. Eng., do hereby certify that:

1. This certificate applies to the Technical Report entitled "Prefeasibility Study Technical Report on the Kutcho Project, British Columbia" with an effective date of June 15, 2017, (the "Technical Report") prepared for Desert Star Resources Ltd.;

2. I am currently employed as Senior Project Manager with JDS Energy & Mining Inc. with an office at Suite 900 – 999 West Hastings Street, Vancouver, British Columbia, V6C 2W2;

3. I am a graduate of the University of Alberta with a B.Sc. in Mining Engineering, 1988. I have practiced my profession continuously since 1988;

   I have worked in technical, operations and management positions at mines in Canada, the United States, Brazil and Australia. I have been an independent consultant for over ten years and have performed mine design, mine planning, cost estimation, operations & construction management, technical due diligence reviews and report writing for mining projects worldwide;

   I am a Registered Professional Mining Engineer in Alberta (#48091) and the Northwest Territories (#1359);

   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. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of NI 43-101;

4. I visited the Kutcho project on September 27-29, 2010;

5. I am responsible for Sections 1 (except 1.4,1.5,1.6,1.9,1.10,1.11,1.12), 2, 3, 12.3,15 ,16, 18 (except 18.11), 19, 20 (except 20.1.4.3), 21, 22, 23, 24, 25, 26, 27, 28, 29 of this Technical Report;

6. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI  43-101;

7. I have had prior involvement with the property that is the subject of this Technical Report. I was QP for “Preliminary Economic Assessment Underground Option Kutcho Project British Columbia” dated September 2, 2009, “Preliminary Economic Assessment Revised Underground Option Kutcho Project British Columbia” dated July 6, 2010; “Kutcho Copper Project Prefeasibility Study British Columbia” dated February 15, 2011;

8. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading; and

9. I have read NI 43-101, and the Technical Report has been prepared in accordance with NI 43-101 and Form 43-101F1.

Effective Date: June 15, 2017
Signing Date: July 31, 2017

**(Original signed and sealed) “Michael Makarenko, P. Eng.”**

Michael Makarenko, P. Eng.

PARTNERS IN ACHIEVING MAXIMUM RESOURCE DEVELOPMENT VALUE

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**CONSENT OF QUALIFIED PERSON**

I, Kelly McLeod, P. Eng., do hereby certify that:

1. This certificate applies to the Technical Report entitled "Prefeasibility Study Technical Report on the Kutcho Project, British Columbia" with an effective date of June 15, 2017, (the "Technical Report") prepared for Desert Star Resources Ltd.;
2. I am currently employed as a Senior Engineer, Metallurgy, with JDS Energy & Mining Inc. with an office at Suite 900 – 999 West Hastings Street, Vancouver, British Columbia, V6C 2W2;
3. I am a Professional Metallurgical Engineer registered with the APEGBC, P.Eng. #15868. I am a graduate of McMaster University with a Bachelors of Engineering, Metallurgy, 1984. I have practiced my profession intermittently since 1984 and have worked for the last 10 years consulting in the mining industry in metallurgy and process design engineering;
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (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. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of NI 43-101;
5. I not personally visited the Kutcho project site;
6. I am responsible for Sections 13 & 17 of this Technical Report;
7. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;
8. I have not had prior involvement with the property that is the subject of this Technical Report;

9. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading; and

10. I have read NI 43-101, and the Technical Report has been prepared in accordance with NI 43-101 and Form 43-101F1.

Effective Date: June 15, 2017
Signing Date: July 31, 2017

**(Original signed and sealed) "Kelly McLeod, P. Eng."**

Kelly McLeod, P. Eng.

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gdkirkham@shaw.ca

**CERTIFICATE OF AUTHOR**

I, Garth David Kirkham, P.Geo., do hereby certify that:

1. This certificate applies to the Technical Report entitled "Prefeasibility Study Technical Report on the Kutcho Project, British Columbia" with an effective date of June 15, 2017, (the "Technical Report") prepared for Desert Star Resources Ltd.;

2. I am a consulting geoscientist with an office at 6331 Palace Place, Burnaby, British Columbia, V5E 1Z6;

3. I am a graduate of the University of Alberta in 1983 with a B. Sc. I have continuously practiced my profession since 1983. I have worked on and been involved with many similar NI43-101 technical reports including Kutcho, Minto, Debarwa and Cerro Las Minitas;

4. I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia.

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 visited the property on April 29th, 2008 and July 17, 2017;

7. In the independent report entitled "NI 43-101 Preliminary Economic Assessment Technical Report on the Cerro Blanco Project, Guatemala" with effective date February 7, 2017, I am responsible for Sections for Sections 6, 7, 8, 9, 10, 11, 12 and 14.

8. In the Technical Report entitled "Prefeasibility Study Technical Report on the Kutcho Project, British Columbia" with an effective date of June 15, 2017, I am responsible for Sections 1.4, 1.6, 4, 5, 6, 7, 8, 9, 10, 11, 12.1 and 14 of this Technical Report;

6331 Palace Place
Burnaby, B.C.
V5E 1Z6
Phone: (604) 529-1070
gdkirkham@shaw.ca

9. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;

10. I have had prior involvement with the property that is the subject of this Technical Report. I was QP for “Preliminary Economic Assessment Underground Option Kutcho Project British Columbia” dated September 2, 2009, “Preliminary Economic Assessment Revised Underground Option Kutcho Project British Columbia” dated July 6, 2010; “Kutcho Copper Project Prefeasibility Study British Columbia” dated February 15, 2011;

11. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading; and

12. I have read NI 43-101, and the Technical Report has been prepared in accordance with NI 43-101 and Form 43-101F1.

Effective Date: June 15, 2017
Signing Date: July 31, 2017

**“(Original signed and sealed) “Garth Kirkham, P.Geo.”**

Garth Kirkham, P.Geo.

**Stantec Consulting Ltd.**
500-4730 Kingsway, Burnaby BC V5H 0C6

July 20, 2017
File: 123220930

**CONSENT OF QUALIFIED PERSON**

I, Daniel Jarratt, EP, P. Eng., do hereby certify that:

1. This certificate applies to the Technical Report entitled "Prefeasibility Study Technical Report on the Kutcho Project, British Columbia" with an effective date of June 15, 2017, (the "Technical Report") prepared for Desert Star Resources Ltd.;

2. I am currently employed as Associate - Senior Air Quality Engineer with Stantec Consulting Ltd. with an office at 200 – 325 25th Street SE, Calgary, Alberta, T2A 7H8;

3. I am a graduate of the Montana College of Mineral Science and Technology with a B.Sc. in Environmental Engineering, 1991.

   I have practiced my profession continuously since 1991;

   I have worked in environmental effects and impacts studies for proposed and operating mines in Canada, Central and South America, Europe and Africa. I have been an environmental consultant for over twenty-five years and have planned and performed environmental (atmospheric) baseline studies and impacts/effects assessments for proposed mines in many jurisdictions;

   I am a Registered Professional Engineer in Alberta (#155200) and British Columbia (#22090);

   I am an Environmental Professional (EP) with the Environmental Careers Organization of Canada – Canadian Environmental Certification Approvals Board (CECAB);

   I am a member of the Air and Waste Management Association (AWMA) (#1007622);

   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. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of NI 43-101;

4. I have not visited the Kutcho project site;

5. I am responsible for Sections 1.12 Environmental Considerations and 20.1.4.3 Meteorology and Air Quality of this Technical Report;

6. I have had prior involvement with the property that is the subject of this Technical Report. I was QP for "Kutcho Copper Project Prefeasibility Study British Columbia" dated February 15, 2011;

7. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading; and

8. The Technical Report sections 1.12 and 20.1.4.3 has been prepared in accordance with NI 43-101.

Effective Date: June 15, 2017
Signing Date: July 20, 2017

**(Original signed and sealed) "Daniel Jarratt, EP, P. Eng."**

Daniel Jarratt, EP, P. Eng.

## CONSENT OF QUALIFIED PERSON

I, Guangwen (Gordon) Zhang, P.Eng., do hereby certify that:

1. This certificate applies to the Technical Report entitled "Prefeasibility Study Technical Report on the Kutcho Project, British Columbia" with an effective date of June 15, 2017, (the "Technical Report") prepared for Desert Star Resources Ltd.;
2. I am currently employed as a Principal Specialist with Tetra Tech Canada Inc., with an office at 14940 – 123 Avenue, Edmonton, AB, Canada, T5V 1B4;
3. I am a graduate of the Dalian University of Technology with a B.Sc. degree in Hydraulic and Water Power Structures, 1988 and a M.Sc. degree in Hydraulic Structure and Geotechnical Engineering, 1991. I am also a graduate of the University of Alberta with a Ph.D. degree in Geotechnical Engineering, 2001.
4. I have practiced my profession continuously since 1991 and have more than 22 years of direct engineering consulting experience in geotechnical and permafrost engineering through design and construction projects in Canada, Russia, and China. My areas of technical expertise include mine tailings and waste management, water management and balance, design and construction of earth structures including dams, geothermal design and evaluation, piled and shallow foundation design, and numerical modelling.
5. I am a Registered Professional Engineer in good standing in Alberta (#70199) and the Northwest Territories and Nunavut (#L1525);
6. 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. I am independent of the issuer as defined in Section 1.5 of NI 43-101;
7. I visited the Kutcho project site on September 27 to 29, 2010;
8. I am responsible for and/or shared responsibility for Sections 1.10, 1.11, and 18.11 of the report entitled "Prefeasibility Study Technical Report on the Kutcho Project, British Columbia" with an effective date of June 15, 2017.
9. I was a Qualified Person for the Technical Report entitled "Kutcho Copper Project Prefeasibility Study British Columbia" dated February 15, 2011, prepared for Kutcho Copper Corporation, and was responsible for Sections 1.8, 1.9, and 19.10 of the report;
10. As of the effective date of this Technical Report, to the best of my knowledge, information, and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading; and
11. I have read National Instrument 43-101, and the Technical Report has been prepared in accordance with National Instrument 43-101 and Form 43-101F1.

Effective Date: June 15, 2017
Signing Date: July 20, 2017

**(Original signed and sealed) "Guangwen (Gordon) Zhang, Ph.D, P. Eng."**

Guangwen (Gordon) Zhang, Ph.D., P.Eng. (AB and NWT/NU)
Principal Specialist, Tetra Tech Canada Inc.
Signed and Dated this 20th day of July 2017, in Edmonton, Alberta

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# 29 Acronyms, Abbreviations, and Units

| Symbol/Abbreviation | Description |
|---|---|
| $/kWh | dollar per kilowatt hour |
| $/liter | dollar per litre |
| $/t | dollar per tonne |
| ° | degree |
| °C | degrees Celsius |
| 3D | three-dimensions |
| AACE | Association for the Advancement of Cost Engineering |
| ABA | acid-base accounting |
| Ag | silver |
| AI | abrasion index |
| ALS | ALS Chemex |
| ANFO | ammonium nitrate/fuel oil |
| AP | acid generation potential |
| ARD | acid rock drainage |
| ARMC | American Reserve Mining Corporation |
| Atna | Atna Resources Ltd. |
| Au | gold |
| Barrick | Barrick Gold Corp. |
| BC CDC | B.C. Conservation Data Centre |
| BC EAO | British Columbia Environmental Assessment Office |
| BCDWS | British Columbia Drinking Water Standards |
| BCWQG | British Columbia water quality guidelines |
| BEC | Biogeoclimatic Ecosystem Classification |
| BQ | drill core diameter of 38 mm |
| BWI | ball mill work index |
| C$ | Canadian dollar |
| C$/ ore t | Canadian dollar per ore tonne |
| C$/t | Canadian dollar per tonne |
| C$M | million Canadian dollars |
| C&F | cut and fill |
| CAPEX | capital cost |
| Capstone | Capstone Mining Corp. |
| CCME | Canadian Council of Ministers of the Environment |
| CEAA | Canadian Environmental Assessment Agency |
| CIM | Canadian Institute of Mining and Metallurgy |
| cm | centimetre |

| Symbol/Abbreviation | Description |
|---|---|
| COG | cut-off grade |
| COSEWIC | Committee on the Status of Endangered Wildlife in Canada |
| COV | cut-off value |
| CRM | certified standard reference material |
| Cu | copper |
| $CuSO_4$ | copper sulphate |
| Desert Star | Desert Star Resources Ltd. |
| DFO | Fisheries and Oceans Canada |
| dt/d | dry tonnes per day |
| dt/h | dry tonnes per hour |
| dt/y | dry tonnes per year |
| EA | environmental assessment |
| EM | electromagnetic |
| EPCM | Engineering, Procurement, and Construction Management |
| ESR | equivalent stress support ratio |
| FWZ | Footwall Zone |
| g | gram |
| G&A | general and administrative |
| $g/cm^3$ | grams per cubic metre |
| g/t | grams per tonne |
| GJ | gigajoule |
| GPS | global positioning system |
| GWh/year | gigawatt hour per year |
| ha | hectare |
| Homestake | Homestake Canada Ltd. |
| hp | horsepower |
| HQ | drill core diameter of 63.5 mm |
| HR | hydraulic ratio |
| Hz | hertz |
| ICP | inductively coupled plasma |
| IRA | inter-ramp angle |
| $IRMR_{90}$ | Laubscher’s In-situ Rock Mass Rating |
| JDS | JDS Energy &  Mining Inc. |
| k dmt | 1,000 (kilo) dry metric tonne |
| KDC | Kaska Dena Council |
| $kg/m^2/h$ | kilogram per square metre per hour |
| kg/t | kilogram per tonne |
| km | kilometre |
| km/h | kilometres per hour |

| Symbol/Abbreviation | Description |
|---|---|
| koz | thousand troy ounces |
| kPa/m | Kilopascal per metre |
| KSA | King Salmon Allochthon |
| kt | kilotonne |
| Kutcho Copper | Kutcho Copper Corp. |
| kV | kilovolt |
| kVA | kilovolt-ampere |
| kW | kilowatt |
| KWh | kilowatt hour |
| kWh/t | kilowatt hours per tonne |
| L | litre |
| L/s | litres per second |
| lb | pound |
| LCT | locked cycle tests |
| LH | longhole |
| LHD | load haul dump units |
| LNG | liquefied natural gas |
| LOM | life of mine |
| LSA | local study area |
| m | metre |
| M | million |
| M$ | million dollar |
| m/day | metres per day |
| m/s | metres per second |
| $m^3$ | cubic metre |
| m3/s | cubic metres per second |
| masl | metres above mean sea level |
| mbsl | metres below sea level |
| MCF | mechanized cut and fill |
| MESL | McElhanney Engineering Services Limited |
| mg/L | milligrams per litre |
| ML/ARD | metal leaching and acid rock drainage |
| Mlb | million pounds |
| mm | millimetre |
| $Mm^3$ | million cubic metres |
| MMER | metal mining effluent regulations |
| MRMR | mining rock mass rating |
| Mt | million metric tonnes |
| Mt/d | million tonnes per day |

| Symbol/Abbreviation | Description |
|---|---|
| MW | megawatt |
| $Na_2SO_3$ | sodium sulphite |
| NI 43-101 | National Instrument 43-101 |
| NONEL | Non-Electric (firing system for explosives) |
| non-PAG | non-potentially acid generating |
| NP | neutralization potential |
| NPV | net present values |
| NQ | drill core diameter of 47.6 mm |
| NSR | Net Smelter Return |
| NTS | National Topographic System |
| OPEX | operating cost estimate |
| oz | ounce |
| P. Geo. | professional geoscientist |
| PAG | potentially acid generating |
| PEA | preliminary economic assessment |
| PEM | Predictive Ecosystem Mapping |
| PFS | preliminary feasibility study |
| PGA | peak ground acceleration |
| PLC | process control system |
| ppm | parts per million |
| psi | pounds per square inch |
| QA/QC | quality assurance/quality control |
| QP | qualified person |
| Rescan | Rescan Environmental Services Ltd. |
| RMR | rock mass rating |
| $RMR_{76}$ | Bieniawski's Rock Mass Rating |
| ROM | run of mine |
| RSA | regional study area |
| RWI | rod mill work index |
| S% | sulphur content |
| SAG | semi-autogenous grinding |
| SG | specific gravity |
| SRF | stress reduction factor |
| SRK | SRK consulting services Inc. |
| t | metric tonne |
| t/d | tonnes per day |
| t/h | tonnes per hour |
| $t/m^2/h$ | tonnes per square metre per hour |
| $t/m^3$ | tonnes per cubic metre |

| Symbol/Abbreviation | Description |
|---|---|
| $t/m^3$ | tonnes per cubic metre |
| TAU | Tuff-Argillite Unit |
| TCL | Teck Cominco Ltd. |
| TEK | Traditional Ecological Knowledge |
| TEM | Terrestrial Ecosystem Mapping |
| TGI | Terasen Gas Inc. |
| the Project | the Kutcho Project |
| TMF | tailings management facility |
| TNPR | Adjusted Total Sulphur based Net Potential Ratio |
| TU | Tahltan and Kaska Dena Traditional Use |
| US$ | dollar (American) |
| US$/dmt | US dollar per dry metric tonne |
| US$/lb | US dollar per pound |
| US$/oz | US dollar per ounce |
| USG | US gallons |
| UTEM | deep penetration EM geophysical surveys |
| UTM | universal transverse mercator |
| VFD | variable frequency drives |
| VHMS | volcanic hosted massive sulphide |
| VMS | volcanic massive sulphide |
| VTEM | versatile time domain electromagnetic |
| WKM | Western Keltic Mines Inc. |
| wt% | percent dry weight |
| Zn | zinc |
| µ | micron |
| µm | micrometre |