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Mining Industry Consultants

NI 43-101 FEASIBILITY STUDY TECHNICAL REPORT

Kudz Ze Kayah Property, Yukon, Canada

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1 Summary

1.1 Introduction

BMC Minerals (No.1) Limited (BMC) commissioned CSA Global Pty Ltd (CSA Global) to compile a Technical Report on the Kudz Ze Kayah Property (“KZK Project” or “KZK Property”) in Yukon, Canada. This report is to comply with disclosure and reporting requirements set forth in National Instrument 43 101 – Standards of Disclosure for Mineral Projects (NI 43-101), Companion Policy 43-101CP, and Form 43-101F1.

This Technical Report discloses material changes to the Property including:

- Recent exploration activities
- An updated Mineral Reserve estimate of the ABM polymetallic deposit
- Results of the KZK Project definitive feasibility study (DFS[1]).

1.2 Property Description and Location

The KZK Property (formerly known as the TAG Property) is located on the northern flank of the Pelly Mountain Range, 260 km northwest of Watson Lake and 115 km southeast of Ross River, Yukon. The project area lies approximately 23 km south of Finlayson Lake and 25 km west of the Wolverine Mine.

BMC holds a total of 879 contiguous Mineral Claims that make up the KZK Project. It is centred at 61°31’N latitude and 130°33’W longitude (416000E 6817000N, NAD83, UTM Zone 9) on NTS map sheets 105G/7–10, within the Watson Lake Mining District. BMC owns 100% of the Property.

The KZK Property lies within the traditional territory of the Kaska First Nation. The KZK Property is covered by a Socio-Economic Participation Agreement (SEPA) that includes all Kaska First Nations.

1.3 Accessibility, Climate, Local Resources, Infrastructure and Physiography

The Property is accessed by the all-weather Robert Campbell Highway which links the towns of Watson Lake and Carmacks. The highway is multi-surfaced with pavement from Carmacks to Faro, gravel from Faro to the Nahanni Range Road turn off and chip seal to Watson Lake. A 24 km-long gated Tote Road extends from the highway to the ABM deposit.

The ABM deposit is located at approximately 1,400 m above sea level elevation in a broad, gently sloping, U-shaped valley, covered by 2–30 m of glacial overburden. Geona Creek, a north-flowing tributary to Finlayson Creek, drains several small ponds which overlie the deposit.

The climate in the area is typically sub-arctic, characterized by cold winter temperatures (minimum mean monthly temperature of –13°C) and low snowfall. Summer is generally mild with maximum mean monthly temperature of 10°C. Rainfall peaks during the summer months with July and August being the wettest months.

The KZK Property is remote and limited infrastructure services are available. The Robert Campbell Highway is 24 km to the north of the ABM deposit and comprises the major transport route between local centres. The Finlayson airstrip is located adjacent to the highway approximately 15 km by road northwest from the KZK Project Tote Road.

[1] The use of DFS is the equivalent to a Feasibility Study as defined in the CIM Definition Standards.

The closest grid power is at Faro, approximately 150 km to the northwest. The nearest year-round deep-water ports for concentrate shipment are 870 km by road to the southwest at Skagway (Alaska) and 905 km by road to the south at Stewart (British Columbia).

1.4 Project History

Cominco (now Teck) conducted a geochemical survey in 1977 across the Finlayson Lake area, however the survey was too wide-spaced to reveal evidence of any volcanic-hosted massive sulphide (VHMS) deposits (alternatively known as volcanogenic massive sulphide deposits). Cominco's interest in the area was reignited in 1992 when soil and silt geochemical sample results from a Cominco reconnaissance program confirmed and expanded upon an anomalous silt sample released in the Geological Survey of Canada's regional geochemistry silt survey for NTS map sheet 105G, Open File 1648 (Hornebrooke and Friske, 1988).

In 1993, a small follow-up program completed by Cominco within the anomalous drainage resulted in the location of a well mineralized, layered sulphide cobble by A.B. Mawer. At the same time, potential source rocks for the mineralized float were recognized. A reconnaissance transient electromagnetic (EM) geophysical survey was immediately implemented over the projected trace of the prospective units where they disappear beneath quaternary cover in the valley floor. This survey identified an EM feature representing a possible source for the mineralized float. The first TAG claims were subsequently staked and recorded on 20 August 1993 to cover the geophysical anomaly. A magnetic survey was also carried out during staking. Further magnetic, horizontal loop electromagnetic (HLEM) and soil surveys were completed later that fall and successfully defined a drill target.

The target was drilled in April 1994 with the first hole completed on 20 April intersecting 22.5 m of massive sulphide rock in two zones. Three additional holes were drilled in April; each intersecting mineralization over significant widths. The weighted average grade of sulphides in the discovery hole was 0.5% Cu, 2.8% Pb, 10.0% Zn, 278 g/t Ag and 2.9 g/t Au over 22.5 m. The sulphide body was named the ABM deposit by the exploration team in recognition of A.B. Mawer's contribution towards the discovery and a distinguished career with Cominco.

In 1995, an additional 133 drillholes totalling 16,178 m were completed at the ABM deposit and on regional targets. Additional exploration soil sampling, minor geological mapping and ground geophysical surveys were completed. Geotechnical investigations, detailed engineering/mine planning, bulk metallurgical sampling, environmental monitoring and archaeological studies were well underway or completed, as well as the construction of a 24 km all-weather Tote Road from the Robert Campbell Highway. A preliminary feasibility study (PFS) was completed in July 1995.

The 1996 exploration program involved regional 1:20,000 scale geological mapping outside the immediate ABM deposit area, ground geophysical surveys and soil geochemistry over the northeast part of the TAG Property. Minor structural mapping and core logging was completed at the ABM deposit.

By the end of 1997, a total of 168 exploration drillholes and 15 metallurgical holes had been completed in the immediate ABM deposit area and another 20 drillholes were completed elsewhere on the property. Other deposit-related work has involved considerable ground and airborne geophysical surveys, detailed geological mapping in the vicinity of the deposit, regional and detailed exploration geochemistry and baseline environmental sampling. Additional geotechnical and metallurgical studies were also undertaken to what at the time was a feasibility study (FS) level.

In 1997, the Fault Creek Zone was discovered within a kilometre of the ABM deposit. It is a high-grade VHMS occurrence with a subtle geochemical and geophysical response.

Cominco's 1998 exploration program resulted in the delineation of another mineralized occurrence; GP4F, located approximately 5 km to the southeast of the ABM deposit.

In March 2000, Cominco announced an agreement in principle to sell the KZK Project to Expatriate Resources Ltd (Expatriate). This option agreement resulted in Expatriate controlling most of the favourable stratigraphy in the Finlayson Lake District. Expatriate amalgamated the ABM and the neighbouring 60% owned Wolverine deposits into the "'Finlayson Project". A positive FS was completed by Hatch Pty Ltd (Hatch) and additional drilling was completed by Expatriate on the Wolverine deposit.

In September 2001, Expatriate terminated the option agreement with Teck Cominco (formed from the merger of Teck Resources and Cominco) for the KZK Project.

BMC acquired the KZK Project from Teck on 24 January 2015.

1.5 Geology and Mineralization

The KZK Project is located with the Finlayson Lake District, a crescent-shaped area approximately 300 km long and 50 km wide that extends from Ross River in the north to Watson Lake in the south.

The Finlayson Lake District predominantly comprises Devonian to Lower Carboniferous (Mississippian) volcanic, intrusive, and sedimentary rocks bounded to the east by Proterozoic and Palaeozoic strata of the Selwyn Basin, representing the ancient North American continental margin, and to the southwest by the Tintina Fault.

Massive sulphide deposits of the Finlayson District are primarily hosted within stratigraphic components of the Big Campbell thrust sheet. Rocks of the Big Campbell thrust sheet include Pre-Late Devonian quartz-rich sedimentary rocks of the North River formation; mafic and felsic volcanic, and carbonaceous clastic rocks of the Upper Devonian Grass Lakes group; Late Devonian to Early Mississippian granitic rocks of the Grass Lakes plutonic suite; carbonaceous clastic and mafic and felsic volcanic rocks of the Lower Mississippian Wolverine Lake group; and carbonaceous clastic rocks and chert of the Lower Permian Money Creek formation.

The Grass Lakes Group has been subdivided into three formations which, from oldest to youngest, are the Fire Lake formation, Kudz Ze Kayah formation, and the Wind Lake formation. The Grass Lakes Group is unconformably overlain by rocks of the Wolverine Lake Group.

The Project area, comprising the Kudz Ze Kayah claim blocks within which the ABM deposit is located, encompasses units of the Grass Lakes Group. The stratigraphy trends easterly, dips moderately to the north and is interpreted as predominantly right way up except locally where it may be overturned in tight, mesoscale F2 folds. The rocks are generally overprinted by greenschist facies metamorphism and a penetrative deformation fabric. The property has been cross-cut by north to northeast-trending brittle faults which cut across all stratigraphic units.

All known massive sulphide mineralization on the KZK Property (e.g. ABM deposit, Fault Creek occurrence, GP4F Zone) occurs within the Kudz Ze Kayah formation, albeit at what appear to be different stratigraphic levels.

The ABM deposit (comprising the ABM Zone and Krakatoa Zone) primarily comprises continuous, shallow-dipping massive sulphide mineralization hosted within a thick felsic package of volcaniclastics and coherent sill/flow complex that locally make up the Kudz Ze Kayah formation.

The base of the Wind Lake formation lies approximately 200 m stratigraphically above the ABM deposit and the Wind Lake formation underlies the majority of the northern half of the KZK Property. The formation comprises carbonaceous and calcareous mudstone intercalated with mafic volcanic rocks along with minor quartzite, siltstone, chert and felsic volcanic rocks.

Massive sulphide of the ABM Zone is hosted within a felsic rock package, whereas the Krakatoa Zone is predominantly hosted by a pre-mineralization mafic sill located within the felsic volcanic package. Mineralization at Krakatoa also occurs in the felsic hangingwall units stratigraphically overlying the mafic sill, in what is broadly interpreted to be the equivalent of the ABM mineralized position. Only scattered vein-style and disseminated mineralization occurs within the mafic sill lying stratigraphically below the ABM Zone.

Massive sulphide mineralization of the ABM Zone is up to 39 m true thickness, extending approximately 700 m along strike and approximately 500 m down dip. It dips to the north-northeast at approximately 35° near surface, transitioning to a dip of approximately 15° at around 200 m depth below the valley floor. The up-dip extent of the deposit is truncated by erosion and covered by approximately 2–20 m of glaciofluvial overburden.

Sulphide mineralization is dominated by pyrite, sphalerite, pyrrhotite (+ marcasite), galena and chalcopyrite, with minor arsenopyrite and a range of sulphosalts predominantly comprising tennantite-tetrahedrite and freibergite. Both the up-dip part of ABM Zone and most of Krakatoa Zone have elevated sulphosalt content relative to the remainder of the ABM deposit.

Krakatoa Zone mineralization, bound to the west by the East Fault and to the east by the Fault Creek Fault, is hosted within Kudz Ze Kayah formation that dips at 35° to the north-northeast. Although of lesser extent, the distribution of mineralization within the Krakatoa Zone is more spatially complex than the ABM Zone due to the stacked mineralized lens system.

Krakatoa Zone mineralization is broadly concordant with stratigraphic layering of the host rocks, extending over approximately 200 m of strike, at least 500 m down dip, and up dip to the base of glacial overburden of 20–30 m thickness.

Host rock types and alteration styles of Krakatoa Zone mineralization are for the most part similar to those encountered in the ABM Zone. The key difference is the degree of mineralization associated with the mafic sill, which below ABM Zone is only poorly mineralized. The Main lens comprises the bulk of mineralization at Krakatoa, with massive sulphide occurring both within the felsic volcanics immediately beneath the mafic sill, and within the mafic sill, after replacement of enclaves of felsic volcaniclastics and/or replacement of the mafic sill itself.

1.6 Exploration

Exploration at the KZK Property has been undertaken by Cominco between 1993 and 1998, and BMC from 2015 to 2019.

Exploration programs undertaken by Cominco included soil sampling, geological mapping, ground geophysical surveys, diamond core drilling (171 holes for 24,928 m), as well as evaluation programs including geotechnical investigations, engineering/mine planning, bulk metallurgical sampling, environmental monitoring, archaeological studies and a detailed PFS in 1995.

Following project acquisition and prior to the beginning of the 2015 field season, BMC conducted extensive data validation, particularly focused on the previously defined ABM deposit.

Commencing in 2015, BMC has conducted a multi-faceted exploration program at the ABM deposit and across the KZK Project including extensive diamond drilling, geophysical surveys, geological mapping, geochemical sampling as well as relogging and resampling of archived historical drill core. Drilling programs undertaken by

BMC have comprised exploration, hydrogeological, metallurgical, resource confirmation/infill, and geotechnical drillholes (148 holes for 25,966 m in 2015, 84 holes for 19,210 m in 2016, 48 holes for 4,929 m in 2017 and 22 holes for 4,055 m in 2018).

1.7 Data Verification, Sampling Preparation, Analysis and Security

The Qualified Person has verified the data disclosed, which underpins the disclosure of the MRE contained in this Technical Report and is of the opinion that data collection and verification procedures adequately support the integrity of the database.

1.8 Mineral Processing and Metallurgical Testing

A large volume of metallurgical testwork was completed by Cominco in the 1990s. While it formed a strong platform for BMC's PFS (CSA Global, 2017) and DFS metallurgical testwork programs, this historical data was not relied upon in predictions of metallurgical performance as the flowsheet and reagent scheme have been modified to optimize metallurgical performance.

ALS Metallurgy completed BMC's PFS and DFS metallurgical testwork in their Perth and Adelaide laboratories in Australia. Five metallurgical domains have been defined for the ABM deposit, based on texture and mineralogy. The metallurgical testwork program investigated comminution and flotation performance for all five domains using domain composite samples together with five variability samples for each domain. Variability samples were primarily selected to assess variability in flotation performance over a range of head grades within each metallurgical domain.

Comminution properties established from testwork for design purposes summarized in Table 1-1 and indicate that the mineralization can be considered "soft" in terms of the "A x b" parameter and of medium hardness in terms of Bond Ball Mill Work Index (BWI).

Table 1-1: Summary of comminution properties

Parameter	Unit	Value
Impact breakage parameter – A x b	-	68.7
Drop Weight Index (DWI)	kWh/m^3	6.05
Bond Ball Mill Work Index (BWI)	kWh/t	12.8
Bond Rod Mill Work Index (RWI)	kWh/t	10.4

The typical flowsheet for DFS testwork included:

- Primary grinding to 80% passing 70 µm
- Copper pre-float rougher flotation, cleaning of the pre-float rougher concentrate without regrinding, to produce final copper concentrate
- Copper rougher flotation, regrinding of rougher concentrate and pre-float cleaner tailings to 80% passing 30 µm, two stages of cleaner flotation
- Lead rougher flotation, regrinding of rougher concentrate and pre-float cleaner tailings to 80% passing 30 µm, two stages of cleaner flotation
- Zinc pre-float rougher flotation, cleaning of the pre-float rougher concentrate without regrinding, to produce final zinc concentrate
- Zinc rougher flotation, regrinding of rougher concentrate and pre-float cleaner tailings to 80% passing 35 µm, two stages of cleaner flotation.

Flotation testwork consisted of open circuit batch flotation tests and locked cycle tests. Data from flotation testwork was used to derive relationships for recovery of economic metals and deleterious elements into concentrates for each domain. Where the data did not support development of a relationship, the average of all tests were used for predicting processing performance. The life of mine (LOM) recoveries determined for the DFS are summarized in Table 1-2. Concentrate grades are predicted to be 25.0% copper, 52.0% lead and 52.0% zinc for copper, lead and zinc concentrates respectively.

Table 1-2: LOM processing recoveries of revenue elements

Concentrate	Copper (%)	Lead (%)	Zinc (%)	Gold (%)	Silver (%)
Copper	73.8	N/A	N/A	27.3	36.8
Lead	N/A	73.5	N/A	29.4	38.2
Zinc	N/A	N/A	85.9	8.1	11.0

Settling and filtration testwork was completed for all concentrates and tailings, with pressure filtration selected for dewatering of concentrates and tailings.

1.9 Mineral Resource Estimates

BMC commissioned CSA Global to undertake an independent, updated MRE for the ABM deposit based on historical datasets and more recent 2015 and 2016 drilling.

The ABM deposit MRE is reported in Table 1-3 (open pit) and Table 1-4 (underground).

Table 1-3: ABM deposit MRE – open pit (at net smelter return (NSR) cut-off grade of CAD$25/t)

Zone	Category	Tonnes (Mt)	NSR (CAD$/t)	Cu (%)	Pb (%)	Zn (%)	Au (g/t)	Ag (g/t)	Cu metal (kt)	Pb metal (kt)	Zn metal (kt)	Au metal (koz)	Ag metal (Moz)
ABM	Indicated	14.6	358	1.0	1.6	6.1	1.3	132	140.9	229.1	886.6	614.0	62.1
	Inferred	0.3	334	1.5	1.5	4.5	1.1	115	4.7	4.9	14.4	10.9	1.2
Krakatoa	Indicated	3.5	443	0.6	3.2	7.2	1.8	213	21.4	113.2	255.5	204.0	24.3
	Inferred	0.1	347	0.6	2.3	6.3	1.3	142	0.1	2.1	5.9	3.8	0.4

Table 1-4: ABM deposit MRE – underground (at NSR cut-off grade of CAD$95/t)

Zone	Category	Tonnes (Mt)	NSR (CAD$/t)	Cu (%)	Pb (%)	Zn (%)	Au (g/t)	Ag (g/t)	Cu metal (kt)	Pb metal (kt)	Zn metal (kt)	Au metal (koz)	Ag metal (Moz)
Krakatoa	Indicated	0.2	397	1.0	2.0	6.1	1.7	170	1.7	3.5	10.5	9.2	0.9
	Inferred	0.4	447	0.8	1.6	9.5	1.2	165	3.2	6.3	37.5	14.9	2.1

Notes:

- *The Mineral Resources in this disclosure were estimated by Aaron Green, Qualified Person.*
- *The effective date of this Mineral Resource is 31 May 2017.*
- *Numbers have been rounded to reflect the precision of an Indicated and Inferred MRE.*
- *The Mineral Resources were estimated using current CIM standards, definitions and guidelines (CIM Council, 2014).*
- *The optimal transition from open pit to underground for the Krakatoa Zone has not been considered when reporting the Mineral Resource. Key modifying factors in determining this transition have been factored into reporting of the Mineral Reserve.*
- *The Indicated Mineral Resources are inclusive of those Mineral Resources modified to produce the Mineral Reserves. Inferred Mineral Resources are, by definition, always additional to Mineral Reserves.*

Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.

A total of 335 diamond drillholes define the ABM deposit for 55,782 m of drilling; 241 assayed drillholes intersect the interpreted mineralization zones. The ABM Zone was sampled using diamond drillholes at nominal 50 m spacing on 25 m spaced north-south oriented sections extending out to 100 m on the peripheries of the deposit. The Krakatoa Zone is sampled targeting pierce points of 25–60 m in the central portion of the deposit to 100 m on the peripheries.

ABM Zone holes were generally angled (–30° to –90°) towards grid south with dip angles set to optimally intersect the mineralized horizon. Approximately 20% of the holes have been drilled vertically. Krakatoa Zone holes were mostly drilled grid southwest and angled at –30° to –90° to avoid the bounding faults. Only one hole was drilled vertically. The orientation of the ABM holes is broadly perpendicular to the mineralization.

A number of geological features including the glacial overburden contact surface, top of fresh rock surface, interpreted faults, mafic and rhyolitic intrusive units, carbonaceous mudstone and the Wind Lake formation cover sequence were modelled for the ABM deposit using drillhole information to assist with resource estimation. Mineralization wireframes were defined primarily by lithological logging of sulphide units and to a lesser extent by Cu, Pb, Zn, Au and Ag assays. Separate mineralization wireframes were defined: 24 wireframes for the ABM Zone and 10 wireframes for the Krakatoa Zone.

Block models constrained by the interpreted mineralized envelopes and geological boundary surfaces were constructed. For the ABM and Krakatoa zones, a parent cell size of 10 m(E) x 10 m(N) x 5 m(RL) was adopted with standard sub-celling to 5 m(E) x 5 m(N) x 2.5 m(RL). Sub-celling was used to maintain the resolution of the mineralized lenses whilst restricting the overall size of the models. Samples composited to 1.5 m (ABM Zone) and 1.0 m (Krakatoa Zone) length were used to interpolate Cu, Pb, Zn, Au, Ag, As, Ba, Bi, Hg, S, Sb and Se grades into the block models using ordinary kriging (OK) interpolation. Block grades were validated both visually and statistically. All modelling was completed using Surpac V6.6 software.

For the ABM deposit, fixed density values were assigned to the block models for each regolith and lithological unit. Fresh felsic rock was assigned a value of 2.76 t/m^3, mafic intrusive rock was assigned a value of 2.80 t/m^3, the mudstone and Wind Lake formation was assigned a value of 2.74 t/m^3, 2.68 t/m^3 was adopted for the rhyolite intrusive (RHYi), and 2.00 t/m^3 for overburden. For the mineralized zones, a tiered approach to the selection of a preferred bulk density value was adopted, and then the bulk density was interpolated into the block model using OK for the mineralized zones and inverse distance cubed (ID3) for the dilution skin. The average bulk densities determined for the ABM stockwork and massive sulphide mineralization were 3.44 t/m^3 and 4.19 t/m^3 respectively, while the average bulk density values for the Krakatoa Zone were 3.86 t/m^3 and 4.09 t/m^3 respectively.

In-ground net smelter return (NSR) values were calculated using assumed metal prices, metallurgical recoveries, smelter terms (including payable factors, concentrate costs and refining charges) and government royalties. No penalties were included. Metal price assumptions were: US$3.50/lb copper, US$1.50/lb zinc, US$1.05/lb lead, US$1,300/oz gold and US$20/oz silver, and an exchange rate of US$0.75 = CAD$1.00. Metal recovery assumptions 92% for copper, 90% for zinc, 70% for lead, 75% for gold (whereby 30% is recovered from copper concentrate, 30% is recovered from lead concentrate and 15% is recovered from zinc concentrate) and 85% for silver (40% from copper concentrate, 30% from lead concentrate and 15% from zinc concentrate).

Based on these assumptions, the formula for the NSR on each block was calculated as:

$$NSR\ US\$/t = (52.84*Cu\_cut) + (9.56*Pb\_cut) + (19.13*Zn\_cut) + (24.41*Au\_cut) + (0.41*Ag\_cut)$$

The US$ NSR was converted to CAD$ using the formula:

$$NSR\ CAD\$/t = (NSR\ US\$/t)/0.75$$

Based on the results of a 2017 Mineral Reserve estimate outlined in the 2017 PFS (CSA Global, 2017), potential open pittable resources were reported above a cut-off NSR of CAD$25/t and potential underground resources reported above CAD$95/t.

To determine the reporting of ABM deposit Mineral Resources as either “open pit” or “underground”, a Whittle™ pit optimization was undertaken. Parameters used for the optimization included:

- Base case metal price assumptions were: US$3.50/lb copper, US$1.50/lb zinc, US$1.05/lb lead, US$1,300/oz gold and US$20/oz silver
- An exchange rate of US$0.75 = CAD$1.00
- Mining recovery of 97%
- Minimum mining width of 25 m
- Overall slope angle of 50°
- Total processing costs (fresh) of CAD$30.60/t
- Plant throughput of 2 million tonnes per annum (Mt/a).

For the ABM Zone, only material reporting inside the selected pit shell (Revenue Factor = 1.00) has been reported above the NSR cut-off of CAD$25/t. For the Krakatoa Zone, mineralized material inside the pit shell has been reported above the NSR cut-off of CAD$25/t, whilst the remainder has been designated as “underground” resource and reported above a cut-off NSR of CAD$95/t.

The ABM Mineral Resource has been classified as Indicated and Inferred and is reported in accordance with the terms set out by the Canadian Institute of Mining, Metallurgy and Petroleum (CIM), as the CIM Definition Standards on Mineral Resources and Mineral Reserves adopted by the CIM council, as amended. The classification level is based upon an assessment of geological understanding of the deposit, geological and grade continuity, drillhole spacing, quality control results, search and interpolation parameters, and an analysis of available density information.

A Mineral Resource is not reported for the GP4F deposit.

1.10 Mineral Reserve Estimate

The Mineral Reserve estimate was prepared for the ABM deposit following the completion of a DFS. This Reserve Estimate has been determined and reported in accordance with NI 43-101 “'Standards of Disclosure for Mineral Projects” (the “Instrument”, June 2011) and the classifications adopted by the CIM Council in November 2014.

The Mineral Reserve estimate for the KZK Project is detailed in Table 1-5. All reserves are classified as “Probable Reserve”, as no Measured Resources have been defined for the Project. Mineral Reserves are inclusive of diluting material that will be mined in conjunction with the Mineral Reserves and delivered to the Process Plant.

Table 1-5: KZK Project Mineral Reserve estimate

Zone/Mine	Category	Ore (Mt)	Cu (%)	Pb (%)	Zn (%)	Au (g/t)	Ag (g/t)
ABM Open Pit	Probable	13.4	0.9	1.5	5.9	1.3	131
Krakatoa Open Pit	Probable	0.6	0.4	3.1	6.3	1.9	246
Total Open Pit	**Probable**	**14.0**	**0.9**	**1.6**	**5.9**	**1.3**	**136**
Krakatoa Underground	Probable	1.7	0.4	2.3	5.0	1.3	147
Total KZK Project	**Probable**	**15.7**	**0.9**	**1.7**	**5.8**	**1.3**	**138**

Notes:

- *The Mineral Reserves in this disclosure were estimated by Karl van Olden, Qualified Person.*
- *The effective date of this Mineral Reserve is 30 June 2019.*
- *Numbers have been rounded to reflect the precision of the estimates.*
- *The Mineral Reserves were estimated using current CIM standards, definitions and guidelines (CIM Council, 2014).*

The Net Smelter Return (NSR) method was used to determine economic mineralization for the KZK Project. NSR values have been calculated from October 29, 2018, long-term consensus metal prices (Table 1-6) using an exchange rate of US$0.792:CAD$1.00, current at the time of commencement of open pit and underground mine design work.

Table 1-6: Commodity prices used in Mineral Reserve estimate

Commodity	Unit	Metal price ($US)
Copper	$/lb	$3.08
Lead	$/lb	$0.94
Zinc	$/lb	$1.10
Gold	$/oz	$1,310.00
Silver	$/oz	$18.42

The Mineral Reserve estimate includes material extracted from the designated open pit and underground excavations that is sourced from the Indicated Mineral Resources only and has a block value greater than the NSR cut-off for the relevant type of mining. All open pit Mineral Reserves are reported to a cut-off NSR value of CAD$29.30/t, while underground Mineral Reserves are reported to a cut-off NSR value of CAD$173.23/t.

All tonnes and grades have been adjusted for planned and unplanned mining dilution and ore loss. Dilution was applied at zero grade.

Reporting and modelling of financial results was completed in June 2019 using current long-term consensus metal prices at June 30, 2019 of US$3.15/lb copper, US$1.10/lb zinc, US$0.95/lb lead, US$1,321/oz gold, US$18.09/oz silver and an exchange rate of CAD$1.00:US$0.78. The Mineral Reserve estimate was reviewed under the revised metal price and exchange rate settings and no adjustments to the calculated Mineral Reserves were considered necessary. The modelling considers all capital, operating and selling costs as defined in the DFS.

1.11 Mining Methods

The ABM deposit will be mined by open pit mining and underground mining methods. Open pit mining of the ABM Zone will be staged into four separate phases to manage overall waste stripping requirements and the adjoining Krakatoa Zone will be mined as a single phase. Underground mining of the Krakatoa Zone beneath the pit will be predominantly via longhole open stoping with paste fill.

Waste material mined from the open pit or underground will be stored permanently on surface. Some waste material will be used for the construction of infrastructure. Further discussion on the waste management strategy is provided in Section 1.13.1.

Open pit mining commences in March 2021 and is planned over a period of 8.6 years, including nine months pre-production mining, plant commissioning and ramp-up. A total of 14.0 Mt of ore will be mined by open pit mining methods.

The underground mine is planned to commence in November 2024 and operate for a period of 60 months, finishing in October 2029. A total of 1.7 Mt of ore will be mined by underground mining methods.

1.12 Recovery Methods

The Kudz Ze Kayah Process Plant and associated facilities have been designed to process run-of-mine (ROM) ore at a rate of 2.0 Mt/a to produce separate copper, lead, and zinc concentrates and tailings; however, the plant will be capable of processing at 270 tonnes per hour (t/h) based on average ore comminution properties and average plant feed grades. The process rate will be varied depending on the grade of the ore.

The process flowsheet consists of the following key stages:

- Crushing, stockpiling and grinding of the ore.
- Pre-float, rougher and cleaner flotation of copper, including regrind of copper rougher concentrate.
- Sequential pre-float, rougher and cleaner flotation of lead, including regrind of lead rougher concentrate.
- Sequential pre-float, rougher and cleaner flotation of zinc, including regrind of zinc rougher concentrate.
- Thickening, filtration, and stockpiling on site of copper, lead, and zinc flotation concentrates. Copper and zinc concentrates will be loaded in bulk onto trucks for transport to port, while lead concentrate will be loaded into sealable containers before transport by truck to port.
- Dewatering of flotation tailings by thickening and pressure filtration.
- Transportation of filtered flotation tailings to the Class A Waste Storage Facility for disposal.

The overall process flow diagram is provided in Figure 1-1.

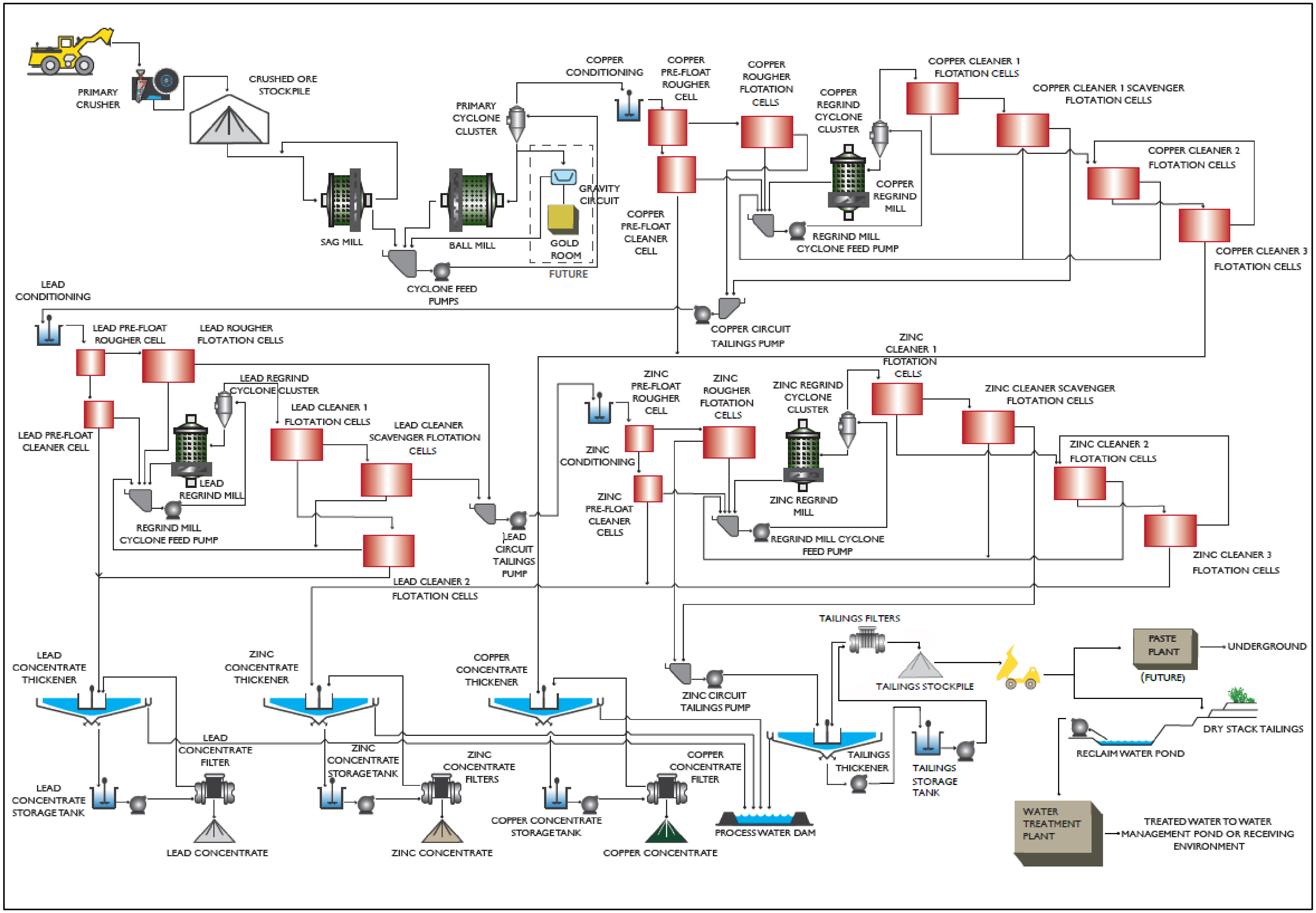

Figure 1-1: Overall process flow diagram

1.12.1 Process Design Criteria

A process design criteria (PDC) and mass balance detail was developed on the basis of annual ore production developed in the PFS. The PDC considered major flows and availability of processes within the facility. The plant will have a design availability of 93% (after ramp-up) which was considered appropriate for a plant of this type and in this location.

1.12.2 Process Description

A Caterpillar 988 front-end loader (FEL) or equivalent will be used to feed the crusher from the ROM pad via a reinforced concrete bridge. The throughput capacity of the crusher was estimated to be 446 t/h, operating 13.5 hours per day.

Crushed ore passes to the Crushed Ore Stockpile via a transfer station. The Coarse Ore Stockpile facility is of the conventional open stockpile type with ore reclaimed via two in-line apron feeders. The Coarse Ore Stockpile has a target live capacity of 12 hours and total stockpile capacity of approximately two days.

The grinding circuit will comprise an open circuit semi-autogenous grinding (SAG) mill and a ball mill in closed circuit with cyclones. Trommel oversize will be returned to the SAG mill feed conveyor by a recycle conveyor system.

A sequential flotation circuit will be used to recover copper, lead and zinc, in that order. Each of the three flotation stages include a pre-float circuit as part of the rougher cells. A three-stage cleaning circuit will be used for copper and zinc flotation, while the lead circuit will operate two-stage cleaning. All three circuits include regrinding to a P_{80} of 30 µm, 30 µm and 35 µm for copper, lead and zinc respectively.

High rate thickeners will be used for concentrate thickening. The copper and lead circuits will each have a 9 m diameter thickener, located inside the plant building, whilst the zinc concentrate duty requires a larger thickener of 14 m diameter which will be located outside.

Vertical plate pressure filters have been selected for concentrate filtration, based on the relatively fine concentrate regrind sizes and the need to consistently achieve the transportable moisture limit. The zinc duty requires two filters operating in parallel to cater for its higher production rate. All filters are standardized to optimize operation, maintenance and spares inventory.

Filtered concentrate (filter cake) will be discharged into individual reinforced concrete bunkers directly below each filter in the Concentrate Storage and Loadout Shed. The copper and zinc filter cake will be removed by FEL and stacked in reinforced concrete bunkers of 5,000 t and 10,000 t respectively. Lead concentrate will be loaded directly into sealed transportable containers. Additional storage capacity of 700 to 1,000t (depending on concentrate type) is available for blending of concentrates to meet concentrate quality requirements.

Copper and zinc concentrate will be bulk loaded into trucks on a weighbridge inside the shed, whilst lead concentrate containers will be loaded directly onto trucks. All trucks will be weighed before leaving site for Stewart port.

A high rate thickener will be used to thicken the flotation tailings stream prior to pressure filtration. The tailings duty requires a 20 m diameter thickener located outside. Two vertical plate pressure filters have been selected to operate in parallel for tailings filtration. The tailings filters will be installed onto an elevated platform within the Process Building. An FEL will be used to reclaim tailings from beneath the filters and load the material directly onto 50-t articulated dump trucks for transport to the Class A Waste Storage Facility.

Majority of the reagent storage and mixing facilities are contained within the Process Building, apart from Lime Slaking and Lead Circuit Depressant 1 (sodium cyanide), which will be contained within separate building structures annexed to the Process Building.

The main operating area of the Process Building is approximately 340 m long and 33 m wide. The Concentrate Storage and Loadout is approximately 54 m long and 78 m wide. The Process Building will be a pre-engineered building, fully clad with profiled insulated sheet metal (sandwich panel type) on the roof and all sides.

A laboratory will be built on site for day-to-day analytical requirements. The laboratory will process around 300 samples per day. Off-site laboratory services will also be used to for less time critical samples from exploration, mining and processing.

An overview of the processing facilities is shown in Figure 1-2.

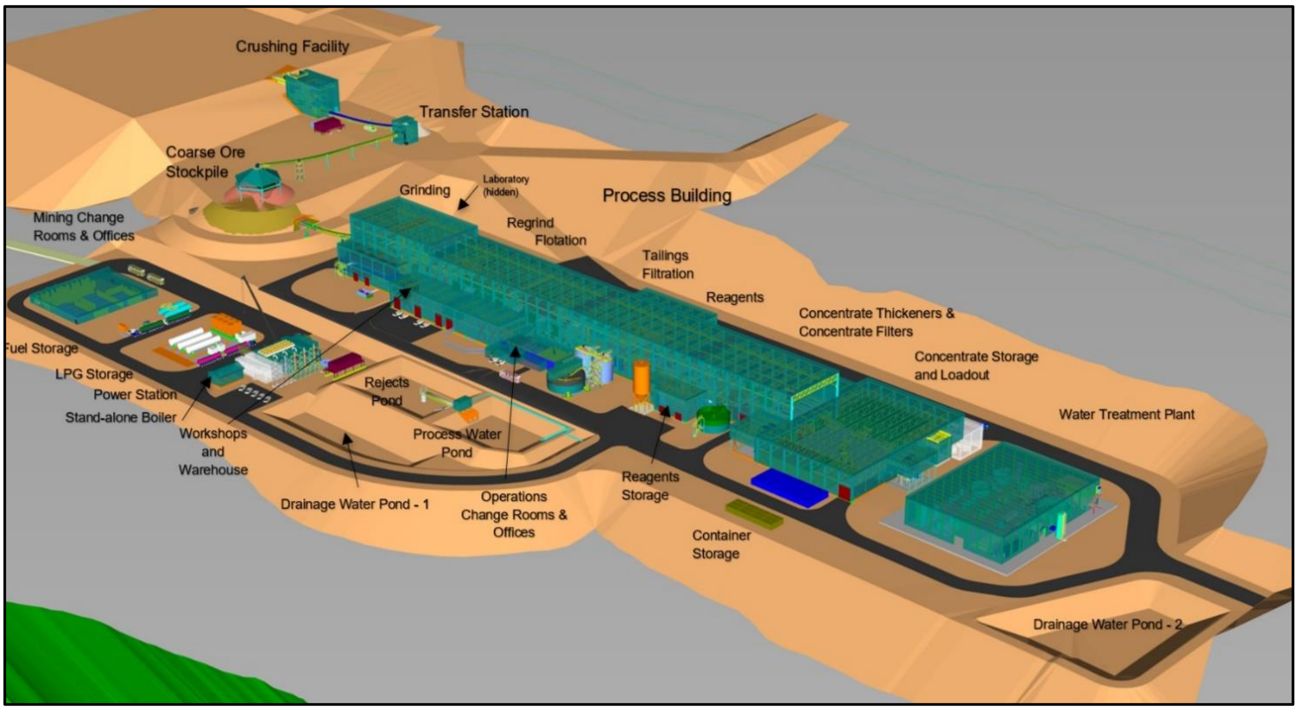

Figure 1-2: Processing facilities overview

1.13 Project Infrastructure

1.13.1 Waste Storage Facilities

Waste rock will be classified as Class A, B or C, based on its potential to produce acid drainage and its metal leaching characteristics and stored accordingly in permanent landforms on surface. In addition, overburden material, comprising glacial till and glaciolacustrine sediments, and topsoil material will both be stockpiled temporarily on surface for later reuse during reclamation of the site.

Class A waste rock is defined as potentially acid generating (PAG) and metal leaching in the near term (i.e. within the life of the operation). The Class A Waste Storage Facility will store all Class A waste rock from the open pit comingled and compacted with filtered tailings trucked from the Process Plant. A buttress will be constructed on the downstream slope of the facility using Class C material. The facility will be fully encapsulated with a liner comprising high-density polyethylene (HDPE) geomembrane and associated low permeability material. During reclamation, the facility will be capped with a minimum of 3 m of Class C material for frost and erosion protection. Overburden and topsoil will be spread over the final formation and the facility will be revegetated.

Class B waste rock is defined as PAG with metal leaching potential over the longer term (after cessation of mining activities). The Class B Waste Storage Facility will use the same composite liner and cover system as the Class A facility. In addition to the closure layer, the Class B facility will include 3 m to 8.5 m of Class C waste rock for frost and erosion protection. Overburden and topsoil will be spread over the final formation before being revegetated.

Class C waste rock is defined as material that is non-reactive or potentially acid consuming, and will have low metal leaching potential; therefore, specific PAG management strategies are not required for this material. Where possible, Class C material will be used for construction purposes around the site as well as capping material for reclamation of the Class A and B facilities. Surplus Class C material will be stored in the Class C Waste Storage Facility. No other encapsulation treatment will be required; however, the facility will be reclaimed by placing a layer of overburden and topsoil material over the final formation surface before being revegetated.

An overburden stockpile will be constructed for the temporary storage of surplus overburden material. Overburden material will be selectively managed and sourced from the stockpile and used during operations as a construction material and in closure for reclamation and select cover material for the other facilities. The stockpile will be completely removed at the cessation of activities and the site revegetated.

Topsoil will be removed and stockpiled from the footprint of each facility prior to its development and construction. Topsoil will be stockpiled around the project areas were terrane can accommodate and will be used in reclamation works.

1.13.2 Water Storage and Management Facilities

A water balance model was developed to assess potential effects of variability on surface and groundwater flows as the project develops. The resulting water management plan involves collecting and controlling site runoff from disturbed areas to ensure maximum recycling of mining and process water and control of water quality discharged to the surrounding environment.

All water in contact with the mine facilities, including the Class A, Class B, Class C, Overburden Waste Storage Facilities, the Open Pit, and the Processing Plant Site and other infrastructure will be collected in water collection ponds for sediment control prior to conveying to either the Upper Water Management Pond or Water Treatment Plant. Both these facilities will then convey water into the Lower Water Management Pond for release to Finlayson Creek and Geona Creek provided contaminants meet project water quality guidelines.

A Pit Rim Pond will be constructed to receive water from the mine workings for the settlement of sediment prior to pumping to the Water Treatment Plant.

A system of diversion drains will be installed around the site to minimize and manage the flow of contact water. A significant diversion ditch will be installed during construction to divert the Fault Creek watercourse to the south to enable Geona Creek to be drained so that mining can extend across the valley floor.

1.13.3 Water Treatment Plant

A Water Treatment Plant will be constructed adjacent to the Process Plant to treat influent water streams from the Class A facility, a portion of the process water stream, ROM pad runoff, process plant sumps, and mine drainage water. Water quality from each of these sources will be monitored to determine if treatment through the Water Treatment Plant is required, and for the purposes of the DFS all these streams were considered to be treated. Treated water will be discharged into the Lower Water Management Pond.

The main contaminants identified as of potential concern for the KZK Project are Se, Al, As, Cd, Cu, Zn and Fe. The proposed treatment system consists of a metals removal circuit and a Selen-IX™ circuit for selenium removal. In Year 6, the Water Treatment Plant metals removal circuit will be modified to include a High-Density Sludge lime neutralization system to treat Class A runoff before it is predicted to turn acidic.

1.13.4 Power Generation and Fuel Supply

Power will be generated onsite by an LNG/Diesel fired Power Station located adjacent to the Process Plant. Up to five 5.5 MW main generators in an N+1 configuration and a diesel generator for black start operation will be installed. Remote facilities such as the camp and pumps in water collection ponds and storage facilities will be powered from individual skid-mounted diesel generators.

Initially, the Power Station will have only three of the five main generators installed, as well as the auxiliary (black-start) generator. At Year 3, the remaining two generators will be added to meet load increases from commencement of the underground mine and surface infrastructure.

Fuel supply and storage on the site includes LNG for the Power Station, and diesel fuel for standalone generators, mining equipment and the Power Station. LNG will be sourced from the vendor’s offsite liquefication facility and trucked to site in LNG tankers. Diesel will also be trucked to site in conventional diesel tankers.

Waste heat from the Power Station will be used by the HVAC system to heat the Process Building through a glycol loop. For the first two years of operation, heat generation will be supplemented by standalone LNG fired boilers until the additional generators are installed.

1.13.5 Mining Infrastructure

Mine workshop facilities will be constructed for both the open pit and underground mining operations by respective mining contractors. A vehicle wash bay will also be constructed adjacent to the mine workshop facilities. The main diesel storage facility, comprising four 100,000-litre tanks will be constructed adjacent to the mine workshop. Explosives will be stored in secure, fenced facilities well separated from the main activity areas. Emulsion agents will be stored in a separate compound, together with the explosive services contractor’s workshop facilities.

1.13.6 Communications

Site communications will be established via microwave link to connect KZK directly to the NorthwesTel terrestrial network through the McEvoy Tower approximately 36 km northeast of the project area. One intermediate repeater site will be required, adjacent to the Tote Road, to relay the signal to the mine site.

1.13.7 Roads

The existing 24 km Tote Road will be upgraded to an all-weather, single lane road with sufficient passing bay pull-outs to safely facilitate two-way traffic. The Tote Road will be the main access to the site for all traffic including personnel transport, supply trucks and concentrate transport trucks.

In and around the site, approximately 25 km of roads will be either upgraded or constructed new in order to services the needs of the operation. This will include the construction of both light vehicle access roads and mine haul roads.

1.13.8 Accommodation Camp

A permanent accommodation camp will be constructed adjacent to the existing exploration camp. It is expected that an initial camp of approximately 100 single-beds will be required for early works, then expanded to 348 dormitory rooms including temporary double-bunks to cater for the peak construction phase.

Catering services at the camp will be outsourced to an experienced catering contractor.

1.13.9 Airstrip

The existing Finlayson airstrip is located approximately 40 km from the site and will be the main facility used to fly all project personnel to and from Whitehorse. It is a gravel strip capable of up to 14-seat capacity aircraft. Personnel will be bussed to site from the airstrip. Should Finlayson airstrip be unable to be used for any reason, contingency options include airstrips at Ross River, Faro and Watson Lake.

1.13.10 Concentrate Haulage

Concentrate will be hauled from KZK to the Port of Stewart along a 905 km southerly route utilizing a combination of the gravel site Access Road, gravelled and sealed secondary highways, and paved primary highways. A portion of the route (Robert Campbell Highway) adopts seasonal payload restrictions of 75% during the spring break up period.

Copper and zinc concentrate will be transported in conventional covered bulk ore style boxes, with modified sealed containers used during seasonal restricted periods. The concentrate will be offloaded at the port, inside the concentrate storage shed, using a truck “Tipper Table”.

Lead concentrate will be transported in sealed containers all year round with the containers being emptied during ship loading. Empty containers will be backhauled by the transport fleet.

The concentrate transport operations will be outsourced, with workshop and support facilities expected to be established at Watson Lake.

1.13.11 Port Facilities

Concentrate will be exported from the port at Stewart using upgraded facilities to be provided by Stewart World Port. The port facilities currently comprise a concrete deck and steel pile jetty with one berth suitable for Handysize and Handymax vessels.

Concentrates from the mine will be stored at a purpose-built storage facility constructed by BMC. Copper and zinc concentrates will be bulk stored until they can be conveyed to ship and discharged into the hold using a ship loader. Lead concentrate will be stored at the port until they can be unloaded inside the ships hold using a container rotating system used in conjunction with the ship’s crane.

1.14 Market Studies and Contracts

Three separate concentrates will be produced during operations: copper, lead and zinc, all with varying levels of precious metal credits and deleterious elements. During the first 18 months of the Project life, all ore will be sourced from a single metallurgical domain. After that time, the mine plan will allow blending of domains. On this basis, two concentrate qualities were estimated for each concentrate product for the purpose of assessing marketability.

All concentrates have been assessed as marketable and are planned to be sold to East Asian markets. Direct marketing has not been completed and BMC has not entered into any contracts for the sale of concentrate at the completion of the DFS. The DFS allows for long term concentrate sales terms with respect to treatment and refining charges, metal payability levels and penalties for deleterious elements. Initial copper concentrate production provides for increased treatment and refining charges to account for higher levels of deleterious elements that are expected to be present in concentrate during the initial commissioning phase of production.

Commodity prices used for the DFS are consensus prices, established by taking the average price forecasts from a range of financial institutions and are presented in Table 1-7.

Table 1-7: Average price forecasts (as at 30 June 2019)

Parameter	Unit	2019	2020	2021	2022	Long term
Copper	US$/lb	$2.94	$3.03	$3.12	$3.31	$3.15
Lead	US$/lb	$0.94	$0.95	$0.95	$0.96	$0.95
Zinc	US$/lb	$1.24	$1.18	$1.15	$1.12	$1.10
Gold	US$/oz	$1,304	$1,335	$1,337	$1,331	$1,321
Silver	US$/oz	$15.74	$16.66	$17.02	$17.46	$18.09
Exchange rate	CAD$/US$	0.758	0.764	0.770	0.779	0.782

1.15 Environmental Studies, Permitting, and Social or Community Impact

The environmental and socio-economic conditions in and around the project area are well characterized. Baseline environmental and socio-economic studies were initiated in 1994/95 by Cominco to support their Initial Environmental Evaluation. These studies included evaluations of climate and hydrology; surface water and groundwater quality; stream sediment quality; aquatic resources (fish, benthic invertebrate and zooplankton characterization); vegetation and terrain mapping; wildlife; archaeological investigation; and socio-economic data collection. Additional baseline studies were conducted in 1996 to support the Type A Water Licence Application. Baseline studies (water quality and aquatic resources) were conducted every two years between 1998 and 2018, to meet the requirements of the water licence. In 2015, BMC initiated a full suite of environmental baseline studies, to support the Environmental Assessment of the KZK Project. The fourth consecutive year of these studies was completed in March 2019.

The KZK Project is subject to an environmental and socio-economic assessment under the *Yukon Environmental and Socio-economic Assessment Act* (YESAA), administered by the Yukon Environmental and Socio-economic Assessment Board (YESAB). BMC submitted a Project Proposal to YESAB on 17 March 2017, which was deemed Adequate in January 2018 and passed through to the Screening stage of assessment, a multi-stage public review process. During the Screening stage, BMC has continued to respond to information requests made by YESAB as the Screening review of the project continues.

At the end of Screening, YESAB will issue a Screening Report to the Yukon Major Projects office and the Department of Fisheries and Oceans (the Decision Bodies). The Decision Bodies will review the Screening Report and will either issue a Decision Document accepting the recommendations or refer the recommendations back to the YESAB Executive Committee for reconsideration.

Once the Screening review of the Project Proposal is completed by YESAB, BMC will submit an application for a Water Licence from the Yukon Water Board, an application for a Quartz Mining Licence under the *Quartz Mining Act*, and other authorizations as required to advance construction and development of the KZK Project.

BMC has initiated consultation and engagement with government agencies, First Nations, various stakeholder groups, and interested parties to introduce the company and to engage and consult these parties regarding the proposed Project. BMC staff meet regularly with Ross River Dena Council and Liard First Nation leadership and officials, as well as holding regular community meetings and providing ongoing financial capacity support to enable First Nation participation in the project development, assessment and permitting. The engagement with First Nations is consistent with and builds upon the existing Socio-Economic Participation Agreement (SEPA).

1.16 Capital and Operating Costs

Pre-production capital costs have been estimated to be CAD$496 million (US$381 million) and sustaining capital costs were CAD$264 million (US$206 million), as summarized in Table 1-8. Total capital over the LOM has been estimated to be CAD$760 million (US$587 million). The capital cost estimate is considered accurate to within the normal limits expected for a FS as defined in the CIM Definition Standards for Mineral Resources and Mineral Reserves. The costs are considered current as of Q4 2018. Cost escalation has not been applied after this date.

Table 1-8: LOM capital cost summary

Capital cost summary	Pre-production (CAD$M)	Sustaining (CAD$M)	Total (CAD$M)
Open Pit Mining	$41	$4	$45
Underground Mining	$0	$81	$81
Process Plant	$197	$13	$211
Water Treatment Plant	$22	$3	$25
Infrastructure	$95	$61	$156
Closure	$0	$102	$102
Subtotal Direct Costs	**$355**	**$264**	**$618**
Owners Costs	$16	Included	$16
Indirect Costs	$78	Included	$78
Subtotal Direct and Indirect Costs	**$449**	**$264**	**$713**
Contingency	$47	Included	$47
TOTAL CAPITAL COST	**$496**	**$264**	**$760**

Notes:

- *Values presented are rounded to the nearest CAD$ 1 million. Totals may not sum precisely.*
- *Currency exchange rate varies over the LOM as detailed in Table 19-3 and averages US$0.77 per CAD$1.00 during Pre-production and US$0.78 per CAD$1.00 during Operations (or the Sustaining Capital period).*
- *Costs are adjusted for asset leasing.*
- *100% equity financing is assumed.*
- *Excludes project finance interest, offtake agreements, reclamation bonding, and other financing arrangements and costs, working capital, exchange rate fluctuations, all licence fees and allowances for special incentives (schedule/safety or others).*
- *Mining of open pit waste is considered to be an operating cost, except for pre-production mining activity.*

LOM operating costs are summarized in Table 1-9 and are considered accurate to within the normal limits expected for a FS as defined in the CIM Definition Standards for Mineral Resources and Mineral Reserves. The costs are considered current as of Q4 2018. The costs presented in Table 1-9 exclude pre-production mining costs that were capitalized and have been included in Table 1-8.

Table 1-9: LOM operating cost summary

Operating cost summary	LOM total (CAD$M)	Unit cost (CAD$/t processed)
Open Pit Mining	$620	$39.42
Underground Mining	$159	$10.10
Processing	$330	$20.96
Water Treatment	$16	$1.04
Administration	$167	$10.60
Road Transport	$354	$22.50
Sea Transport and Port Operations	$212	$13.48
Equipment Leases	$78	$4.96
First Nations (Administration and Profit Share)	$50	$3.18
Royalties	$222	$14.09
Treatment and Refining Charges, Penalties	$679	$43.21
Total Operating Cost	**$2,886**	**$183.53**

1.17 Economic Analysis

Economic analysis of the project as presented in this Technical Report demonstrates that the KZK Project is commercially viable given the Base Case economic results, with key indicators as follows:

- LOM gross revenue of US$4,064 million
- LOM EBITDA of US$1,981 million
- Pre-production capital expenditure of US$381 million (including owners' costs and contingency)
- LOM project free cash flow of US$901 million (after tax)
- Net present value (NPV) (7% discount rate) of US$527 million (after tax June 30, 2019 valuation)
- An internal rate of return (IRR) of 40% (after tax June 30, 2019 valuation)
- Payback period of 2.0 years from commencement of production.

LOM metal production contained in concentrate is detailed in Table 1-10.

Table 1-10: LOM metal production

Metal	Unit	Produced
Zinc	thousand tonnes	786.3
Copper	thousand tonnes	100.2
Lead	thousand tonnes	195.4
Gold	thousand ounces	432.0
Silver	million ounces	59.8

The sensitivity of changes in key project variables on project NPV was determined by simple factoring of these elements. Most variables were assessed on a standard ±10% change. The relative sensitivity to project NPV, for the most sensitive variables, is shown in Figure 1-3.

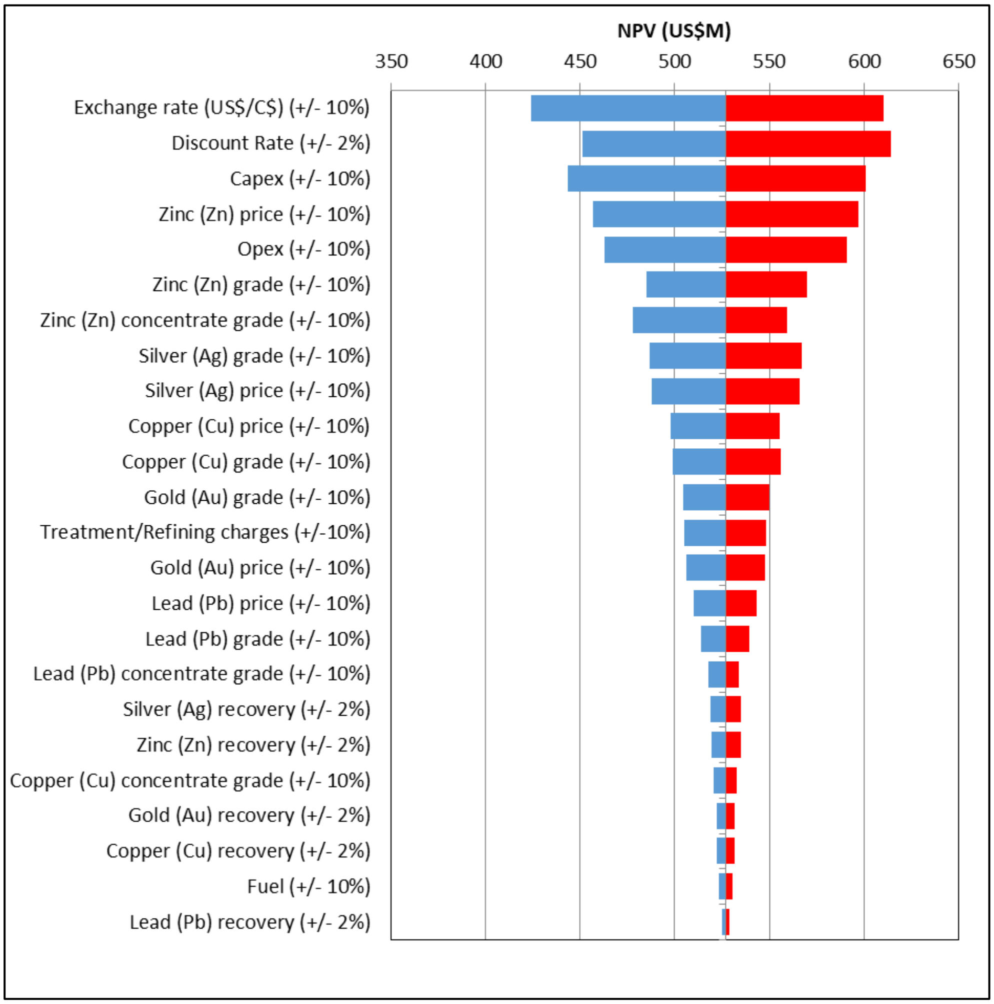

Figure 1-3: Sensitivity to project variables

1.18 Conclusions

The KZK Property is located in a region known to contain economically significant VHMS deposits. The project comprises a wide range of base metal exploration targets from near grass-roots geochemical anomalies, conceptual geological and geophysical targets to drill-ready targets (Fault Creek Zone, northwest ABM, Krakatoa extensions, GP4F) and advanced stage targets consisting of Inferred and Indicated Mineral Resources (ABM and Krakatoa zones). In addition to this, large sections of ground within the KZK Property remain under-explored.

CSA Global considers that data collection techniques are consistent with industry good practice and suitable for use in the preparation of MREs to be reported in accordance with CIM Definition Standards for Mineral Resources and Mineral Reserves. Quality control data supports the integrity of the analytical data which has been utilized.

The ABM deposit remains open (i.e. Krakatoa down-dip extent) and additional drilling is required to fully define the extents of mineralization.

Metallurgical testwork has proved that the Kudz Ze Kayah ore can be processed using a conventional wet grinding circuit followed by sequential flotation to produce marketable concentrates of copper, lead and zinc. The concentrates will also contain significant precious metal credits.

Waste rock will be stored in separate facilities according to expected acid generation and metal leaching potential, with long term closure planning considered from the outset. Tailings from ore processing will be produced as a filtered tailing product that will be deposited in the Class A Waste Storage Facility together with Class A waste rock.

A composite basin liner system incorporating a geomembrane liner will be constructed at the base of storage facilities that have the potential for acid generation and metal leaching characteristics. These waste storage facilities also have a composite liner system placed on top of each facility for closure to minimize exposure to oxygen and water as well as promote active revegetation of the facilities.

All infrastructure has been designed to be situated within a single watershed to minimize impacts on the broader environment. Water that does not come into contact with the KZK Project footprint will be diverted around the site for discharge. Contact water not requiring chemical treatment for discharge will be kept separate from water that does in order to minimize chemical treatment requirements. Reuse of water within the mining and processing facilities will be prioritized to limit the amount of water that will require treatment prior to discharge from the site. A Water Treatment Plant will be constructed to treat water to meet site discharge quality measures.

Concentrates will be hauled to the Port of Stewart, BC for shipping to market. The Portland waterway remains ice-free all year round. The DFS proposes that all concentrates will be sold into the East Asian region at generally standard commercial arrangements for sale of concentrates.

The pre-production capital cost to develop the project is estimated to be CAD$496 million, including a contingency of CAD$47 million. The capital cost estimate has been prepared to an accuracy within the normal limits expected for a FS as defined in the CIM Definition Standards for Mineral Resources and Mineral Reserves. Sustaining capital costs are estimated to be CAD$162 million and closure costs CAD$102 million.

The average operating cost over the life of the project is estimated to be CAD$184/t[2] of ore processed.

The time required to develop the project was estimated to be 31 months from the commencement of initial engineering works. Onsite construction of the KZK Project was estimated to be completed within approximately 13 months.

The KZK Project, as described in this Technical Report is environmentally and technically feasible, delivering a positive case on which the project can move forward. The project presents a viable development scenario for open pit mining of the ABM Zone and upper Krakatoa Zone and underground mining the lower portion of the Krakatoa Zone. Mining will be completed within an 8.6-year period.

1.19 Recommendations

Based on the economic results of this DFS, it is recommended that BMC progress the KZK Project to detailed engineering and construction. During the course of the DFS, the following items were identified as having the potential to further improve the economics of the KZK Project and/or reduce risk to project development and should be pursued as part of the detailed engineering:

- Commence tender process for EPCM contractor to facilitate award of contract as soon as practicable following project sanction. Consideration should be given to the benefits of early contractor involvement.
- Commence negotiations with identified customers for the sale of Quality A concentrates with the view to establishing binding off take agreements as soon as practicable.

[2] Includes all site operating costs, concentrate transportation to smelters, concentrate treatment, refining and penalty charges, government royalties and First Nations payments.

- In order for the KZK Project to progress to higher Mineral Resource classification levels (Measured and Indicated), further infill, or grade control drilling will be required. This drilling should also be planned to improve confidence in the prediction of Class A, B and C waste material classifications.
- A geotechnical drilling program should be completed to support detailed design for the underground mine, prior to commencement of underground mining operations. Given that the underground mine commences approximately 3.5 years after open pit mining operations commence, this work can be completed after the commencement of open pit mining.
- Additional assays of existing variability flotation concentrates for minor elements (Bi, Cd, Hg, Se) are recommended to improve the dataset for predicting concentrate qualities.
- Complete current laboratory treatability testwork for the Water Treatment Plant processes to improve confidence in meeting discharge quality targets.
- Final process plant foundation designs to be confirmed against geotechnical data and water balance models to ensure they are fit for purpose prior to construction.

Further recommendations are provided in Section 26.

2 Introduction

2.1 Issuer

BMC Minerals (No.1) Limited (BMC) commissioned CSA Global Pty Ltd (CSA Global) to compile a Technical Report on the Kudz Ze Kayah Property (“KZK Project” or “KZK Property”) in the Yukon, Canada. This report is to comply with disclosure and reporting requirements set forth in National Instrument 43 101 – Standards of Disclosure for Mineral Projects (NI 43-101), Companion Policy 43-101CP, and Form 43-101F1.

BMC is a private company with its headquarters located in Vancouver, British Columbia (BC). CSA Global is a privately-owned consulting company that has been operating from Perth, Western Australia for more than 30 years.

The principal author of this report is Karl van Olden, CSA Global Mining Manager. Mr van Olden has more than five years’ experience in the field of Mineral Reserve estimation and is a Qualified Person according to NI 43-101 standards.

2.2 Terms of Reference

This Technical Report discloses material changes to the KZK Property including:

- Recent exploration activities
- An updated Mineral Reserve estimate of the ABM polymetallic deposit
- Results of the KZK definitive feasibility study (DFS).

The DFS utilizes the Mineral Resource estimate (MRE) for the ABM polymetallic deposit previously reported in CSA Global’s NI 43-101 Prefeasibility Study Technical Report for the Kudz Ze Kayah Project Yukon Territory, Canada. Report № R103.2017 (CSA Global, 2017).

The DFS is based on mining and processing of the ABM deposit; a polymetallic volcanic-hosted massive sulphide (VHMS) deposit containing economic concentrations of copper, lead, zinc, gold and silver. Mining is planned to be conducted via both open pit and underground mining methods, with ore processed into separate copper, lead and zinc concentrates via sequential flotation through a nominal 2.0 million tonnes per annum (Mt/a) processing plant. Tailings will be deposited in a dry stack tailings facility, while waste rock will be stored according to acid generation and metal leaching potential.

The mine is planned to operate for a minimum of eight years, producing approximately 200,000 dry tonnes zinc concentrate, 60,000 dry tonnes copper concentrate and 50,000 dry tonnes lead concentrate each year of full production. Concentrate will be transported to the port of Stewart in BC for sale to market.

2.2.1 Independence

Neither CSA Global, nor the authors of this report, has any material present or contingent interest in the outcome of this report, nor do they have any pecuniary or other interest that could be reasonably regarded as being capable of affecting their independence in the preparation of this report. The report has been prepared in return for professional fees based upon agreed commercial rates and the payment of these fees is in no way contingent on the results of this report. No member or employee of CSA Global is, or is intended to be, a director, officer or other direct employee of BMC. No member or employee of CSA Global has, or has had, any shareholding in BMC. There is no formal agreement between CSA Global and BMC regarding CSA Global undertaking further work for BMC.

2.2.2 Notice to Third Parties

CSA Global has prepared this report having regard to the particular needs and interests of its client, and in accordance with their instructions and in compliance with NI 43-101 Technical Reporting. This report is not designed for any other person's particular needs or interests. Third party needs and interests may be distinctly different to BMC's needs and interests, and the report may not be sufficient, fit or appropriate for the purpose of the third party, other than its prescription in relation to NI 43-101.

2.2.3 Results are Estimates and Subject to Change

The ability of any person to achieve forward-looking production and economic targets is dependent on numerous factors that are beyond CSA Global's control and that CSA Global cannot anticipate. These factors include, but are not limited to, site-specific mining and geological conditions, management and personnel capabilities, availability of funding to properly operate and capitalize the operation, variations in cost elements and market conditions, developing and operating the mine in an efficient manner, unforeseen changes in legislation and new industry developments. Any of these factors may substantially alter the performance of any mining operation.

2.2.4 Element of Risk

The interpretations and conclusions reached in this report are based on current geological theory and the best evidence available to the author at the time of writing. It is the nature of all scientific conclusions that they are founded on an assessment of probabilities and, however high these probabilities might be, they make no claim for absolute certainty. Any economic decisions which might be taken on the basis of interpretations or conclusions contained in this report will therefore carry an element of risk.

2.3 Principal Sources of Information

The preparation of the Technical Report has been coordinated and completed by CSA Global largely based on information provided by the Issuer (BMC) in conjunction with various specialist, independent consultants required to complete all aspects of the DFS. These consultants included:

- CSA Global Pty Ltd (CSA Global) – Estimation of Mineral Resources, open pit and underground mine design, scheduling, capital and operating cost estimation, and Mineral Reserve estimation
- Dempers & Seymour Pty Ltd (D&S) – Mining geotechnical assessment
- Allnorth Consultants Ltd (Allnorth) – Process Plant and associated site services and facilities engineering design and capital and operating cost estimation, concentrate haulage
- Minnovo Pty Ltd (Minnovo) – Sub-consultant to Allnorth specifically responsible for the Process Plant design and metallurgical testwork
- Knight Piésold Ltd (KP) – Tailings and waste rock storage engineering design, surface water management engineering design and capital and operating cost estimation
- Integrated Sustainability Consultants Inc. (ISC) – Water Treatment Plant design and capital and operating cost estimation
- JDS Mining & Energy Inc. (JDS) – Port concentrate storage facility capital cost estimation
- Braemar Technical Services LLC (Braemar) – Port and shipping operating cost estimation
- StoneHouse Consulting Inc. (StoneHouse) – Concentrate marketing assessment
- Alexco Environmental Group (AEG) – Environmental baseline monitoring, geochemical assessment of waste rock and tailings and environmental management and closure cost estimation

- Tetra Tech Inc. (Tetra Tech) – Groundwater monitoring
- Onsite Engineering (OSE) – Engineering design and cost estimate to upgrade Tote Road to a Mine Access Road.

A full listing of the principal sources of information is included in Section 27 of this report.

2.4 Qualified Person Section Responsibility

This report was prepared by or under the supervision of the Qualified Persons identified in Table 2-1 for each of the sections of this report.

Table 2-1: Qualified Person section responsibility

Section(s)	Section title	Qualified Person
1	Summary	Karl van Olden
2	Introduction	Karl van Olden
3	Reliance on Other Experts	Karl van Olden
4	Property Description and Location	Aaron Green
5	Accessibility, Climate, Local Resources, Infrastructure and Physiography	Aaron Green
6	History	Aaron Green
7	Geological Setting and Mineralization	Aaron Green
8	Deposit Types	Aaron Green
9	Exploration	Aaron Green
10	Drilling	Aaron Green
11	Sample Preparation, Analyses and Security	Aaron Green
12	Data Verification	Aaron Green
13	Mineral Processing and Metallurgical Testing	John Fleay
14	Mineral Resource Estimates	Aaron Green
15	Mineral Reserve Estimates	Karl van Olden
16	Mining Methods	See subsections
16 (except 16.4.1, 16.5.2)	Mining Methods	Karl van Olden
16.4.1, 16.5.2	Open Pit Geotechnical, Underground Geotechnical	Paul Hughes
17	Recovery Methods	John Fleay
18	Project Infrastructure	see subsections
18.1, 18.9	Waste Storage Facilities, Site Roads	Les Galbraith
18.2	Water Storage and Management Facilities	Jaime Cathcart
18.3	Water Treatment Plant	AJ MacDonald
18.4, 18.5, 18.6, 18.8, 18.11, 18.12, 18.14, 18.15	Power Generation and Electrical Distribution, Fuel Supply, Heat Recovery for HVAC, Communications, Accommodation Camp, Airstrip, Concentrate Haulage, Security	Grant Morgan
18.7	Mining Infrastructure	Karl van Olden
18.10	Access Road	Jeremy Araki
18.13	Port Facilities	Bader Diab
19	Market Studies and Contracts	Karl van Olden
20	Environmental Studies, Permitting and Social or Community Impact	See subsections
20.1	Environmental Assessment and Permitting	Karl van Olden
20.2	Environmental Studies	See subsections

Section(s)	Section title	Qualified Person
20.2.1, 20.2.2, 20.2.7, 20.2.8, 20.2.9, 20.2.10	Climate, Terrain, Aquatic Ecosystems and Resource, Wildlife, Archaeology and Heritage Resources, Vegetation and Soils	Karl van Olden
20.2.3, 20.2.5, 20.2.6	Hydrological Assessment, Surface Water Quality, Water Quality Modelling	Cheibany Elemine
20.2.4	Hydrogeological Assessment	Guy Roemer
20.3	Community Engagement	Karl van Olden
20.4	Mine Closure	Cheibany Elemine
21	Capital and Operating Costs	Karl van Olden
22	Economic Analysis	Karl van Olden
23	Adjacent Properties	Karl van Olden
24	Other Relevant Data and Information	Geoff Davidson
25	Interpretation and Conclusions	Karl van Olden
26	Recommendations	Karl van Olden
27	References	Karl van Olden
28	Certificates	Karl van Olden
29	Abbreviations and Acronyms	Karl van Olden

2.5 Qualified Person Site Inspections

A site visit was conducted by Aaron Green (Qualified Person) and Neil Martin (BMC (UK) Limited – Technical Director) from 11 to 13 October 2015 (three days). The purpose of the site visit was to:

- Inspect operating drill rigs
- Review current drilling and sampling procedures
- Verify the location of selected drill collars and downhole surveys
- Inspect site geological data collection systems (mapping, logging etc.)
- Review site geology
- Review sample storage facilities including historical core storage farm
- Discuss quality assurance with geological personnel
- Discuss data storage and review the drillhole database.

Site sample storage facilities and the analytical laboratory in Vancouver (SGS) were also inspected by Aaron Green and Dennis Arne of CSA Global (Vancouver), and Robin Black (BMC – Exploration Manager) on Thursday 22 October 2015 (one day).

A subsequent site visit was conducted Aaron Green, Neil Martin and Robin Black on 26 July 2017 (one day). No active drilling was being undertaken at the time of the site visit. Limited geological outcrops around the ABM deposit were visited as well as the core farm.

A site visit was conducted by Karl van Olden (Qualified Person) and Jim Newton (Chief Mining Engineer – BMC) from 15 to 18 August 2016 (four days). The site visit achieved the following:

- Inspect the proposed mine site.
- Inspect:
  - the site layout
  - access to site
  - proposed mining, processing plant and infrastructure locations and key local features.

- Discuss geological and geotechnical data and interpretation with the on-site technical personnel.
- Discuss environmental and social elements of the Project.
- Discuss mine planning considerations with the BMC technical staff and consultants.

Site visits were conducted by Paul Hughes (Qualified Person) from 28 to 30 July 2017 (three days) and 25 to 29 April 2018 (five days). The purpose of the site visits was to conduct quality assurance and quality control (QAQC) geotechnical logging of selected boreholes for both the open pit and underground mine geotechnical studies. Samples were also selected while on site for a geotechnical laboratory test program.

A site visit was conducted by Geoff Davidson (Qualified Person) on 15 October 2017 (one day). The purpose of the site visit was to inspect the proposed location of the mine and associated infrastructure. View diamond drill core indicative of the subsurface mining environment. Discuss environmental and social elements of the project.

Site visits were conducted by Jeremy Araki (Qualified Person) on 26 May 2015 (one day) and on 12 September 2016 (one day). The purpose of these site visits was to inspect the site for its existing condition and to field check the current road and road infrastructure designs for the requirements to construct and operate proposed infrastructure relating to the project. There were no negative outcomes from the above inspection, and all proposed locations and local features were deemed fit for use in the DFS.

A site visit was conducted by Les Galbraith (Qualified Person) on 9 May 2016 (one day). The purpose of the site visit was to inspect the site for its existing conditions, inspect the proposed facility locations, evaluate locations for geotechnical investigations, and to better understand the requirements to construct and operate the proposed infrastructure.

There were no negative outcomes from any of the above inspections, and all samples and geological data, proposed locations and local features and project planning were deemed fit for use in the DFS.

A site visit was not conducted by John Fleay, Jaime Cathcart, AJ MacDonald, Grant Morgan, Bader Diab, Guy Roemer or Cheibany Elemine. Where required, these Qualified Persons have relied on information from other Qualified Persons or experienced professionals within their respective organizations who have attended the site.

3 Reliance on Other Experts

The Qualified Persons have prepared this Technical Report from a range of sources including their personal work, contributions from other CSA Global and BMC personnel, and from a range of external consultants. Where input has been received from these sources, the Qualified Persons have reviewed and verified the contained assumptions and conclusions. The Qualified Persons do not disclaim responsibility for this information.

With regard to environmental matters reported on in Section 20.2 (excluding Sections 20.2.3, 20.2.4, 20.2.5 and 20.2.6), the primary Author and Qualified Person (Karl van Olden) has relied upon the opinion and information provided by Ms Kelli Bergh of GTK Environmental Management Ltd (BMC, 2019b).

The author and Qualified Person has not reviewed the status of BMC's tenure agreements pertaining to the KZK Property and has relied on information provided by BMC regarding the legal title to the mineral concessions (Section 4.2).

The author and Qualified Person is not qualified to provide comment or opinion on any legal issues associated with the KZK Project. Assessment and reporting of these aspects rely on information provided by BMC and has not been independently verified by CSA Global.

4 Property Description and Location

4.1 Project Location

The KZK Project area (formerly known as the TAG Property) is located on the northern flank of the Pelly Mountain Range, 260 km northwest of Watson Lake and 115 km southeast of Ross River, Yukon (Figure 4-1). The Project area lies approximately 23 km south of Finlayson Lake and 25 km west of the Wolverine Mine (Figure 4-2).

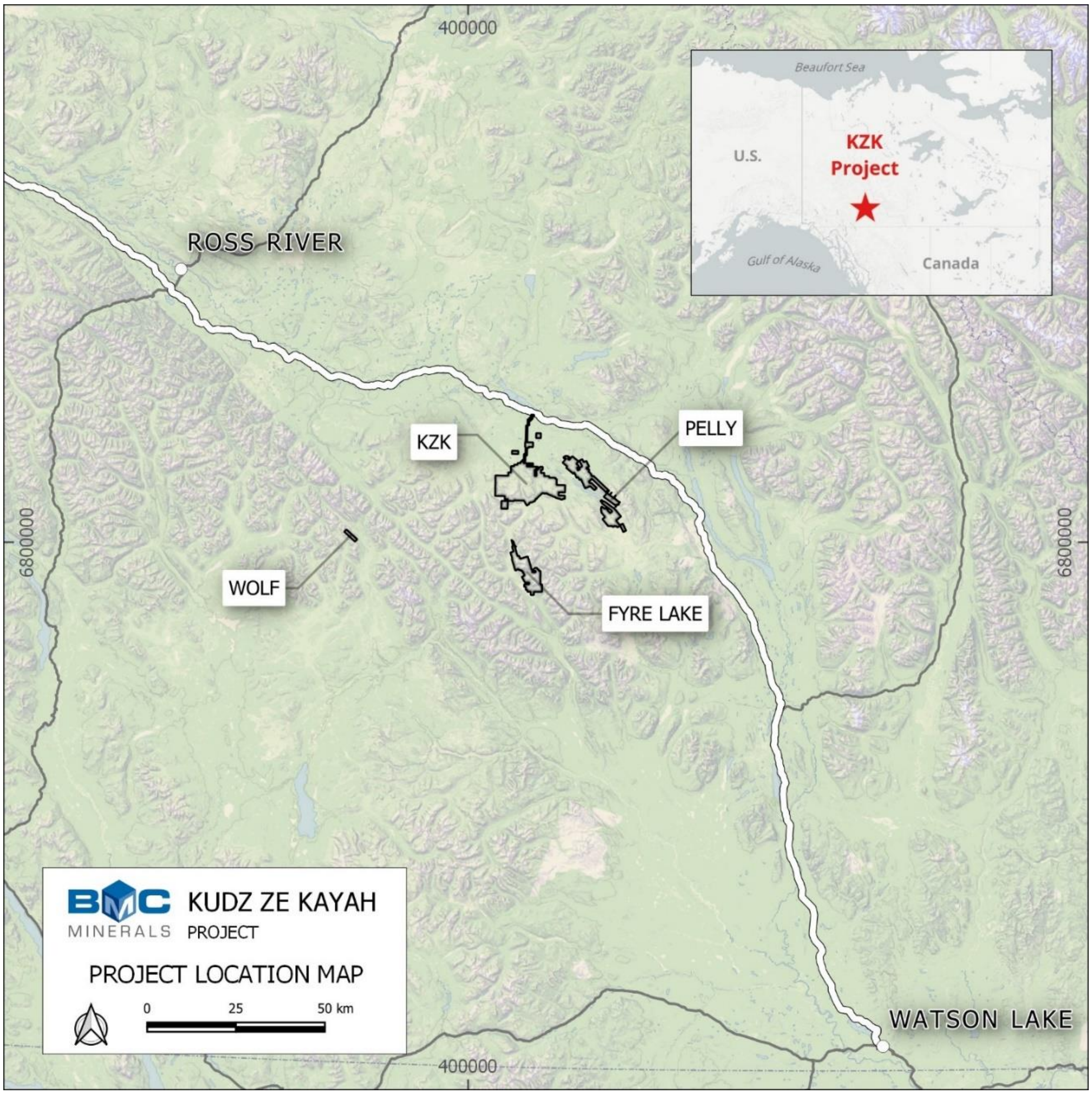

Figure 4-1: BMC Mineral Claim blocks showing the location of the KZK Project

The KZK Property is accessed by the, all-weather Robert Campbell Highway which links the towns of Watson Lake and Carmacks. The highway is multi-surfaced with chip seal from Carmacks to Faro, gravel from Faro to the Nahanni Range Road turn off and chip seal the remainder of the way to Watson Lake. The Property is centred at 61°31’N latitude and 130°33’W longitude (416000E 6817000N, NAD83, UTM Zone 9) on NTS map sheets 105G/7–10, within the Watson Lake Mining District.

4.2 Mineral Tenure and Surface Rights

BMC holds several Mineral Claim blocks in the Finlayson Lake District, either through 100% ownership or under option, and these are shown in Figure 4-1. BMC is 100% owner of the 879 Mineral Claims, covering 161.3 km^2 and held under the *Quartz Mining Act* (Yukon) and associated Regulations, that constitute the KZK Project. These are the subject of this report and are shown in Figure 4-2 and detailed in Appendix 1.

BMC acquired the KZK Project from Teck on 24 January 2015. The reader is referred to Section 6.1 for a discussion of the ownership history of the Project.

CSA Global has viewed a recent tenement summary from the Government of Yukon's Department of Energy, Mines and Resources (EMR), provided by BMC, covering these Mineral Claims and which provided the details of each Mineral Claim (Appendix 1). All contiguous claims over the ABM deposit do not expire until 2 April 2035.

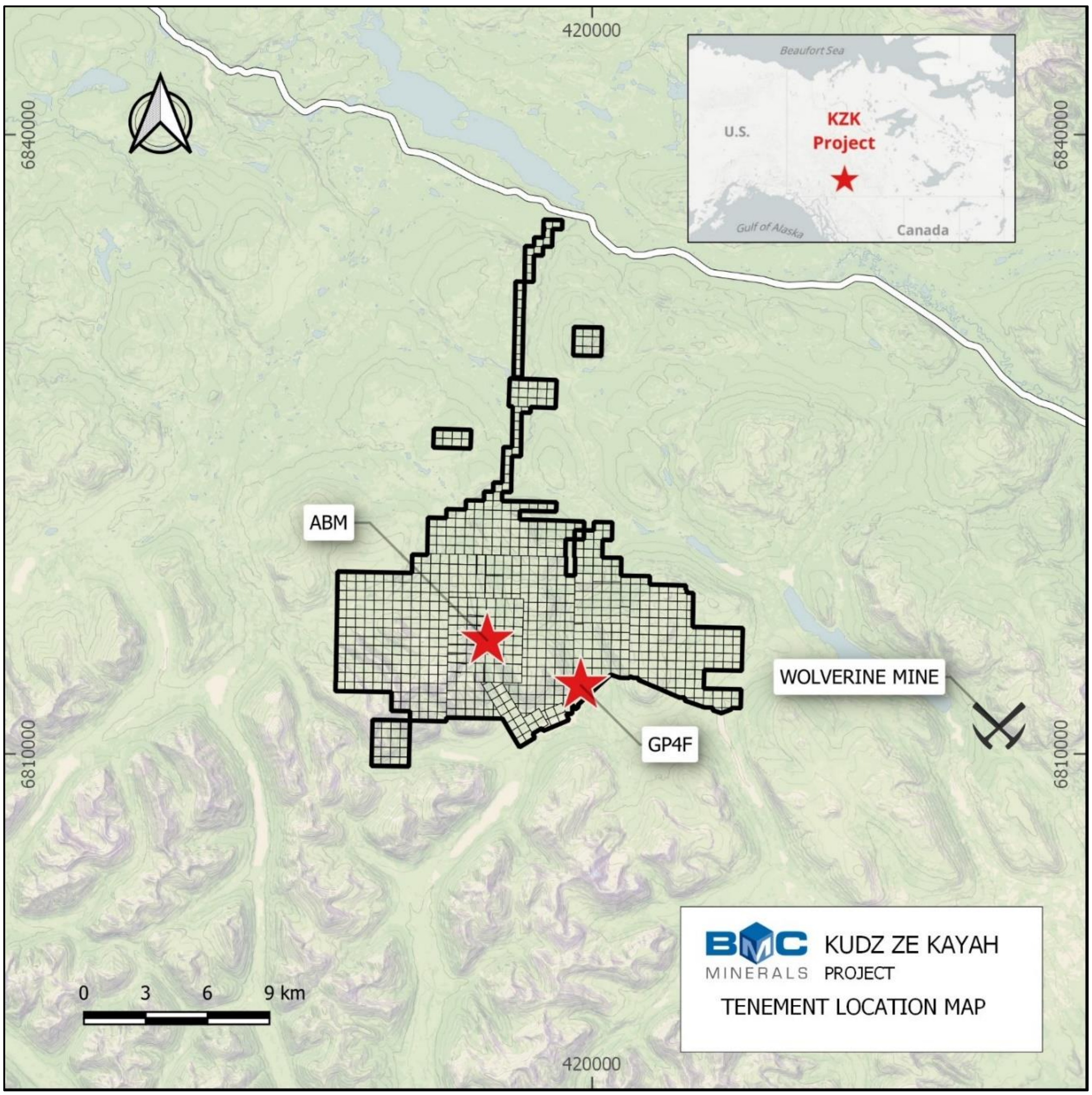

Figure 4-2: Location of the BMC Mineral Claims and the ABM and GP4F deposits

The KZK Property lies within the traditional territory of the Kaska First Nation (comprising Ross River Dena Council, Liard River First Nation, Dease River First Nation and Kwadacha First Nation). Portions of the Property, including the ABM deposit and the proposed infrastructure are covered by a Socio-Economic Participation Agreement (SEPA).

BMC is the 100% owner of Lease 105G07-001, granted under the *Territorial Lands (Yukon) Act* and associated Regulations for a five-year period commencing 1 May 2015. The Lease relates to a 47.24 hectare (ha) parcel of land that covers the Kudz Ze Kayah Access Road (KZK Tote Road) and the Gatehouse. The Lease is renewable in five-year terms, and a security deposit of CAD$260,000 bond has been lodged in relation to the Lease.

Mineral claims confer title to hard rock mineral tenure only. Surface rights are held by the Crown, as administered by the Yukon Territory. Trapping rights over the western portion of the KZK Property are held by the Ross River Dena Council under Group Trapline #405, while trapping rights over the rest of the Property are held under Single Holder Trapline #250. KZK overlaps with part of Outfitter Concession #20, held by Yukon Big Game Outfitters. There are several parcels of land in the vicinity of the KZK Property which have been reserved for a future land claim settlement with the Ross River Dena Council. Staking of new mineral tenure is currently not permitted within the Ross River Dena Council traditional territory surrounding the KZK Project due to a government moratorium. The moratorium is set to expire on 31 July 2019 but may potentially be extended beyond that date.

During a detailed field survey of the wooden stakes utilized to mark out Mineral Claims on the ground, a small number of "Claim Fractions" (i.e. narrow gaps in mineral tenure) were identified. The footprint of the proposed mine, processing facility and related infrastructure as outlined in this Technical Report lie within a fully contiguous set of Mineral Claims (i.e. without gaps), with the exception of a narrow strip of ground that underlies a small portion of the Class C waste facility that is not essential to the viability of the Project. To acquire surface rights over this area, BMC intends to apply for a surface lease under the *Territorial Lands (Yukon) Act*.

4.3 Datum and Projection

All grid coordinates reported herein (unless otherwise specified) use Universal Transverse Mercator (UTM) System Projection, Zone 9 NAD83.

4.4 Royalties

BMC is 100% owner and no third-party royalties or other encumbrances exist over the KZK Mineral Claims covering the Mineral Resources described in this report except as outlined below.

A 1.2% net smelter return (NSR) royalty is jointly held by Teck and Nyrstar NV over the mineral claims ON 21-85, ON 87-101, ON 104-113, ON 116-125 ON 162-173 and ON 197-198. These claims do not cover the Resources described in this report.

4.5 Environmental Liabilities

There has been no mining activity on the KZK Mineral Claims and hence there are no environmental liabilities, other than those which may be related to recent exploration activities undertaken by BMC.

4.6 Permitting

A Type A Water Licence (QZ97-026-1), was granted to Cominco for a period of 20 years on 17 September 1998 and assigned to BMC in January 2015. The Water Licence at the time of grant allowed for construction and operation of the mine as proposed by Cominco and was more recently sufficient for BMC's water use in support of exploration activities. This licence expired on 28 September 2018.

Prior to expiry of the Type A Water Licence, BMC was granted a Type B Water Licence (QZ16-085) on 15 June 2017 to allow for continuity of activities after expiry of the Type A licence. This licence expires on 6 June 2027.

A Class 3 Exploration Permit (LQ00424), covering the contiguous claims over and adjacent to the ABM deposit, (the KZK claim block) was granted to BMC for a five-year period from an effective date of 29 June 2015. This permit was subsequently amended 15 June 2017 to a 10-year period resulting in an expiry of 28 June 2025.

BMC lodged KZK Project Proposal documentation with the Yukon Environmental and Socio-economic Assessment Board (YESAB) on 24 March 2017 and the Adequacy Review phase of the project assessment commenced on 27 March 2017. YESAB deemed the Project Proposal to be adequate on 10 January 2018 and the Project Proposal then entered the Screening phase.

4.7 Other Significant Factors and Risks

Environmental, permitting, legal, title, taxation, socio-economic, marketing, and political or other relevant issues could potentially materially affect access, title or the right or ability to perform work on the Property. However, as of the Effective Date of this report, the author and Qualified Person is unaware of any such potential issues affecting the Property.

5 Accessibility, Climate, Local Resources, Infrastructure and Physiography

5.1 Accessibility

The ABM deposit is located in the upper end of the Geona Creek valley. Road access to the site is via a 24 km-long, 4 m-wide, all weather road (the Tote Road: Figure 5-1) connecting the site to the Robert Campbell Highway.

Figure 5-1: View of the Tote Road (looking south toward the Process Plant and Mining Area)

5.2 Climate and Physiography

The KZK Project occurs within the Pelly River and Pelly Mountain ecoregions. It is located within the northern foothills of the Pelly Mountains of the Yukon Plateau, on the east side of the divide between the Pelly River and the Liard River drainage basin. The topography of the area consists of high rolling hills, locally with ponds and lakes occupying valley bottoms. Elevations range from 1,000 m near Finlayson Lake to over 2,000 m at the peaks of the mountains southwest of the ABM deposit.

The ABM deposit is located at approximately 1,400 m elevation in a broad, gently sloping, U-shaped valley, covered by 2–30 m of glacial overburden. A north-flowing tributary to Finlayson Creek, Geona Creek, drains several small ponds which, in part, overlie the deposit (Figure 5-2). The GP4F occurrence is located on a hillside 5 km to the southeast.

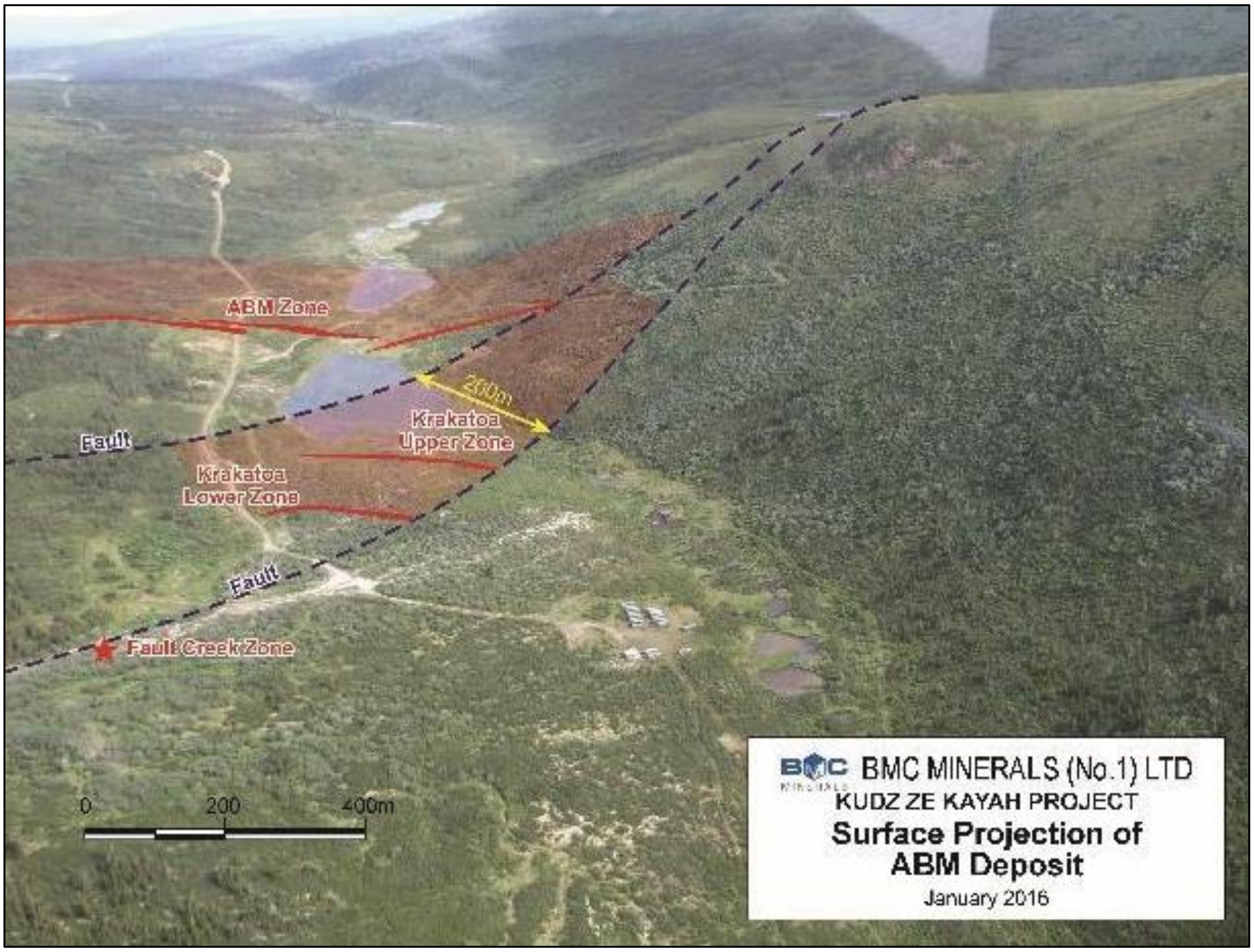

Figure 5-2: Aerial view of the ABM deposit
Deposit outline is projected to surface and Tote Road from the exploration camp, with the old Cominco core yard in the foreground. Source: BMC, 2016

Below the tree line (1,350–1,500 m), white and black spruce are the most common tree types. Black spruce is usually dominant in wetter areas whereas white spruce predominates in drier areas. Paper birch, aspen, balsam and lodgepole pine are also present. Alpine fir grows near the tree line. In dense coniferous stands, feather moss dominates the understory, but in more open areas, willows and heath-like shrubs become prevalent.

Sedge or sphagnum tussocks are common in wetlands and under black spruce. Shrub birch and willow occur in the subalpine and extend well above the tree line. The region has intermittent permafrost with moist depressions comprising peat plateaus, patterned fen and bog complexes.

The climate in the area is typically sub-arctic, characterized by cold winter temperatures (minimum mean monthly temperature of –13°C) and low snowfall (Figure 5-3). Summer is generally mild with maximum mean monthly temperature of 10°C. Precipitation falls fairly even throughout the year, predominantly as rain from

May through September, and snow for the balance of the year. The long-term mean annual precipitation is 520 mm.

Groundwork on the Property is possible year-round; however, snow can impact activity and may cover parts of higher elevations late into the summer. Drilling can be conducted year-round by using heated water for drilling during the colder months. During the production phase, the operations will be conducted on a 24-hour, 7-day basis year-round. Habitable buildings will be appropriately insulated and air conditioned to enable this operation.

Figure 5-3: View of the valley hosting the ABM deposit along the Tote Road (looking north)
Source: Green, 2015b

5.3 Local Resources and Infrastructure

The KZK Project area is considered a relatively remote site. The nearest local population centres include the following:

- Ross River approximately 130 km north of the Tote Road with a population of 409 (Yukon Government (YG), 2019)
- Faro approximately 190 km north of the Tote Road with a population of 410 (YG, 2019)
- Watson Lake approximately 230 km south of the Tote Road with a population of 1,482 (YG, 2019).

Travel through the area is primarily by road along the Robert Campbell Highway (Highway 4), 24 km north of the Project area, and is the major transport route between local centres. Each of the local population centres are also serviced by an airstrip, with Watson Lake being the only sealed runway. Air travel for the Project will be via the Finlayson airstrip, located adjacent to the Robert Campbell Highway, approximately 14 km north of the Tote Road. Alternative airstrips are available at Faro and Watson Lake. No other transport infrastructure exists in the area.

The closest grid power is located at Ross River. Given the distance to this facility, it was determined to be more viable to generate power on site for the KZK operation.

Water for the operations will be sourced from groundwater produced from dewatering the mine and surface run-off captured within the footprint of the operation.

Through consultation with local communities and First Nations, BMC has identified that unemployment and underemployment within local communities is high (BMC, 2018). BMC therefore propose to preferentially hire local people for the operation. Scholarship and training programs are already underway to encourage participation and improve local skills. Regionally, BMC propose the point of hire for most personnel for the operations will be in Yukon, with transportation to site being a combination of road and/or air transport. Recent mine closures have meant that experienced labour is currently available in Yukon. Contractors and other third parties will provide majority of labour for the Project, which may be sourced either locally or from further afield as required.

Sufficient suitable areas are available for the permanent storage of all tailings and waste rock that will be produced during the operations. The storage areas are located along the side of the valley above Geona Creek and the proposed landforms have been determined to be permanently stable.

A number of locations for the Process Plant were considered during the PFS, with the final location decided during the DFS. The proposed site for the Process Plant will be located on the western side of the valley above Geona Creek, approximately 1.5 km north of the mining operation. There is enough area available to construct all required infrastructure for the Process Plant; however, earthworks will be required to create a level and stable site on which to construct these facilities.

BMC has full title over the proposed mining area. Further discussion on surface rights is provided in Section 4.2.

6 History

6.1 Project and Exploration History

Cominco conducted a geochemical survey in 1977 across the Finlayson Lake area; however, the survey was wide spaced and did not reveal evidence of any VHMS deposits.

Cominco's interest in the area was reignited in 1992 when soil and silt geochemical sample results from a reconnaissance program confirmed and expanded upon anomalous silt sample data released in the Geological Survey of Canada's regional geochemistry silt survey for NTS map sheet 105G, Open File 1648 (Hornebrooke and Friske, 1988).

In 1993, a follow-up program within the anomalous drainage resulted in the location of a well mineralized, layered massive sulphide cobble by A.B. Mawer. At the same time potential host rocks for the mineralized float were recognized. A reconnaissance transient electromagnetic (EM) geophysical survey was immediately implemented over the projected trace of the prospective units where they disappear beneath quaternary cover in the valley floor. This survey identified an EM feature representing a possible source for the mineralized float. The first TAG claims were subsequently staked and recorded on 20 August 1993 to cover the geophysical anomaly. A magnetic survey was also carried out during staking. Further magnetic, horizontal loop electromagnetic (HLEM), soil surveys and staking were completed later that fall and successfully defined a drill target.

The target was drilled in April 1994 with the first hole completed on 20 April intersecting 22.5 m of massive sulphide rock in two zones. Three additional holes were drilled in April; each intersecting mineralization over significant widths. The weighted average grade of sulphides in the discovery hole was 0.5% Cu, 2.8% Pb, 10.0% Zn, 278 g/t Ag and 2.9 g/t Au over 22.5 m. The sulphide body was named the ABM deposit by the exploration team in recognition of A.B. Mawer's contribution towards the discovery and a distinguished career with Cominco. By the end of 1994, 52 drillholes totalling 8,485 m were completed along with ground and airborne geophysical surveys, detailed mapping in the vicinity of the deposit, regional and detailed exploration geochemistry, and baseline environmental sampling. In 1995, an additional 133 drillholes totalling 16,178 m were completed at the ABM deposit and on regional targets. Additional exploration soil sampling, minor geological mapping and ground geophysical surveys were completed. Geotechnical investigations, detailed engineering/mine planning, bulk metallurgical sampling, environmental monitoring and archaeological studies were well underway or completed, as well as the construction of a 24 km all-weather Tote Road from the Robert Campbell Highway. A preliminary feasibility study (PFS) was completed in July 1995.

Although majority of the work by Cominco post-dating discovery of the ABM deposit was focused on mining studies and support for permitting, additional exploration work was being undertaken to identify additional resources nearby. From the period 1994 to 1999, Cominco completed 286.0 line km of TEM, 53.7 line km of HLEM, 35.8 line km of gravity and 45.1 line km of ground magnetic surveys in addition to the collection of 2,856 soil samples and property wide mapping at 1:20,000 scale (Holroyd, 1995; Vanderkly, 1995; Schultz, 1995). Cominco also completed 12,362 line km of DIGHEM airborne surveys covering the KZK Property as well as a large part of the Finlayson District outside of the property boundaries. The HLEM surveys were carried out using an Apex MaxMin I-10 system, with a 100 m coil separation. The HLEM readings were taken at 25 m intervals along the lines and four frequencies (440 Hz, 880 Hz, 1,760 Hz and 3,520 Hz) were recorded.

The magnetics survey was carried out using GEM GSM-19 magnetometers. A base station was established at the KZK camp and the total field magnetic readings were corrected for diurnal variations. The base and field magnetometers were synchronized to record simultaneously. Total field magnetic readings were taken at 12.5 m intervals along the grid lines.

Gravity readings were taken with a LaCoste Romberg gravity meter, Model “G”, S/N 494. Base stations were established on the grid and by utilizing base station readings (at least two per day), all gravity readings were corrected for diurnal drift and levelled to this common base. Gravity readings were corrected for latitude and elevation (including both free-air and Bouguer corrections). The data was then processed for a Bouguer density of 2.67 g/cc.

All soil and silt samples were analysed for Cu, Pb, Zn, Ag, As, Cd, Co, Ni, Fe, Mo, Cr, Bi, Sb, V, Sn, W, Sr, Y, La, Mn, Mg, Ti, Al, Ca, Na and K by inductively coupled plasma (ICP) Selected samples were analysed for Au by aqua regia decomposition/atomic absorption spectrophotometry (AAS) and Ba by x-ray fluorescence (XRF) at Cominco Exploration Research Laboratory (CERL) in Vancouver. Soil samples were typically collected at 100 m spacing in either contour or wide-spaced grid lines with grid soil coverage on the eastern portions of the property where prospective geology occurs.

At the end of 1997, a total of 168 exploration drillholes and 15 metallurgical holes had been completed in the immediate ABM deposit area and another 20 drillholes were completed elsewhere on the KZK Property. This drilling campaign resulted in the identification of the Fault Creek Zone within a kilometre of the ABM deposit. The discovery hole (K97-181) targeted a HLEM/magnetic feature and intersected a 6.4 m thick interval of 5.15% Cu, 1.02% Pb, 5.59% Zn, 140.5 g/t Ag and 2.4 g/t Au from 12.9 m depth (MacRobbie and Bannister, 1998). Follow-up drilling with two 50 m step-outs along strike and a single step-out 200 m down dip did not intersect additional massive sulphide.

Cominco’s 1998 exploration program resulted in the delineation of an additional occurrence of mineralization; GP4F, located approximately 5 km to the southeast of the ABM deposit. The prospect was first drilled in 1995 by Cominco (DDH K95-167) to follow up a HLEM/magnetic geophysical anomaly; however, the hole was completed without intersecting significant mineralization (approximately 15–20 m above the GP4F mineralized horizon). Nine diamond holes were drilled in the 1998 program and all intersected the GP4F mineralized horizon.

Cominco Ltd. (1998) reported an Inferred MRE for GP4F of 1.5 Mt grading 6.4% Zn, 3.10% Pb, 0.10% Cu, 90 g/t Ag and 2.0 g/t Au and confirmed the potential for other VHMS deposits in the area (Expatriate, 1999 Annual Report). A Qualified Person has not undertaken sufficient work to classify the historical estimate as a current Mineral Resource and CSA Global does not treat the historical estimate as a current Mineral Resource.

In March 2000, Cominco announced an agreement in principal to sell the KZK Project to Expatriate Resources Ltd (Expatriate). This agreement resulted in Expatriate controlling most of the favourable stratigraphy in the Finlayson Lake District. Expatriate amalgamated the ABM and the neighbouring 60% owned Wolverine deposits into the “Finlayson Project”. A positive FS was completed by Hatch Pty Ltd (Hatch). Additional drilling was completed by Expatriate on the Wolverine deposit and ground geophysical surveys (TEM, magnetics) were completed in the area south of the ABM deposit, extending their survey south of the Fault Creek target but no significant results were encountered.

In September 2001, Expatriate allowed the option with Teck Cominco (formed from the merger of Teck Resources and Cominco) for the KZK Project to lapse.

BMC acquired the KZK Project from Teck on 24 January 2015.

6.2 Previous Mineral Resource Estimates

CSA Global and BMC are not treating the following historical resources, as listed in Table 6-1 as current Mineral Resources; they are presented for informational purposes only. The Qualified Person has not undertaken sufficient work to classify the historical resources as current mineral resources, or comment on the reliability of the estimates. The current MRE for the ABM deposit presented in this report supersedes all past estimates and benefits from the additional information summarized in Section 14.

Numerous historical MREs for the ABM/KZK deposit were completed "in-house" by Cominco. Very little documentation describing the estimation methodology has been identified. It appears that the major difference between the reported tonnages of the early estimates relates to different bulk density values applied to the mineralized zone, whether the resource was reported as "global" or "in pit", and in part whether the peripheral resource defined by widely-spaced drilling (approximately 100 m) was included or not.

MREs prepared by Cominco in 1995 were conducted during their PFS and were described as "mineable in-pit" resources, hence the lower reported tonnages. Subsequent resource estimates were also reported according to the then current NI 43-101 standards by Teck Cominco in 2001 and 2006. It is likely that the 2001 and 2006 resource estimates reported by Teck Cominco are restatements of the Cominco resource estimate completed in 1998.

CSA Global completed an MRE in 2014 as part of BMC's due diligence process. This MRE was subsequently publicly reported on 18 January 2016. Following the 2015 field season, an updated MRE undertaken by CSA Global was reported by BMC on 22 January 2016.

Table 6-1 summarizes the previous ABM resource estimates.

Table 6-1: Summary of previous MREs for the ABM deposit (no cut-off)

Company	Reporting code	Method	Classification	Tonnes (Mt)	Cu (%)	Pb (%)	Zn (%)	Ag (g/t)	Au (g/t)
Cominco (November 1994)#1	N/A	Sectional	-	14.04	0.88	1.33	5.61	125.0	1.17
Cominco (November 1994)#1	N/A	ID2	-	13.85	0.97	1.30	5.45	126.9	1.19
Cominco (June 1995)#2	N/A	ID2#8	-	11.4	1.01	1.64	6.4	143	1.46
Cominco (1998)#3	N/A	ID2#8	-	11.3	0.90	1.50	5.90	133	1.30
Teck Cominco (2001)#4	N/A	ID2#8	-	11.3	0.90	1.50	5.90	-	1.30
Teck Cominco (2007)#5	N/A	ID2#8	-	12.8	0.81	1.7	5.9	-	1.38
CSA Global (2015)#6	JORC	ID2	Inferred	13.9	0.9	1.6	6.1	140	1.4
CSA Global (January 2016)#7	JORC and CIM	OK	Indicated	16.7	0.9	1.8	6.2	144	1.4
			Inferred	3.4	0.7	2.8	7.1	191	1.5

Notes:
- *N/A – Not applicable. Historical estimates prepared prior to the adoption of NI 43-101 reporting standards.*
- *JORC – Joint Ore Reserves Committee Code (2012 Edition).*
- *CIM – Canadian Institute of Mining, Metallurgy and Petroleum Definition Standards (CIM, 2014).*
- *ID2 – Inverse distance squared.*
- *OK – Ordinary kriging.*
- *#1 Cominco Ltd, 1994.*
- *#2 Cominco Ltd, 1995.*
- *#3 Cominco Ltd, 1998.*
- *#4 Teck Cominco Ltd, 2001.*
- *#5 Teck Cominco Ltd, 2007.*
- *#6 Green, 2015b.*
- *#7 Green, 2016. Includes Krakatoa Zone discovery.*
- *#8 The estimation method is assumed but could not be confirmed.*

6.3 Historical Production

No past mining or production has occurred at the KZK Property.

7 Geological Setting and Mineralization

7.1 Regional Geology

The KZK Project is located with the Finlayson Lake District, a crescent-shaped area approximately 300 km long and 50 km wide that extends from Ross River in the north to Watson Lake in the south (Figure 7-1).

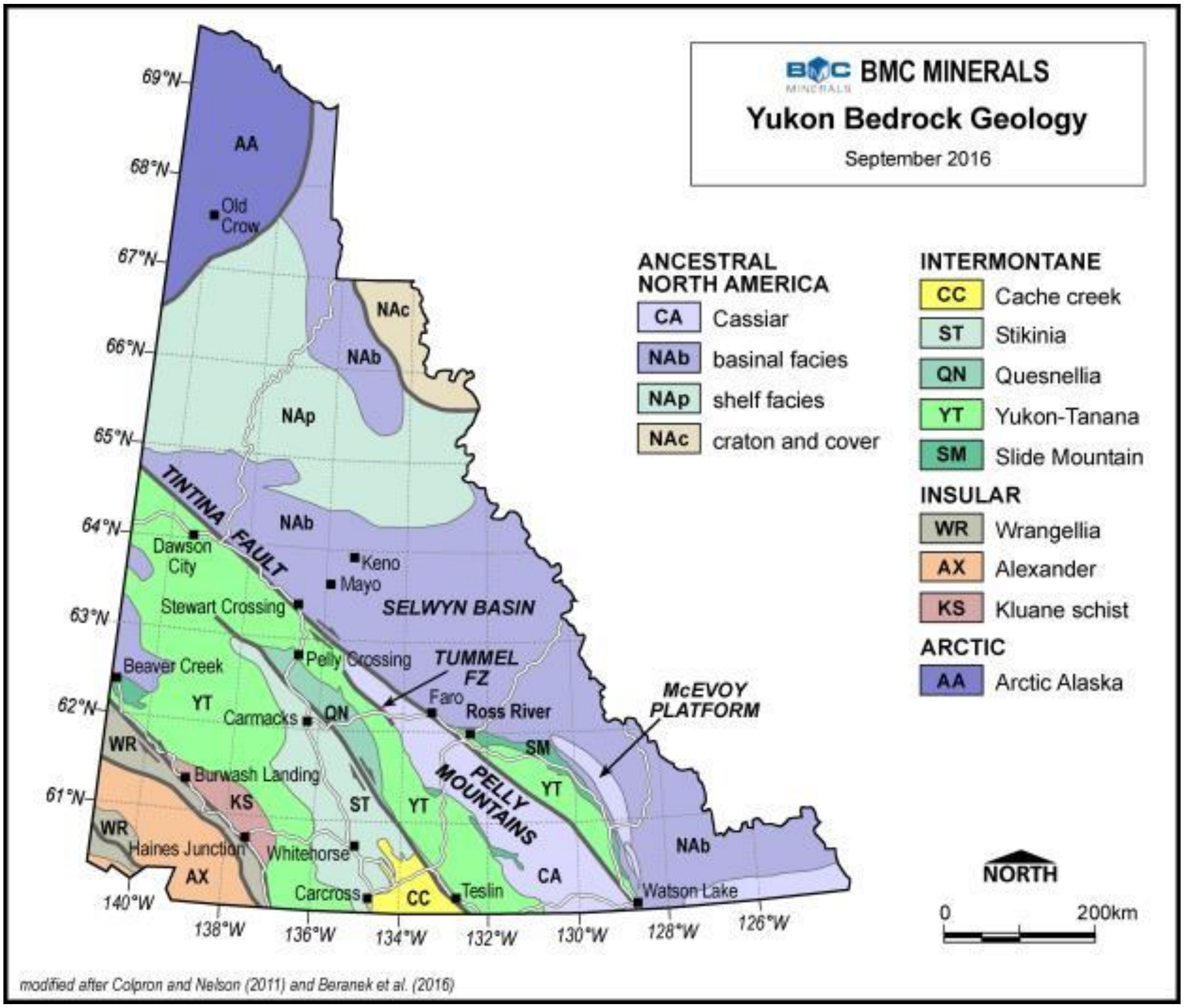

Figure 7-1: Yukon bedrock geology and terrane map

Modified after Colpron and Nelson (2011) and Beranek et al. (2016)

The Finlayson Lake District predominantly comprises Devonian to Lower Carboniferous (Mississippian) volcanic, intrusive, and sedimentary rocks bounded to the east by Proterozoic and Palaeozoic strata of the Selwyn Basin, representing the ancient North American continental margin, and to the southwest by the Tintina Fault. Rocks of the Finlayson Lake District comprise several fault- and unconformity-bound groups and formations of early Mississippian to Early Permian age (Murphy et al., 2006) (Figure 7-2 and Figure 7-3). The Yukon-Tanana and Slide Mountain terranes, which together with minor allochthonous elements that make up the Finlayson Lake District, are separated from the ancient continental strata to the northeast by the Inconnu Thrust (Mortensen and Jilson, 1985; Plint and Gordon, 1996; Tempelman-Kluit, 1979; Figure 7-2).

Within the Finlayson Lake District, the Jules Creek Fault separates the Yukon-Tanana terrane from the Slide Mountain terrane. The Yukon-Tanana terrane of the Finlayson Lake District is interpreted to be contiguous with the main body of the Yukon-Tanana terrane, which underlies most of west central Yukon, after reconstruction of an approximately 425 km right-lateral, strike-slip movement of Late Cretaceous age along the Tintina Fault (e.g. Mortensen, 1992; Peter et al., 2007).

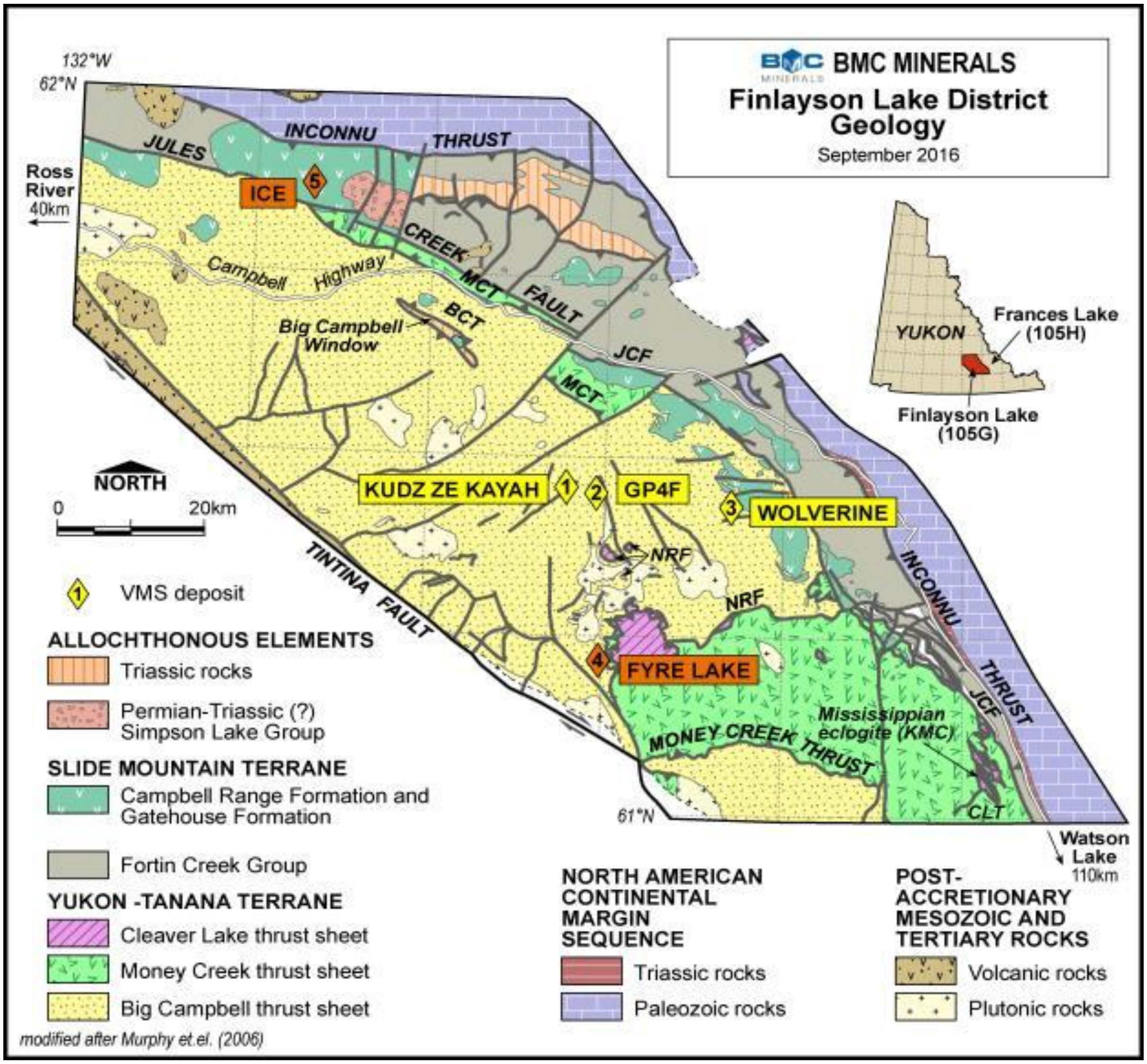

Figure 7-2: Tectonostratigraphic subdivisions of the Finlayson Lake District
Source: Murphy et al. (2006)

Massive sulphide deposits of the Finlayson District are primarily hosted within stratigraphic components of the Big Campbell thrust sheet (Figure 7-2 and Figure 7-3). The exception is the Ice deposit, which is hosted within the Campbell Range formation, a stratigraphic component of the Slide Mountain terrane.

Rocks of the Big Campbell thrust sheet include Pre-Late Devonian quartz-rich sedimentary rocks of the North River formation; mafic and felsic volcanic, and carbonaceous clastic rocks of the Upper Devonian Grass Lakes group; Late Devonian to Early Mississippian granitic rocks of the Grass Lakes plutonic suite; carbonaceous clastic and mafic and felsic volcanic rocks of the Lower Mississippian Wolverine Lake group; and carbonaceous clastic rocks and chert of the Lower Permian Money Creek formation (Murphy et al., 2006) (Figure 7-3).

The Grass Lakes Group comprises strongly foliated and lineated layered sedimentary and volcanic rocks positioned in a roof setting above and between bodies of Early Mississippian granitic orthogneiss and weakly foliated mid-Cretaceous granite (Murphy, 1997). The Grass Lakes Group has been subdivided into three formations which, from oldest to youngest, are the Fire Lake formation, Kudz Ze Kayah formation, and the Wind Lake formation (Peter et al*.,* 2007). Each formation is described below:

- The Upper Devonian (c. 365 Ma) Fire Lake formation is a mafic volcanic sequence comprising mainly chloritic phyllite with some carbonaceous phyllite and rare muscovite-quartz phyllite of probable felsic volcanic protolith. Intrusions and sills of mafic and serpentinized ultramafic plutonic rocks occur within the Fire Lake formation (Peter et al., 2007).
- Stratigraphically overlying the Fire Lake formation is the Late Devonian (c. 360–356 Ma) Kudz Ze Kayah formation, a dominated by what is interpreted as sequence of felsic volcanic and volcaniclastic and sedimentary rocks. It predominantly comprises feldspar-muscovite-quartz phyllite and augen phyllite of probable felsic volcanic and volcaniclastic origin, and lesser fine-grained carbonaceous and siliciclastic sedimentary rocks (Peter et al., 2007).
- The Wind Lake formation forms the uppermost unit of the Grass Lakes Group and comprises carbonaceous phyllite, quartzite, and chloritic phyllite of probable alkalic mafic volcanic and intrusive protolith (Peter et al., 2007).

Coeval with the Kudz Ze Kayah and Wind Lake formations are peraluminous plutonic granitoids of the Grass Lakes Suite which are interpreted as the subvolcanic intrusive equivalents to the felsic volcanic host rocks of the ABM deposit and are as old as 363 ± 3.3 Ma (Mortensen, 1992). These rocks are deformed and were intruded by younger, late-kinematic plutonic rocks prior to deposition of the Wolverine Lake Group (Peter et al*.,* 2007).

The Grass Lakes Group is unconformably overlain by rocks of the Wolverine Lake Group (Figure 7-3), and comprises a basal unit of conglomerate, grit, sandstone, and carbonaceous argillite, a middle unit of quartz-feldspar phyric felsic volcanic rocks, rare chert and sandstone, and an upper unit of aphyric rhyolite, argillite, magnetite iron formation, and mafic volcanic and intrusive rocks (Murphy et al*.,* 2006; Peter et al*.,* 2007).

A second unconformity separates the Wolverine Lake Group from the overlying carbonaceous clastic rocks (carbonaceous phyllite, chert-pebble conglomerate, quartzofeldspathic sandstone to pebble conglomerate, and locally, matrix-supported diamictite) and dark grey to black chert of the Lower Permian Money Creek formation (Peter et al*.,* 2007).

Both the Grass Lakes Group and Wolverine Lake Group occur in the footwall of the Money Creek thrust and record two cycles in the evolution of a Late Devonian to early Mississippian ensialic back-arc (Murphy and Piercey, 2000; Piercey et al*.,* 2001, 2006). The unconformity separating these groups marks a period of deformation, uplift, and erosion (Peter et al*.,* 2007).

Uranium-lead geochronology places an upper age limit of 356.9 ± 0.5 Ma for the host rocks to the Wolverine deposit (Mortensen, 1992; Piercey et al*.,* 2008), and the immediate stratigraphic hangingwall is dated at 346 ± 2.2 Ma (Piercey, 2001), indicating that Wolverine is younger than Kudz Ze Kayah (Peter et al*.,* 2007).

The Campbell Range formation is a mafic-dominated sequence comprising basalt, chert, and argillite which unconformably overlies rocks of the Wolverine Lake Group. Radiolarians and c. 273–274 Ma U-Pb ages on gabbros and plagiogranites indicate a Pennsylvanian to Permian age (Murphy et al*.,* 2006; Peter et al*.,* 2007).

The rocks of the Finlayson Lake District indicate formation and emplacement in a variety of tectonic settings, including rifted frontal arc, continental back-arc, and oceanic back-arc that range in age from 365 Ma to 275 Ma (Peter et al*.,* 2007).

The Finlayson Lake District is characterized by a central core of higher-grade metamorphic rocks (to lower amphibolite facies) surrounded by lower grade, low greenschist facies rocks. This relatively simple metamorphic distribution is interpreted to be a consequence of Cretaceous dynamothermal events (Murphy, 2004). Also occurring during the Cretaceous was the emplacement of late syn- to post-kinematic granite plutons that are regionally associated with W-Mo-Au-Bi occurrences and the Tsa Da Glisza emerald occurrence north of the Kona deposit.

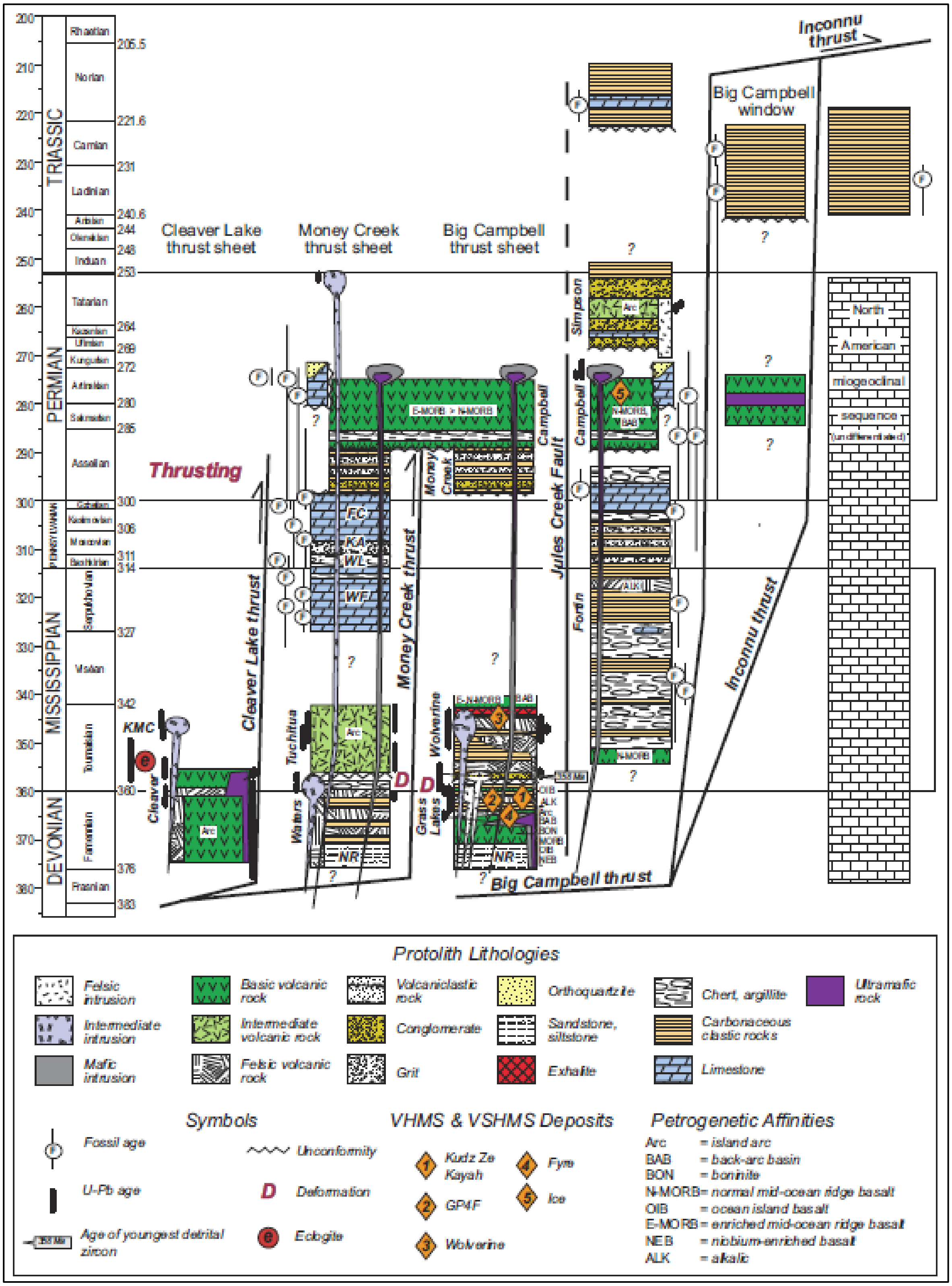

Figure 7-3: Structural and stratigraphic relationships in the Finlayson Lake District

Abbreviations are as follows: FC=Finlayson Creek limestone; KA=King Arctic formation; KMC=Klatsa metamorphic complex; NR=North River formation; WF=Whitefish limestone; WL=White Lake formation.

Source: Peter et al. (2007) modified after Murphy et al. (2006)

7.1.1 Regional Mineralization

The Finlayson Lake District hosts numerous base metal sulphide deposits that collectively contain in excess of 45 Mt of base and precious-metal rich sulphide mineralization (Green, 2016; Traynor, 2005; Tucker et al., 1997), and these occur as a result of four distinct mineralization events. The principal deposits and their tectonic setting (Figure 7-4) are summarized below:

- The Besshi-type Kona (Fyre Lake) Cu-Co-Au-rich massive sulphide deposit is interpreted as the stratigraphically oldest of the massive sulphide deposits, hosted within mafic volcanic rocks of the Fire Lake formation. Kona is interpreted to be situated at the transition from mafic volcanic rocks to overlying turbiditic sedimentary rocks emplaced in a fore-arc setting (Hunt, 2002; Peter et al., 2007).
- The Kuroko-type ABM Zn-Pb-Cu-Ag-Au-rich massive sulphide deposit occurs within the predominantly felsic volcanic rocks that comprise the Kudz Ze Kayah formation, stratigraphically above Kona.
- The Bathurst-type Wolverine Zn-Pb-Cu-Ag-Au-rich massive sulphide deposit is hosted by rhyolitic volcanic rocks and carbonaceous argillite of the Wolverine formation at a stratigraphic position above a regional unconformity and higher than both the ABM and GP4F deposits (Hunt, 2002; Tucker et al., 1997). As the deposit is hosted by graphitic shales and lesser felsic volcanic and volcaniclastic rocks, it may alternatively be classified as a volcanic-sediment-hosted massive sulphide (VSHMS) deposit (Peter et al., 2007).
- The Cyprus-type Ice Cu-Au-rich massive sulphide deposit is hosted within late Palaeozoic mafic volcanic and associated sedimentary rocks of the Campbell Range formation (Hunt, 2002; Peter et al., 2007).

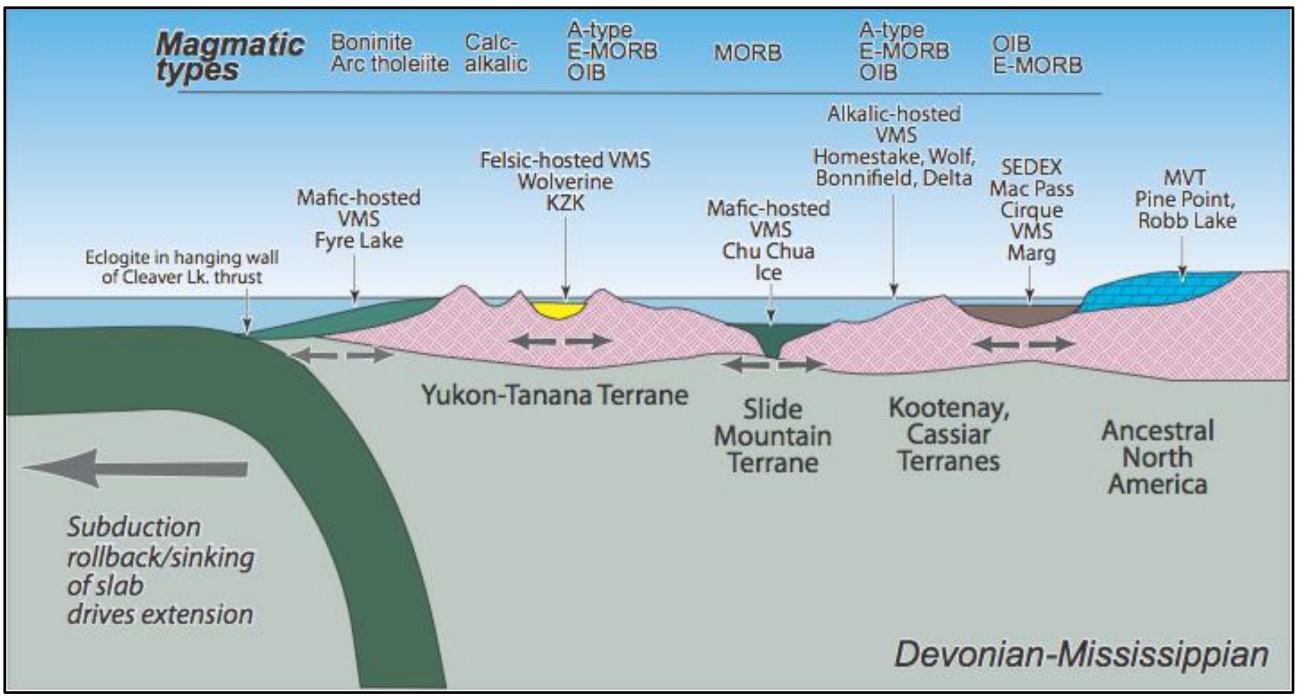

Figure 7-4: Interpreted tectonic setting of Devonian-Mississippian mineral deposits in the Yukon-Tanana and adjacent terranes
Source: Piercey, 2015

Although interpreted to have been deposited in similar tectonic environments, a range of recent studies undertaken on the mineralization and host rocks of the Wolverine and ABM deposits highlight key features which clearly differentiate the two mineralization events, as summarized in Table 7-1.

Table 7-1: Summary of key mineralization event features

<table>
<tr><th>Features</th><th colspan="4">Wolverine deposit</th><th colspan="4">ABM deposit</th><th>Comments</th></tr>
<tr><td>Age</td><td colspan="4">347–346 Ga
(Piercey et al., 2008)</td><td colspan="4">362.4–362.9 Ga
(Manor et al., 2018)</td><td>Approximately 16 Ma age difference and separated by mapped unconformity</td></tr>
<tr><td>Metal source
(Pb isotopes)</td><td colspan="4">1.7 Ga (Proterozoic)
(Peter et al., 2007)</td><td colspan="4">approximately 3.14 Ga (Archaean)
(Peter et al., 2007)</td><td>Different basement metal sources</td></tr>
<tr><td>Host rocks</td><td colspan="4">Carbonaceous shale dominant</td><td colspan="4">Felsic volcanic & volcaniclastic dominant</td><td>Markedly different host rocks to mineralization</td></tr>
<tr><td>Mineral Reserve</td><td colspan="4">5.2 Mt @ 9.7% Zn, 1.3% Pb, 0.9% Cu, 282 g/t Ag, 1.4 g/t Au
(Yukon Zinc, 2007)</td><td colspan="4">15.7 Mt, 5.8% Zn, 1.7% Pb, 0.9% Cu, 138 g/t Ag, 1.3 g/t Au
(Section 15)</td><td>Essentially same range of metals – similar fluids associated with formation</td></tr>
<tr><td>Selenium source
(Se isotopes)</td><td colspan="4">Footwall and coeval carbonaceous shales (broad and strongly negative $\delta^{82}Se_{NIST}$)
(Layton-Matthews et al, 2013)</td><td colspan="4">Magmatic (leaching or degassing) (narrow and mildly negative $\delta^{82}Se_{NIST}$)
(Layton-Matthews et al, 2013)</td><td>Different host rock/basement source control on selenium content of mineralization</td></tr>
<tr><td rowspan="3">Se content, mineralization (ppm)</td><td>N=</td><td>Min.</td><td>Max.</td><td>Mean</td><td>N=</td><td>Min.</td><td>Max.</td><td>Mean</td><td rowspan="3">Selenium content of Wolverine approximately 7.4 times higher</td></tr>
<tr><td>171</td><td>1</td><td>4,605</td><td>1,161</td><td>3,181</td><td>0.5</td><td>2,620</td><td>157</td></tr>
<tr><td colspan="4">Wolverine contains clausthalite
(Bradshaw, et al., 2008; Layton-Matthews et al, 2008)</td><td colspan="4">(BMC drill dataset)</td></tr>
</table>

7.2 Property Geology

The Project area, comprising the Kudz Ze Kayah claim blocks within which the ABM deposit is located, encompasses units of the Grass Lakes group. The surface geology of the property was initially mapped by Cominco in 1996 at 1:20,000 scale (Schultz & Hall, 1997) and subsequently by BMC at 1:20,000 scale in 2017 (Baker et al., 2017). It is from the latter report that the following summary is primarily derived.

Within the KZK Property, the stratigraphy trends easterly, dips moderately to the north and is interpreted as predominantly right way up except locally where it may be overturned in tight, mesoscale F2 folds. The rocks are generally overprinted by greenschist facies metamorphism and a penetrative deformation fabric. The property has been crosscut by north to northeast-trending brittle faults (e.g. Fault Creek and East Creek faults) which cut across all stratigraphic units. These faults can be traced as trends in aeromagnetic data for kilometres to tens of kilometres, and structures of similar timing and orientation can be readily identified in regional mapping datasets.

All known massive sulphide mineralization on the KZK Property (e.g. ABM deposit, Fault Creek occurrence, GP4F Zone) occurs within the Kudz Ze Kayah formation, albeit at what appear to be different stratigraphic levels.

The major components of Grass Lakes group stratigraphy on the KZK Property (Figure 7-5) include, from stratigraphic base to top:

1. Fire Lake formation: comprising mafic flows and ultramafic intrusions;
2. Kudz Ze Kayah formation: comprising
    a. A lower sedimentary component of the Kudz Ze Kayah formation, and
    b. An upper felsic volcanic component, which is host to the VHMS mineralization; and
3. Wind Lake formation: equivalent to “upper sedimentary and mafic volcanic sequence” of MacRobbie and Holroyd (2000), and DMm and DMcp units of Murphy et al. (2001).

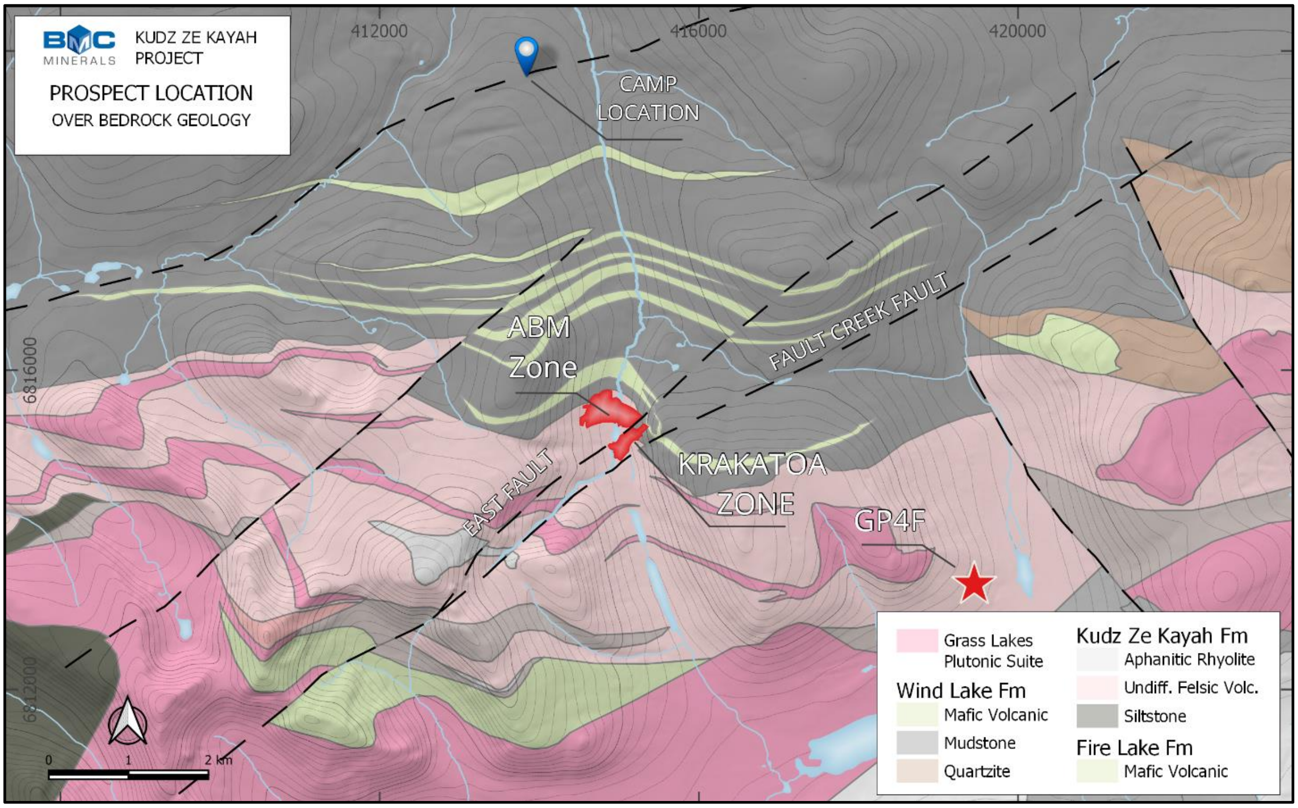

Figure 7-5: Property-scale bedrock geology map

Within the property, the lower sedimentary component of the Kudz Ze Kayah formation primarily comprises siltstone, phyllitic schist, light grey quartzite and massive tuffaceous wacke, with minor mudstone, carbonaceous mudstone and felsic tuff, all of which are interfingered with feldspar porphyry intrusions.

The felsic volcanic component of the upper Kudz Ze Kayah formation has, on the basis of relict primary textures, been broadly subdivided into units comprising felsic volcaniclastics, coherent felsic flows and intrusives, and aphanitic to feldspar-quartz porphyritic intrusives. A relatively minor component of mafic intrusive rocks and mudstone is also present. Rhyolitic volcaniclastic units appear most abundant, comprising thinly bedded tuffs, lapilli tuffs and crystal tuffs that are locally intercalated with thin layers of mudstone. With increasing grain size these fine-grained felsic tuffs grade into crystal-rich ash tuffs containing up to 35% light grey to white feldspar and clear grey quartz phenocrysts. Units of what are interpreted as rhyolitic flows and intrusives are thickest, most abundant and best exposed at Rhyolite Peak, southwest of the ABM deposit. These coherent units typically appear siliceous and aphanitic in outcrop. Some outcrops along Fault Creek contain up to 70% spherulites ranging from 5 mm to 25 mm in diameter. Feldspar-quartz porphyry intrusions are characterized by 15–30% feldspar and/or 5–20% quartz phenocrysts set in a very fine-grained quartz-sericite-feldspar ± chlorite-ankerite groundmass. The feldspar phenocrysts are euhedral, light grey and range from 3 mm to 18 mm in size, whereas the quartz phenocrysts are subhedral, clear grey and 2–4 mm in diameter. The coherent porphyry units are likely cogenetic with the adjacent volcaniclastic units.

Mafic intrusions are present within the Kudz Ze Kayah formation, the best example of which is a large mafic sill in the footwall of the ABM deposit and comprising strongly foliated porphyroblastic amphibole-chlorite-biotite-calcite schist. Other examples of mafic intrusion comprise coarse-grained mafic schist containing less deformed zones displaying relict gabbroic texture.

Occurrences of biotite-rich units with quartz, feldspar and/or calcite have previously been interpreted as both mafic intrusive and sedimentary rocks but are reinterpreted now as “pelite” on the basis of their mineralogy and typically finely intercalated contacts with enveloping felsic volcanic rocks. Mafic tuffs, comprising fine-grained chlorite-calcite phyllitic schist, occur as a minor component of the Kudz Ze Kayah formation.

The base of the Wind Lake formation lies approximately 200 m stratigraphically above the ABM deposit and the Wind Lake formation underlies the majority of the northern half of the KZK Property. The formation comprises carbonaceous and calcareous mudstone (MDS) intercalated with mafic volcanic rocks (MAFt, MAFw) along with minor quartzite, siltstone, chert and felsic volcanic rocks.

Recent preliminary high-precision uranium-lead zircon dates (Manor et al., 2018) taken from the hanging wall of the ABM deposit provide an upper limit to the age of mineralization and felsic volcanism within the Kudz Ze Kayah formation to c. 362.8 Ma. The transition from felsic volcanism in the Kudz Ze Kayah formation to alkalic mafic volcanism in the Wind Lake formation is similarly constrained to c. 362.4 Ma (Manor et al., 2018).

7.3 Deposit Geology

7.3.1 Summary

The ABM deposit (comprising the ABM Zone and Krakatoa Zone) primarily comprises continuous, shallow-dipping massive sulphide mineralization hosted within a thick felsic package of volcaniclastics and coherent sill/flow complex that locally make up the Kudz Ze Kayah formation (Figure 7-6).

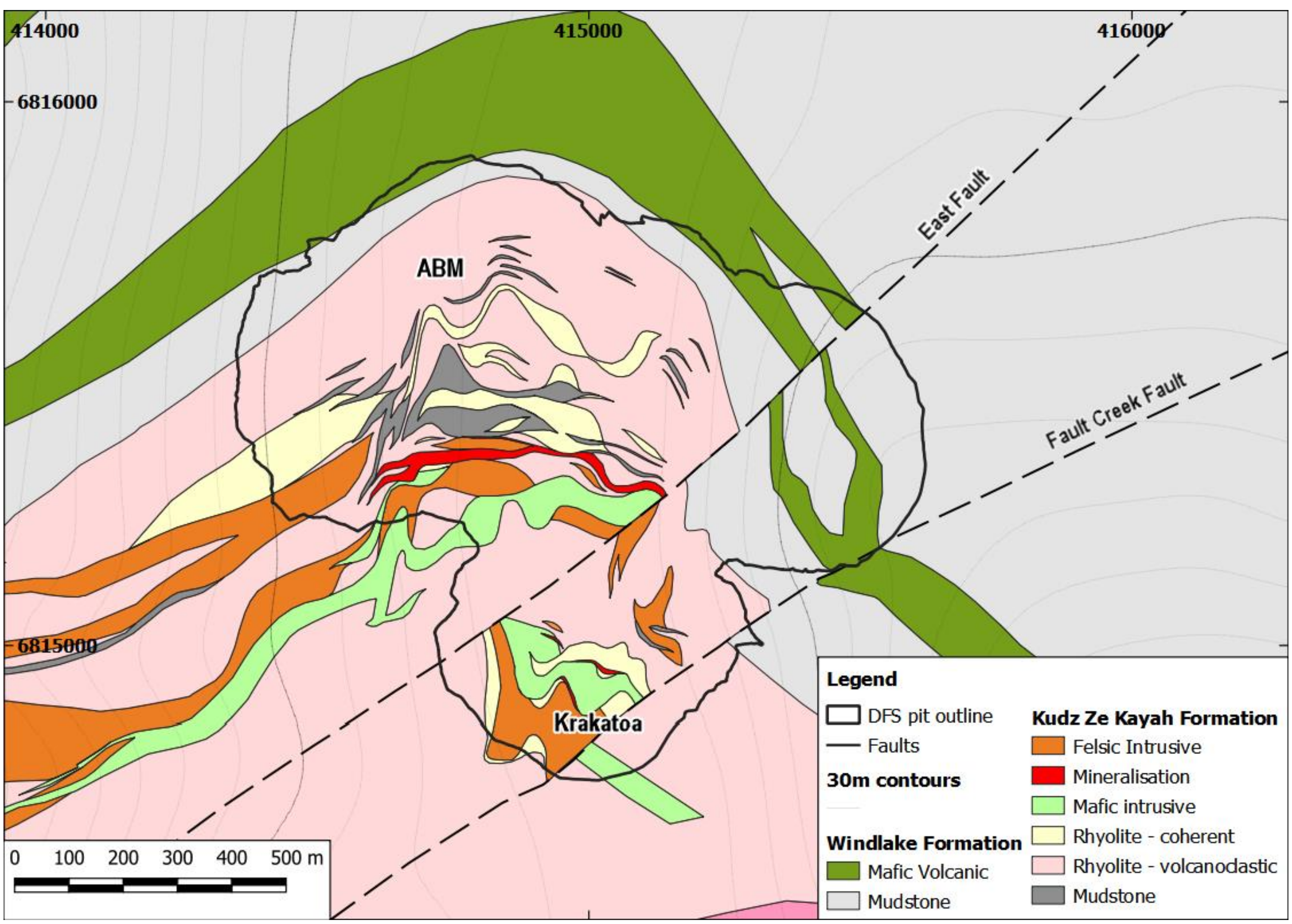

Figure 7-6: Surface geological map of the Kudz Ze Kayah deposit area

Massive sulphide of the ABM Zone is hosted within a felsic rock package, whereas the Krakatoa Zone is predominantly hosted by a pre-mineralization mafic sill located within the felsic volcanic package. Mineralization at Krakatoa also occurs in the felsic hangingwall units stratigraphically overlying the mafic sill, in what is broadly interpreted to be the equivalent of the ABM mineralized position. Only scattered vein-style and disseminated mineralization occurs within the mafic sill lying stratigraphically below the ABM Zone.

The upper limits of the ABM and Krakatoa Zones are truncated near surface and overlain by glaciofluvial sediments. The massive sulphide mineralization at ABM occurs under approximately 2–20 m of glaciofluvial overburden and is up to approximately 30 m in true thickness, whereas the Krakatoa Zone occurs under approximately 30 m of glaciofluvial overburden and is up to approximately 22 m in true thickness. The down-dip margin of the ABM Zone appears to transition into a mixed and variably carbonaceous felsic volcano-sedimentary package, whereas the Krakatoa Zone currently remains open at depth beyond the down-dip extent of the mafic sill.

A post-mineralization brittle fault zone (East Fault) offsets the ABM and Krakatoa zones, and angular clasts of sulphide are to be found within the fault zone breccias (Figure 7-7). The south-eastern margin of Krakatoa is cut by another late brittle fault zone of the same generation (Fault Creek Fault). There exists a marked difference in the stratigraphy east of the Fault Creek Fault, with recent drilling having identified a package to the east of the fault, dominated by felsic volcaniclastics and minor felsic intrusives, which transitions conformably up into the overlying Wind Lake formation. The marked change in volcanic stratigraphy across the Fault Creek Fault, despite little of no evidence of a significant offset of the Wind Lake formation basal contact, could potentially indicate the presence of a syngenetic fault structure along the south-eastern limit of the Krakatoa Zone. Recent attempts at a reconstruction of the ABM Zone and Krakatoa Zone into a single massive sulphide unit are also not consistent with the degree of movement along the East Fault that is observed at the Wind Lake formation basal contact. These features could indicate the presence of syngenetic fault structures either side of the Krakatoa Zone that was later the locus of late-stage brittle faulting. As such, the area east of the Krakatoa Zone (Figure 7-8) remains a significant exploration target. A schematic geological cross-section through both the ABM and Krakatoa zones (Section B-B’ in Figure 7-8) is shown in Figure 7-9.

Figure 7-7: Angular to sub-rounded primary sulphide and sulphidized clasts within East Fault breccia (hole K15-292)
Source: Green, 2015b

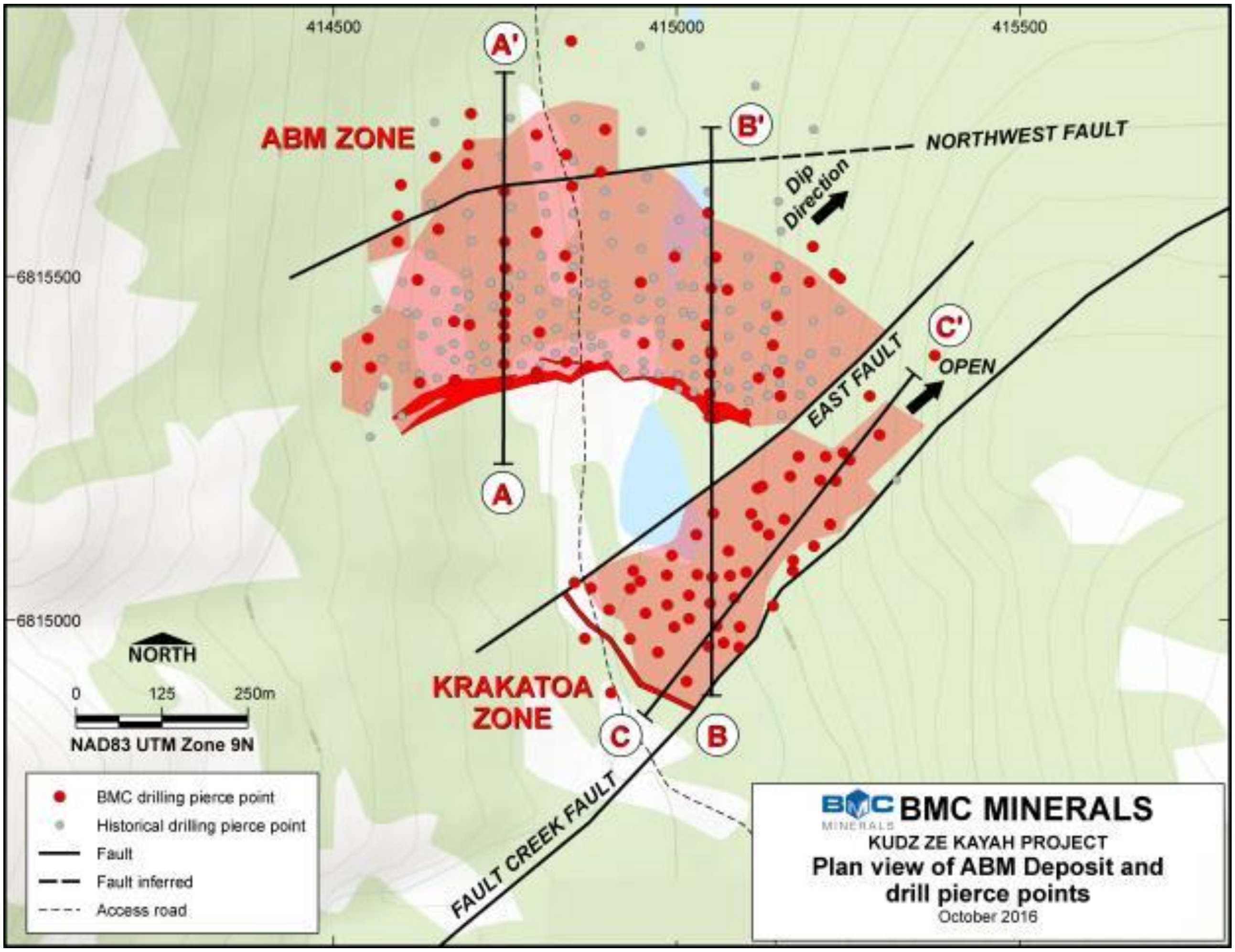

Figure 7-8: Plan view of ABM deposit showing both ABM and Krakatoa zones

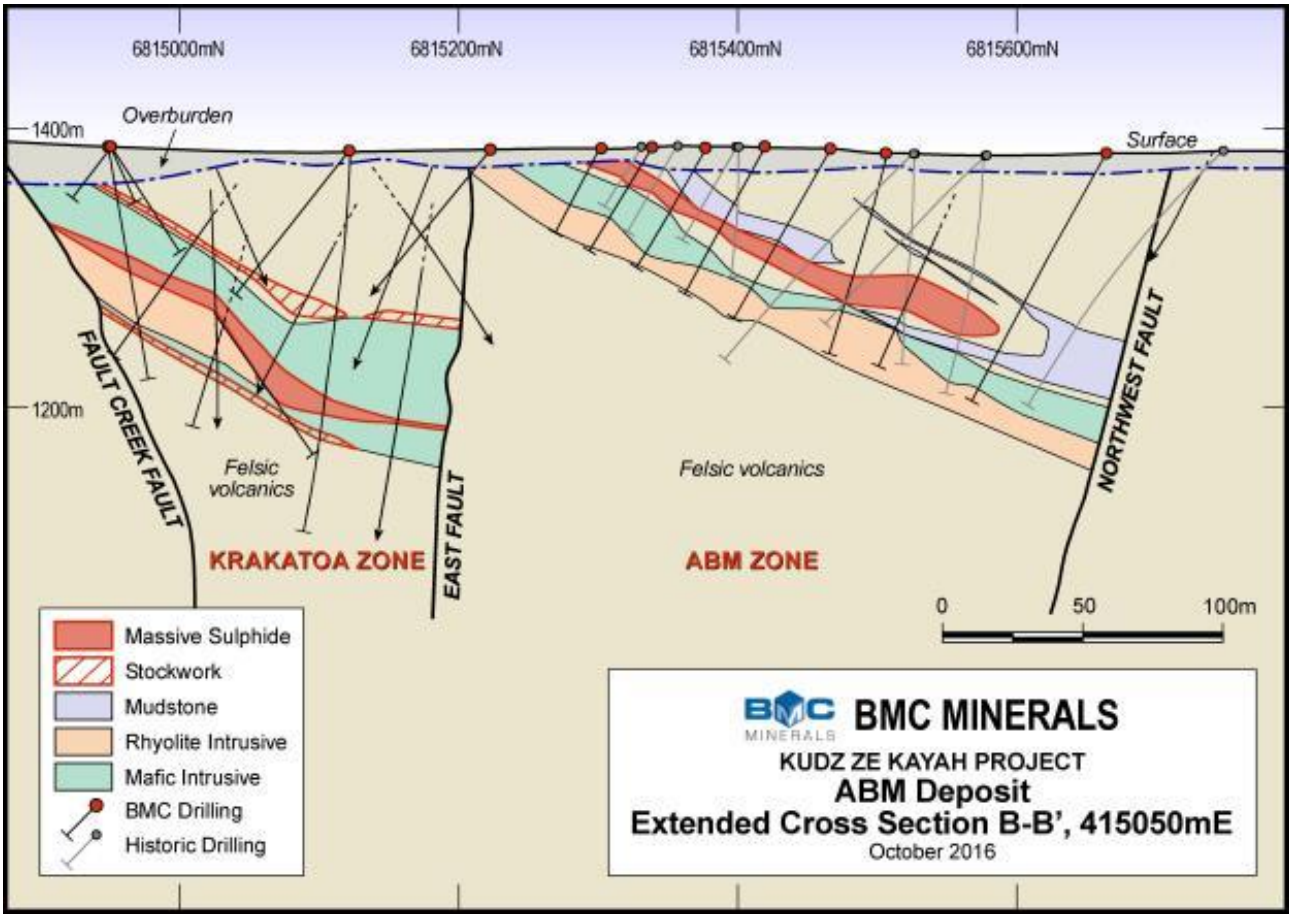

Figure 7-9: Schematic cross-section view looking west through both the ABM and Krakatoa zones showing their spatial relationship

Note: BMC drill traces shown in black and Cominco/Teck drill traces in grey. Section B-B' in Figure 7-8.

7.3.2 Stratigraphy

ABM Deposit

Geological logging of diamond drill core and field mapping undertaken at the ABM deposit, both historically by Cominco and more recently by BMC, has focused on providing a broad lithological framework for future mining activities rather than for the generation of a detailed stratigraphic model. General results of this work are summarized below and principal component lithologies are summarized in Table 7-2.

The numerous component lithologies identified in Table 7-2 have been aggregated into a smaller set of more continuous rock units which are discussed below.

Table 7-2 ABM deposit lithological units

Stratigraphic unit	Geology detail code	Geology detail sub-code	Lithology	Description
	OVBN	-	Overburden	Unconsolidated overburden
Wind Lake Formation	MAFt	-	Mafic volcaniclastic	Confined to Upper Sedimentary and Mafic Volcanic Sequence
	MDU	-	Mudstone and tuffaceous mudstone	Calcareous carbonaceous and mafic tuffaceous mudstones of the Upper Sedimentary and Mafic Volcanic Sequence
Kudz Ze Kayah formation	SED	-	Undifferentiated sedimentary rock	Variable; includes locally calcareous sandstone, wacke, recrystallized limestone
	MDS	-	Mudstone, tuff and rhyolite	Undifferentiated carbonaceous mudstone, tuffaceous mudstone and carbonaceous coherent rhyolite
		MDSt	Rhyolite tuff-dominant mudstone	<15% carbonaceous component, heterolithic appearance with rare mudstone laminae
		MDSc	Carbonaceous mudstone	>15% carbonaceous component, weakly heterolithic to semi-massive mudstone with slaty cleavage
		MDSw	Coherent rhyolite with minor carbonaceous mudstone	<15% carbonaceous component, laminae of mudstone intercalated with coherent rhyolite
	PEL	-	Pelite	Interpreted sedimentary protolith. Biotite rich rock with quartz, feldspar and/or calcite
		PELc	Calcite-rich pelite	Biotite and calcite dominant sedimentary rock
		PELq	Quartz-rich pelite	Biotite and quartz dominant sedimentary rock
	CHT	-	Chert	
		CHTc	Chert with interlayered carbonaceous mudstone	
		OA	Magnetite-bearing massive sulphide	Massive sulphide with abundant disseminated euhedral medium-grained magnetite or laminated magnetite. Commonly includes PY+MG+SP+GL±CP±PO
		OB	Wispy laminate, fine buckshot textured, non-magnetite bearing massive sulphide	Fine to coarse-grained massive sulphide consisting of PY+SP+GL±CP±PO±MG. Magnetite is typically trace or absent
		OF	Pyrrhotite-rich sulphide	Massive sulphide with >50% pyrrhotite. Sulphide assemblage of PO+MG±CP±PY±SP±GL
		OH	Pyrite-rich massive sulphide	Fine-grained, homogeneous pyrite dominated (>80% pyrite) massive sulphide
		OC	Chalcopyrite + pyrrhotite net-textured massive sulphide	Fine to coarse-grained massive sulphide comprised of dominantly macroscopic CP+PO comprising >10% to <50%
		OD	Brecciated sulphide	Sulphide with crackle breccia to mosaic breccia texture. Sulphide assemblage of PY+SP+GL±CP±PO±MG
		OG	Chalcopyrite-rich sulphide	Semi-massive to massive sulphide with >30% CP. Typical sulphide assemblage of CP+PO+PY+SP±GL±MG
		OI	Heavily disseminated sulphide in schistose host rock	Stringer and disseminated sulphide to semi-massive sulphide mineralization with >20% sulphide

Stratigraphic unit	Geology detail code	Geology detail sub-code	Lithology	Description
		OJ	Heavily disseminated sulphide in proximal altered rock	Stringer sulphide to semi-massive sulphide hosted within altered rocks with an alteration assemblage of CL+CI Sulphide assemblage of PY+CP+SP±PO±GL
		OK	Heavily disseminated sulphide and/or stringer style mineralization associated with silicate gangue	Heavily disseminated sulphide and/or stringer style mineralization associated with barite ± quartz ± carbonate gangue
	RHY	-	Undifferentiated rhyolite	Typically altered and difficult to identify
	RHYc	-	Coherent rhyolite	Undifferentiated coherent rhyolite
		RHYcw	Curdy-textured and/or flow-banded	Flows or sub-volcanic intrusions with definitive siliceous flow banding or relict flow banding textures
		RHYcf	Feldspar ± quartz porphyry	Medium to coarse-grained feldspar and quartz porphyritic texture rhyolite
		RHYcq	Quartz porphyry	Medium to coarse-grained quartz-phyric texture
	RHYi		Aphanitic rhyolite	Aphyric massive to semi-massive siliceous rhyolite interpreted as intrusive dykes and sills
	RHYv	-	Volcaniclastic rhyolite	Undifferentiated fine to medium-grained volcaniclastic
		RHYvl	Lapilli tuff	Heterolithic tuff with >15% lapilli typically in an ash-dominated ground mass
		RHYva	Coarse-grained to ash tuff	Phaneritic ash tuff with >10% of crystals occurring as >1-<2 mm typically in an ash-dominated ground mass
		RHYvx	Quartz ± feldspar crystal tuff	Quartz, ± feldspar-phyric (>10%) in an ash-dominated ground mass
	MAFi	-	Mafic intrusions	Chlorite-altered mafic intrusion, primarily in the footwall but also occurs as narrow dykes or sills within the felsic sequence
	FLZ		Fault Zone	

Abbreviations: PY = pyrite, PO = pyrrhotite, MG = magnetite, CP = chalcopyrite, SP = sphalerite, GL = galena, CL = chlorite, CI = cordierite.

Surficial Geology and Overburden

Regional mapping (Jackson, 1993) delineated the extent of morainal deposits and colluvial aprons on the slopes above the northerly trending Geona Creek valley, and glaciofluvial sediments in the valley floor. The morainal deposits comprise silty to sandy gravels with cobbles and minor boulders, and the colluvial aprons comprise granular soils with clastic components to boulder size. Glaciofluvial sediments in the valley bottom comprise predominantly sands and gravels with minor silt.

Investigations of the surficial geology undertaken across the mine project area (summarized in: Golder Associates, 1996; Alexco Environmental Group, 2015) include surface mapping, test pitting, and geotechnical and hydrogeological drilling. Glacial sediments comprising ablation till have been mapped in the valley floor, locally to 30 m in thickness and progressively thinning up the valley slopes. Several small eskers and a glacioflucial terrace have also been identified in the project area and may provide a source of construction aggregate.

An active alluvial fan is present at the confluence of Geona and Fault Creeks south of the ABM deposit area. Material in the fan comprises loose to compact gravelly sand with rare cobbles.

Wind Lake Formation

The Wind Lake formation outcrops on topographic highs to the west, north and east of the ABM deposit (Figure 7-5 and Figure 7-6). It is encountered in holes collared above the Krakatoa Zone and eastern parts of the ABM Zone where the Wind Lake formation stratigraphy is juxtaposed against Kudz Ze Kayah formation by late faulting. The Wind Lake formation comprises undifferentiated black and grey carbonaceous and calcareous to weakly calcareous mudstone, siliceous siltstone and chert that is locally intercalated with green-grey to olive green fine-grained mafic volcaniclastic units occurring as massive intervals up to 7 m in thickness. The mafic volcaniclastic rocks are interpreted to be epiclastic in nature, derived from a distal mafic volcanic source. The Wind Lake formation is fissile and typically rubbly in drill core, and the contact with the underlying Kudz Ze Kayah formation appears conformable.

Kudz Ze Kayah Formation

The Kudz Ze Kayah formation both structurally and stratigraphically underlies the Wind Lake formation in the vicinity of the ABM deposit and is host to the massive sulphide mineralization. The Kudz Ze Kayah formation in the vicinity of ABM comprises a thick sequence of felsic volcanic schist interbedded with variably carbonaceous metasedimentary and calcareous mafic schist units. Although in places recognisable, primary volcanic and sedimentary textures in these rocks have typically been poorly preserved due to the overprinting impact of hydrothermal alteration, metamorphism and polyphase deformation. Several packages of rocks have been identified on the basis of the primary textures which can be identified in drill core and it is these that have been utilized in generating a 3D geological model for the ABM deposit:

- Carbonaceous units (MDS):
  - A range of lithologies (MDS) occur within the upper part of the Kudz Ze Kayah formation sharing the commonality of a carbonaceous component. This includes rhyolite tuff-dominated mudstone, carbonaceous mudstone and coherent rhyolite with minor carbonaceous mudstone. The units have variably undergone widespread isoclinal folding with transposition of minor fold limbs, and where strongly carbonaceous the rock is fissile and rubbly in drill core. These lithologies are not typically mineralized; however, locally where the mudstones are in direct contact with massive sulphide, they may be partially mineralized over approximately 1 m from the massive sulphide contact. Near the edges of the deposit, and particularly in the down-dip portions, mudstones and related ash tuffs are interpreted to be lateral equivalents to the massive sulphide horizon on the basis of alteration, mineralization and structural position.
- Coherent rhyolite:
  - Various units mapped as coherent rhyolite (RHYc) are evident, predominantly comprising quartz-sericite schist for which the protoliths are interpreted to have been rhyolitic flows and shallow intrusions. Flow-banding is locally preserved, as are peperitic and hyaloclastite textures. Variation in the feldspar and quartz phenocryst components of the bodies indicates a multi-phase magmatic emplacement history.
- Aphanitic rhyolite:
  - Massive grey, weakly to un-foliated aphanitic rhyolite (RHYi), which may locally contain quartz and/or feldspar phenocrysts and rare 1–4 mm amygdales, is interpreted as a largely shallow intrusive unit (although the variation in phenocryst components probably indicates multiple intrusive components to this unit). To the southeast the unit occurs as intrusive masses with aphanitic interiors that grade

outwards into flow-foliated margins displaying peperitic contacts indicative of shallow sub-seafloor emplacement into surrounding unconsolidated volcaniclastic sediments. The relative timing of emplacement of this aphanitic rhyolite unit in relation to the mafic sill in the footwall of the ABM Zone is not definitive, but the locally peperitic contacts of this unit and a lack of unambiguous peperitic contacts in the case of the mafic sill may support the contention that emplacement of the mafic sill post-dates the aphanitic intrusive.

  - Much of the unit is mineralized with a fine dusting of pyrite that imparts a grey to dark grey colour, as well as medium to coarse-grained pyrite and rare sphalerite within irregular brittle quartz and/or calcite stringer veins, and this indicates emplacement occurred pre- to syn-mineralization.

- Volcaniclastic rhyolite:
  - Rhyolitic rocks which are interpreted to be primary fragmental in nature, and now predominantly comprising quartz-sericite schist, have been logged as rhyolitic volcaniclastics (RHYv). The most common volcaniclastic rock identified in logging is lapilli tuff, with well-preserved fragmental texture rarely evident. The upper portion of the Kudz Ze Kayah formation in the southeast corner of the ABM deposit comprises a significant thickness of massive fine-grained ash tuff with heterogeneous alteration domains that are interpreted as possibly after flattened pumice clasts. Coarse-grained ash tuffs are interpreted elsewhere in the deposit; however, individual units rarely display lateral continuity. Fine-grained volcaniclastics with quartz phenocrysts, and less commonly feldspar phenocrysts, are most abundant in the upper and northwest part of the deposit, stratigraphically below ash tuff in the southeast and beneath the mafic footwall intrusives. The volcaniclastics are probably predominantly epiclastic in nature, however it is likely that a significant component also comprises hyaloclastite facies equivalents related to the coherent rhyolite units. It is also possible that some rocks grouped with the volcaniclastics may be the result of heterogeneous hydrothermal and diagenetic alteration of coherent rhyolite units with subsequent compaction and deformation leading to the development of pseudo-fragmental textures.

- Footwall mafic and other intrusions:
  - Mafic intrusive rocks (MAFi) are present lower in the stratigraphy, comprising numerous approximately 0.5 m to 5 m thick dykes and sills, with a large (to 50 m thick) and more continuous mafic sill located in the footwall to massive sulphide mineralization of both the ABM Zone and the shallower lenses of the Krakatoa Zone. The same thick mafic sill hosts minor sulphide mineralization beneath the ABM Zone and a significant component of the massive sulphide mineralization that comprises the Krakatoa Zone (Duncan, 2015). Smaller dykes and sills range from aphanitic, green and chlorite-bearing to biotite-rich intrusions that are typically calcareous. Contacts are generally sharp with chilled margins. The mafic bodies are very rarely amygdaloidal, and in the southeast of the deposit there were observed some questionable examples of peperitic contacts with surrounding felsic volcaniclastics.
  - The large footwall sill at Krakatoa varies from a medium-grained, banded, chlorite-biotite-carbonate schist to schist with dark chlorite and/or amphibole-bearing patches that may represent a relict gabbroic texture. Proximal to massive sulphide mineralization it can be quite sericite-altered.
  - The volcanogenic massive sulphide mineralization hosted by the thick mafic sill, combined with possible peperitic margins and amygdales, would indicate emplacement of the sill into the felsic volcanic pile at a shallow depth below the seafloor prior to mineralization. The mix of peperitic and sharp mafic contacts may suggest that the felsic pile was variably lithified at the time of emplacement, however doubt surrounding the interpreted peperitic contacts would favour emplacement post-

intrusion of the aphanitic rhyolite. A lack of pillow textures or reworked mafic clasts would indicate that the mafic unit was not emplaced as a flow on the seafloor.

7.3.3 Structure and Metamorphism

Host rocks to the ABM deposit have been deformed and metamorphosed to upper greenschist to lower amphibolite facies (Peter et al*.,* 2007), and almost without exception display one or more penetrative deformation fabrics.

S_0 is most commonly recognized as bedding in mudstone and tuff units assigned to the Wind Lake formation, and locally within the Kudz Ze Kayah formation overlying and lateral to the ABM deposit.

A sub-horizontal to moderately north to northeast dipping, penetrative deformation fabric (S_1) is best developed in phyllites, fine-grained schists and areas of increased phyllosilicate alteration. S_1 is typically subparallel to the original bedding (S_0) with a mean dip direction of 007° and a dip of 24.3°. The S_1 foliation may represent a compaction fabric (MacRobbie, 1995; Baker et al*.,* 2017).

An S_2 crenulation cleavage defined by fine folds of S_1 generally strikes similarly to S_1 but is more steeply dipping to the north or northeast. In local, mesoscale folds, S_1 and S_2 may be oriented at high angle to one another. Development of S_2 crenulations is not uniform – it is highly localized and varies from intense to broadly-spaced. More felsic and well foliated (S_1) rocks with abundant sericite tend to exhibit the most intense and finely developed D_2 crenulations whereas these folds are broader in coarser-grained and less felsic rock. F_2 folds are typically mesoscale, south-verging, and appear to fold S_0 and S_1 only locally (Baker et al*.,* 2017).

Kink bands (S_3) are rare across the Property and defined by cm-scale kinking of the main (S_1) schistosity. Although based on a small subset of data, S_3 planes have two dominant orientations, subvertical with a northwest-southeast strike and sub-horizontal and all their hinge lines plunge gently to moderately to the north. This suggests that these D_3 structures form conjugate sets of kink bands.

Although small, presumably parasitic, folds are observable in drill core, no large-scale fold patterns have been identified in the field. Cominco interpreted the ABM deposit as being overturned within isoclinally folded host stratigraphy based on the occurrence of intense chlorite-cordierite alteration and chalcopyrite-pyrrhotite-rich mineralization in both hangingwall and footwall, and on the assumption that mineralization was emplaced on the seafloor. The current interpretation is one of mineralization being emplaced in a sub-seafloor position, with which the above alteration and mineralization patterns which would be consistent without the requirement for isoclinal folding. The lack of stratigraphic repeat above and below the ABM deposit (e.g. the large footwall mafic sill) and the conformable contact with the overlying Wind Lake formation does not support deposit-scale isoclinal folding in the vicinity of the ABM deposit.

Several possible growth faults have been identified within Krakatoa Zone, striking west-northwest and dipping steeply north-northeast. These structures have been identified based on apparent offsets and rapid changes in the thickness and extent of coherent mafic and felsic intrusive bodies, rapid changes in the thickness and extent of massive sulphide mineralization, and localized zones of elevated copper-gold mineralization and intense chlorite alteration. It is most likely that the interpreted growth faults predate mineralization and were later conduits for hydrothermal fluids during emplacement of ABM sulphide mineralization.

Host rocks, mineralization and pervasive deformation fabrics are cut by two significant late faults; East Fault and Fault Creek Fault, both which are filled with cataclasite (including sulphide clasts, Figure 7-7) and gouge (Figure 7-10). These faults can be traced as trends in aeromagnetic data for kilometres to tens of kilometres. Structures of similar timing and orientation can be identified in regional datasets. The East Fault is sub-vertical, strikes 052°, and is interpreted to truncate and offset the eastern end of the ABM Zone. Using the basal Wind Lake contact, plus the ABM and Krakatoa sulphide deposits as tenuous stratigraphic equivalents, and assuming no earlier faults offsets having occurred, then the East Fault has displaced the Krakatoa Zone at least 200 m downward (i.e. south-block down). Alternatively, an interpreted dextral strike slip movement along East Fault could provide a similar apparent offset.

Figure 7-10: East Fault characterized by polylithic fault breccia and minor gouge (K15-262 at approximately 127.2 m)
Source: Hughes and Baknes, 2015

The Fault Creek Fault intersects the topographic surface approximately 290 m southeast of the East Fault and dips to the northwest at an average of approximately 72˚. The near-surface dip is somewhat shallower but steepens with depth. The degree of displacement across the Fault Creek Fault is unclear, however contact between Wind Lake formation and Kudz Ze Kayah formation in outcrop and drilling would indicate a potentially negligible offset.

The East Fault and Fault Creek Fault define a fault-block containing the massive sulphide-bearing volcanic and sedimentary package that comprises the Krakatoa Zone. This fault block is dissected by a number of smaller-scale late-stage faults that do not appear to extend beyond the larger bounding faults and have offsets of less than 20 m. These smaller-scale structures are characterized by up to several metres core length of cataclasite and gouge.

The Wind Lake formation and Kudz Ze Kayah formation both display a moderate to intense foliation through all rock types, with increased foliation intensity within rocks with a high phyllosilicate and/or carbonaceous content. In the upper part of the Kudz Ze Kayah formation, for example, the units have variably undergone widespread isoclinal folding with transposition of minor fold limbs.

Massive sulphide mineralization typically displays fine displays fine millimetre scale banding, augen and pressure shadows around more competent components, and occasionally durchbewegung textures, all evidence of sulphide deformation. Petrological examination indicates recrystallization of all sulphide minerals has occurred during metamorphism, except for pyrite or those minerals contained within pyrite. Overall the massive sulphide lenses behaved as competent bodies in contrast to adjacent phyllosilicate-rich host rocks, principally due to the high pyrite content. This resulted in increased strain developed adjacent to massive sulphide margins.

At the deposit scale it appears that the stratigraphic sequence is not overturned, and there is no evidence to support the interpretation of large-scale folding of the ABM Zone. Although vein-style/disseminated sulphide mineralization occurs both in the footwall and hanging wall of the massive sulphide ores, this is not of itself evidence of orebody folding, particularly if the sulphide mineralization is emplaced below the seafloor. There is no fold repeat of the pre-mineralization footwall mafic sill at either ABM or Krakatoa, and the down-dip transition from massive sulphide into partially mineralized clastic horizon displaying textures indicative of replacement would indicate that the down-dip extent of ABM Zone is not a fold hinge.

7.4 Mineralization

ABM Zone

Massive sulphide of the ABM Zone (Figure 7-9) is up to 39 m true thickness, extending approximately 700 m along strike and approximately 500 m down dip. It dips to the north-northeast at approximately 35° near surface, transitioning to a dip of approximately 15° at around 200 m depth below the valley floor. The up-dip extent of the deposit is truncated by erosion and covered by approximately 2–20 m of glaciofluvial overburden.

Massive sulphide mineralization of the ABM Zone occurs as several stacked massive sulphide lenses to the west, transitioning to a single massive horizon at around 415,025 m E and extending to approximately 415,250 m E where it is then truncated by the post-mineralization East Fault. Stockwork and disseminated mineralization occurs equally both in the hanging wall and footwall to massive sulphide, and to a lesser degree between massive sulphide lenses.

Sulphide mineralization is dominated by pyrite, sphalerite, pyrrhotite (+ marcasite), galena and chalcopyrite, with minor arsenopyrite and a range of sulphosalts predominantly comprising tennantite-tetrahedrite and freibergite (Figure 7-11). Both the up-dip part of ABM Zone and most of Krakatoa Zone have elevated sulphosalt content relative to the remainder of the ABM deposit.

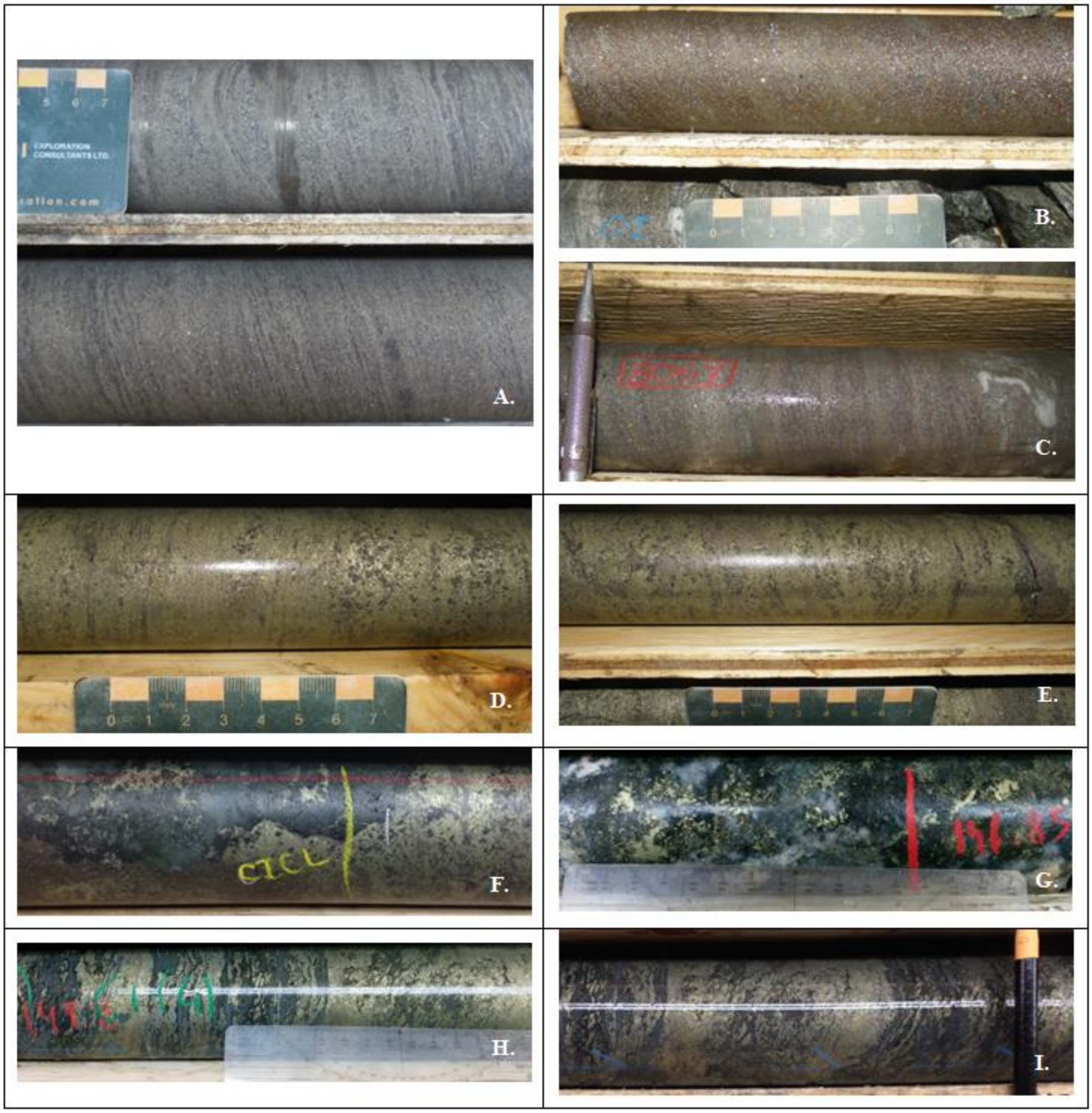

Figure 7-11: Photographs of ABM deposit massive and disseminated sulphide types

Photo A: OA with medium-grained magnetite forming thin bands within massive pyrite.

Photos B and C: OB showing disseminated red-brown sphalerite in medium- to coarse-grained pyrite and quartz-carbonate gangue.

Photos D and E: OG consisting of massive chalcopyrite intergrown with clots and narrow bands of pyrrhotite.

Photos F and G: Variations of OC showing coarse-grained net-textured chalcopyrite and pyrrhotite associated with strong chlorite alteration and quartz-carbonate gangue.

Photos H and I: OJ consisting of deformed stringer-style chalcopyrite-pyrrhotite mineralization in intense chlorite-cordierite alteration.

Source: Hughes and Baknes, 2015

Krakatoa Zone

Krakatoa Zone mineralization (refer Figure 7-8 and Figure 7-9), bound to the west by the East Fault and to the east by the Fault Creek Fault, is hosted within Kudz Ze Kayah formation that dips at 35° to the north-northeast. Although of lesser extent, the distribution of mineralization within the Krakatoa Zone is more spatially complex than the ABM Zone due to the stacked mineralized lens system. Massive sulphide mineralization occurs within three principal mineralized horizons:

1. The “Upper lens”, broadly interpreted as the stratigraphic equivalent to the ABM lens.
2. The “Main lens”, the major component of Krakatoa Zone in terms of sulphide mineralization with a true thickness up to 22 m.
3. A less pronounced and semi-continuous “Lower lens”.

Krakatoa Zone mineralization is broadly concordant with stratigraphic layering of the host rocks, extending over approximately 200 m of strike, at least 500 m down dip, and up dip to the base of glacial overburden of 20–30 m thickness.

Several pre- to syn-mineralization growth faults are thought to influence the massive sulphide bodies at Krakatoa in terms of offsets and changes in massive sulphide thickness. Both the large coherent mafic and aphanitic rhyolite bodies appear to have an influence on the spatial distribution of sulphide mineralization. As these bodies are themselves mineralized, it is likely that these coherent bodies acted as fluid aquacludes during mineral deposit formation.

Host rock types and alteration styles of Krakatoa Zone mineralization are for the most part similar to those encountered in the ABM Zone. The key difference is the degree of mineralization associated with the mafic sill, which below ABM Zone is only poorly mineralized. The Main lens comprises the bulk of mineralization at Krakatoa, with massive sulphide occurring both within the felsic volcanics immediately beneath the mafic sill, and within the mafic sill, after replacement of enclaves of felsic volcaniclastics and/or replacement of the mafic sill itself.

7.4.1 Alteration

At ABM, the massive sulphide mineralization is surrounded by an alteration halo which changes outward from the massive sulphide in a broadly systematic fashion at the deposit scale, with the following trends evident moving away from the massive sulphide:

- Massive sulphide – stringer and disseminated sulphide – disseminated sulphide dominated pyrite with minor pyrrhotite
- Ankerite/siderite (Figure 7-12M) – calcite
- Albite/celsian – chlorite (Figure 7-12J) – sericite/muscovite (Figure 7-12K).

Hydrothermal alteration extends into the footwall, hangingwall and down dip of massive sulphide. In detail, the alteration distribution and intensity vary to some degree with the underlying lithology (mafic versus felsic) and is locally variable (millimetre to tens of metre scale).

In addition to this pervasive alteration, restricted zones of cordierite alteration are present with cordierite spatially associated with proximal chlorite alteration as well as in more distal fine-grained carbonaceous and tuffaceous mudstones and ash tuffs with or without chlorite. The mafic intrusives, especially at Krakatoa, are characterized by elevated arsenic within calcite veining.

Syn- to post-deformation quartz ± calcite veining is observed throughout the drilling. Distal to the massive sulphide where the syn-mineralization alteration is subdued, it can be seen that these veins are variably associated with haloes of phyllosilicate (sericite, biotite), calcite and tourmaline. This veining is interpreted to be related to emplacement of Cretaceous intrusions in the district.

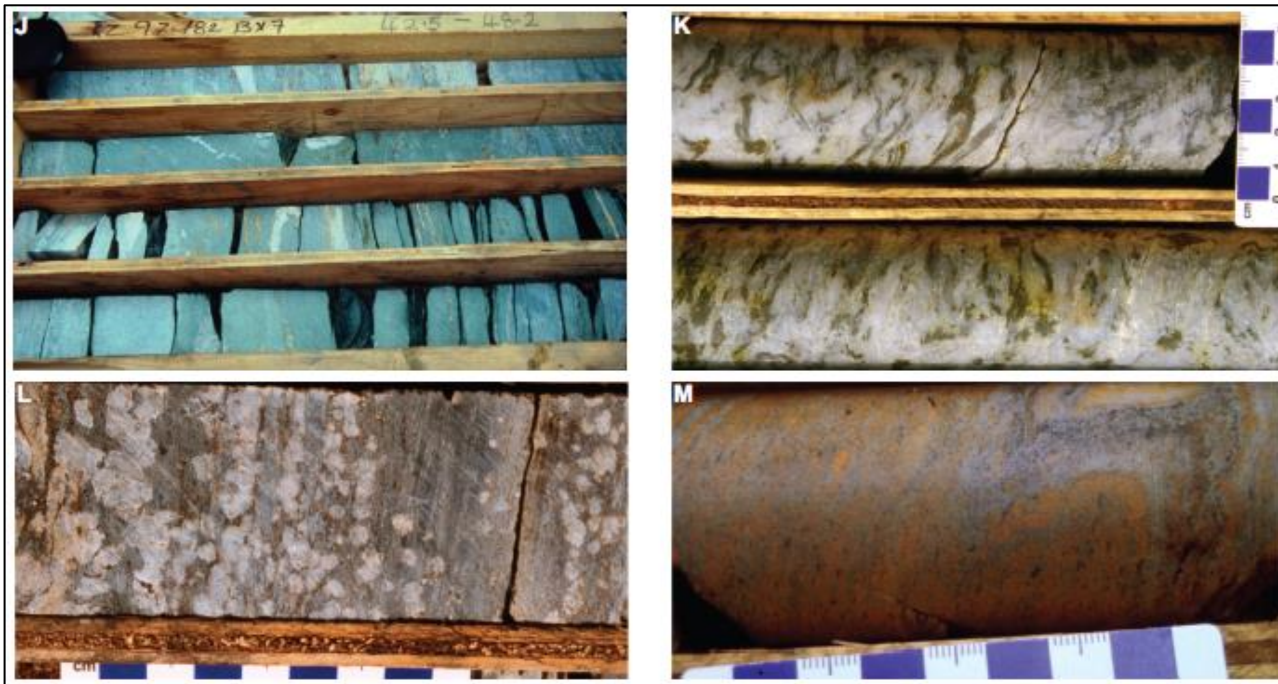

FIGURE 9 CONTINUED. Photographs of Kudz Ze Kayah deposit host rocks and mineralization. (J) Strongly chloritized felsic volcaniclastic rock, stratigraphic footwall (DDH K97-182; 42.5-48.2 m). The diameter of the core is 4.5 cm. (K) Sericite-altered felsic volcaniclastic rock, stratigraphic footwall. (L) Chlorite-albite altered felsic volcaniclastic rock from immediate stratigraphic footwall to the massive sulphides. (M) Fe-carbonate-altered felsic volcaniclastic rock, stratigraphic hanging wall. Unless stated, the locations of individual samples are not specified.

Figure 7-12: Photographs of ABM deposit host rocks and alteration
Source: Peter et al., 2007

7.4.2 Metallurgical Domains

The sulphide mineralization can be readily divided on the basis on texture, style and significant mineral components. Stringer vein, stockwork and disseminated sulphide style mineralization (MET8) occurs primarily in both the footwall and hangingwall to massive sulphide mineralization. The massive sulphide itself can be visually separated into pyrrhotite-magnetite-dominant (MET2-4) and pyrite-dominant (MET5-7) domains. Examination of geochemical data from the diamond drilling dataset (Green, 2016) in the context of the metallurgical domains defined in the field demonstrates a clear geochemical differentiation between the domains.

- MET2-4: elevated Cu, Fe, Bi, Se; moderate Pb, Zn, Sb; reduced Ag, Au, As, Ba, Hg
- MET5-7: elevated Ag, Au, As, Ba, Hg, Sb; moderate Fe, Pb, Zn, Bi, Se; reduced Cu
- MET8: elevated Cu; and low Ag, Au, Fe, Pb, Zn, As, Ba, Bi, Hg, Sb and Se.

However, when the geochemical data is normalized with respect to iron content, to take into account the increased gangue component, it can be seen that sulphide mineralization in MET8 is in fact characterized by elevated Cu, Bi, Se and moderate Ag, Au, Pb, Zn, As, Ba, Hg, Sb (Table 7-3) in a similar fashion to the pyrrhotite-magnetite-rich domain.

Table 7-3: Geochemical data by metallurgical domain

Met domain	Statistic	Ag ppm	Au ppm	Cu %	Fe %	Pb %	S %	Zn %	As ppm	Ba ppm	Bi ppm	Hg ppm	Sb ppm	Se ppm
Massive sulphide	Number	1,823	1,812	1,823	1,823	1,823	866	1,823	1,807	1,725	1,807	1,043	1,806	1,447
	Minimum	-0.30	-0.07	-0.01	0.76	-0.01	-1.00	-0.01	-1	-1	-1.00	-1.00	-1	-1
	Maximum	1,181.50	19.16	18.00	49.50	14.90	49.20	29.60	39,740	256,805	608.00	248.00	9,927	1,290
	Mean	**164.76**	**1.63**	**0.92**	**29.96**	**1.97**	**31.32**	**6.55**	**3,077**	**17,981**	**50.63**	**24.08**	**596**	**109**
	SD	113.72	1.27	1.82	10.91	1.53	13.70	3.89	3,464	32,412	54.80	27.23	726	170
	CV	0.69	0.78	1.99	0.36	0.78	0.44	0.59	1.13	1.8	1.08	1.13	1.22	1.56
Stockwork	Number	449	438	450	449	450	139	450	416	343	416	205	416	250
	Minimum	-0.30	-0.07	-0.01	1.50	-0.01	-1.00	0.00	-1	-1	-1.00	-1.00	-1	-1
	Maximum	619.20	4.65	12.60	48.00	8.35	44.45	17.35	17,892	104,896	367.00	80.10	4,146	799
	Mean	**68.48**	**0.63**	**1.08**	**16.45**	**0.63**	**9.65**	**2.59**	**1,025**	**5,600**	**45.08**	**6.13**	**164**	**87**
	SD	79.39	0.72	1.41	10.42	1.04	10.59	3.22	2,194	10,376	44.33	11.47	422	144
	CV	1.16	1.14	1.3	0.63	1.64	1.1	1.24	2.14	1.85	0.98	1.87	2.57	1.66
Magnetite	Number	576	576	570	576	576	576	195	566	524	566	340	566	397
	Minimum	1.60	0.02	-0.01	1.90	-0.01	-1.00	0.04	-1	2	-1.00	-1.00	-1	-1
	Maximum	859.00	5.83	10.78	49.80	6.83	48.80	18.40	20,322	70,000	696.00	66.70	3,760	1,840
	Mean	**89.41**	**0.95**	**1.09**	**38.83**	**1.11**	**33.81**	**6.43**	**1,508**	**2,864**	**79.67**	**6.54**	**203**	**149**
	SD	70.65	0.78	1.17	7.20	1.13	11.23	3.70	2,500	7,718	55.60	6.86	361	223
	CV	0.79	0.82	1.08	0.19	1.02	0.33	0.57	1.66	2.69	0.7	1.05	1.78	1.5

Notes:

- *Geochemical data for MET2-4 (magnetite), MET5-7 (massive sulphide) and MET8 (stockwork) domains, inclusive of all ABM and Krakatoa Zone ores.*
- *Abbreviations: SD = standard deviation; CV = coefficient of variation.*
- *Negative values shown represent levels that are below detection limit for the analysis undertaken. The value of the numeric displayed, after removal of negative sign, is the analytical method detection limit.*

Although not visually evident in hand specimen, petrographic studies (Macleod, 1994a-b; 1995a-d; Macleod 1997; Macleod, 1999; Townend & Townend, 2015) reveal that ABM Zone sulphide mineralization above approximately 1,340 m RL and the Krakatoa Zone comprise mineralization with a significantly elevated silver-rich sulphosalt content. In both cases, the predominant mineralization type is pyrite-rich.

On this basis, the ABM mineralization was divided into five principal metallurgical domains for the purpose of metallurgical testwork:

- MET2-4:
  - Magnetite-pyrrhotite-rich massive sulphide (pyrrhotite often partially replaced by marcasite)
  - Low galena, sulphosalt and arsenopyrite content.
- MET5-7:
  - Pyrite-rich massive sulphide, often with barite
  - Elevated galena, sulphosalt and arsenopyrite content
  - Visible gold/electrum in thin section.
- MET8:
  - Stringer/disseminated sulphide, typically with chlorite-rich gangue
  - Similar mineralogy to MET2-4.

- MET +1340 RL:
  - Elevated tetrahedrite, arsenopyrite and visible gold content
  - Predominantly pyrite-rich mineralization with minor stringer and pyrrhotite-magnetite component, often with barite.
- MET Krakatoa:
  - Elevated freibergite content
  - Predominantly pyrite-rich mineralization with minor stringer and pyrrhotite-magnetite component, often with barite.

A summary of the relative proportion of the metallurgical domains by tonnage, derived from the January 2016 resource estimate, is summarized in Table 7-4.

Table 7-4: Summary of proportion of metallurgical domains for the ABM deposit

Zone	MET2-4	MET5-7	MET8	Comment
ABM Zone (All)	20%	65%	15%	Comprises 100% of ABM Zone
ABM (above 1340 m RL)	18%	75%	7%	Comprises 26% of ABM Zone
ABM (below 1340 m RL)	20%	61%	19%	Comprises 74% of ABM Zone
Krakatoa Zone (All)	**9%**	**80%**	**11%**	**Comprises 100% of Krakatoa Zone**
ABM + Krakatoa (All)	**17%**	**69%**	**14%**	**100% of ABM + Krakatoa zones**
	MET + 1,340 RL comprises 16% of combined ABM + Krakatoa zones			

7.4.3 Oxidation

Evidence of oxidation due to weathering along fractures and faults in the host rock at ABM, indicated primarily by iron-staining, typically extends to a vertical depth of around 18 m below surface and in places to around 50 m below surface.

A total of 132 core samples from 12 drillholes along the full strike of the ABM deposit, extending from the just below the base of the glaciofluvial overburden to a vertical depth of approximately 60 m, were submitted for ethylene diamine tetra acetic acid (EDTA) analysis. No soluble Cu or Zn was identified, but a modest level of soluble Pb was found to be present (maximum 668 mg/l, median 140 mg/l). A further 26 samples were also taken from the deepest part of the ABM Zone, well below any visual sign of weathering, and analysed by the same method. The analysis also returned no evidence of soluble Cu or Zn but indicated a similar level of soluble Pb (maximum 534 mg/l, median 197 mg/l). This result is interpreted to indicate that soluble lead evident in the mineralization is not the result of weathering and oxidation but is instead related to the inherent solubility of lead-rich minerals in the sulphide mineralization. On this basis, there is no evidence of near-surface oxidation which may impact on metallurgical recoveries.

7.4.4 Proposed Genetic Model (ABM Deposit)

The ABM deposit is interpreted to have formed through the mixing of heated metalliferous hydrothermal fluids with cold ambient seawater within a felsic pile at a likely depth of around 200 m or less (Doyle & Allen, 2003). This interpretation is based on the following features:

- Relics of host lithologies are commonly preserved in the massive sulphide mineralization
- Mineralization and intense alteration occur within the thick mafic sill which was emplaced below the seafloor (Krakatoa)

- The down-dip extent of the ABM Zone displays a gradational transition from massive sulphide to sediments (i.e. a replacement front)
- Massive sulphide mineralization appears discordant to the host stratigraphy which itself comprises a "volcanic pile" and is not a "layer-cake" in nature
- Intense alteration and mineralization of a similar style and intensity extend into both the footwall and hangingwall of the massive sulphide lenses
- There is a lack of reworked sulphide clasts, sulphidized fossils or chimney fragments evident in volcaniclastic units rocks overlying or adjacent to the deposit, despite evidence that a major component of the host rocks was emplaced on the seafloor as stratified mass flow units and extrusive flows.

An interpreted sequence for formation of the ABM deposit is outlined in Figure 7-13 to Figure 7-16.

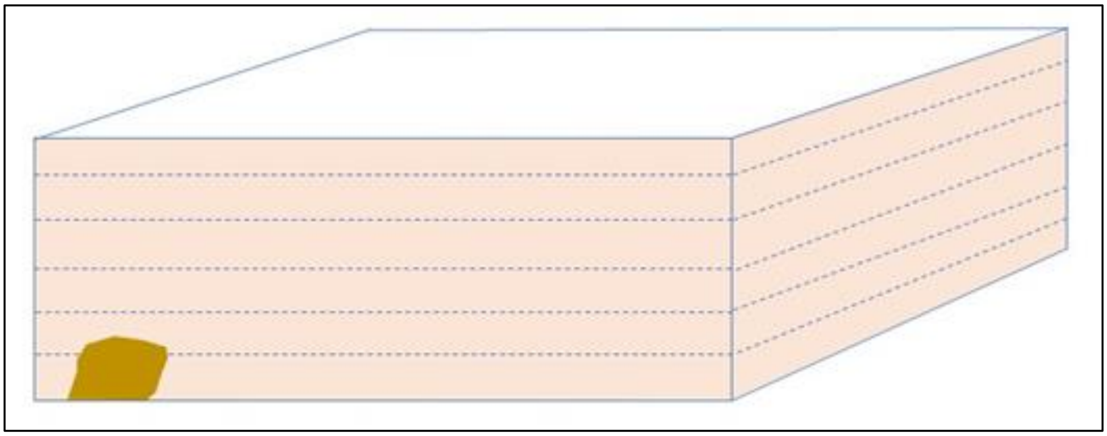

Figure 7-13: Deposition of stratified volcaniclastics as mass flows distal to a felsic volcanic source

Deposition of stratified volcaniclastics as mass flows distal to a felsic volcanic source, but not so distal so as to generate turbiditic features. Age date of ~363.25 Ma for GP4F host rocks (Manor pers. comm., 2018) may be indicative of timing. Volcaniclastics ultimately derived from same magmatic source as Grass Lakes plutonic suite which persists until late some time prior to deposition of Wind Lake formation.

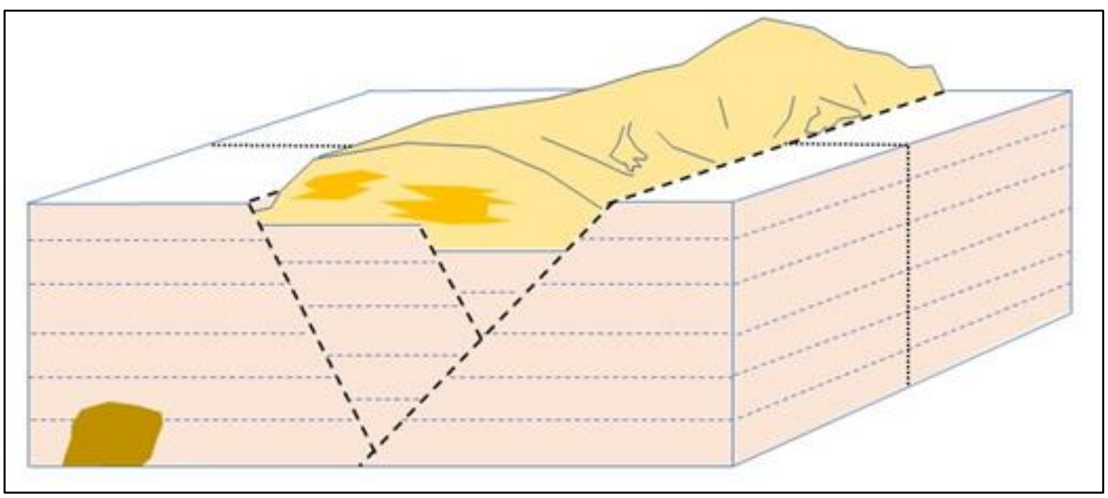

Figure 7-14: Normal and transfer faulting coeval with emplacement of felsic pile

The pile comprises a chaotic mix of shallow intrusives and lava flows (± flow banding, peperitic margins), autoclastic breccias (hyaloclastite) and locally derived mass flows (fine to coarse volcaniclastics). Multiple phases of magmatic intrusion as evidenced by phenocryst component of coherent units. Intrusion of Grass Lakes plutonic suite ongoing. Age dates for ABM of ~362.4 to 362.8 Ma (Manor et al, 2018) are indicative of timing.

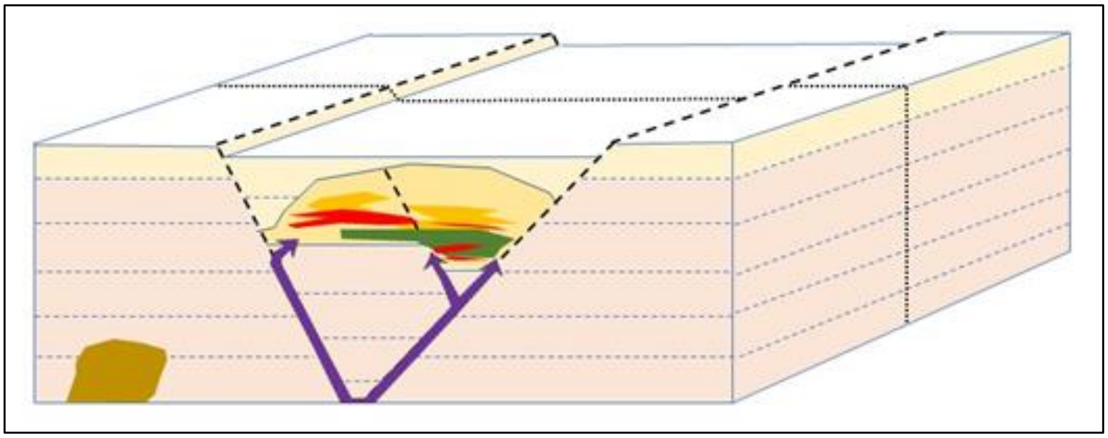

Figure 7-15: Normal faulting continues and graben deepens

Proximal felsic eruption ceases, fine-grained stratified volcaniclastics deposited (more distal to source), and mafic sill is emplaced into a variably lithified felsic pile (faulting and differences in local stratigraphy influence extent and thickness of intrusion). Subsequent to this, hydrothermal fluids pass up along the faults with resultant deposition of sulphides as hydrothermal fluids mix with seawater within the felsic pile (and mafic sill). Depth of mineral deposit formation ~200 m below seafloor. Intrusion of Grass Lakes plutonic suite ongoing, with subvolcanic intrusion providing the heat required to drive regional hydrothermal fluid circulation.

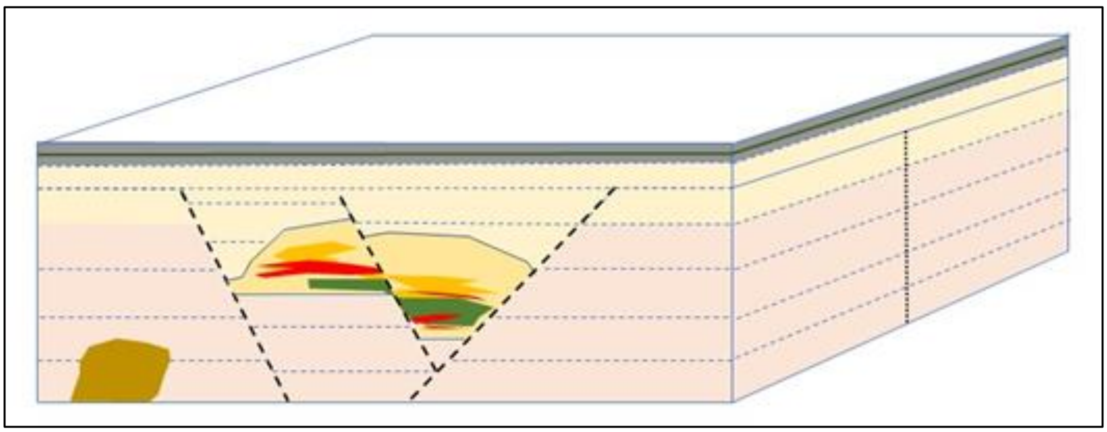

Figure 7-16: Mineralization ceases

Mineralization ceases and a final phase of movement along normal faults offsets the ABM and Krakatoa zones (probably tens of metres). Felsic mass flows (± carbonaceous component) continue to be emplaced before transitioning to an interbedded sequence of felsic mass flows and carbonaceous mudstones (ambient sedimentation). This is then conformably followed by emplacement of an interbedded sequence of carbonaceous mudstones and mafic volcaniclastics (distal to source). The geochemistry of the mafic volcaniclastics differs from the pre-mineralization mafic sill.

8 Deposit Types

8.1 Deposit Style

Volcanic-hosted massive sulphide (VHMS) deposits, also known as volcanic-associated, volcanogenic (VMS), and volcanosedimentary-hosted massive sulphide (VSHMS) deposits, are major sources of Zn, Cu, Pb, Ag, and Au, and significant sources for Co, Sn, Se, Mn, Cd, In, Bi, Te, Ga, and Ge. They typically occur as lenses of polymetallic massive sulphide that form at or near the seafloor in submarine volcanic environments, and are classified according to base metal content, gold content or host-rock lithology (Galley et al., 2007).

They occur submarine volcanic terranes that range in age from 3.4 Ga to actively forming deposits in modern seafloor environments. The most common feature among all types of VMS deposits is that they are formed in extensional tectonic settings, including both oceanic seafloor spreading and arc environments. Most ancient VMS deposits that are still preserved in the geological record formed mainly in oceanic and continental nascent-arc, rifted arc, and back-arc settings. Primitive bimodal mafic volcanic-dominated oceanic rifted arc and bimodal felsic-dominated siliciclastic continental back-arc terranes contain some of the world's most economically important VMS districts (Galley et al., 2007).

Most, but not all, significant VMS mining districts are defined by deposit clusters formed within rifts or calderas. Their clustering is further attributed to a common heat source that triggers large-scale subseafloor fluid convection systems. These subvolcanic intrusions may also supply metals to the VHMS hydrothermal systems through magmatic devolatilization (Galley et al., 2007).

As a result of large-scale fluid flow, VHMS mining districts are commonly characterized by extensive semi-conformable zones of hydrothermal alteration that intensifies into zones of discordant alteration in the immediate footwall and hanging wall of individual deposits. VMS camps can often be further characterized by the presence of thin, but laterally extensive, units of ferruginous chemical sediment formed from exhalation of fluids and distribution of hydrothermal particulates (Galley et al., 2007).

8.2 Concepts Underpinning Exploration

Any exploration approach to be considered in the case of VHMS deposits is ultimately linked to the processes involved in their formation and the properties inherent in the resultant mineral deposit and variably altered host rocks. The deposits are typically formed on the seafloor at or near volcanic or hydrothermal vents as a product of circulating hydrothermal fluids precipitating metals and trace elements leached from the underlying crust. This results in lithological, structural and geochemical fingerprints that are unique to the environment of formation. Moreover, the physical properties of the resultant mineralization relative to the host rocks provide unique geophysical signatures.

Field mapping and prospecting can be undertaken to identify rock sequences that may be amenable to hosting VHMS mineralization, in conjunction with techniques such as whole rock, rare earth and trace element geochemical fingerprinting as a guide to the likely tectonic setting of the host stratigraphy. Additional, more detailed field mapping and scout drilling can be utilized to identify areas of interest, including altered volcanic rocks, syngenetic fault structures and proximal-to-vent volcanic facies. The accumulation of metals and various trace elements in the mineral deposit, the enrichment and/or depletion of a range of major and trace elements in the surrounding host rocks, and in some cases the enrichment of metals and trace elements in associated exhalite units, all lend themselves to the use of geochemical exploration techniques. In the case of the KZK Project this has to date included geological mapping of alteration, and soil, silt and rock chip geochemical surveys. The mineralization itself may, as in the case of the ABM deposit which comprises a mix of heavy, and variably magnetic and conductive mineral accumulations, be amenable to gravity, magnetic and electrical geophysical (e.g. surface, airborne and downhole EM, IP) techniques, all of which have been utilized at the Project to date.

All of these techniques have been applied with an iterative approach to identify broader areas of prospectivity followed by subsequent surveys of increasing detail until there is a target defined by multiple encouraging features from several independent datasets. Once targets are generated, they are drilled using diamond drilling methods to the point that the extents of the mineralization are known and are then infill drilled to increase geological confidence.

9 Exploration

Mineral exploration conducted by previous operators within the Project area is discussed in Section 6 (History). Following project acquisition and prior to the beginning of the 2015 field season, BMC undertook extensive data validation, particularly focused on the previously defined ABM and GP4F deposits. Using this information, CSA Global reported BMC's maiden Inferred Mineral Resource for the ABM deposit (Table 6-1).

Throughout 2015 and 2016, majority of the work at the KZK Project was focused on future development of the ABM deposit but did include modest exploration programs near the ABM deposit with the aim to add resources to the mine plan. Elsewhere on the property, exploration built on the work completed previously by Cominco by expanding the property-wide data for developing targets. Emphasis toward developing property-wide targets has increased from 2017 onwards.

To date, two targets that incorporate modelled plates from the VTEM™ survey (Section 9.1.1) have been drill tested. These include the Santorini target located approximately 300 m south of and in the footwall to the ABM Zone and the Rhyolite Peak (Tarawera) target located 1,000 m west-southwest of ABM. Two modelled VTEM™ plates are located in the Santorini target area; however, only one has been tested. Hole K15-327 returned intercepts of 2.65 m of 0.6% Cu, 1.5% Pb, 3.3% Zn and 30 g/t Ag starting at 65.3 m downhole and 1.1 m of 0.6% Cu, 0.2% Pb, 5.5% Zn and 30 g/t Ag starting at 83.0 m downhole. A second hole, hole K15-328 returned 0.8 m of 0.15% Cu, 0.06% Pb, 2.6% Zn and 8 g/t Ag starting at 84.7 m downhole. In the Rhyolite Peak (Tarawera) target area, a single hole K16-415, tested a shallow, moderately north dipping VTEM™ modelled plate and returned 0.52 m of 0.8% Cu, 0.3% Pb, 9.0% Zn and 26 g/t Ag starting at 46.5 m depth.

9.1 Geophysics

9.1.1 Versatile Time-Domain Electromagnetic

In 2015, BMC contracted Geotech to fly a 267 line km VTEM™ survey over the Kudz Ze Kayah formation on the KZK Property (Figure 9-1). The survey lines were flown at an azimuth of 015° with a traverse line spacing of 150 m and tie lines were flown perpendicular to the traverse lines at a spacing of 1,500 m. Subsequent modelling of the 2015 VTEM™ data developed 24 plates of varying size and orientation. The modelled plates are not necessarily the result of mineralization and hence follow-up drilling is required to test the targets.

In 2016, BMC expanded the VTEM™ coverage by 902 line km to cover the entire KZK Property using the same survey parameters and procedures used in the 2015 survey (Figure 9-1). Results from the expanded survey indicate broad formational responses, especially for the Wind Lake formation and structurally influenced magnetic lows. Subsequent modelling of the 2016 VTEM™ data resulted in a further 45 plates for follow up and integration into property-wide target ranking, which is ongoing.

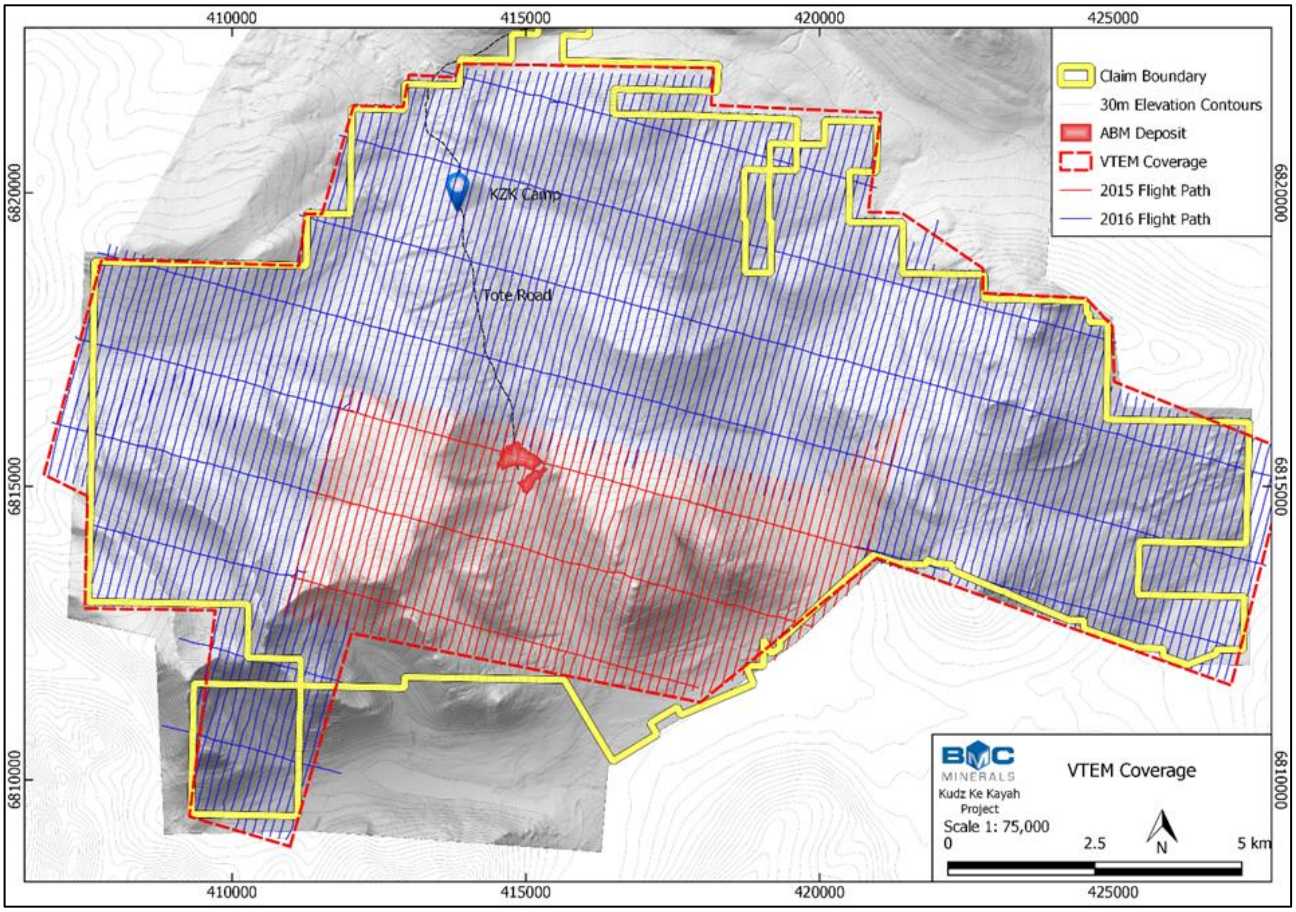

Figure 9-1: VTEM coverage

9.1.2 Ground Electromagnetic

Several fixed loop transient electromagnetic (FLTEM) surveys were conducted through the project area during the 2017 and 2018 field seasons (Figure 9-2). The purpose of these surveys was to target potential mineralization associated with defined geochemical anomalies throughout the project area, as well as stratigraphically down dip from the ABM deposit. Transmitter loops were planned to keep potential survey locations inside the loop and positioned to have good coupling with expected conductor geometry. All loops were modified on the ground to account for topography, potential avalanches and other obstacles.

The 2017 survey, conducted by Discovery Geophysics International out of Saskatoon, SK, Canada, consisted of two loops which covered an area of approximately 6 km^2. A total of 32.9 line-km and 376 stations were surveyed in a north-south line orientation at 200 m line spacing and 50–100 m station spacing.

The following year, Aurora Geoscience out of Whitehorse, YT, Canada was contracted to complete another large survey which consisted of four large loops which covered approximately 25.4 km^2 at various locations through the KZK claim block. A total of 40 lines with 1,451 stations were surveyed over a two-phase program spanning late spring and early summer 2018 (Table 9-1).

The instrument for both 2017 and 2018 surveys consisted of an Electromagnetic Imaging Technology (EMIT) SMART Fluxgate sensor coupled with an EMIT SMARTem24 Receiver unit and involved measurements of the X, Y, and Z components of the secondary EM field (B-field).

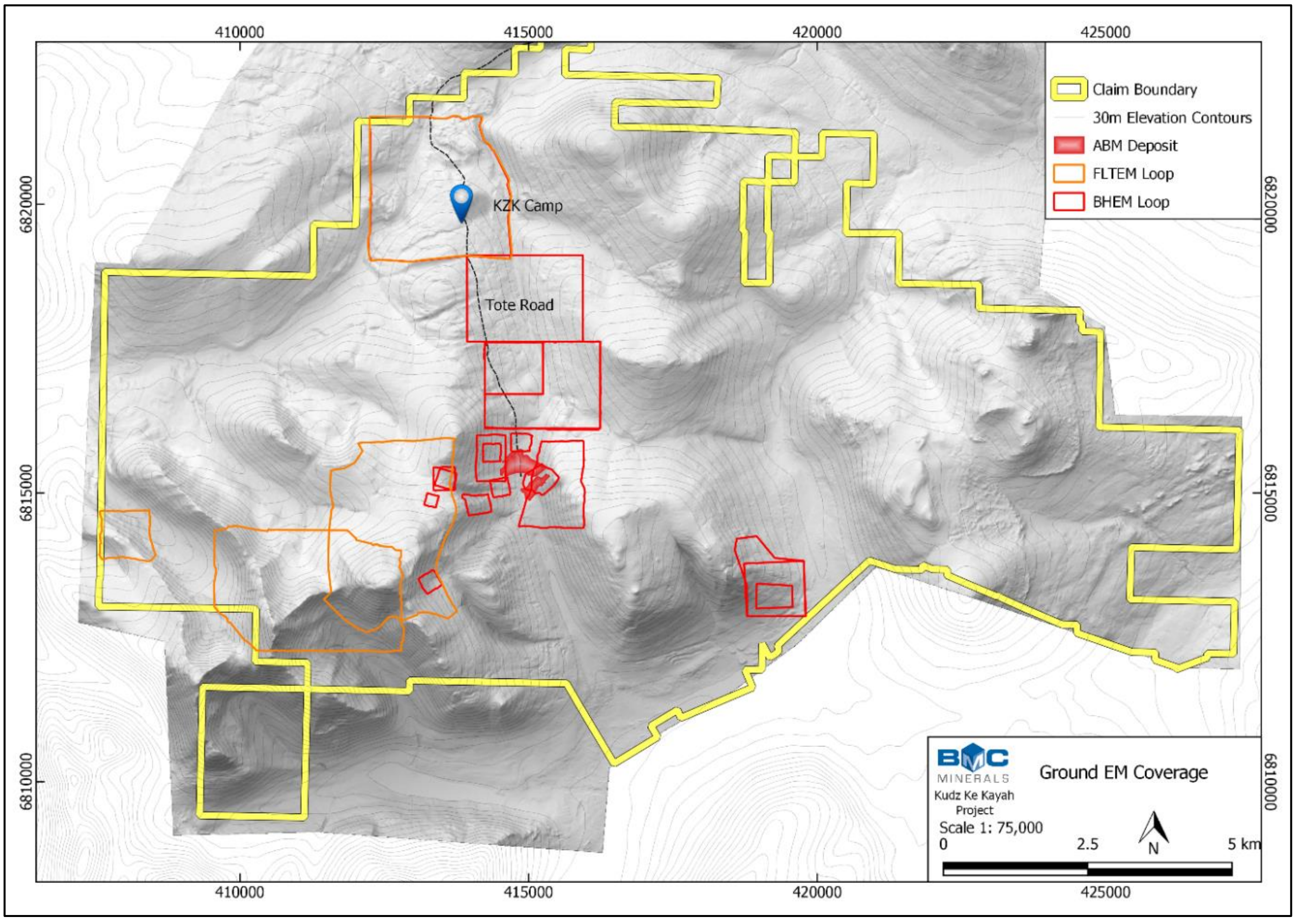

Figure 9-2: FLTEM and BHEM coverage

Table 9-1: FLTEM surveys completed by BMC since 2015

Year	No. of loops	Survey coverage (km^2)	Lines – stations	Survey company
2017	2	6 km^2	10 – 376	Discovery Geoscience International
2018	4	25.4km^2	40 – 1,451	Aurora Geoscience
Total	**6**	**31.4**	**50 – 1,827**	

9.1.3 Ground Gravity

BMC completed a ground gravity survey in 2015 collecting 1,734 readings over a 6 km x 2 km area extending from the ABM deposit to the southeast along a portion of the same prospective stratigraphy that was the focus of the VTEM™ survey. Gravity readings were collected every 50 m along lines spaced 200–300 m apart and trending at an azimuth of 015°. No significant targets derived from the gravity data were deemed worthy of immediate follow up.

9.1.4 Bore Hole Electromagnetic

Ongoing bore hole electromagnetic (BHEM) surveys have also been an integral aspect of BMC's exploration methodology throughout the KZK Project. Aurora Geoscience and Discovery Geoscience International conducted several surveys since 2015 for a total of 14,874 m of drilling in 33 holes over 17 loops (Table 9-2, Figure 9-2).

BHEM surveys used an EMIT SMART Fluxgate sensor coupled with an EMIT SMARTem24 Receiver unit and involved measurements of the X, Y, and Z components of the secondary EM field (B-field) with a base frequency of 5 Hz similar to the surface FLTEM surveys. The BHEM measurements were captured using an EMIT DigiAtlantis borehole probe which measured all three components of the primary EM (B) field; A (axial downhole), U (orthogonal to A and parallel to the azimuth of the drillhole), and V (orthogonal to A and U). The probe was lowered through the hole using a either a winch frame and borehole cable reel or by the wireline on the drill rig. Measurements were made at 20 m intervals throughout the hole with additional infill readings made at 5 m and 10 m increments at the operator's discretion.

Table 9-2: BHEM surveys done by BMC since 2015

Year	No. of loops	No. of holes	Metres surveyed	Survey company
2015	4	8	2,575.5	Aurora Geoscience
2016	4	11	6,790.5	Aurora Geoscience
2017	2	3	2,200	Discovery Geoscience International
2018	7	11	3,308	Aurora Geoscience
Total	**17**	**32**	**14,874**	

9.1.5 Seismic

One two-dimensional (2D) high-resolution seismic reflection survey line was completed on the KZK claim block in 2017 by HiSeis Pty Ltd out of Perth, Western Australia. The primary objective was to image the massive sulphides as well as the regional structural architecture.

The survey line stretched for approximately 10,510 m, dissecting the KZK claim block in a north-south orientation (Figure 9-3). Survey configuration consisted of a 5 m receiver and 10 m source spacing produced by one 50,000 lb IVI HEMI-50 Vibroseis truck. A total of 1,051 unique source points were recorded along the line with final data referenced in NAD83 Z9 UTMs coordinates.

The 2D line was processed using a conventional processing flow adapted to the appropriate site parameters, survey objectives and geology known to exist through the KZK claim block. After trialling a variety of migration algorithms, the final product was a pre-stacked time migration. This product provided the most information and correctly positioned data in space and time.

Depth conversion was guided by the velocity models used in seismic data processing. The downhole seismic and rock property measurements were valuable for use in checking the depth conversion due to the reflectivity present. Caution was used to ensure no artificial structure was introduced by variations within the velocity model.

The result of the survey was positive and appeared to image both the massive sulphide mineralization at the ABM deposit as well as multiple geologic structures in the area. A future 3D seismic survey is being considered over the deposit region.

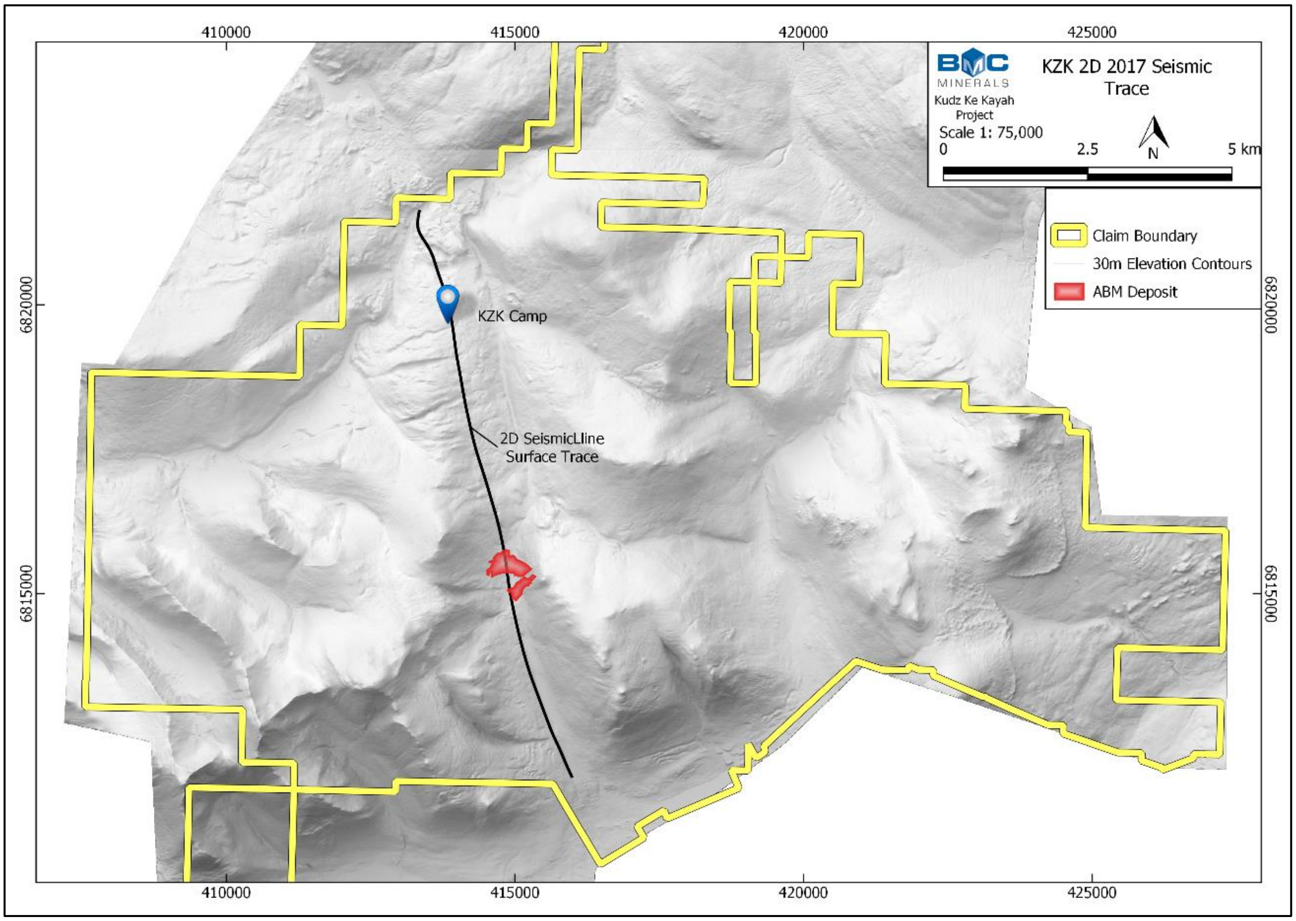

Figure 9-3: Location of the seismic reflection survey line

9.2 Soil Geochemistry

A total of 6,289 B-horizon soils samples have been collected throughout the KZK claim block by the BMC since 2017. The soil grid covers approximately 62.45 km^2 with sample spacing at 100 m x 100 m for most of the grid. The spacing was reduced to 50 m x 50 m within infill regions where anomalism was identified either from historical surveys or in camp analyses using a portable XRF handheld analyser. The soil grid covered the prospective stratigraphy of the Kudz Ze Kayah formation within the southern half of the KZK claim block (Figure 9-4).

Field data was captured on "write-in-the-rain" sheets and then transferred daily into a Microsoft Excel spreadsheet, along with the NAD83 Z9 UTM coordinates that were measured with a handheld Garming global positioning system (GPS) map 62 unit. Data captured at each sampling station includes the soil horizon, colour, texture and moisture content, as well as the station ID, sample depth and vegetation. Station IDs match the geochemical IDs.

Quality assurance and quality control (QAQC) was monitored by inserting blanks (silica powder) and field duplicates into the sample stream, at rates of approximately one for every 40 samples.

Geochemical analyses of all 2017 and 2018 samples collected as part of the soil sampling programs included aqua regia digest with an inductively coupled plasma – mass spectrometry (ICP-MS) finish as well as fire assay/AAS (for gold) at SGS Burnaby lab, BC, Canada (SGS). The SGS lab in Burnaby, BC, Canada is an ISO 9001:2008 certified laboratory (accredited laboratory No. 744) that conforms to the requirements of

CAN-P-1579, CAN-P-1578 and CAN-P-4E (ISO/IEC 17025-2005). The pH values for each sample were recorded at the KZK camp.

In addition to the wet chemistry analysis, all 2017 soil samples were re-analyzed by portable XRF at the SGS lab to evaluate correlation with ICP data and produce data comparable with the 2018 portable XRF work. Following some minor sample preparation at the lab, analysis was done with a Model 800 Bruker S1 Titan portable XRF analyzer, using "SOIL mode" and a run time of three minutes. The portable XRF analyses returned 38 elements, including Cu, Pb, Zn and Ba.

Soil samples and field blanks collected on the 2018 program were also analyzed using an Olympus Vanta portable XRF based at the KZK camp. Each sample was first analyzed in "GEOCHEM" mode for 40 seconds at 40 kV followed by 40 seconds at 15 kV, then in "REE" mode for 30 seconds at 50 kV and 10 seconds at 10 kV. Most elements were captured by the GEOCHEM mode with barium captured in REE mode.

Synthesis of the soil results are based on the wet chemistry analyses from SGS. Percentiles were calculated with the PERCENTRANK function in Microsoft Excel, then binned into six groups to facilitate plotting in QGIS software. The six bins are equal to the <50th percentile, 50–80, 80–90, 90–95, 95–99 and >99th (bins 1–6 respectively). A geochemical anomaly has been defined by:

- Two adjacent samples with ≥90th percentile
- A single sample >99th percentile.

A total of 27 areas of anomalous geochemistry (Figure 9-4) have been identified ranging in extents from approximately 26,000 m^2 to 870,000 m^2.

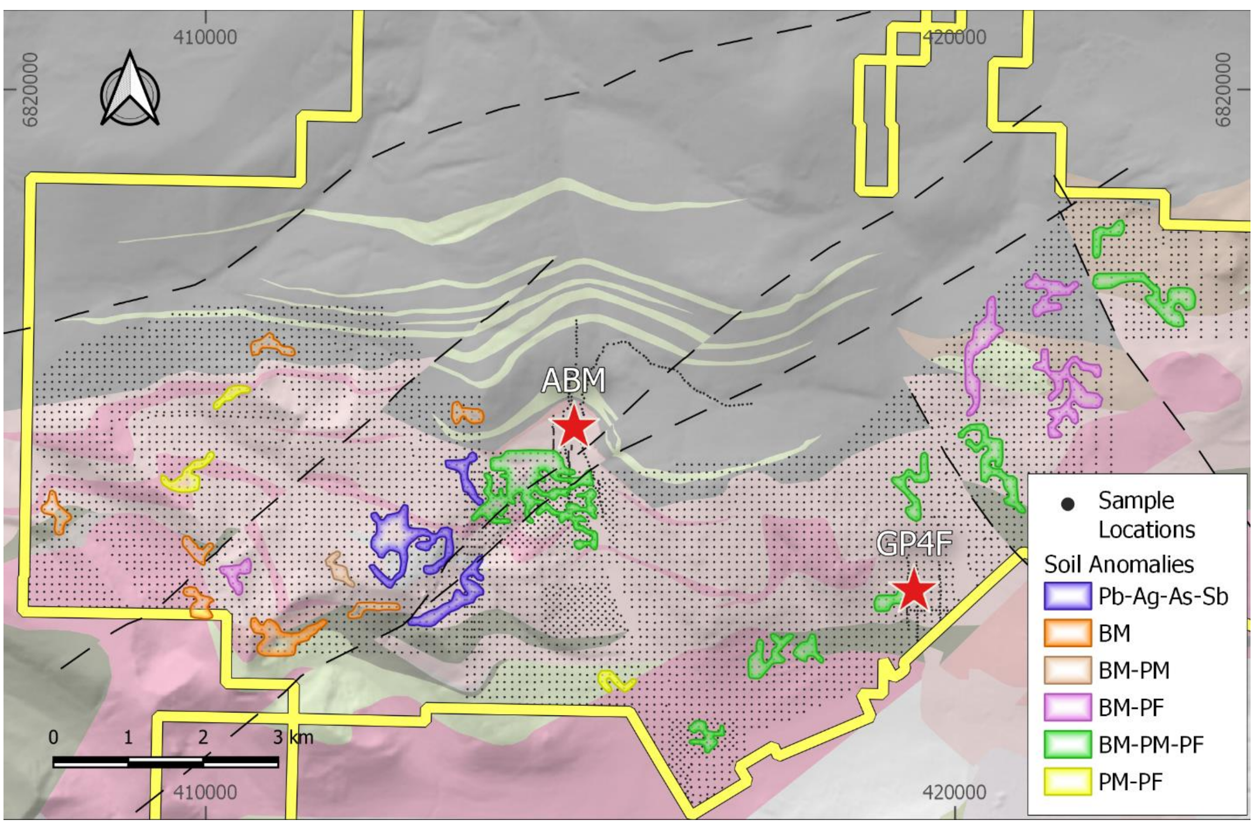

Figure 9-4: Soil geochemistry samples from the KZK Project and discrete multi-point anomalies

Soil anomalies are divided into several associations: Base metal (BM: Cu, Pb and Zn), precious metals (PM; Au and Ag), and pathfinder elements (PF; As, Bi, Sb, Sn, Tl and W).

10 Drilling

10.1 Drilling Summary

10.1.1 Historical (Pre-2015)

The ABM deposit and GP4F Zone have been sampled using diamond drilling (DD) only.

All drilling at the KZK Project prior to 2015 was completed by Cominco.

DD completed by Cominco in 1994 and 1995 targeting the ABM deposit was carried out by DJ Drilling of Surrey, British Columbia. In 1994, helicopter movable Boyles 25A and Longyear LF70 rigs were used, whilst in 1995 two Longyear 38 drill rigs were operating. Drilling in 1997 and 1998 appears to have also been conducted by DJ Drilling, although details of the drill rigs have not yet been located. Majority of drilling conducted in 1998 was situated at GP4F.

Historical DD drilling by Cominco is NQ size. Core was generally sampled at 1.5 m lengths across all drilling programs.

10.1.2 BMC (2015 Onwards)

The 2015, 2016 and 2017 ABM drilling programs consisted of hydrogeological, metallurgical, resource confirmation/infill, and geotechnical drillholes (Table 10-1 to Table 10-3). Some drillholes were designed to test several program objectives. 2018 drilling focus shifted to drill testing multiple exploration targets throughout the KZK claim block (Table 10-4).

Table 10-1: Summary of the 2015 KZK Project drilling program

Drilling type	No. of holes	Total metres	Comments
Exploration	21	6,548	GP4F, ABM, FCZ, Santorini, Krakatoa
Hydrogeological	11	325	
Metallurgical	29	3,406	Includes twin and wedge holes
Resource definition	78	14,732	ABM, Krakatoa, GP4F
Geotechnical	9	955	ABM
Total 2015 Program	**148**	**25,966**	

Source: OMI database

Table 10-2: Summary of the 2016 KZK Project drilling program

Drilling type	No. of holes	Total metres	Comments
Exploration	16	8,155	ABM, Krakatoa, Sebesi, Tarawera
Hydrogeological	9	267	Site infrastructure
Metallurgical	7	1,055	ABM, Krakatoa
Resource definition	37	8,462	ABM, Krakatoa, GP4F
Geotechnical	15	1,270	Krakatoa, site infrastructure
Total 2016 program	**84**	**19,210**	

Source: OMI database

Table 10-3: Summary of the 2017 KZK Project drilling program

Drilling type	No. of holes	Total metres	Comments
Exploration	4	2,451.0	ABM down dip, ABM
Hydrogeological	13	511.8	Site infrastructure
Geotechnical	31	1,965.9	Site infrastructure, proposed ABM pit
Total 2017 program	**48**	**4,928.7**	

Source: OMI database

Table 10-4: Summary of the 2018 KZK Project drilling program

Drilling type	No. of holes	Total metres	Comments
Exploration	12	3,722.4	Rhyolite Peak, Kuril, Kermadec, ABM NW
ARD Drilling	9	332.5	Proposed pit footprint
Total 2018 Program	**48**	**4,054.9**	

Source: BMC database

Most of the 2015 drilling was completed by Geotech Drilling Ltd of Prince George, BC, Canada utilizing four skid-mounted diamond drill rigs (two Zinex A5 and two Hydrocore 2000 machines). The remaining hydrogeological holes were completed by Midnight Sun Drilling of Whitehorse, YT, Canada (Hughes and Baknes, 2015).

Majority of drilling in the 2016 field season was completed by Hytech Drilling Ltd of Smithers, BC, Canada, utilizing skid-mounted Tech 5000 diamond drill rigs and the remaining holes were completed by New Age Drilling Solutions of Whitehorse, YT, Canada using two Zinex A5 diamond drills.

Drill programs in 2017 and 2018 were completed predominately by New Age Drilling of Whitehorse, YT, Canada utilizing both a skid mount and helicopter supported Zinex A5 diamond drill rig. Midnight Sun Drilling of Whitehorse, YT, Canada also supported the 2017 geotechnical drilling with a track mounted Prospector drill rig.

A summary of all drilling within the ABM deposit is provided in Table 10-5.

Table 10-5: Summary of drilling at the ABM deposit – ABM and Krakatoa zones

Year	Type	No. of holes	Total metres
1994	DD	51	8,382
1995	DD	112	14,046
1997	DD	8	2,501
2015	DD	124	21,546
2016	DD	40	9,308
2017	DD	1	182
Total		**335**	**55,964**

Source: OMI database

Drilling at the GP4F Zone in 2015 consisted of 10 holes for a total of 3,291 m and aimed to follow up historical Cominco intercepts, along with testing for extensions of the defined mineralization. Drilling was designed to reduce the drill spacing to nominal 50–75 m pierce points and to test the variability in mineralization between widely spaced historical drillholes. The program was also designed to expand the GP4F Zone to the east and down dip of the historically-defined mineralized footprint.

The 2016 GP4F drilling program consisted of seven holes for a total of 1,554 m. Two holes (K16-393 and K16-400) were designed to test the up-dip extension of the main mineralized horizon. The remaining holes were drilled as infill holes.

A summary of all drilling programs undertaken at the GP4F Zone is shown in Table 10-6.

Table 10-6: Summary of GP4F Zone drilling programs

Year	Company	No. of holes	Total metres	Hole IDs
2015	BMC	10	3,291	K15-224, -234, -247, -261, -268, -280, -285, -294, -302, -306
2016	BMC	7	1,554	K16-380, -388, -393, -396, -400, -403, -407
Total		**27**	**4,845**	

Source: OMI database

Drill core was HQ3, HQ and NQ3 in size. Mineralized samples had a nominal sample length of 1.0 m adjusted to geological boundaries to a minimum sample length of 0.3 m, with barren or poorly mineralized host rock samples typically having maximum and minimum sample lengths of 1.5 m and 0.5 m respectively.

Figure 10-1 shows a Geotech Drilling Hydracore 2000 skid-mounted rig drilling at the ABM Zone in 2015. Figure 10-2 shows resource drilling being completed at the GP4F Zone in 2015.

Figure 10-1: Geotech Drilling HC2000 drill rig at the ABM Zone on hole K15-291
Source: Green, 2015a

Figure 10-2: Geotech Drilling HC2000 drill rig at the GP4F deposit during the 2015 field season
Source: Green, 2015a

The 2017 drill program focused predominantly on geotechnical drilling around the proposed mine site infrastructure and pit design. Additional to the geotechnical program, four exploration drillholes were drilled, three of which targeted the down-dip continuation of the KZK stratigraphy to the north of the ABM deposit. The fourth drillhole targeted ABM stratigraphy and mineralization within the ABM deposit to characterize the acoustic impedance of these features and determine that seismic reflection profiling is a viable exploration technique through the KZK claim block.

The 2018 drilling focus switched towards testing priority drill targets defined by geochemical and geophysical survey within 3–4 km from the ABM deposit (Table 10-7). A total of 12 exploration drillholes were completed on five exploration targets throughout the KZK claim block. Two sub-economic intersections returned from the drilling including two separate zones at the Rhyolite Peak Prospect consisted of massive and semi-massive sulphide mineralization. Drillhole K18-484, located 1.6 km west-southwest of the ABM deposit, intercepted massive sulphide with similar characteristics as ABM mineralization and returned assays of 3.8 m @ 0.4% Cu 1.3% Pb, 5.1% Zn, 128 g/t Ag and 0.8 g/t Au from 7.5 m. Approximately 1.4 km to the south of K18-484 and 2.5 km southwest of the ABM deposit, K18-480 also intercepted semi-massive mineralization consisting of bands of sphalerite in carbonaceous mudstones. Assays returned 12.61 m @ 0.13% Cu, 0.8% Pb, 1.97% Zn, 24.4 g/t Ag and 0.05 g/t Au from 20.3 m (Table 10-8).

Table 10-7: Exploration diamond drillhole specifications

Hole ID	Prospect	Easting (m)	Northing (m)	Elevation (m)	Azimuth (°)	Dip (°)	Length (m)	Comments
K15-327	Santorini	414639	6815178	1,442	195	-60	221.0	Mineralized
K15-328	Santorini	414678	6815205	1,429	195	-60	236.0	No significant results
K15-329	FCZ	414801	6814638	1,422	180	-55	200.0	No significant results
K15-332	FCZ	414797	6814638	1,422	180	-83	182.0	No significant results
K16-368	Sebesi	415221	6815020	1,422	233	-65	145.0	No samples taken
K16-372	Sebesi	415124	6814798	1,388	240	-60	597.0	No significant values
K16-374	Sebesi	415450	6814995	1,513	245	-60	801.0	No significant values
K16-384	Sebesi	415661	6815386	1,563	224.9	-65.1	400.0	No significant values
K16-391	Sebesi	415655	6815385	1,568	224.8	-64.9	101.0	No significant values
K16-394	Sebesi	415655	6815385	1,568	233	-65	231.6	Mineralized
K16-408	Sebesi	415424	6815263	1,548	228	-67	491.0	No samples taken
K16-413	Sebesi	415800	6815170	1,595	240	-65	147.0	No samples taken
K16-414	Sebesi	415800	6815170	1,595	245	-65	339.0	No samples taken
K16-415	Rhyolite Peak	413575	6815210	1,712	195	-60	854.5	Mineralized
K16-416	Sebesi	415440	6815276	1,550	183	-68	854.5	No samples taken
K18-467	Kermadec	413413	6820445	1,332	161.86	-61.04	590.0	No significant results
K18-468	Kermadec	413670	6820318	1,341	177.67	-70.29	56.0	No significant results
K18-469	Kermadec	413672	6820318	1,341	172.12	-70.49	692.0	No significant results
K18-480	Rhyolite Peak East	413374	6813450	1,732	147.7	-69.5	215.4	Mineralized
K18-481	Kuril	414060	6814770	1,642	169.03	-51.35	300.0	No significant results
K18-482	ABMNW	414354	6815675	1,515	180	-60	401.0	No significant results
K18-483	Rhyolite Peak North	413609	6815315	1,715	195.86	-60.15	297.0	No significant results
K18-484	Rhyolite Peak North	413310	6814850	1,737	196	-61	135.0	Mineralized
K18-485	Rhyolite Peak East	413249	6813442	1,770	151.13	-69.94	201.0	No significant results
K18-486	Kuril	414184	6814844	1,560	169	-51	251	No significant results
K18-487	Santorini	414511	6815092	1,485	168.85	-49.67	359.0	Mineralized
K18-488	Santorini	414509	6815092	1,485	252.28	-89.58	225.0	Mineralized

Table 10-8: Significant exploration drilling Intercepts

Hole ID	Prospect	From (m)	To (m)	Interval (m)	Estimated true width (m)	Cu (%)	Pb (%)	Zn (%)	Ag (g/t)	Au (g/t)
K15-327	Santorini	65.3	67.9	2.6	2.6	0.6	1.5	3.3	30	0.04
and		142.7	143.1	0.4	0.4	0.3	1.0	4.8	51	0.05
K15-328	Santorini	84.67	85.5	0.8	0.8	0.2	0.1	2.6	8	0.0
K16-394	Sebesi	797.4	797.9	0.5	unknown	0.50	0.20	9.30	8	0.00
K16-415	Rhyolite Peak	46.5	47.0	0.5	unknown	0.80	0.30	9.00	26	0.10
K18-480	Rhyolite Peak East	20.3	33.0	12.6	12.6	0.1	0.8	2.0	24	0.05
including		31.0	33.0	2.0	2.0	0.0	2.9	7.1	89	0.13
K18-484	Rhyolite Peak North	7.5	11.3	3.8	3.8	0.4	1.3	5.1	128	0.80
K18-487	Santorini	47.0	52.0	5.0	5.0	0.4	0.0	0.2	9	0.02
K18-488	Santorini	68.9	71.2	2.3	2.3	1.1	0.2	1.3	35	0.03

Other significant findings include strong alteration and weak mineralization at ABM NW and several chlorite-altered and/or mineralized zones on the Santorini-Kuril trend. Further drilling is recommended to follow up on the 2018 drilling results at the Rhyolite Peak, ABM NW, and Santorini-Kuril prospects.

10.2 Collar Surveying

Many of the early drillholes appear to have been drilled on a truncated regional grid (with first few digits removed for ease of use). Following completion, the 1994 Cominco drillhole collars were surveyed by qualified surveyors, McElhanney Consulting Services Limited of Vancouver, BC. The holes were surveyed using static GPS vectors and adjusted by least squares to within two decimal places and are considered accurate; this is supported by differential GPS pickups of many of the historical collars during the 2015 field season (Figure 10-3).

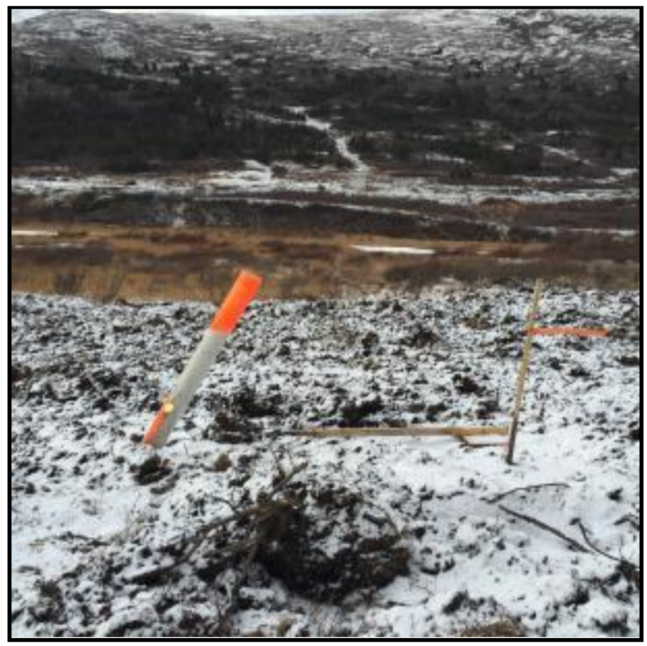

Figure 10-3: Historical drill collar (left) and 2015 collar (right) at the ABM deposit
Source: Green, 2015b

Details of surveying for post-1994 (but pre-2015) drilling have not been identified; however, the majority of these holes were located and resurveyed during the 2015 field season.

A total of 84 Cominco collars (66 from ABM) were located, verified and surveyed by Challenger Geomatics Ltd (Challenger) of Whitehorse, Yukon in 2015 using Leica Viva real-time kinematic (RTK) GNSS resulting in location accuracy of 0.25 m.

A total of 158 holes drilled in the 2015 and 2016 field seasons by BMC at ABM were surveyed by Challenger using Leica Viva (RTK) GNSS resulting in location accuracy of 0.25 m. This survey was completed with an RTK differential GPS with radio base stations set up in proximity to drilling sites to provide real time kinematic corrections. The remaining six drillholes were located via an azimuth positioning system (APS) for X and Y coordinates and Z from a DEM derived from a light detection and ranging (LiDAR) survey. The APS unit is capable of accuracy down to less than 1 m and the vertical precision of the LiDAR survey is 0.1 m.

During the 2016 field program, BMC resurveyed all Cominco collar locations at GP4F, with the exception of K95-167, using a RTK-GPS system the results of which confirmed the accuracy and location of the historical surveys. The single 1995 hole (K95-167) was surveyed at the collar and end-of-hole using a "single shot" survey.

BMC drilling indicates no significant major issue with Cominco drillhole survey data.

All surveys were completed in UTM Zone 9 NAD83.

10.3 Downhole Surveying

10.3.1 ABM Deposit

Based on the supplied database, it would appear that the majority of historical holes used "single-shot", acid-etch style surveys taken approximately every 30 m downhole. Exact details of the historical downhole survey methods for the various drilling programs have not been located.

For the 2015 drilling, the drill rig was aligned to the planned azimuth with the help of a Reflex APS. The APS is a GPS-based compass that is not affected by local magnetic interference (natural or manmade) and produces true north azimuth measurements to within 0.5° with good GPS integrity.

Downhole surveys were completed using a Reflex EZ-Shot system, a "single-shot" high precision magnetic instrument that measures the drillhole azimuth relative to magnetic north as well as the drillhole dip and magnetic field strength. Magnetic north azimuth readings are corrected to grid north azimuths by adding 22.5°. The first downhole survey was completed once the drillhole had penetrated several metres into bedrock, followed by 25 m intervals for the rest of the hole. Downhole surveys were not accepted if the corrected azimuth was significantly different from the azimuths on either side of it, which is usually a result of localized magnetic field interference. In general, magnetic field strengths of 5600–6000 nT were accepted whereas those below 5600 nT and above 6000 nT indicated magnetic interference (Hughes and Baknes, 2015).

In 2016, 30 holes were surveyed using a Reflex Gyro non-magnetic Instrument upon completion of the holes. The use of the Gyro orientation tool continued for all inclined holes in bedrock for the 2017 and 2018 exploration seasons.

10.3.2 GP4F Zone

Based on the supplied database, it would appear that the majority of historical holes used "single-shot", acid-etch style surveys taken approximately every 30 m downhole. Exact details of the historical downhole survey methods for the various drilling programs have not been located.

For the 2015 drilling, the drill rig was aligned to the planned azimuth with the help of a Reflex APS. The APS is a GPS-based compass that is not affected by local magnetic interference (natural or manmade) and produces true north azimuth measurements to within 0.5° with good GPS integrity.

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In 2016, all holes at the GP4F Zone were surveyed using a Reflex Gyro non-magnetic instrument upon completion of the holes. No drilling has been undertaken at GP4F since 2016.

10.4 Drilling Orientation

10.4.1 ABM Deposit

The ABM Zone was drilled towards grid south at angles ranging from –30° to vertical (–90°) to intersect the mineralized zones close to perpendicular for the bulk of the deposit.

The Krakatoa Zone was drilled towards grid southwest at varying angles to obtain close to true width intersections. This drilling orientation was also selected to avoid drilling down the bounding faults, an issue that occurred in the 1997 drilling program which caused abandonment of at least one drillhole and contributed to missing some drill targets. It is evidence that once a drill string has entered the faults at an oblique angle, the drill string will remain within the bounds of the fault zone.

10.4.2 GP4F Zone

GP4F drillholes were generally angled (–45° to –90°) towards grid south with dip angles set to optimally intersect the mineralized horizon. Of the 27 holes drilled at GP4F, two were drilled vertically.

10.5 Drill Sample Recovery

10.5.1 Historical (Pre-2015)

The drilling database only contains sample recovery data for holes drilled in 1998. Core loss was recorded over some mineralized intervals, although the weighted average recovery recorded for all recorded intervals was 93%.

The 1995 PFS document reported:

- "Preliminary rock quality data interpreted from drill core, indicates that the western and central portions of the deposit yield fair (50–75%) to good (75–90%) rock quality designation (RQD) values while the eastern portion yields poor (25–50%) to fair (50–75%) RQD values. Artesian conditions were encountered at depth in several boreholes though the deposit area."

Recovery records for the remaining holes are not in the database.

10.5.2 BMC (2015 Onwards)

For all drilling programs, recovery and RQD was recorded for all holes. Rock quality was good with recovery values averaging greater than 90% and RQD values confirm the same distribution delineated in the historical data. Special attention was paid to recovery through mineralized intervals. Several mineralized intervals were redrilled to achieve better recoveries.

10.6 Logging

10.6.1 Historical (Pre-2015)

Mineralization and host rock lithologies were logged in detail in 1994 to develop a legend suitable for coding in GEORES (database software for resource modelling) format. Special attention was paid to defining variations in sulphide types within the mineralized intervals and to describing immediate hangingwall and footwall units (Cominco Ltd, 1995).

All core from the 1994 drill program, with the exception of the first eight holes, was photographed prior to sampling. Logging procedures for post-1994 holes were not recorded. In 2015, BMC scanned the historical core photographs (generally incomplete sets) and re-photographed key zones (wet and dry) from the historicial core. Both datasets were stored as digital "jpg" files.

Logging procedures for post-1994 holes have not been recorded in the information provided to date; however, it is assumed they were logged in detail according to methods adopted in earlier programs. Summary logs for all 1998 drillholes are presented in MacRobbie and Holroyd (2000).

10.6.2 BMC (2015 Onwards)

Geological drillhole data was captured by the geologists' using GeoSpark software, a Microsoft Access-based relational database system that stores drillhole and geological data in individual data tables.

For the ABM drilling program, it was necessary that geological information was logged at the appropriate resolution for resource modelling and geo-metallurgical interpretations. Attention was paid to identifying mineralization types (principally based on the original Cominco logging of "ore" types), deposit alteration types, hangingwall and footwall lithology as well as sulphide minerals, carbonate content and oxidation intensity for use in potential acid-generating (PAG) geodomain interpretations (Baker, 2015).

All drill core was photographed (wet and dry) prior to sampling and stored as digital "jpg" files. Core logging facilites are shown in Figure 10-4.

Figure 10-4: Core logging facilities at the BMC KZK exploration camp
Source: Green, 2015b

10.6.3 Historical Relogging Program

A relogging program of archived drill core from all historical holes at the KZK Project were undertaken ahead of and during the 2015 drilling program. The program sought to design a system of standardized logging for both relogging and new drilling, bringing all historical logs up to the new simplified standard and allow for the creation of a new geological model (Voordouw et al., 2016). Relogging was accomplished from a fly camp established adjacent to the historical core archive (Figure 10-5).

Figure 10-5: Historical Cominco core storage yard at KZK exploration camp
Source: Green, 2015b

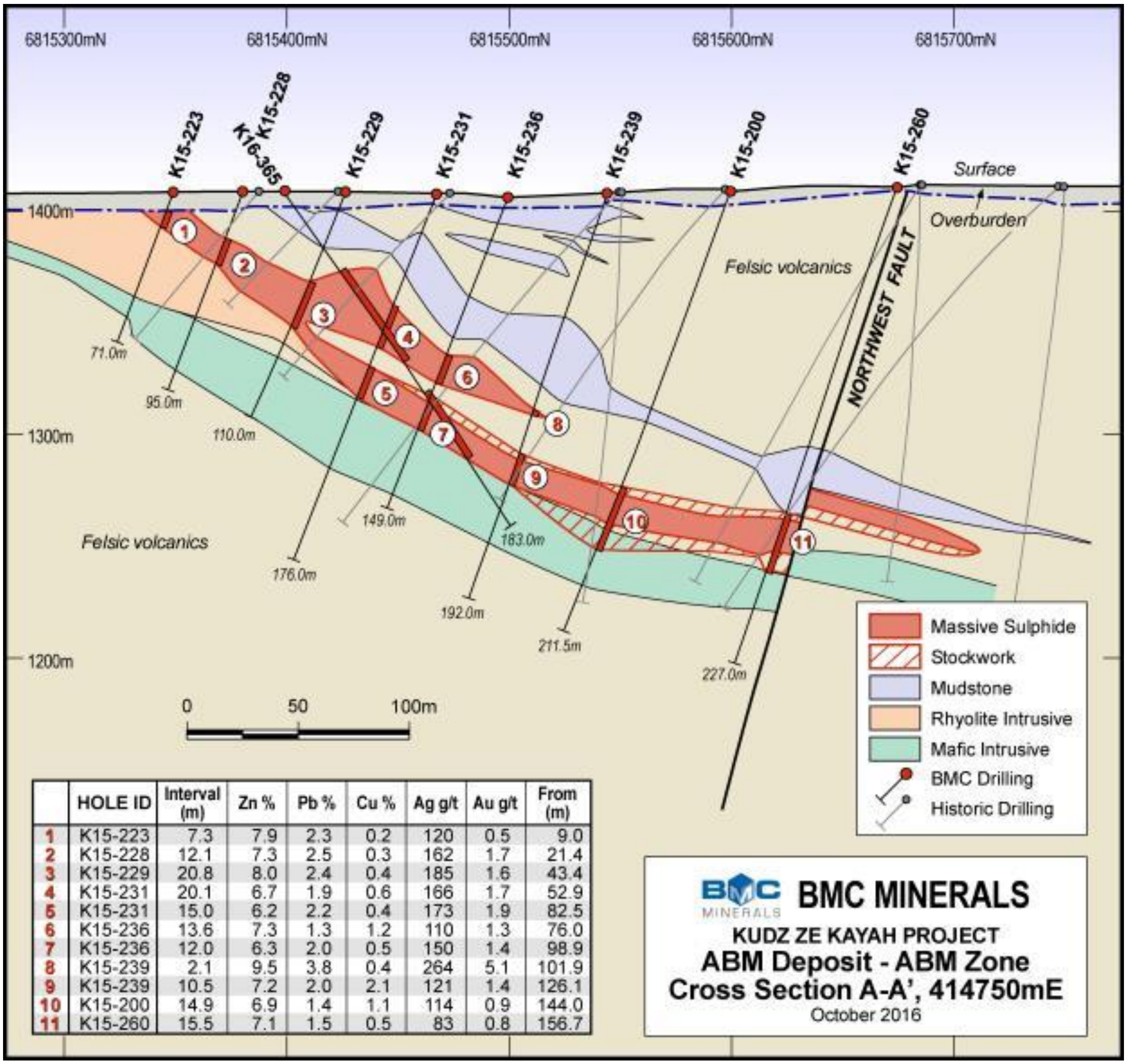
The relogging program also included selective sampling of massive sulphides and adjacent host rock to confirm historical grades such that historical analytical data could be included in new resource estimates and future reserves.

A total of 174 holes were relogged for a total of 24,953 m, comprising most of the historical ABM and Fault Creek Zone drilling as well as two holes from GP4F. Six ABM holes were not re-logged, including three holes that could not be located (K94-001, K94-002, K95-169), one hole that was abandoned at 10 m depth (K97-183A), one hole with too much missing core (K95-168) and one hole that was found but inadvertently not logged (K97-176). Besides relogging drill core, the 2015 program also included logging the footage (core) blocks, relabelling with aluminium tags, completing an inventory of boxes and creating a map of the core storage area (Baknes, 2015).

The program was completed by the end of 2016. All logs were combined with new drill logs into a complete standardized dataset.

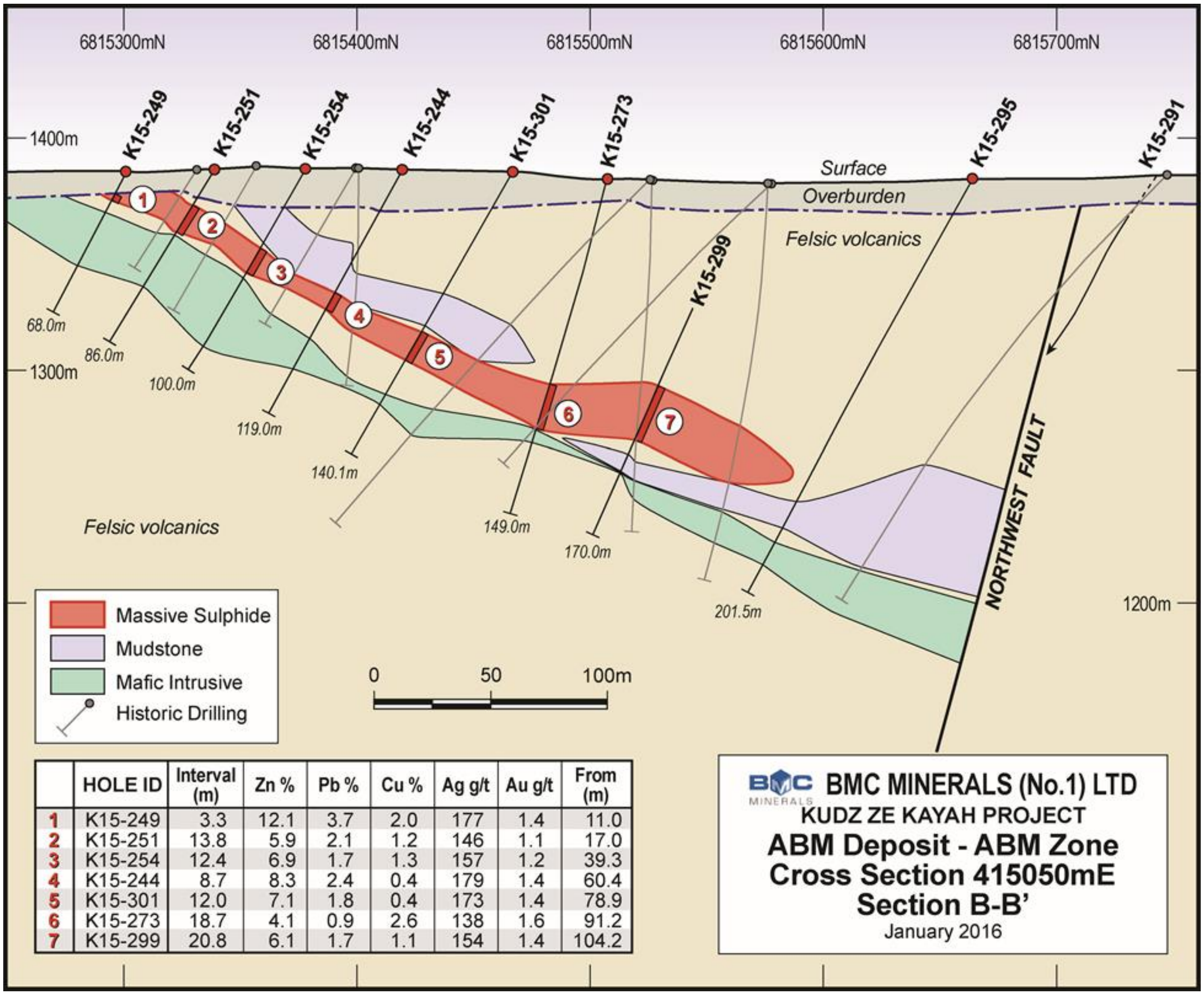
10.7 Significant Intercepts

Some significant intercepts from the 2015 drilling campaign are shown on the schematic cross-sections below (Figure 10-6 to Figure 10-8).



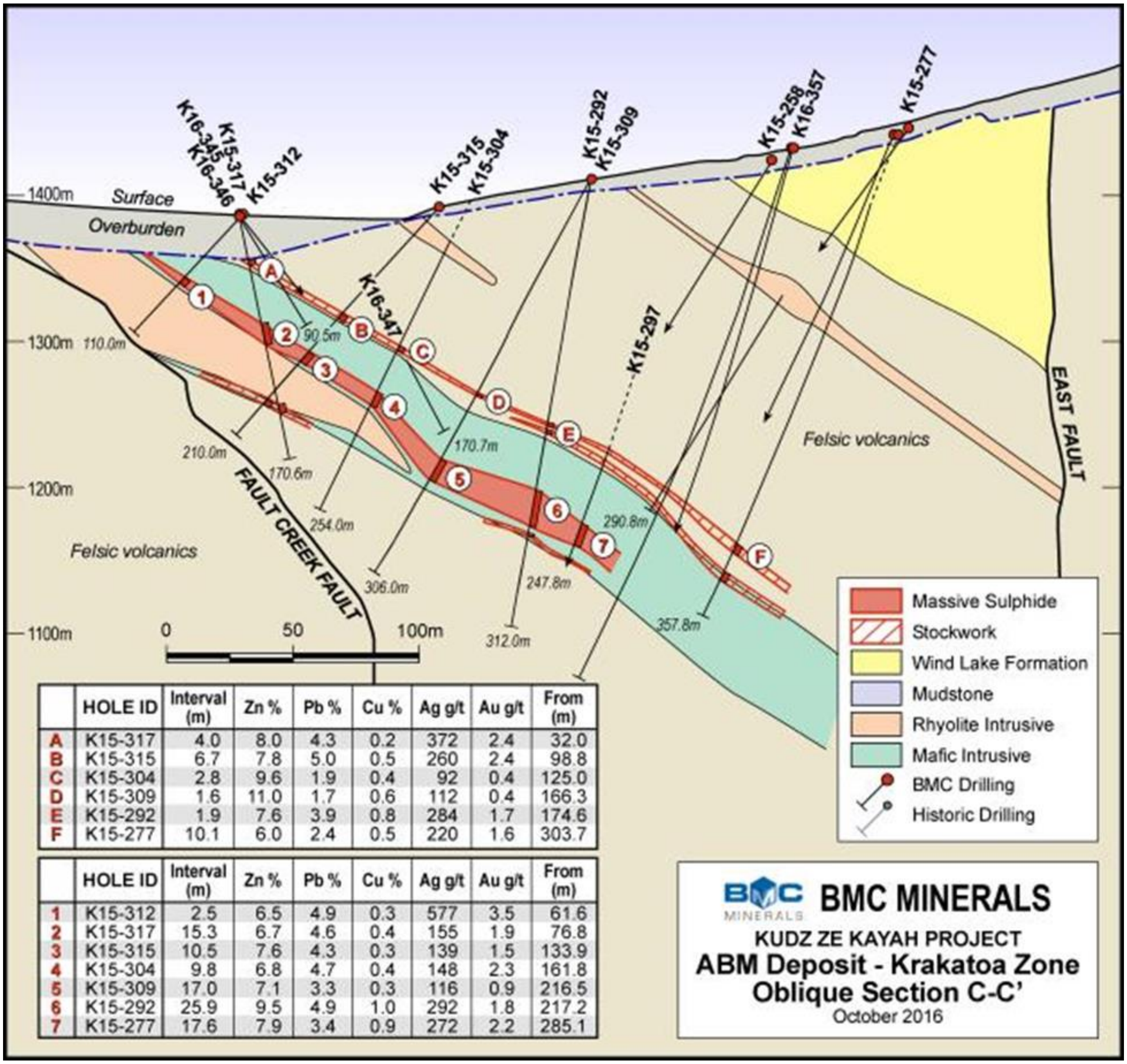
	HOLE ID	Interval (m)	Zn %	Pb %	Cu %	Ag g/t	Au g/t	From (m)
1	K15-223	7.3	7.9	2.3	0.2	120	0.5	9.0
2	K15-228	12.1	7.3	2.5	0.3	162	1.7	21.4
3	K15-229	20.8	8.0	2.4	0.4	185	1.6	43.4
4	K15-231	20.1	6.7	1.9	0.6	166	1.7	52.9
5	K15-231	15.0	6.2	2.2	0.4	173	1.9	82.5
6	K15-236	13.6	7.3	1.3	1.2	110	1.3	76.0
7	K15-236	12.0	6.3	2.0	0.5	150	1.4	98.9
8	K15-239	2.1	9.5	3.8	0.4	264	5.1	101.9
9	K15-239	10.5	7.2	2.0	2.1	121	1.4	126.1
10	K15-200	14.9	6.9	1.4	1.1	114	0.9	144.0
11	K15-260	15.5	7.1	1.5	0.5	83	0.8	156.7

Figure 10-6: Schematic cross-section 414,750 m E looking west through the ABM Zone with selected 2015 downhole intercepts



	HOLE ID	Interval (m)	Zn %	Pb %	Cu %	Ag g/t	Au g/t	From (m)
1	K15-249	3.3	12.1	3.7	2.0	177	1.4	11.0
2	K15-251	13.8	5.9	2.1	1.2	146	1.1	17.0
3	K15-254	12.4	6.9	1.7	1.3	157	1.2	39.3
4	K15-244	8.7	8.3	2.4	0.4	179	1.4	60.4
5	K15-301	12.0	7.1	1.8	0.4	173	1.4	78.9
6	K15-273	18.7	4.1	0.9	2.6	138	1.6	91.2
7	K15-299	20.8	6.1	1.7	1.1	154	1.4	104.2

Figure 10-7: Schematic cross-section 415,050 m E looking west through the ABM Zone with selected 2015 downhole intercepts



	HOLE ID	Interval (m)	Zn %	Pb %	Cu %	Ag g/t	Au g/t	From (m)
A	K15-317	4.0	8.0	4.3	0.2	372	2.4	32.0
B	K15-315	6.7	7.8	5.0	0.5	260	2.4	98.8
C	K15-304	2.8	9.6	1.9	0.4	92	0.4	125.0
D	K15-309	1.6	11.0	1.7	0.6	112	0.4	166.3
E	K15-292	1.9	7.6	3.9	0.8	284	1.7	174.6
F	K15-277	10.1	6.0	2.4	0.5	220	1.6	303.7

	HOLE ID	Interval (m)	Zn %	Pb %	Cu %	Ag g/t	Au g/t	From (m)
1	K15-312	2.5	6.5	4.9	0.3	577	3.5	61.6
2	K15-317	15.3	6.7	4.6	0.4	155	1.9	76.8
3	K15-315	10.5	7.6	4.3	0.3	139	1.5	133.9
4	K15-304	9.8	6.8	4.7	0.4	148	2.3	161.8
5	K15-309	17.0	7.1	3.3	0.3	116	0.9	216.5
6	K15-292	25.9	9.5	4.9	1.0	292	1.8	217.2
7	K15-277	17.6	7.9	3.4	0.9	272	2.2	285.1

Figure 10-8: Schematic oblique cross-section looking northwest through Krakatoa Zone (parallel to bounding faults), intersections are measured downhole

11 Sampling Preparation, Analysis and Security

11.1 Sampling Techniques

11.1.1 Historical (Pre-2015)

The ABM deposit and GP4F Zone have been sampled using DD only. Historical DD drilling by Cominco is NQ size.

Cominco sampling practices are detailed in its PFS document (1995) for all drilling prior to June 1995. Subsequent Cominco drilling programs were reported in the annual "Year End" reports although minimal detail was provided on drilling and sampling techniques.

In 1994, all sulphide intersections were sawn with an open circulation rock saw, typically into 1.5 m-long samples, and sent to CERL in Vancouver. Samples were subjected to a three-stage crushing procedure, pulverized, and screened to –150 mesh prior to aqua regia digestion (and solvent extraction in the case of Au) and assaying.

In 1995, all samples were split by hydraulic splitter, typically to 1.5 m lengths, and subject to the same preparation and analytical procedures as described above.

Details of the post-1995 sampling and preparation procedures have not been located but the Qualified Person considers that it is reasonable to assume that similar procedures have been followed to those used previously.

11.1.2 BMC (2015 Onwards)

The 2015 ABM deposit resource confirmation and infill drilling program was designed to twin mineralized intersections to confirm the existence and tenor of massive sulphide in historical holes on sections 414,750 m E, 414,850 m E and 415,050 m E, and to permit resampling of the historical core on these sections for geochemical analysis as an aid to establishing the veracity of the historical geochemical dataset.

Subsequent ABM drilling programs consisted of hydrogeological, metallurgical, resource confirmation/infill, and geotechnical drillholes. Drill core included HQ3, HQ and NQ3 sizes.

Geologists identified all core samples with a unique sample identification number and marked all sample intervals with sample tags in the core box. All mineralized intersections were sampled, including approximately 10 m of the immediate hangingwall and footwall host rock in an effort to characterize waste rock dilution that could be encountered during mining. Primary core samples conformed to lithological boundaries where possible, with core loggers making an attempt to constrain alteration and mineralization features within the lithological boundaries as well. Mineralized samples had a nominal sample length of 1.0 m and a minimum length of 0.3 m with barren or poorly mineralized host rock samples typically having maximum and minimum sample lengths of 1.5 m and 0.5 m respectively.

Drill core samples were cut using an open circulation rock saw. Split core was consistently sampled from the same side and placed into labelled plastic sample bags that were sealed with a plastic zip-tie and shipped in labelled rice bags sealed with individually numbered security tags. A list of required quartz washes was submitted to the analysis laboratory that would follow suspected high-grade samples in order to avoid contaminating adjacent lower grade samples. These quartz washes were completed at the crush and pulverizing stage of sample preparation.

GP4F Zone drilling used the same sampling techniques as for the ABM deposit.

11.2 Sample Security

No information was available for historical Cominco drilling sample security.

For recent drilling (2015–2018), sample chain of custody is managed wholly by BMC. All samples were placed in poly sample bags labelled with unique sample numbers and equivalent bar-coded sample tags included in the bag. Samples were then packaged in lots of five to 10 in white poly rice sacks. The rice sacks were sealed using fibre tape and uniquely numbered non reusable security seals. Sacks were then palletized and shrink wrapped for shipment to the laboratory. Tracking numbers, bag inventory and security tag information is then provided to the laboratory with instructions to notify upon receipt and of any compromised bags.

All remaining core samples are stored in trays and racks in the core yard at the Exploration Camp (Figure 11-1). Access to the camp and core yard is provided by a single gated tote road that is manned throughout the field season and locked during winter.

Figure 11-1: Core storage yard for BMC drilling at KZK exploration camp

11.3 Dry Bulk Density Determinations

11.3.1 Methodology

In 1994, bulk density determinations were completed on the first 40 drillholes within the ABM deposit outline. A representative 10 cm-long sample was selected from each assayed interval (usually 1.5 m) of mineralization and flanking or intervening waste. Each sample was suspended from a metric balance and weighed in air and again immersed in water ("water immersion method") (Cominco Ltd, 1995). No historical bulk density measurements were taken by Cominco from GP4F drill core.

Bulk densities from the ABM and GP4F deposits were measured in the field by BMC staff on new core samples over the entire length of the sample interval using the water immersion method. This approach differed from that employed by past explorerers, when used only a portion of the drill core from a sample interval yielding potentially unrepresentative data. For this reason, only bulk densities measured during the 2015 and 2016 field season as part of the current drill program were used for resource modelling. Data that were clearly erroneous were culled.

Individual measurements were completed on the entire half-core sample submitted for analysis and on representative 10–15 cm long whole-core samples of various hangingwall and footwall host rock lithologies. The entire half-core samples ranged in weight from as little as 0.14 kg to as much as 9.61 kg, with a median value of 2.15 kg.

Each sample was weighed in air on a metric balance and then suspended below the balance and weighed whilst immersed in water (i.e. the "water immersion method"). When half-core samples were either too friable or broken, bulk density measurements were not completed as material could be lost and would not be representative.

In 2015, calibration readings were taken on a 20 cm long piece of metal rebar approximately every 10th reading and as the last measurement for each drillhole. These calibrations were recorded in sequence with the bulk density measurements. However, the metal rebar was replaced for the 2016 field season with two samples: sealed massive sulphide and sealed rhyolite.

Bulk density determinations for 8,393 ABM samples were originally included in the final database, including repeat analyses of known rock types and a standard reference material. Removal of quality control (QC) samples left 7,556 samples for analysis. Bulk density determinations for 568 GP4F samples were included in the final database, of which 99 samples were situated within the mineralized envelopes.

11.3.2 Results

In order to create a reliable bulk density dataset for estimation, different methods were evaluated to predict bulk densities for samples from the ABM deposit (Arne, 2015a). A tiered approach to the selection of a preferred bulk density value was adopted using the following order of preference:

1. Field bulk measurements, following the removal of statistical outliers.
2. Pycnometer data where no field measurement was available (two samples).
3. Bulk densities calculated by multiple regression analysis using S data, where available, optimized for the highest coefficient of determination.
4. Where S data were absent, bulk densities were calculated using weighted Fe-Cu-Pb-Zn data with the simple exponential regression (1.0*Cu%)+(1.81*Pb%)+(0.97*Zn%)+(1.20*Fe%). This was completed for the cleaned bulk densities for zone 8 and for samples having a bulk density <2.75 g/cm^3 in zones 5, 6 and 7.

Based on the parameters detailed above, calculated bulk densities were derived for 1,027 core samples from 54 holes at ABM, and 1,121 core samples from 24 holes at Krakatoa for interpolation (Arne, 2015a).

For Krakatoa data, measured bulk densities were available for all samples within the mineralization wireframes.

For the mineralized domains, bulk density was estimated using OK, utilizing variogram parameters that were derived for Fe in order to honor the relationship between density and Fe. The average bulk densities determined for the ABM stockwork and massive sulphide mineralization were 3.44 t/m^3 and 4.19 t/m^3

respectively, while the average bulk density values for the Krakatoa Zone were 3.86 t/m³ and 4.09 t/m³ respectively.

Fixed density values were assigned into the block model for each regolith and lithological unit, setting fresh felsic material to 2.76 t/m³ (based on the median of the normal histogram from the measured bulk density dataset), 2.80 t/m³ for the mafic intrusive rock, 2.74 t/m³ for the mudstone and Wind Lake formation, 2.68 t/m³ for the rhyolite intrusive (RHYi), and 2.00 t/m³ for overburden.

11.3.3 Quality Assurance – Density

Detailed analysis of the 2015 QC results for bulk density were outlined in Arne (2015b) and summarized in Green (2016). The following sections only report on the 2016 QAQC program.

Details in the section below are taken from Arne (2016a).

A previous recommendation by CSA Global was to identify one or two suitable rock samples that could be used routinely for QC on the measurement of bulk density in the field. Two such samples were identified and analyzed repeatedly at the beginning of the 2016 drilling program. These included a sample of massive sulphide (MXSX) with an average bulk density of 4.63 g/cm³ that was painted to prevent oxidation, and a sample of rhyolite having an average density of 2.65 g/cm³.

After sufficient data had been collected using a variety of analysts under different conditions, standard deviations were determined from 82 analyses of the MXSX standard and 250 determinations of the rhyolite standard and are 0.058 g/cm³ and 0.015 g/cm³, respectively. These provide a basis upon which to establish control limits for future quality assurance (QA) of bulk density measurements in the field.

The results of plotting the bulk densities for the standards are given in Figure 11-2 and Figure 11-3. The bulk density determinations show good repeatability throughout the 2016 field season, but there are two clear failures that indicate an error in measurement. These represent a very small proportion of the analyses (<1%).

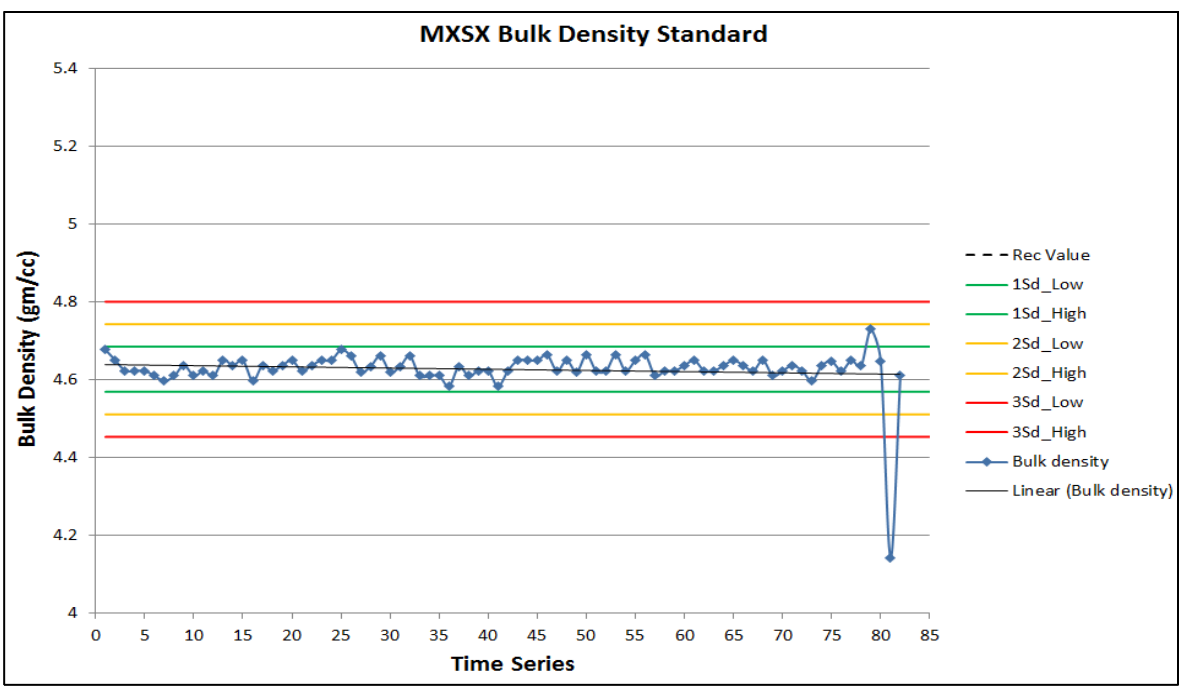


Figure 11-2: Control chart for bulk density determinations of a massive sulphide standard throughout 2016
Source: Arne, 2016a

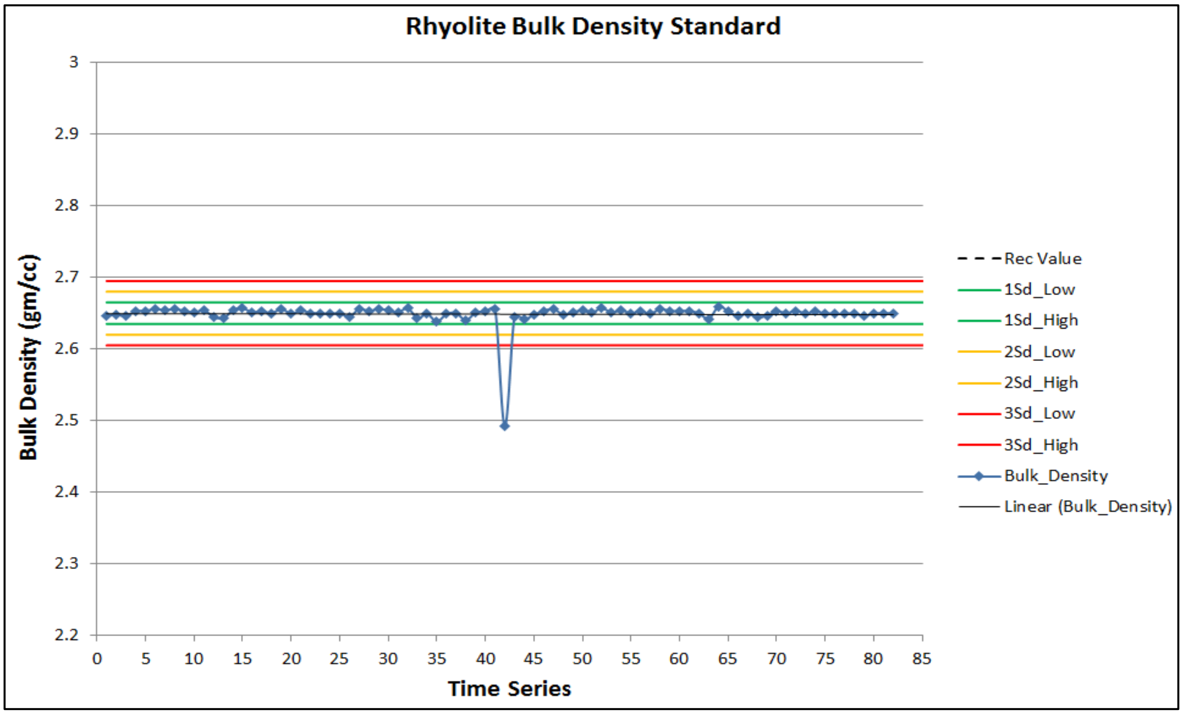


Figure 11-3: Control chart for bulk density determinations of a rhyolite standard throughout 2016
Source: Arne, 2016a

Overall, the QC data show good accuracy and precision and are considered to be more than adequate to support the assay and bulk density data used for the Mineral Resource update of the ABM deposit.

11.4 Sample Analysis

11.4.1 Historical (Pre-2015)

Cominco's 1994, 1995, 1997 and 1998 samples were analysed at it's non-independent Cominco Exploration Research Laboratory (CERL) in Vancouver. It is unknown to the Author and Qualified Person if CERL possessed any certifications at that time.

Following aqua regia digestion (and solvent extraction in the case of Au), all 1994 samples were analysed by AAS for Cu, Pb, Zn, Ag, Au and Fe. Base metals and Fe were then determined using standard wet chemical assay procedures and precious metals were fire assayed. If the sample recorded above the upper detection limit, a second "ore grade" analysis was undertaken through dilution of aliquots.

Ba was determined by pressed pellet/XRF. All samples were also analysed by multi-element ICP.

Similar assaying procedures were used in 1995, although Ba and Fe were not assayed.

In 1997, a total of 349 core samples were collected. Of these, 320 were analysed for 27 elements by ICP, Au by aqua regia decomposition/AAS and Ba by XRF, in addition to whole rock major and minor oxides by XRF and trace elements Zr and Y by pressed pellet AAS (MacRobbie, 1998).

For drilling conducted in 1998, a total of 197 core samples were collected and were analysed for Cu, Pb, Zn, Ag, As, Cd, Co, Ni, Fe, Mo, Cr, Bi, Sb, V, Sn, W, Sr, Y, La, Mn, Mg, Ti, Al, Ca, Na and K by ICP, Au by aqua regia decomposition/AAS and Ba by XRF. Intervals with greater than 1% Pb, Zn or Cu were assayed for Cu, Pb, Zn, Fe (total), Ag (AAS), Au (fire assay with AA and gravimetric finish) and Se by ICP and XRF.

11.4.2 BMC (2015 Onwards)

A total of 2,906 ABM deposit samples were collected and analysed for Cu, Pb, Zn, Ag, Au, Fe; 2,256 samples for Ba and 2,315 samples analysed for S using a Na-peroxide fusion and ICP-AES finish. The Na-peroxide fusion is considered to be a complete digestion. A sum of 1,145 samples were analysed for As, Bi, Hg, Sb and Se using aqua regia digest with ICP-MS finish.

From BMC drilling at GP4F, samples were collected and analysed for Cu, Pb, Zn, Ag, Au, Fe, Ba and S using a Na-peroxide fusion and ICP-AES finish. The Na-peroxide fusion is considered to be a complete digestion. All samples were analysed for As, Bi, Hg, Sb and Se using aqua regia digestion with ICP-MS finish.

Au was analysed by 30 g fire assay with an AAS finish and Ag analysed by ICP with an AAS finish on a 2 g two-acid digest aliquot. Samples that returned >4% Ba were analysed by XRF. Au and Ag over-limits were triggered at Au >5 g/t and Ag >150 g/t respectively, resulting in re-analysis using a 30 g fire assay with gravimetric finish.

All samples were analysed at SGS (Vancouver). SGS and its employees are independent from BMC. The SGS Vancouver laboratory is accredited to the requirements of ISO/IEC 17025. Detection limits for each analytical method used are shown in Table 11-1.

Table 11-1: Analytical methods and range for the ABM deposit drilling by BMC

Element	Analytical method	Analytical range
Au	30 g fire assay/AAS finish	0.005–10 ppm
Au over limit (>5 g/t): 30 g fire assay/gravimetric finish		0.5–3,000 ppm
Ag	2 g two-acid digest/ICP-AAS	0.3–300 ppm
Ag over limit (>150 ppm): 30 g fire assay/gravimetric finish		10–5,000 ppm
Cu	Na peroxide fusion/ICP-AES	0.01–30%
Pb	Na peroxide fusion/ICP-AES	0.01–30%
Zn	Na peroxide fusion/ICP-AES	0.01–30%
Fe	Na peroxide fusion/ICP-AES	0.05–30%

11.4.3 EDTA Analysis

In an effort to characterize surface oxidation of sulphide minerals at ABM, 90 sample pulps of shallow (<50 m) sulphide mineralization from the 2015 sampling program were submitted for EDTA leach analysis. EDTA provides a quantitative measure of the degree of oxidation by forming complexes with the oxidation products of sulphide minerals (Bicak and Ekmekci, 2012). Later in the 2015 program, a second set of 68 sample pulps taken at depth within the ABM resource envelope were analysed by EDTA leach analysis.

Results of all EDTA analyses indicate no significant soluble Cu or Zn; however, a soluble Pb component is present. Discussion with the consultant metallurgist undertaking testwork on ABM mineralization indicates that this can be managed during mineral processing. The EDTA analyses indicated no significant weathering effects near surface.

11.5 Quality Assurance/Quality Control

11.5.1 Methodology

No documented QAQC procedures have been located for the Cominco drilling programs. Detailed and systematic programs do not appear to have been in place during the Cominco drilling; however, it may be that the documentation has not yet been located.

To check for assay accuracy, one in 10 samples from the 1995 drill program were selected for "umpire" analysis by Chemex Laboratories using the same assay methods. Au assays compared well up to 3 g/t; however, 32 of 133 analyses above this threshold showed some scatter. Ag showed good correlation at all levels (Cominco Ltd., 1995).

The BMC field QAQC program entailed submission of coarse blank material every 20th sample and Certified Reference Material (CRM) every 20th sample. CRMs were selected with the aim of covering a wide range of base and precious metal grades and with a matrix similar to the mineralogy of the ABM deposit.

Approximately 3% of samples analysed in 2015 and 2016 were submitted to ALS Minerals Vancouver laboratory for umpire analyses via Na peroxide fusion and ICP-OES finish. ALS Minerals and its employees are independent from BMC. The ALS Vancouver laboratory is accredited to the requirements of ISO/IEC 17025 .

Additional quartz wash was inserted in the pulverizing stage where high-grade mineralization was suspected. Quartz wash residues were retained for possible later analyses. Wet screen analyses was completed every 50th sample to ensure consistent crush size.

In addition, as part of the relogging program, a total of 417 half-core samples drilled by Cominco within the resource area were resampled by BMC.

Detailed analysis of the 2015 QC results were outlined in Arne (2015b) and summarized in Green (2016).

11.5.2 Blanks

Cross contamination of samples has been monitored using coarse garden stone ("blank") sourced from Premier Tech Home & Garden in Brantford, Ontario containing negligible base and precious metals.

Data from 149 coarse blanks have been evaluated using two threshold values — three times the lower limit of detection (3xDL) and 10 times the lower limit of detection (10xDL) (Figure 11-4). The 3xDL threshold is appropriate for exploration programs where low-level regional geochemical anomalies are sought. The higher threshold value of 10xDL has been employed in this review. There is only one clear case of probable cross contamination involving Zn, with a blank giving a value of 0.12% (Figure 11-5). Overall, the percentage of samples potentially affected by cross contamination is less than 1% (Arne, 2015b).

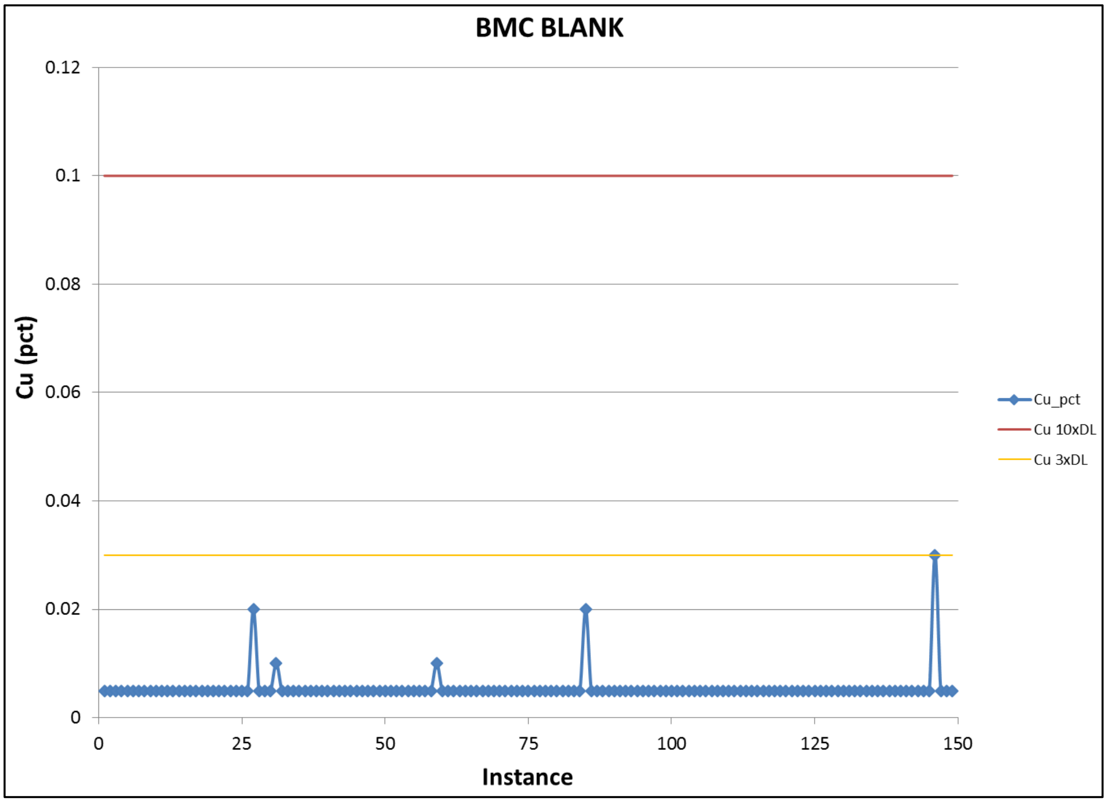


Figure 11-4: Blank control chart for Cu (%)

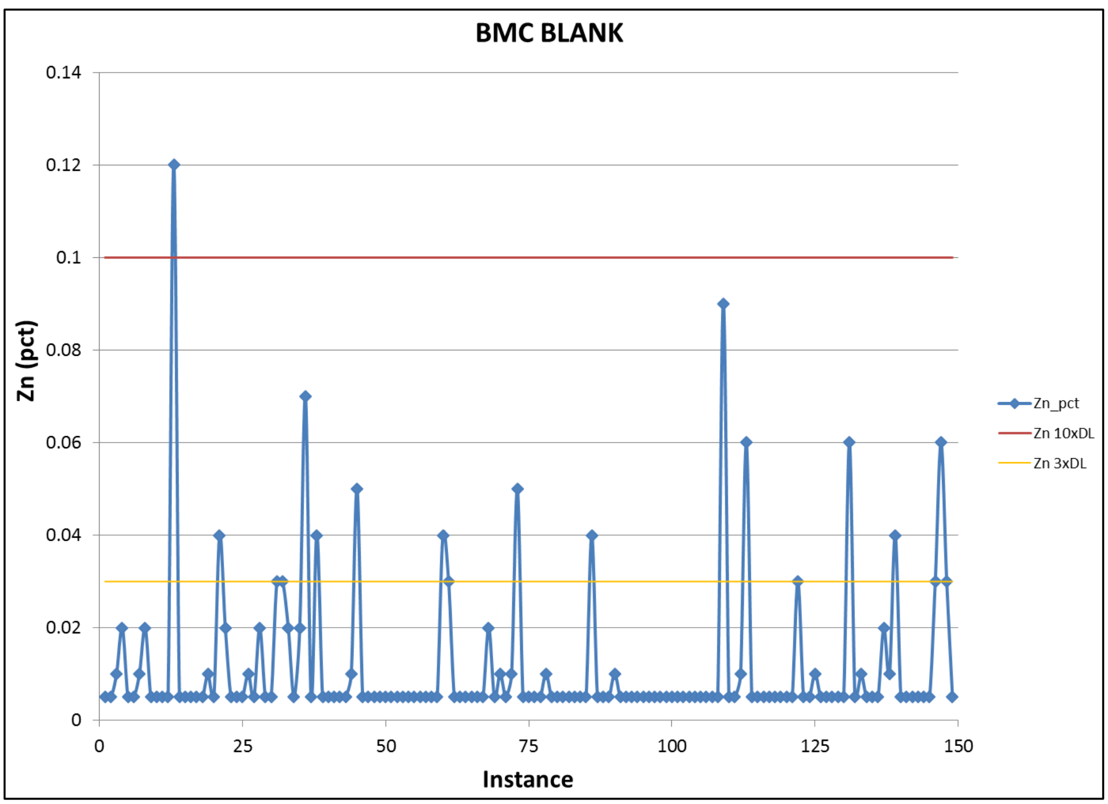


Figure 11-5: Blank control chart for Zn (%)

11.5.3 Certified Reference Materials

CRM selection aimed to get coverage over a range of base and precious metal grades reflected from historical assay datasets with a minimal number of CRMs. CRM selection was significantly hampered by what was commercially available through a range of international suppliers.

BMC 2015

Details in this section are taken from Arne (2015b).

Data from a total of six CRMs inserted into the sample stream by BMC were reviewed. There is a clear positive bias in data from five of the CRMs certified for Zn and for four certified for Pb (Table 11-2). In contrast, data for Cu, Ag and Au show negative biases, although this is not consistent across all CRMs certified for these elements (Table 11-2). There is no relationship between grade and Pb or Zn bias.

The base metal CRMs used by BMC are largely derived from VHMS deposits and so are considered to be matrix appropriate for the KZK Project. Ag was analysed following a two-acid digestion whereas CDN-ME-1311 and OREAS 113 are certified for Ag by a four-acid digestion only, and the analytical method used from GBM310-16 is not described. This may in part explain the negative bias for Ag, which is particularly pronounced in these CRMs.

CDN-ME-1311 has a high SiO_2 content which would likely lend itself to having a component of the certified metals "locked up" as inclusions in the silicates that would not be accessible with a two-acid digest. This may explain in part the negative bias for this standard. Petrographic work consistently shows that the ABM deposit has a much lower silicate content and a significant carbonate content, which is in contrast to the CRM.

Table 11-2: Summary of average biases from CRMs for 2015 program

CRM	CRM recommended values			Average bias (%)		
	Cu %	**Pb %**	**Zn %**	**Cu**	**Pb**	**Zn**
CDN-ME-1311	44.90	0.84	0.47	1.12	n/a	-1.89
OREAS 623	-1.23	-4.69	3.91	3.45		1.01
OREAS 621	1.25	3.71	-1.49	2.58		1.69

**n/a – not applicable. Source: Arne, 2015b*

Subsequently it was discovered that Pb was under-reporting by the Na-peroxide fusion employed where the samples contained >1% Ba, as barite was precipitating out of the fusion and taking Pb with it. The difference was significant enough that these samples have been re-assayed using a reduced aliquot weight.

Typical control charts are presented in Figure 11-6 and Figure 11-7 below.

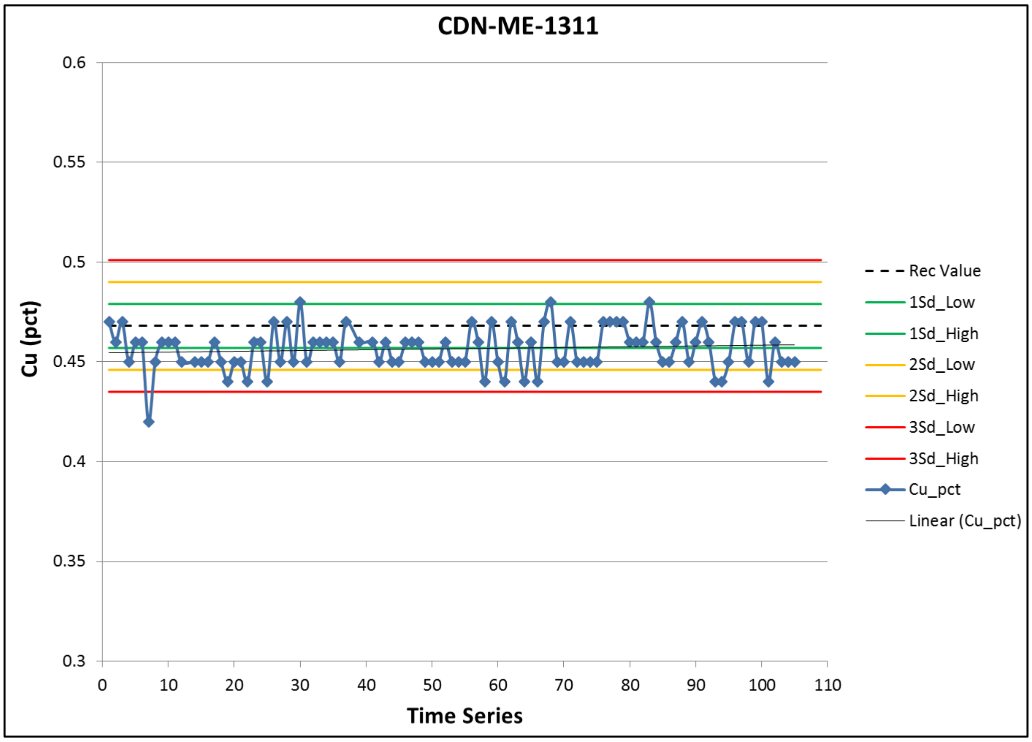


Figure 11-6: 2015 CRM (CDN-ME-1311) control chart for Cu (%) showing negative bias

Note: 1Sd = 1 Standard Deviation, 2Sd = 2 Standard Deviations, 3Sd = 3 Standard Deviations as defined for the individual CRM.

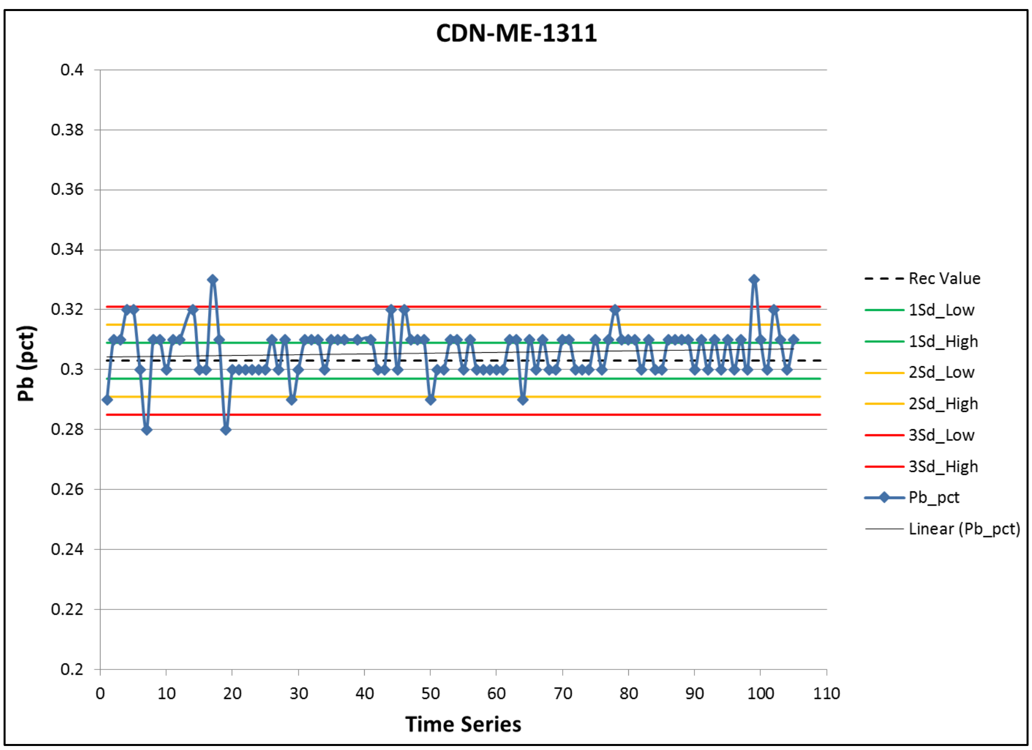


Figure 11-7: 2015 CRM (CDN-ME-1311) control chart for Pb (%) showing slight positive bias

Note: 1Sd = 1 Standard Deviation, 2Sd = 2 Standard Deviations, 3Sd = 3 Standard Deviations as defined for the individual CRM.

BMC 2016

Details in this section are taken from Arne (2016a).

ABM Deposit

Data accuracy has been assessed using data for Au, Ag, Cu, Pb, Z, S and Fe from three CRMs, CDN-ME-1311, OREAS 621 and OREAS 623. Aside from Cu data from OREAS 621, all average biases are within 2% of the certified values (Table 11-3). While Pb and Zn both show slight positive biases, these are lower than those previously displayed by the same CRMs in the 2015 data (Arne, 2015b). Cu data in OREAS 621 show negative biases similar to those shown by the same CRMs in the 2015 QC data.

The negative bias previously noted in the 2015 Ag data for CRM OREAS 623 is not apparent in the 2016 data because the four-acid certification results have been used. The method description provided for the Ag analyses (GE-AAS12E) incorrectly stated that aqua regia digestion was being used, when in fact SGS uses a reverse aqua regia digestion consisting of 3:1 $HCl:HNO_3$ designed to provide a more aggressive attack of sulphides in the sample. The four-acid digestion certified value is less than the aqua regia for OREAS 623 and more in line with the analyses generated by SGS.

The 2016 CRM data for the ABM Zone are illustrated in Figure 11-8, Figure 11-9, and Figure 11-10 for CRMs CDN-ME-1311, OREAS 621 and OREAS 623 respectively, using Z-scores. Z-scores are calculated using the following formula:

$$Z - Score = \frac{Difference\ between\ observed\ and\ certified\ value}{Standard\ deviation\ of\ certified\ value\ from\ certificate}$$

Z-score values typically vary between 1 and –1, and so most data lie within one standard deviation of the certified value for each element.

In conclusion, the data are considered to be very accurate based on the CRMs inserted by BMC with the 2016 samples from ABM. Overall average relative biases are all less than 2%.

Table 11-3: Summary of average biases from ABM deposit CRMs for 2016 program

CRM element		CRM recommended values						Average bias (%)					
	N	Ag (ppm)	Au (ppm)	Cu %	Pb %	Zn %	Fe %	Ag	Au	Cu	Pb	Zn	Fe
CDN-ME-1311	40	44.90	0.84	0.47	0.30	1.12	n/a	-0.22	1.54	-0.75	1.65	1.58	n/a
OREAS 623	24	20.40	0.83	1.73	0.25	1.03	13.5	-0.49	-0.57	-0.55	1.51	1.09	0.57
OREAS 621	24	68.00	1.25	0.37	1.33	5.22	3.71	-0.18	1.73	-2.24	1.24	0.18	0.49
Overall bias								-0.30	0.90	-1.18	1.47	0.49	0.28

**n/a – not applicable. Source: Arne, 2016a*

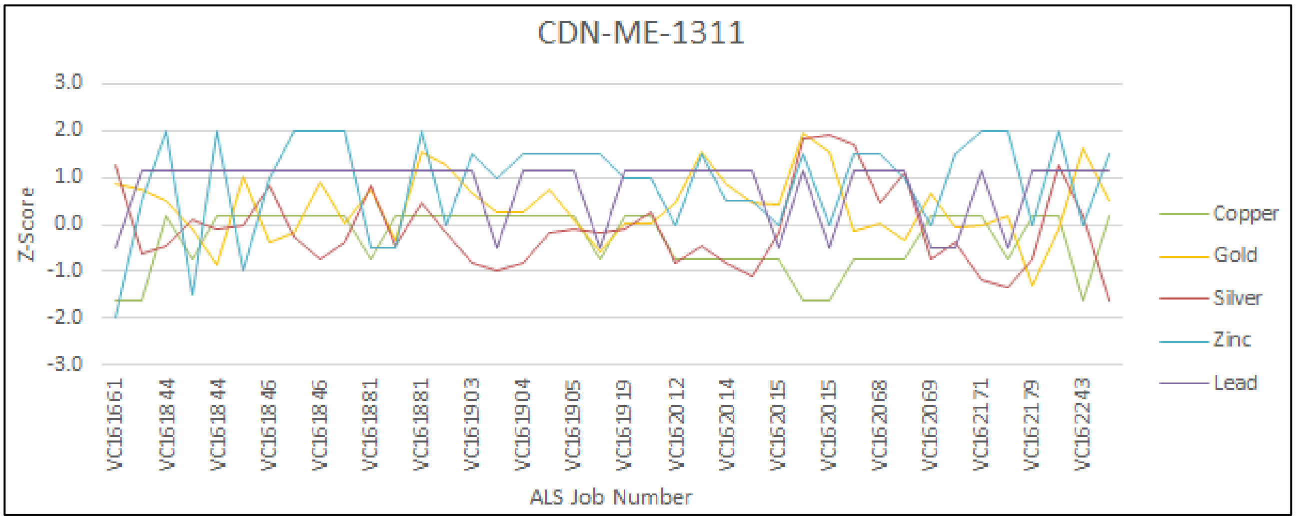


Figure 11-8: 2016 ABM Zone summary of Z-score for CDN-ME-1311

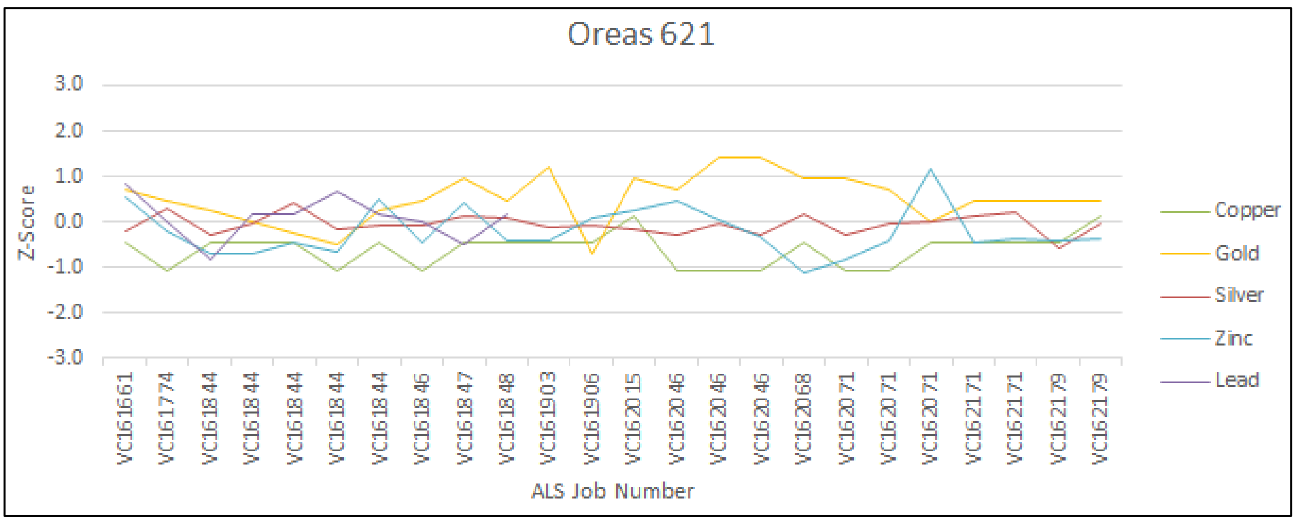


Figure 11-9: 2016 ABM Zone summary of Z-score for OREAS 621

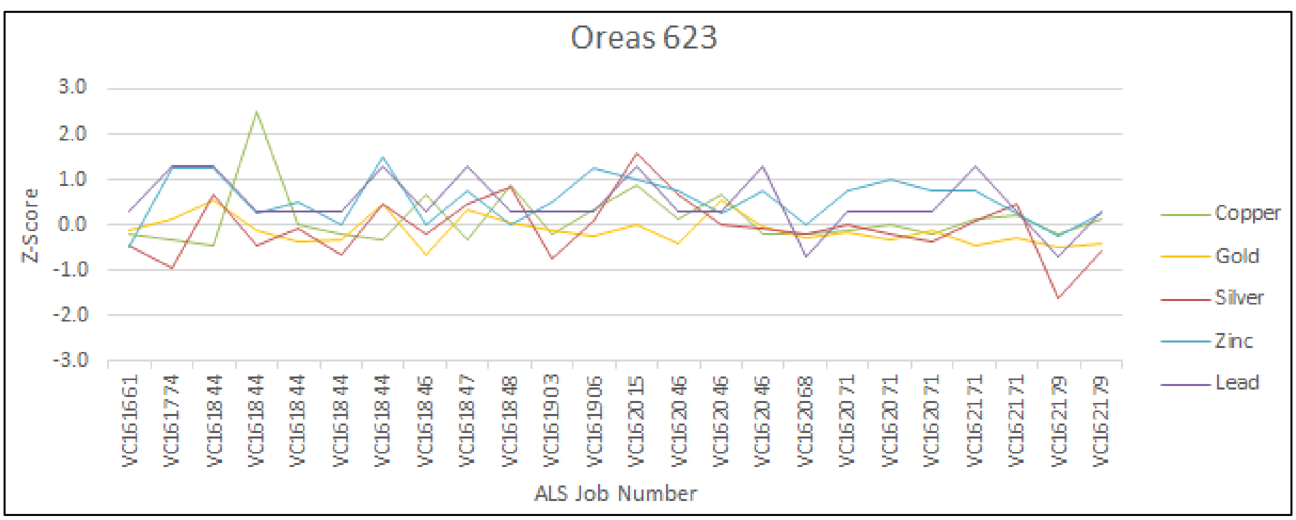


Figure 11-10: 2016 ABM Zone summary of Z-score for OREAS 623

GP4F Zone

Data from 27 analyses of CRM CDN-ME-1311, inserted into the sample stream by BMC during the 2016 drilling program at the GP4F Zone, have been reviewed (Table 11-4). Data from an additional six samples each of OREAS 621 and OREAS 623, as well as a single sample of GBM 310-16, have also been reviewed.

Table 11-4: Summary of average biases from GP4F Zone CRMs for 2016 program

CRM		CRM recommended values	Average bias (%)					
			Cu %	Pb %	Zn %	Cu	Pb	Zn
CDN-ME-1311	27	44.9	1.12	n/a*	-0.02	-0.11	-1.3	1.21
OREAS 623	6	20.40	0.25	1.03	13.5	-0.82	4.72	0.54
OREAS 621			0.37	3.71	0.29	0.93	-2.24	1.50

**n/a – not applicable. Source: Arne, 2016b*

The 2016 CRM data for the GP4F Zone from CDN-ME-1311 are illustrated in Figure 11-11 using Z-scores.

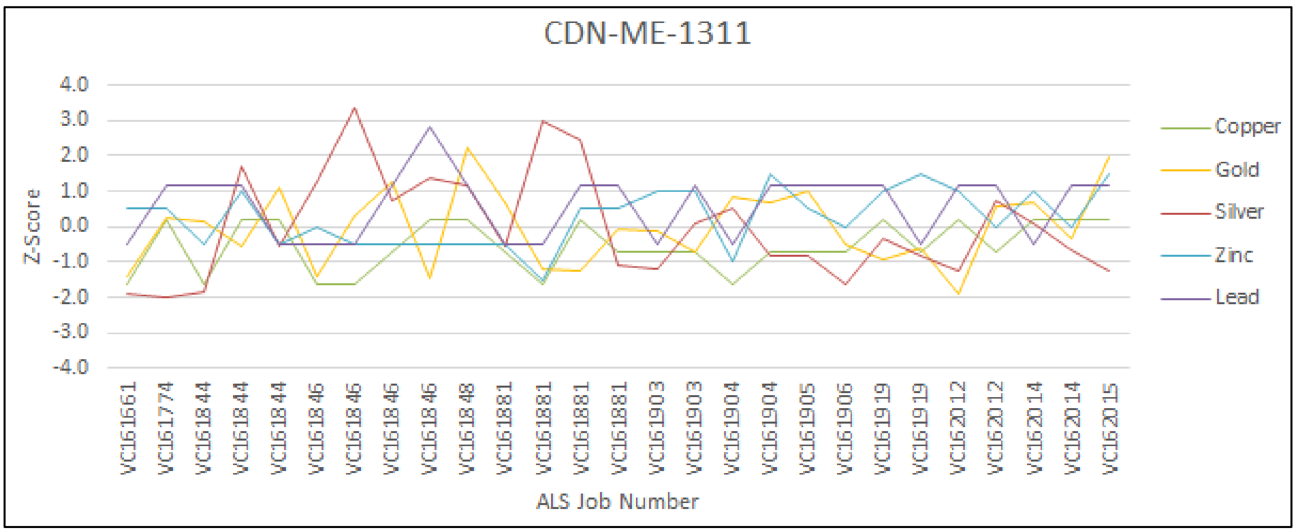


Figure 11-11: 2016 GP4F Zone summary of Z-score for CDN-ME-1311

In conclusion, the data are considered to be very accurate based on the CRMs inserted by BMC with the 2016 samples from the GP4F Zone. The average relative biases are all less than 2%, except for Ag, which is under-reporting by approximately 2%.

11.5.4 Umpire Laboratory Results

BMC 2015

Details in this section are taken from Arne (2015b).

Pulps from a total of 150 samples from the 2015 program were submitted to ALS (Vancouver) for check assay using the same analytical methods as those employed at SGS.

The data have been examined using a series of scatter plots with fits to the data obtained by ordinary least squares regression. The data are compared in Table 11-5 using the coefficient of variation (CV), the correlation coefficient and an estimate of relative bias derived from the percentage difference, i.e. (original assay-check assay)/original assay*100. Negative values in Table 11-5 indicate that the SGS data have an average negative bias relative to the ALS data from the check assays.

Overall, there is very good agreement between Cu and Zn values from SGS and ALS, with SGS values slightly lower than the ALS data and little scatter between the two datasets. Both Pb and Ag show strong positive correlations between the SGS and ALS data, but with much more scatter than the Cu or Zn data (Table 11-5). Ag is slightly lower in the ALS samples, on average, whereas Pb is slightly higher in the ALS samples compared to the SGS data (Table 11-5). Variability in the Ag data is due in part to the nuggetty distribution of some of the Ag at ABM in fine-grained electrum grains and sulphosalts, as well as slight differences in the strength of aqua regia used at the two laboratories. Lower Pb in the SGS samples probably reflects under-reporting due to the presence of barite, even though both labs used a Na-peroxide digestion for the base metals analyses.

Table 11-5: Summary of check assay statistics for 2015 program

Element	CV	Correlation coefficient	SGS bias relative to ALS
Ag	7.18 %	0.99	+3.31 %
Au	21.3 %	0.93	-2.83 %*
Cu	2.79 %	0.99	-0.11 %
Pb	7.92 %	0.95	-0.44 %
Zn	2.56 %	0.99	-1.00 %

**Data are imprecise and this bias estimate is not considered to be reliable. Source: Arne, 2015b*

As previously discussed, data for Au is the least precise and this imprecision is reflected in both the high CV and low R values for the two datasets. LA-ICPMS and petrography demonstrate that Au occurs as electrum grains and within minor arsenopyrite component of the mineralization, both of which display a heterogenous distribution. Overall, Au is slightly lower in the SGS samples, although this conclusion must be moderated by the poor precision of the combined data.

A total of 15 QC samples were submitted with the check assays, including six blanks, three CDN-ME-1311, four OREAS 621 and three OREAS 623 CRMs. The CRM data all lie within 1SD or 2SD of the recommended values and the blanks are all within acceptable limits. The quality of the ALS data is therefore considered to be acceptable, although it must be borne in mind that there are far fewer QC samples than were submitted to SGS.

However, as was the case with the SGS data, there are clear positive biases in the Pb and Zn data, as well as negative biases in the Cu and Au data for some of the CRMs. These biases are generally similar to or slightly more extreme than those displayed by the SGS data for the BMC CRMs, albeit based on a much smaller dataset. This observation is consistent with the relative biases for Pb and Zn between the SGS and ALS check assays described previously, as well as being consistent with a slight positive bias in the SGS data relative to the historical assays, as discussed in the following section. The Ag assays remain problematic. The ALS Ag data for the BMC CRMs generally show a positive bias, and yet the ALS data are lower, on average, than the SGS data for the check assays. This contradiction may in part lie in the variable nature of the Ag bias observed in SGS assays over time, particularly for CDN-ME-1311, as well as the small number of QC samples available for the check assays.

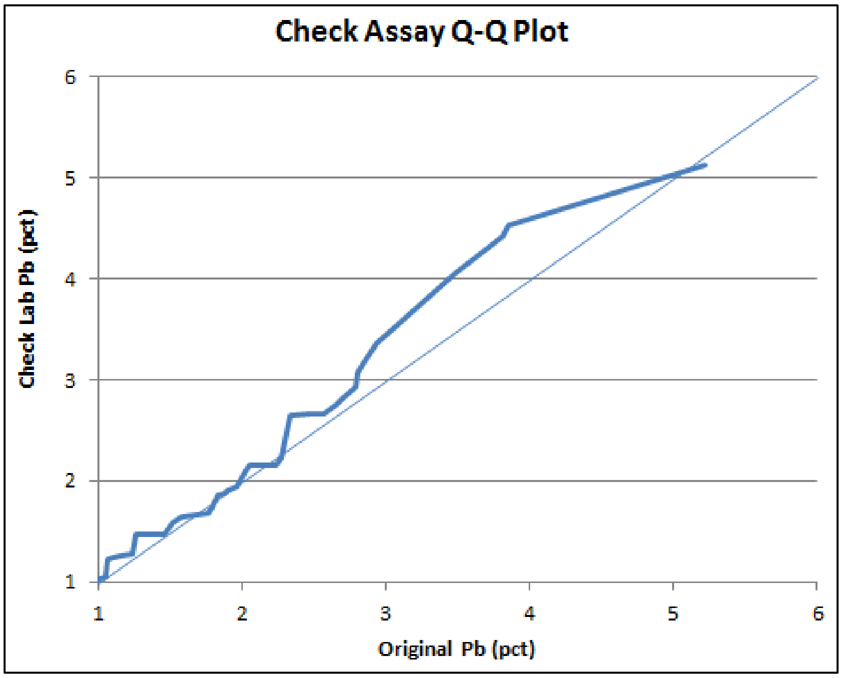


Figure 11-12: Quantile-quantile (Q-Q) plot using each data pair as a quantile and showing different distributions for Pb data from ALS and SGS
Source: Arne, 2015b

BMC 2016

Details in this section are taken from Arne (2016a,b).

Pulps from a total of 38 samples from the ABM deposit and 44 samples from the GP4F deposit were submitted to ALS in 2016 for check assay using the same analytical methods as those employed at SGS. The data have been examined using a series of scatterplots with fits to the data obtained by ordinary least squares regression. The data are compared in Table 11-6 for the ABM deposit and Table 11-7 for the GP4F Zone using the CV, the correlation coefficient and an estimate of relative bias derived from the percentage difference (i.e. (original assay-check assay)/original assay*100). Negative values in Table 11-6 and Table 11-7 indicate that the SGS data have an average negative bias relative to the ALS data from the check assays.

Table 11-6: Summary of ABM deposit check assay statistics for 2016 program

Element	CV	Correlation coefficient	SGS bias relative to ALS
Ag	8.5%	0.99	0.6%
Au	21.3%	0.94	-16.4%*
Cu	3.6%	0.99	2.9%
Pb	3.0%	0.99	2.5%
Zn	3.2%	0.99	-2.7%

** Data are imprecise and this bias estimate is not considered to be reliable. Source: Arne, 2016a*

Table 11-7: Summary of GP4F Zone check assay statistics for 2016 program

Element	CV	Correlation coefficient	SGS bias relative to ALS
Ag	3.1%	0.99	1.6%
Au	50.7%	0.92	-25.8%*
Cu	2.9%	0.98	1.2%
Pb	4.1%	0.96	5.0%
Zn	2.0%	0.99	0.2%

** Data are imprecise and this bias estimate is not considered to be reliable. Source: Arne, 2016b*

For the ABM deposit, there is very good agreement between Ag values from SGS and ALS, which is an improvement from the previous assessment (Arne, 2015b). With the exception of Au, all the data examined show strong positive correlations between the SGS and ALS data, but with a positive bias in the SGS data Cu and Pb data relative to ALS, and a negative bias for Zn. Overall, Au is significantly lower in the SGS samples compared to the previous review of the 2015 data, although this conclusion must be moderated by the poor precision of the combined data. It is worth noting that higher Au grades have been encountered at the ABM deposit during the most recent drilling and these likely indicate the presence of coarse Au in the samples.

For the GP4F deposit, with the exception of Au, all the data examined show strong positive correlations between SGS and ALS data, but with slight positive biases in the SGS Cu, Pb and Zn data relative to ALS. In particular, the ALS data show higher Ag and Pb values relative to the SGS data. This may be in part due to differences in the digestion used for the Ag determinations (SGS use a two-acid rather than an aqua regia digestion), and Pb determinations at SGS by Na peroxide fusion are known to have been underestimated in 2015 in the presence of >1% Ba in the samples.

These biases are not entirely consistent with the biases evident from the assessment of CRMs discussed in a previous section, and so the discrepancy may be due to biases in the ALS data. No CRMs submitted with the ALS check samples have been reviewed, but they are likely to be insufficient in number to adequately constrain bias.

As previously discussed, data for Au is the least precise and this imprecision is reflected in both the high CV and low R^2 values for the datasets. It is not possible to place any emphasis on average bias estimates from Au data with such a high CV using a least squares regression.

11.5.5 Historical Core Resampling

An important aspect of the historical core re-logging program was to establish the quality of historical assay results so those results could be incorporated into the new resource estimate. Historical core was resampled using the same mineralized intervals as Cominco from the remaining half core for all holes on sections 414,750E, 414,850E and 415,050E. Those sections and holes were determined to be representative of mineralization over the breadth of the ABM deposit. In addition, samples of weakly to unmineralized wall rock were collected from 4.5 m to 6 m outward from the massive sulphide contacts where historical sampling had not been completed (Baknes, 2015).

The following details are taken from Arne (2015b):

- Remaining half drill cores for 417 samples previously sampled and analysed by Cominco (CERL) were resampled during the 2015 field program by BMC and analysed by SGS (Vancouver) using the same methods employed on the 2015 drillholes (Na-peroxide digestion with an ICP-OES finish). Historical drill core was stored on site under cover (Figure 10-5) and no significant oxidation of the core was noted.

No historical core resampling was undertaken during 2016.

The historical assays undertaken by CERL involved an aqua regia digestion followed by an AAS or ICP finish. Above detection limit material is believed to have been re-analysed following dilution. Ag and Au were also analysed by fire assay and Ba was analysed by XRF.

Scatterplots and Q-Q plots were generated for each element and an example for zinc is shown in Figure 11-13 and Figure 11-14.

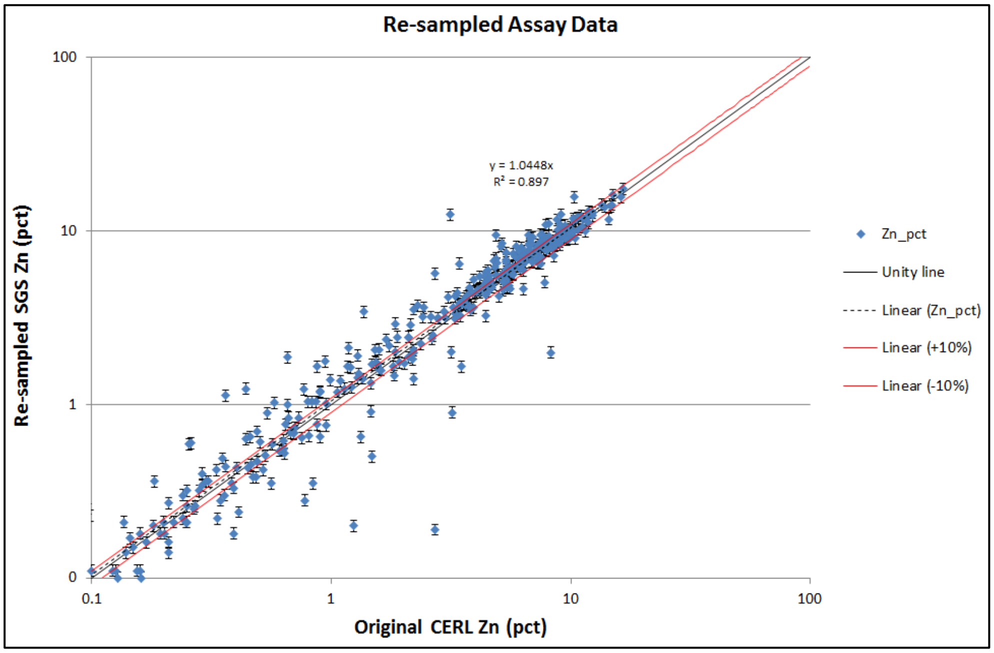


Figure 11-13: Scatterplot of historical and 2015 data for Zn

Note: Error bars for the SGS data are estimates of precision from preparation duplicate pairs expressed at 2SD.
Source: Arne, 2015b

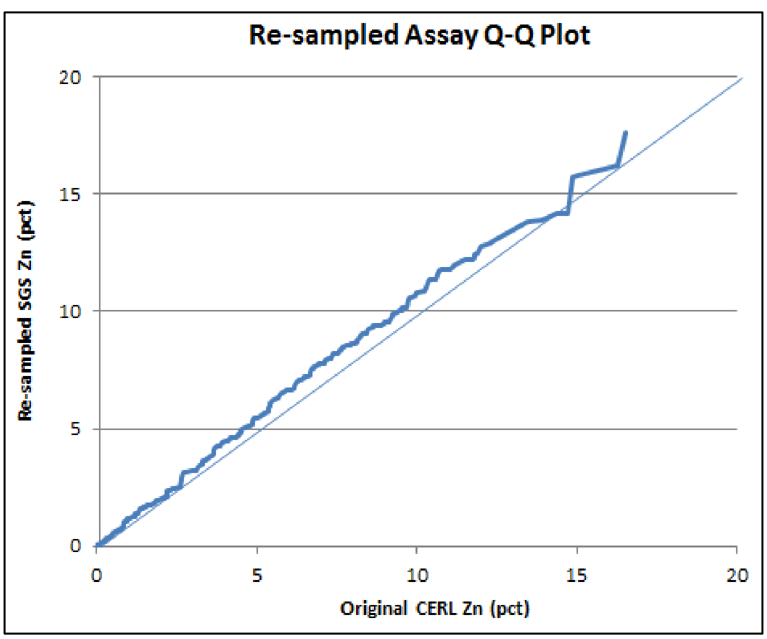


Figure 11-14: Q-Q plot for historical and 2015 data for Zn
Source: Arne, 2015b

The main conclusions of the comparison of the historical CERL data (1990s era) obtained using an aqua regia digestion for the base metals and the 2015 data obtained from SGS using a sodium peroxide fusion are summarized in point form below:

1. Cu – good agreement up to 3%.
2. Pb – slight positive bias in SGS data, once samples with>1% Ba were re-assayed.
3. Zn – positive bias in SGS data >2.6% to 15%.
4. Ag – good agreement to 250 ppm.
5. Au – good to fair agreement to 3 ppm.
6. Fe – good agreement in the range 8–15%; positive bias in SGS data <8%; negative bias in SGS data between 15% and 20%.

The observed agreement in Cu data up to 3% supports the conclusions reached from a comparison of the SGS and ALS check assays that the Cu data are generally reproducible between laboratories. Positive Zn biases in the SGS Pb and Zn data relative to the CERL data is consistent with positive biases seen in the SGS Pb and Zn data relative to a number of CRMs. The slight positive bias seen in the ALS check Pb and Zn assays relative to the SGS data indicate that ALS would produce an even stronger positive biases relative to the CERL Pb and Zn data. The Ag data are in good agreement to 250 ppm when both SGS and CERL data were obtained using an aqua regia digestion. The agreement breaks down above 250 ppm Ag, close to the upper detection limit of 300 ppm for this method at SGS.

Given the particulate nature of Au observed in petrographic work on ABM, poor agreement between SGS and CERL above 3 ppm Au is not surprising, but the systematic positive bias observed above this value in the CERL data, rather than random scatter, suggests a fundamental difference in the way the samples were prepared and analysed by fire assay.

11.6 Summary Opinion of Qualified Person

Based on an assessment of the historical drilling results and the recent drilling by BMC, the Qualified Person considers the sample preparation and analytical procedures to have been adequate at the time undertaken. More recent drilling by BMC, including extensive infill drilling has confirmed the width and tenor of the historical results, as well as the geological interpretation. Therefore, the Qualified Person considers the entire dataset to be acceptable for resource estimation with assaying posing minimal risk to the overall confidence level of the MRE.

The Qualified Person considers that adequate procedures were in place to ensure security of drill core and samples from the drill rig to the laboratory.

12 Data Verification

12.1 Site Visit

A site visit was conducted by Aaron Green (Qualified Person) and Neil Martin (BMC (UK) Limited – Technical Director) from 11 to 13 October 2015. The purpose of the site visit was to:

- Inspect operating drill rigs
- Review current drilling and sampling procedures
- Verify the location of selected drill collars and downhole surveys
- Inspect site geological data collection systems (mapping, logging etc)
- Review site geology
- Review sample storage facilities including historical core storage farm
- Discuss QA with geological personnel
- Discuss data storage and review the drillhole database.

Majority of data, drilling and geological records were found to be well maintained by BMC and comprehensive field procedures have been developed. The following conclusions were made from the site visit:

- All drill crews were observed operating to a very high, professional standard and all equipment was presented in excellent condition. All procedures relating to drilling including environmental, safety, sampling and surveying appeared to be followed.
- All staff including the drillers, offsiders, field assistants and geologists seemed to be comfortable with the drilling and sampling procedures.
- Preliminary verification of the drill collar coordinates by CSA Global indicated an acceptable level of accuracy, although further verification will be necessary once a final database is produced. Subsequently, hole collars for all drilling (with the exception of a small number of holes) were surveyed by the end 2016 to a requisite level of accuracy. Confirmation of historical collar locations was achieved to a sensible level.
- The geologists and field assistants seemed to understand QA procedures.

Site sample storage facilities and the analytical laboratory in Vancouver (SGS) were also inspected by Aaron Green and Dennis Arne of CSA Global (Vancouver), and Robin Black (BMC – Exploration Manager) on Thursday 22 October 2015.

There were no negative outcomes from any of the above inspections, and all samples and geological data were deemed fit for use in Mineral Resource estimation.

A subsequent site visit was conducted Aaron Green, Neil Martin and Robin Black (BMC – VP Exploration) on 26 July 2017. No active drilling was being undertaken at the time of the site visit. Limited geological outcrops around the ABM deposit were visited as well as the core farm.

12.2 Database Verification and Validation

From 2015 to November 2017, the drillhole database was managed off site by OMI Pty Ltd (OMI) based on information provided by BMC, Equity Exploration Consultants (Equity Exploration) and the laboratories. Original “hard copy” data was located by BMC and entered by OMI into a Microsoft Access database. Results from the 2015 and 2016 exploration programs were managed by OMI and loaded directly into the master Microsoft Access database.

CSA Global compared the original "hard copy" data (drill collars, laboratory assay reports, geological logs, downhole surveys) for approximately 10% of the total drillholes with the database provided by OMI. No significant issues were noted.

In addition to checking "hard copy" data, relevant tables from the database were imported into Surpac software. Validation of the final data import by CSA Global included checks for:

- Missing data for entire holes
- Missing collar coordinates
- Overlapping sample intervals
- Samples interval exceeding the hole depth
- Missing sample intervals
- Missing downhole survey data
- Azimuth or dip changes >5.00°
- "From" depths greater than or equal to "To" depths
- "From" depth does not start from 0.

The data in the database is comprehensive and of a high standard and all issues noted were minor and were corrected by OMI prior to commencement of the MREs.

Since December 2017, CSA Global has managed the KZK database. All data for the KZK Project has been migrated to an acQuire 4 GIM Suite database. The database is managed out of CSA Global's United Kingdom office in Horsham.

12.3 Verification of Sampling and Assaying

12.3.1 Visual Inspection

Historical and 2015 drill core was viewed extensively by the Qualified Person (Aaron Green) and BMC's Technical Director (Neil Martin) during the October 2015 site visit. Visual validation of mineralization against assay results was undertaken for several holes and verified the presence of significant sulphide mineralization as reported. Significant intercepts appear to correlate with the intensity of mineralization logged in the field.

12.4 Audits and Reviews

A review of the sampling techniques and data was carried out by CSA Global during the October 2015 site visit. Visual validation of the drillhole locations and mineralized intersections was undertaken against hard copy drill sections. Relative to each other and the cross sections provided, the drillholes used as the basis for the MRE update were considered acceptable for classification and reporting under National Instrument (NI) guidelines.

The analytical laboratory, SGS (Vancouver), was inspected by Aaron Green (Qualified Person) and Dennis Arne of CSA Global, and Robin Black (BMC – Exploration Manager) on 22 October 2015. The laboratory visit found no significant issues at SGS with the site well presented, clean and with excellent procedures and equipment in place to produce high-quality assays.

The Qualified Person has verified the data which underpins the resource estimate contained in this Technical Report. The Qualified Person is of the opinion that data verification procedures undertaken on the data adequately support the integrity of the data and its use in the MRE.

13 Mineral Processing and Metallurgical Testing

13.1 Introduction

Cominco completed a large program of metallurgical testwork on the ABM deposit in the 1990s to advance development of the project. This testwork is noted briefly for completeness of the record of testing. While it formed a strong platform for BMC to commence a new metallurgical testwork program, the results from historical testwork have not been used in predictions of metallurgical performance for the DFS as the flowsheet and reagent scheme have been modified to optimize metallurgical performance.

Historical metallurgical testwork was completed by Lakefield Research, Met Engineers, Process Research Associates, G&T Metallurgical and Cominco's own in-house facilities between October 1994 and July 1997. The flowsheet developed through Cominco's work established a three-stage sequential flotation circuit to produce copper, lead and zinc concentrates, with regrind of copper and zinc rougher concentrate.

BMC commenced metallurgical testwork in 2016 for the PFS (CSA Global, 2017), initially with SGS in Burnaby, Canada before moving the testwork program to ALS Metallurgy in Perth, Australia. ALS Metallurgy in Kamloops, Canada also completed confirmatory comminution testwork. DFS metallurgical testwork continued in 2017 and 2018 with ALS Metallurgy in Adelaide, Australia. The DFS metallurgical testwork program was supervised by Minnovo, with testwork results and interpretation reported in Allnorth's report (Allnorth, 2019).

13.2 Orebody and Metallurgical Domains

The ABM deposit comprises the ABM Zone and the adjacent Krakatoa Zone. Three major metallurgical domains were recognized for the ABM Zone in diamond core based on texture and mineralogy:

- MET2-4 – massive sulphide mineralization with significant magnetite component
- MET5-7 – massive sulphide mineralization lacking significant magnetite component
- MET8 – vein/stockwork mineralization with significant silicate gangue component.

A fourth metallurgical domain for the ABM Zone (+1340 RL) was defined by spatially controlled mineralogical zonation above 1340 m RL, with a higher level of copper bearing sulphosalt mineralization (primarily tennantite-tetrahedrite) observed than that of mineralization below this elevation.

A single metallurgical domain (Krakatoa) was recognized for the Krakatoa Zone.

13.3 Samples

Ten composite samples were generated for the 2016 PFS testwork and were used for comminution and flotation testing. The PFS samples were:

- ABM master composites 1 and 2
- ABM life of mine (LOM) composite
- ABM domain composites (four samples): +1340 RL, MET2-4, MET5-7, and MET8
- Krakatoa composites (three samples): In Pit Main, In Pit Upper, and -1250 RL.

These samples were selected by BMC from cores that were drilled in 2015 and 2016 and were large samples, typically from three or four holes and included approximately 10% dilution. Some of these samples were reused for the initial DFS flowsheet optimization testwork performed in 2017.

Twenty-five composite samples, being five samples for each of the five metallurgical domains (also known as the domain variability samples) were generated for the DFS variability testwork performed in 2017 and 2018. These samples were typically from a single continuous downhole interval to represent typical mine blocks that would be processed. Each sample included dilution, which ranged from 8.0% to 13.1% by weight, with the average dilution across all samples being 9.4%. The five domain variability samples for each metallurgical domain were also combined to generate five domain main composite samples. The DFS samples were:

- Main composites (five samples): Krakatoa, +1340 RL, MET2-4, MET5-7, and MET8
- Krakatoa variability composites (five samples): Krakatoa-1 through Krakatoa-5
- ABM variability composites (five samples): +1340 RL-1 through +1340 RL-5
- ABM variability composites (five samples): MET2-4-1 through MET2-4-5
- ABM variability composites (five samples): MET5-7-1 through MET5-7-5
- ABM variability composites (five samples): MET8-1 through MET8-5.

These samples were selected by BMC based on input from Minnovo and were primarily intended to assess variability in flotation performance over a range of head grades, although comminution properties were also studied. Sample selection also considered spatial distribution through the deposit.

A summary of the head grades of the DFS samples are shown in Table 13-1.

Table 13-1: Summary of DFS sample head grades

Domain	Sample	Cu (%)	Pb (%)	Zn (%)	Au (ppm)	Ag (ppm)	As (ppm)	Sb (ppm)	Se (ppm)	Hg (ppm)
+1340 RL	Main Composite	0.50	1.77	5.73	1.68	142	3,360	599	115	32
	Variability minimum grades	0.19	0.87	2.02	0.59	74	2,200	533	65	14
	Variability maximum grades	1.18	2.95	9.30	2.02	248	6,200	1,034	365	58
MET2-4	Main Composite	0.88	1.23	6.30	0.73	84	1,530	271	345	6
	Variability minimum grades	0.63	0.16	2.13	0.36	26	40	4	280	1
	Variability maximum grades	1.43	2.83	10.2	1.18	202	5,000	531	515	10
MET5-7	Main Composite	1.13	1.24	6.25	1.06	118	1,850	247	180	52
	Variability minimum grades	0.26	0.51	2.41	0.43	64	290	29	155	4
	Variability maximum grades	4.35	1.78	12.0	2.25	160	8,400	851	305	205
MET8	Main Composite	1.00	0.88	3.59	0.52	66	880	138	180	13
	Variability minimum grades	0.24	0.40	0.96	0.14	14	680	20	40	2
	Variability maximum grades	2.21	1.14	7.36	1.23	174	3,930	476	400	23
Krakatoa	Main Composite	0.35	2.18	5.04	1.53	184	4,230	497	65	27
	Variability minimum grades	0.15	1.11	3.75	1.29	114	1,400	202	30	7
	Variability maximum grades	0.84	3.63	8.38	2.21	292	8,000	1,038	180	41

Five waste samples were selected by BMC for comminution testing only.

13.4 Comminution

The SMC Test® was used in the design of the comminution circuit. Eleven composites were submitted for SMC Testing during the PFS. The five DFS domain composites were also submitted for SMC Testing. A summary of the consolidated statistics for the SMC Tests is shown in Table 13-2.

Table 13-2: Mineralization composites SMC testwork summary

	DWI (kWh/m³)	A x b	t_a	SG (t/m³)	SCSE (kWh/t)
Minimum	2.76	68.7	0.42	3.15	5.72
Maximum	6.05	145.3	0.94	4.58	8.15
Average	4.40	94.9	0.62	3.99	6.97
80th percentile	5.43	75.8	0.74	-	7.50
85th percentile	5.63	73.8	0.76	-	7.55

Notes: DWI – Drop Weight Index. A x b – Impact breakage parameters. t_a – Abrasion breakage parameter. SG – Specific gravity. SCSE – SAG circuit specific energy.

For the DFS, the extreme values for A x b and DWI of 68.7 and 6.05 kWh/m^3 respectively were used for design purposes due to the broadness (long downhole intervals) of many of the composite samples that were tested and the potential for reduced variability in these samples.

Six of the 11 PFS composites, five of the DFS domain composites, and four of the DFS variability samples were submitted for Bond Work Index testing. Bond Work Index test data from Cominco's metallurgical testwork was also considered and a summary of the consolidated statistics for Bond Work Index testwork is shown in Table 13-3.

Table 13-3: Bond Work Index testwork summary

	BWI @ 75 µm (kWh/t)	RWI (kWh/t)	AI (g)
Minimum	7.6	6.1	0.0425
Maximum	14.7	10.7	0.3683
Average	11.8	8.9	0.115
Standard Deviation	1.5	1.7	0.078
80th percentile	12.8	10.3	-
85th percentile	13.3	10.4	-

Notes: BWI – Bond Ball Mill Work Index. RWI – Bond Rod Mill Work Index. AI – Abrasion Index.

Minnovo would normally recommend the use of 80th percentile RWI and BWI values for calculation of ball mill power. For design purposes the 85th percentile RWI value was selected for use due to the broadness (long downhole intervals) of many of the composite samples that were tested and the potential for reduced variability in these samples and the limited number of RWI results.

The 80th percentile BWI value was used for design purposes due to the greater number of BWI results.

Five waste samples were also submitted for BWI and SMC testing and results are summarized in Table 13-4. The SMC Test results showed that waste is slightly harder than mineralization on the basis of BWI values. Typically, plant feed (containing mineralization and waste dilution) can be expected to require more grinding energy in the ball mill than mineralization alone. The testwork results indicated that mineralization with waste dilution during mining is not expected to cause a material change to grinding energy requirements.

Table 13-4: Waste samples SMC and BWI testwork summary

	DWI (kWh/m³)	A x b	t_a	SG (t/m³)	SCSE (kWh/t)	BWI @ 75µm (kWh/t)
Minimum	3.08	63.4	0.58	2.64	7.19	7.7
Maximum	4.45	85.9	0.84	2.82	8.25	15.1
Average	3.79	73.6	0.69	2.75	7.73	12.8

13.5 Flotation

Flotation testing in the PFS and DFS included:

- Batch open circuit and locked cycle tests on the PFS composites
- Large batch open circuit tests on ABM Master Composite 2 to produce concentrate for settling and filtration tests
- Large batch open circuit tests on ABM Master Composite 1, +1340 RL composite, and MET5-7 composite to produce additional flotation tailings for filtration tests
- Batch open circuit and locked cycle tests on the DFS domain composites
- Batch open circuit tests on the DFS variability samples.

The DFS batch open circuit flowsheet typically included:

- Primary grinding to 80% passing 70 µm
- Copper pre-float rougher flotation, cleaning of the pre-float rougher concentrate without regrinding, to produce final copper concentrate
- Copper rougher flotation, regrinding of rougher concentrate and pre-float cleaner tailings to 80% passing 30 µm, two stages of cleaner flotation
- Lead rougher flotation, regrinding of rougher concentrate and pre-float cleaner tailings to 80% passing 30 µm, two stages of cleaner flotation
- Zinc pre-float rougher flotation, cleaning of the pre-float rougher concentrate without regrinding, to produce final zinc concentrate
- Zinc rougher flotation, regrinding of rougher concentrate and pre-float cleaner tailings to 80% passing 35 µm, two stages of cleaner flotation.

The DFS variability sample initial batch float tests generally provided results in line with the DFS main composite results, although some samples showed less than optimal results. Follow-up tests were performed with increased collector addition which, in nearly all cases, resulted in significantly improved results.

One DFS variability sample (+1340 RL-3) showed limited improvement with increased collector addition. This sample was tested at a finer primary grind of 60 µm and higher collector addition with greatly improved results.

A summary of the optimized DFS variability sample batch open circuit tests is shown in Table 13-5.

Table 13-5: Summary of DFS variability sample batch open circuit test optimized results

Variability sample	Head grade					Cu concentrate		Pb concentrate		Zn concentrate		Au, Ag rec. to Cu, Pb concentrates	
	Cu %	Pb %	Zn %	Au ppm	Ag ppm	Grade %Cu	Rec. %Cu	Grade %Pb	Rec. %Pb	Grade %Zn	Rec. %Zn	%Au	%Ag
+1340 RL – 1	0.19	0.87	2.02	0.59	74	20.4	77.3	51.6	61.9	49.0	91.6	69.8	83.3
+1340 RL – 2	0.40	1.37	4.22	1.03	150	27.3	62.6	49.6	53.6	51.8	81.4	36.2	69.1
+1340 RL – 3	1.18	1.69	5.98	2.01	126	24.7	69.1	44.8	48.1	46.7	85.8	29.0	60.0
+1340 RL – 4	0.47	1.75	7.03	1.98	208	23.9	71.5	63.3	55.7	51.2	93.3	52.1	76.6
+1340 RL – 5	0.24	2.95	9.30	2.02	248	26.2	60.6	55.4	83.2	52.5	94.3	57.4	83.4
MET2-4 – 1	1.43	0.16	2.13	0.36	26	27.8	93.0	5.6	27.4	29.8	61.9	53.6	74.2
MET2-4 – 2	0.63	0.97	5.22	0.48	60	24.3	46.1	51.0	67.9	49.3	91.0	22.1	38.2
MET2-4 – 3	0.64	1.44	6.11	1.01	98	26.7	56.6	40.8	53.7	42.0	79.6	31.6	49.4
MET2-4 – 4	1.34	0.54	6.85	1.05	88	25.9	73.3	25.8	40.4	46.0	90.8	47.8	41.7
MET2-4 – 5	0.64	2.83	10.2	1.18	202	25.9	65.5	47.0	77.0	53.1	85.3	38.2	73.0
MET5-7 – 1	0.67	0.72	2.41	0.62	64	21.4	63.9	30.5	18.9	47.1	83.1	34.0	38.0
MET5-7 – 2	4.35	0.51	3.31	2.25	142	30.7	96.8	38.0	29.5	51.1	91.2	89.1	87.9
MET5-7 – 3	0.26	1.63	6.76	2.14	160	25.7	40.8	45.2	63.8	50.0	89.0	40.5	63.7
MET5-7 – 4	0.37	1.78	6.47	1.13	142	31.8	64.9	67.6	73.3	58.0	95.2	42.6	83.0
MET5-7 – 5	0.91	0.59	12.0	0.43	104	22.0	79.4	28.6	53.9	52.4	96.2	51.2	62.7
MET8 – 1	0.24	0.40	0.96	0.14	14	27.5	86.9	78.2	46.0	48.3	90.1	35.7	52.4
MET8 – 2	1.23	0.55	2.54	0.36	60	30.2	90.7	60.4	44.6	48.5	93.8	72.8	67.9
MET8 – 3	2.21	1.05	4.50	1.23	174	30.7	78.2	48.4	76.6	57.3	94.0	57.2	80.3
MET8 – 4	0.46	0.97	4.64	0.62	84	26.8	82.8	59.5	80.3	58.3	91.1	71.0	83.7
MET8 – 5	0.36	1.14	7.36	0.74	74	22.5	63.1	67.3	73.8	57.6	95.3	41.9	69.1
Krakatoa – 1	0.84	1.93	3.80	1.98	114	29.1	86.9	65.5	72.1	52.1	92.2	53.9	81.3
Krakatoa – 2	0.19	1.11	3.75	1.95	186	21.1	41.2	29.3	66.6	49.8	88.4	65.0	75.5
Krakatoa – 3	0.15	1.64	4.92	1.29	208	16.9	50.2	36.7	70.7	53.0	86.0	54.9	69.6
Krakatoa – 4	0.25	3.63	5.58	2.21	292	22.2	43.4	55.7	82.3	50.5	84.1	50.3	79.1
Krakatoa – 5	0.32	2.33	8.38	1.77	190	23.5	64.1	55.6	71.4	50.2	86.0	50.3	70.6

The DFS locked cycle tests were satisfactory except for MET5-7 main composite. The test was repeated at increased collector addition with improved results.

A summary of the final locked cycle test results for all DFS metallurgical domain composites is shown in Table 13-6.

Table 13-6: Summary of locked cycle test results

Domain	Head grade					Cu concentrate		Pb concentrate		Zn concentrate		Au, Ag rec. to Cu, Pb concentrates	
	Cu %	Pb %	Zn %	Au ppm	Ag ppm	Grade %Cu	Rec. %Cu	Grade %Pb	Rec. %Pb	Grade %Zn	Rec. %Zn	%Au	%Ag
+1340 RL	0.50	1.77	5.73	1.68	142	24.5	73.6	53.9	76.8	54.8	83.0	52.0	75.6
MET2-4	0.88	1.23	6.30	0.73	84	24.8	76.2	45.2	66.4	50.9	88.4	52.7	64.3
MET5-7	1.13	1.24	6.25	1.06	118	27.7	85.0	48.0	67.4	53.8	93.4	64.4	79.3
MET8	1.00	0.88	3.59	0.52	66	29.1	85.9	64.6	64.8	55.5	91.5	59.9	74.1
Krakatoa	0.35	2.18	5.05	1.53	184	27.3	58.5	57.9	80.2	53.1	91.0	57.5	73.1

The optimized variability results were considered suitable as the basis for predictive recovery and concentrate grade models.

The locked cycle test results correlated well with the optimized batch test results. The differences in performance between the batch and locked cycle results were applied to the predictive models developed using the batch results and are considered to provide a reasonable basis for prediction of plant-scale performance.

13.6 Gold and Silver Recovery

ABM and Krakatoa mineralization contains gold and silver values that are recovered to the copper, lead, and zinc concentrates during sequential flotation as a result of true flotation and as encapsulated particles. Recovery of gold and silver into copper and lead concentrates is preferable than into zinc concentrate as payability levels are higher than for zinc concentrates.

Bench-scale tests were performed with the aims of:

- Characterizing the deportment of gold and silver to the various concentrates
- Increasing the overall recovery of gold and silver to flotation and/or gravity concentrates, as compared to the typical overall recovery to copper and lead flotation concentrates.

These tests included:

- Amalgamation of milled mineralization and of lead circuit tailings as a diagnostic assessment of free gold and silver
- Gravity concentration of mineralization and lead circuit tailings in an attempt to minimize gold and silver deportment to the zinc concentrate and final tailings
- Flotation tests using a gold specific collector in the copper flotation circuit to maximize gold recovery to copper concentrate.

The testwork showed the following:

- A high proportion of the gold and silver is recovered into the copper and lead flotation concentrates. The average aggregate recoveries were 60.9% for gold and 70.9% for silver.
- It appears that a very low proportion of free gold is present in the mineralization, with a resultant low recovery to gravity concentrate.
- The average recovery of gold to gravity concentrate for all mineralized samples tested was 9.4%. The addition of gravity concentration to the flowsheet is generally unlikely to significantly increase the aggregate gold and silver recovery, although opportunities remain to be investigated with the +1340 RL metallurgical domain.

The gold is likely very fine and is distributed mainly in (or associated with) copper, iron, and lead sulphides. The gold shows essentially linear distribution by size fraction.

Using a gold specific collector in the copper circuit increases the proportion of gold and silver recovered to the copper concentrate and reduces the proportion recovered to the lead concentrate but does not increase the aggregate recovery of gold and silver. Improvements in the recovery of gold using a gold specific collector for the +1340 RL metallurgical domain may be possible.

The aggregate recovery of gold and silver to gravity, copper, and lead flotation concentrates was largely consistent, regardless of the use of gravity concentration or a gold specific collector, with the exception of the +1340 RL metallurgical domain.

13.7 Dewatering

Outotec performed dynamic settling and filtration test work on copper, lead, and zinc concentrate samples produced during the 2017 PFS. Microanalysis Australia determined the transportable moisture limit (TML) of these samples.

The concentrate samples achieved very high solids concentration in the underflow, ranging from 75% to 83%, at low flocculant addition rates of 5–10 g/t, and produced overflow with low suspended solids (<100 ppm).

Competent concentrate filter cakes were formed, with low moisture content ranging from 7% to 9%, and with filtration rates ranging from 756 kg/h/m^2 to 1,240 kg/h/m^2. Concentrate filter cake TML ranged from 8% to 10%.

Outotec performed dynamic settling and filtration testwork on the (PFS) ABM Master Composite 2 bulk batch flotation tailings sample.

The bulk tailings sample achieved very high solids concentration in the underflow, ranging from 73% to 75%, at low flocculant addition rates of 5–10 g/t, and produced overflow with low suspended solids (<120 ppm).

The PFS selected vacuum disc filters for dewatering the tailings. During the DFS, the use of vacuum disc filters was identified as a significant project risk, based on the reported inability of vacuum filtration to consistently produce suitable (low moisture) filter cake.

Outotec also performed filtration testwork on DFS flotation tailings samples from the +1340 RL, MET5-7, and ABM Master Composite 1 samples, including repeat tests on the ABM Master Composite 1 sample at a finer grind. The testing included three vacuum filtration methods and two pressure filtration methods.

For the initial DFS tests, the vacuum filtration methods compared to pressure filtration, showed:

- Higher cake moisture
- Inconsistent cake moisture
- Greater filtration rate
- Inconsistent filtration rate.

For the repeat tests, vacuum filtration showed a higher moisture content and lower filtration rate for the finer sample compared to the initial tests. Pressure filtration showed consistent moisture and filtration rates for the finer sample and was selected for the DFS design.

Based on observations made during the filtration testwork, by repeated tapping of filter cake to encourage liquefaction, a cake moisture less than 14% will be required for reliable operational performance which is achievable with the pressure filters selected for the DFS. Outotec testwork achieved cake moisture of 10–11% for the selected filters.

13.8 Flotation Recovery and Grade Models

Mathematical models for each metallurgical domain were investigated from open circuit batch tests for:

- Major base metal recoveries to concentrates, as functions of Cu, Pb, and Zn head grades
- Flotation concentrate grades (Cu, Pb, Zn), as functions of Cu, Pb, and Zn head grades
- Recoveries of gold and silver to flotation concentrates
- Recoveries of Pb, Zn, Fe, and S to copper concentrate, as functions of various head grade ratios (e.g. S/Cu)
- Recoveries of Cu, Zn, Fe, and S to lead concentrate, as functions of various head grade ratios (e.g. S/Pb)

- Recoveries of Cu, Pb, Fe, and S to zinc concentrate, as functions of various head grade ratios (e.g. S/Zn)
- Grades of contaminants (As, Sb, Bi, Cd, Hg, Se) in flotation concentrates, as functions of various head grade ratios (e.g. Sb/Cu).

Data from locked cycle tests were used for model validation, as noted later in this section.

Most of the models showed moderate to strong correlation coefficients (R^2 in the range 0.3 to 0.99) and were adopted for predicting metallurgical performance. Certain datasets did not demonstrate a reliable relationship and in these instances the weighted average has been used. Weighted averages were adopted for:

- +1340 RL copper recovery to copper concentrate
- Krakatoa zinc recovery to zinc concentrate
- Gold and silver recovery to all base metal concentrates
- Arsenic grade in lead concentrate.

It is believed that the presence of tennantite-tetrahedrite sulphosalts, which contain varying gold and varying silver up to 30% Ag, is partly responsible for the difficulty in deriving relationships of gold and silver recovery to head grade.

Recovery curves based on open circuit batch tests for copper into copper concentrate, lead into lead concentrate and zinc into zinc concentrate are shown in Figure 13-1 to Figure 13-3.

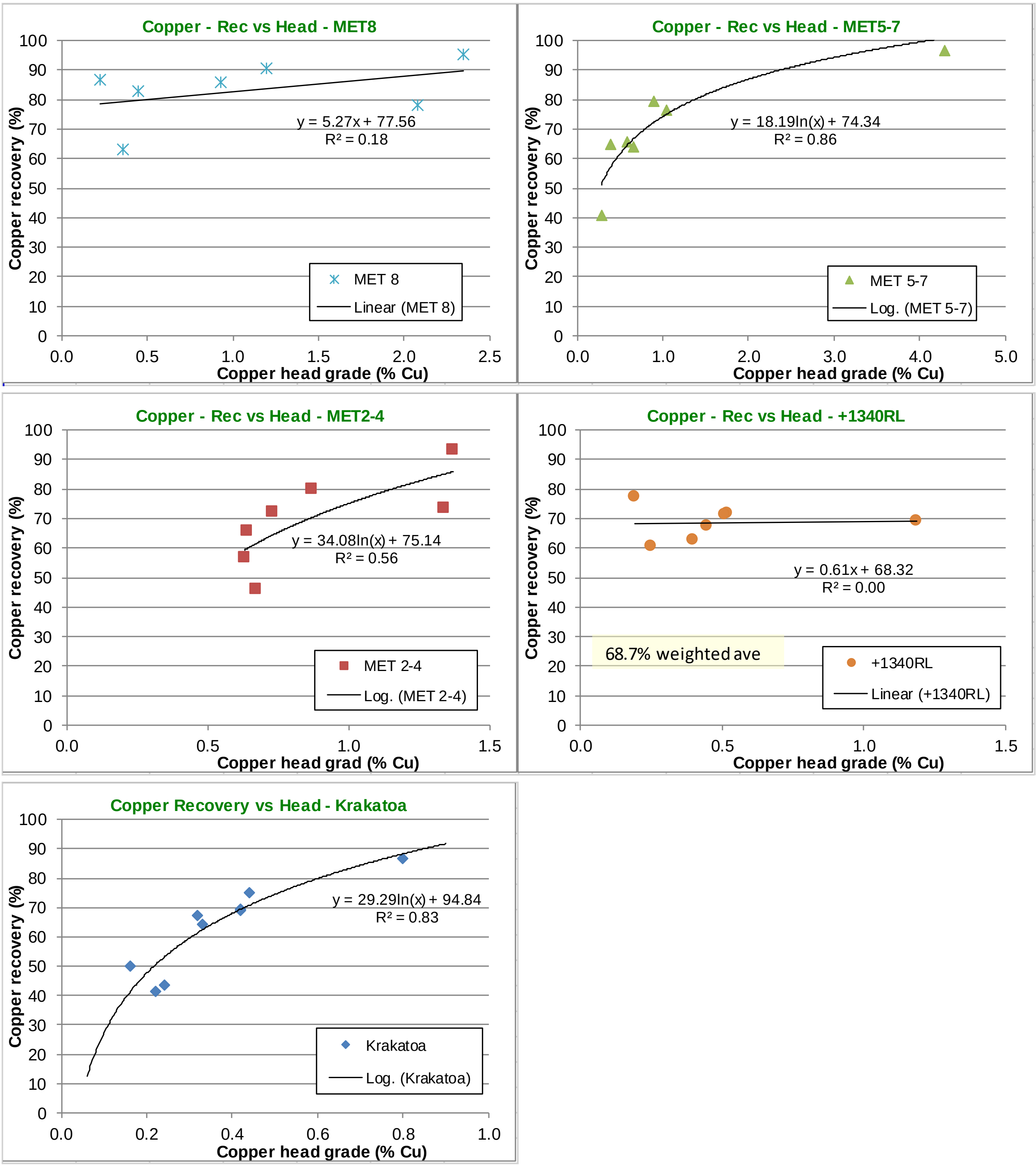


Figure 13-1: Copper head grade vs recovery

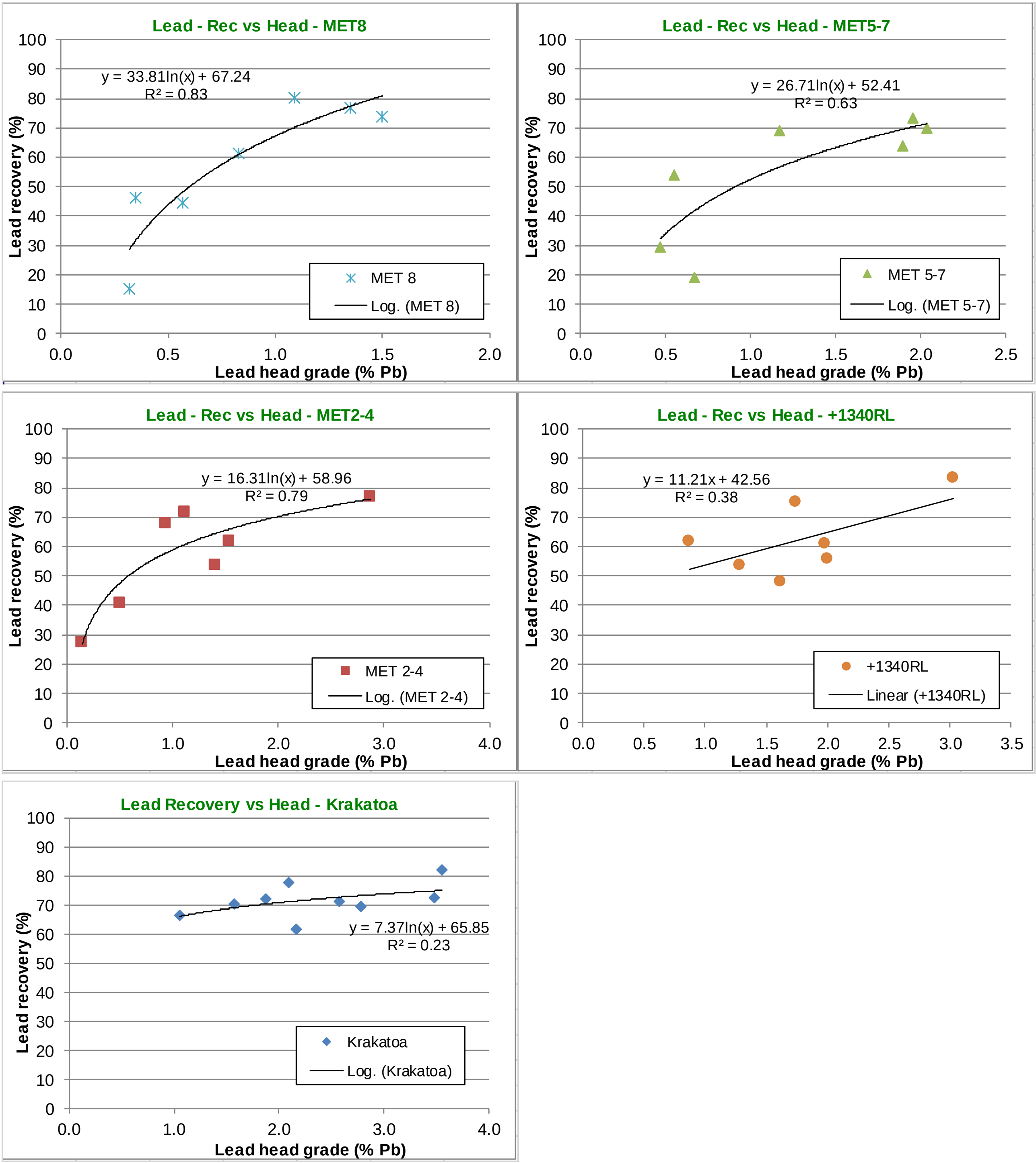


Figure 13-2: Lead head grade vs recovery

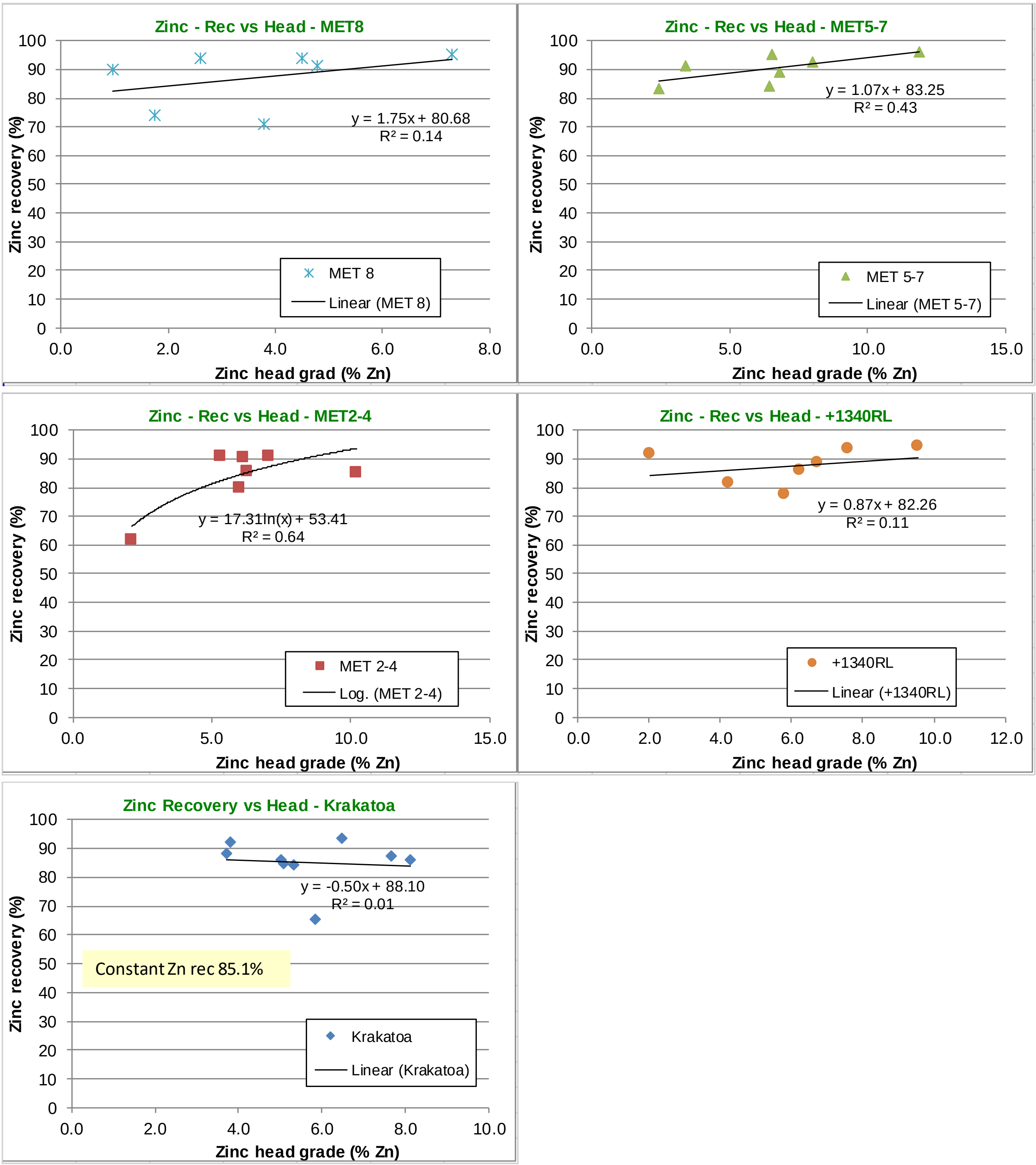


Figure 13-3: Zinc head grade vs recovery

All models were validated by examining the predicted performance for locked cycle test samples against the actual locked cycle test results, with offsets applied to align batch test predictions to locked cycle test results.

The final recovery models, inclusive of offsets, for base and precious metals are detailed in Table 13-7.

Table 13-7: Final process recovery models

Concentrate	Met domain	Base metal recovery	Gold recovery	Silver recovery
Copper Concentrate	+1340 RL	67.5	18.6	33.9
	MET2-4	34.08*ln(Cu head)+79.54	34.2	36.6
	MET5-7	18.19*ln(Cu head)+69.04	27.3	34.2
	MET8	5.27*(Cu head)+82.16	56.0	31.1
	Krakatoa	29.29*ln(Cu head)+85.84	29.8	49.1
Lead Concentrate	+1340 RL	11.21*(Pb head)+56.36	35.5	37.2
	MET2-4	16.31*ln(Pb head)+59.76	20.5	40.6
	MET5-7	26.71*ln(Pb head)+62.41	33.1	42.2
	MET8	33.81*ln(Pb head)+61.44	5.7	25.2
	Krakatoa	7.37*ln(Pb head)+71.45	27.2	36.5
Zinc Concentrate	+1340 RL	0.87*(Zn head)+77.66	6.8	11.0
	MET2-4	17.31*ln(Zn head)+57.01	9.5	17.0
	MET5-7	1.07*(Zn head)+81.35	9.1	11.2
	MET8	1.75*(Zn head)+78.58	9.2	7.7
	Krakatoa	89.2	7.5	9.0

13.9 Processing Schedule

Ore has been scheduled to be processed at a maximum throughput rate of 2.2 Mt/a. As noted in Section 17.1, two primary design cases have been considered in the process design. When the scheduled zinc grade processed is in excess of that considered in the design cases, process plant throughput has been reduced so that the design case metal content (2.0 Mt/a throughput rate @ 6.52% zinc grade, or 357 t of contained zinc per day) is not exceeded.

Copper and lead grades are typically inversely related (high copper with low lead and vice versa). Capacity constraints are associated with copper rather than lead and when high copper grades are scheduled lead grades are typically well below the design capacity indicating surplus capacity in the lead circuit. It is proposed that when periods of high copper grades are planned in the mine schedule that some of the lead circuit capacity could be repurposed to provide additional copper circuit capacity.

An ore commissioning plan has been prepared for the commissioning and ramp up of the processing facilities. Ore commissioning will commence with mineralized waste to facilitate testing the operational capacity of the grinding and tailings circuits. Flotation circuits will then be progressively commissioned starting with the copper circuit followed by lead and then zinc. The progressive increase in throughput rates, processing recoveries and concentrate grades are detailed in Table 13-8.

Table 13-8: Process Plant commissioning schedule

Month	Plant feed[1] (%)	Copper concentrate				Lead concentrate				Zinc concentrate			
		Recovery (%)[2]			Concentrate grade (%)[3]	Recovery (%)[2]			Concentrate grade(%)[3]	Recovery (%)[2]			Concentrate grade(%)[3]
		Cu	Au	Ag		Pb	Au	Ag		Zn	Au	Ag	
Dec 2021	10.7 (Mineralized Waste Only)												
Jan 2022	36.0	48.9	48.9	48.9	73.7	14.5	14.5	14.5	53.4	0.0	0.0	0.0	0.0
Feb 2022	64.2	79.2	79.2	79.2	92.4	48.9	48.9	48.9	80.0	61.8	61.8	61.8	83.4
Mar 2022	79.0	91.1	91.1	91.1	99.2	68.0	68.0	68.0	95.7	79.7	79.7	79.7	92.6
Apr 2022	89.3	97.0	97.0	97.0	100	80.1	80.1	80.1	99.1	88.0	88.0	88.0	95.6
May 2022	92.4	99.7	99.7	99.7	100	88.0	88.0	88.0	99.9	94.9	94.9	94.9	97.7
Jun 2022	95.5	100	100	100	100	95.3	95.3	95.3	100	99.2	99.2	99.2	99.6
Jul 2022	96.8	100	100	100	100	99.3	99.3	99.3	100	100	100	100	100
Aug 2022	100	100	100	100	100	100	100	100	100	100	100	100	100
Sep 2022	100	100	100	100	100	100	100	100	100	100	100	100	100

Notes: 1. Percent of designed plant throughput. 2. Percent of expected recovery. 3. Percent of expected concentrate grade.

The LOM processing schedule Process Plant recoveries and proportions of each metallurgical domain are summarized in Table 13-9.

While relationships were developed for predicting concentrate grades for each domain, constant concentrate grades have been applied in the financial model of 25%, 52% and 52% for copper, lead and zinc concentrates respectively. The process plant will be operated to target a steady concentrate grade and the financial model reflects this operating practice.

The predictive recovery models were applied to the life of mine processing schedule to predict concentrate qualities for those elements where sufficient data exists (the economic elements and those deleterious elements that will attract penalty costs for marketing). For the purposes of assessing the marketability of concentrates, two separate concentrate qualities have been defined and are detailed in Section 19.

In the first 18 months of the project, the +1340 RL metallurgical domain will be the sole source of ore for processing (Type A Concentrate Quality), until the open pit reaches sufficient depth to access other metallurgical domains and enable blending (Type B Concentrate Quality).

Estimates of concentrations of the other elements in each concentrate were prepared by reviewing ICP data of the various concentrate products produced from locked cycle laboratory testing.

Table 13-9: LOM processing schedule

Parameter	Units	Total	2021	2022					2023					2024	2025	2026	2027	2028	2029
			Q4	Q1	Q2	Q3	Q4	Total	Q1	Q2	Q3	Q4	Total						
Ore processed	Mt	**15.7**	-	0.3	0.5	0.5	0.5	1.8	0.5	0.5	0.5	0.5	2.0	2.0	2.2	2.1	2.1	2.2	1.4
Mineralized waste	Mt	**0.0**	0.0	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Zinc grade	% Zn	**5.8**	-	6.6	6.5	6.6	6.4	6.5	6.3	6.3	6.6	7.1	6.6	6.45	5.3	5.9	5.7	5.3	4.7
Copper grade	% Cu	**0.9**	-	0.4	0.5	0.5	0.7	0.5	0.9	0.8	0.9	0.8	0.9	1.1	1.2	0.9	0.8	0.7	0.7
Lead grade	% Pb	**1.7**	-	2.1	2.0	2.0	1.9	2.0	1.8	1.7	1.7	1.8	1.7	1.7	1.3	1.8	1.7	1.7	1.8
Gold grade	g/t Au	**1.3**	-	1.7	1.7	1.7	1.6	1.7	1.6	1.5	1.4	1.4	1.5	1.4	1.2	1.3	1.1	1.2	1.3
Silver grade	g/t Ag	**138**	-	171	167	166	158	165	157	149	1.42	145	148	145	119	136	121	131	143
Iron grade	% Fe	**29**	-	27	28	29	29	29	30	30	31	33	31	32	32	29	27	26	24
Arsenic grade	ppm As	**2,579**	-	3,110	3,116	3,513	3,077	3,212	3,233	3,280	3,393	3,173	3,271	2,722	2,211	2,775	2,021	2,038	2,535
Mercury grade	ppm Hg	**18**	-	20	21	25	25	23	21	25	22	27	24	22	18	16	12	15	15
Antimony grade	ppm Sb	**463**	-	657	624	758	785	712	710	686	631	568	651	513	369	465	329	332	347
Selenium grade	ppm Se	**197**	-	143	151	158	175	158	206	195	207	221	207	201	184	182	212	227	198
Zinc concentrate																			
Zinc recovery	%	**86**	-	48	78	83	83	76	84	83	85	89	85	88	86	88	89	88	88
Gold recovery	%	**8**	-	4	6	7	7	6	7	7	7	9	7	9	9	9	9	9	8
Silver recovery	%	**11**	-	6	10	11	11	10	11	11	11	12	11	12	12	12	11	10	9
Copper concentrate																			
Copper recovery	%	**74**	-	53	67	68	68	65	68	68	72	74	69	77	80	74	72	71	74
Gold recovery	%	**27**	-	15	18	19	19	18	19	19	21	29	21	29	30	30	31	31	33
Silver recovery	%	**37**	-	27	34	34	34	32	34	34	34	35	34	35	34	37	40	41	44
Lead concentrate																			
Lead recovery	%	**74**	-	40	69	79	77	69	77	76	75	76	76	75	70	76	74	74	75
Gold recovery	%	**29**	-	18	31	35	36	31	35	36	34	30	34	30	29	29	27	28	25
Silver recovery	%	**38**	-	19	33	37	37	33	37	37	38	42	39	41	40	41	39	38	36
Ore processed by domain																			
+1340 RL domain	%	**24**	-	100	100	100	100	100	96	100	75	0	69	11	17	0	0	0	0
MET2-4 domain	%	**12**	-	0	0	0	0	0	2	0	7	24	8	23	19	24	12	5	1
MET5-7 domain	%	**38**	-	0	0	0	0	0	2	0	18	76	23	59	50	55	44	44	21
MET8 domain	%	**10**	-	0	0	0	0	0	0	0	0	0	0	7	14	7	15	17	26
Krakatoa domain	%	**15**	-	0	0	0	0	0	0	0	0	0	0	0	0	15	29	33	51

14 Mineral Resource Estimates

14.1 Introduction

The ABM MRE has an effective date of 31 May 2017 as previously reported in the PFS Technical Report (CSA Global, 2017). The MRE is re-reported herein and is in accordance with the Canadian Securities Administrators' NI 43-101. The MRE is generated in conformity with generally accepted CIM "Estimation of Mineral Resource and Mineral Reserves Best Practice Guidelines" (CIM Council, 2003) and CIM "Definition Standards for Mineral Resources and Mineral Reserves" (CIM Council, 2014).

Previous MREs generated for the deposit are described in Section 6.2. The current MRE presented in this report supersedes all past estimates.

Reported Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. There is no guarantee that all or any part, of a Mineral Resource will be converted into a Mineral Reserve.

The resource estimation methodology for the ABM deposit comprised the following procedures:

- Model mineralized wireframes based on logged lithology and sample grade values
- Generate geological wireframes for the overburden surface, mafic intrusive units and faults based on logged geology and field observations
- Define resource domains
- Verify the drilling data against the LiDAR topographic surface
- Data compositing and declustering for geostatistical analysis, variography and validation
- Application of top-cuts based on geostatistical analysis
- Construct block model following kriging neighbourhood analysis (KNA)
- Grade interpolation using standard techniques such as OK, ID2 or ID3
- Application of a NSR to define a reporting cut-off and provide a basis for "reasonable prospects of economic extraction"
- Resource classification, validation and reporting
- Technical resource report on the MRE.

14.2 Database Cut-Off

The current ABM resource model was prepared using all drilling data available at 11 September 2016. The data included historical drilling results from the KZK Project as well as results from the 2015 and 2016 exploration program. The data was stored in a Microsoft Access database and named "Kzk_resource_database_20161022.mdb".

Additional drilling has been completed at the KZK Project after the effective date (as detailed in Section 10.1.2); however, these holes tested exploration targets outside the ABM deposit area or have no material effect on the reported ABM Mineral Resource.

14.2.1 Data Excluded

Drillholes were flagged under the "res_inclusion" field of the Collar table of the Microsoft Access database as either "y" (yes) or "n" (no) for inclusion into the resource estimate.

A total of 32 holes were excluded from the ABM MRE dataset (Table 14-1). The majority of holes excluded were metallurgical holes and/or wedges. Two holes; K95-168 and K95-169, were excluded from the MRE as exact collar locations of these historical, Cominco-drilled holes could not be confirmed and the downhole surveys appeared questionable. These holes were drilled down plunge of the Main Zone and showed poor correlation with surrounding holes.

Table 14-1: Listing of excluded drillholes from the ABM deposit MRE

Hole ID	Hole depth (m)	Easting	Northing	Elevation	Reason	Parent hole ID
K95-168	171	414702.59	6815342.8	1,418.65	Drilled down plunge	
K95-169	157	414650.99	6815351.12	1,430.48	Drilled down plunge	
K15-201	35	414795.321	6815362.914	1,400.253	Metallurgical twin	K15-202
K15-205	146	414849.957	6815542.83	1,395.479	Metallurgical twin	K15-203
K15-213	99	414625.136	6815357.987	1,436.3	Metallurgical	
K15-216W1	182	414845.596	6815743.656	1,394.852	Metallurgical wedge	K15-216
K15-221	44	414675.698	6815358.001	1,424.236	Metallurgical twin	K15-218
K15-225	25	414752.728	6815351.293	1,408.902	Metallurgical twin	K15-223
K15-226W1	182	414850.978	6815676.392	1,396.44	Metallurgical wedge	K15-226
K15-230	41	414871.81	6815378.906	1,390.693	Metallurgical twin	K15-227
K15-237	119	414750.629	6815496.738	1,406.768	Metallurgical twin	K15-236
K15-238W1	194	414901.586	6815741.065	1,388.746	Metallurgical wedge	K15-238
K15-241	35	414952.382	6815425.373	1,383.307	Metallurgical twin	K15-240
K15-241R	65	414952.057	6815422.013	1,383.379	Metallurgical twin	K15-240
K15-243W1	200	414801.777	6815776.435	1,403.945	Metallurgical wedge	K15-243
K15-245	70	415050.945	6815416.155	1,386.662	Metallurgical twin	K15-244
K15-246	129	415134.011	6815441.807	1,400.731	Metallurgical twin	K15-242
K15-252	41	415048.949	6815336.681	1,386.143	Metallurgical twin	K15-251
K15-256	32	415100.747	6815309.079	1,393.181	Metallurgical twin	K15-255
K15-260W1	196	414749.033	6815674.459	1,411.257	Metallurgical wedge	K15-260
K15-262	348	415101.047	6815372.083	1,391.439	Geotech	
K15-264W1	176.7	414800.261	6815624.607	1,401.87	Metallurgical wedge	K15-264
k15-266	110	414698.867	6815466.42	1,418.903	Metallurgical twin	K15-272
K15-269	72.72	414698.872	6815466.422	1,418.874	Resource/Met	
K15-270	170	415151.29	6815552.105	1,402.992	Metallurgical twin	K15-267
K15-275	122	415051.127	6815507.001	1,382.274	Metallurgical twin	K15-273
K15-276	110	414675.458	6815453.463	1,425.014	Metallurgical twin	K15-274
K15-278W1	161	414625.354	6815538.706	1,437.145	Metallurgical wedge	K15-278
K15-281W1	199.6	414594	6815656	1,447.51	Metallurgical wedge	K15-281
K15-283	190.7	415026.341	6815454.157	1,382.958	Metallurgical twin	K15-279
K15-284W1	191	414653.695	6815652.409	1,430.397	Metallurgical wedge	K15-284
K15-288	86	414800.352	6815436.95	1,400.507	Metallurgical twin	K15-287

14.3 Preparation of Wireframes

14.3.1 Mineralization

Each 25 m spaced cross section (or 50 m spaced oblique section for Krakatoa) was displayed in Surpac together with drillhole traces which were colour-coded according to logged lithology and sample grade values. Separate sets of strings were generated for the polymetallic (Cu-Pb-Zn-Au-Ag) massive sulphide, stockwork/disseminated sulphide mineralization, mafic volcanic footwall unit, overburden surface, top of fresh rock surface and interpreted faults.

The following techniques were employed whilst interpreting the mineralization:

- Each cross section was displayed on screen with a clipping window equal to a half distance from the adjacent sections.
- All interpreted strings were snapped to either lithology and/or assay drillhole intervals.
- Internal waste within the mineralized envelopes was not interpreted and modelled (with the exception of a small internal waste zone in the Krakatoa Zone). Instead, it was either included in the interpreted envelopes or split using bifurcation techniques where supported by surrounding drill information.
- If a mineralized envelope did not extend to the adjacent drillhole section, it was projected halfway to the next section, and terminated. The general direction and dip of the envelopes was maintained, although the lens thickness was reduced from the last known intersection.
- Where no drillhole was present down dip, the mineralization was extended approximately 25–40 m down dip (roughly half the drill spacing on section).
- If a mineralized lens extended to the overburden surface, it was extended, at the same width as the last drillhole, above the surface to ensure there would not be any gaps between the lens and the overburden when the block model was built.

Figure 14-1 shows an example of an interpreted cross section with mineralization and geological features.

The interpreted strings were used to generate 3D solid wireframes for the mineralized envelopes. Every section was displayed on-screen along with the closest interpreted section. If the corresponding envelope did not appear on the next cross section, the former was projected halfway to the next section, where it was terminated.

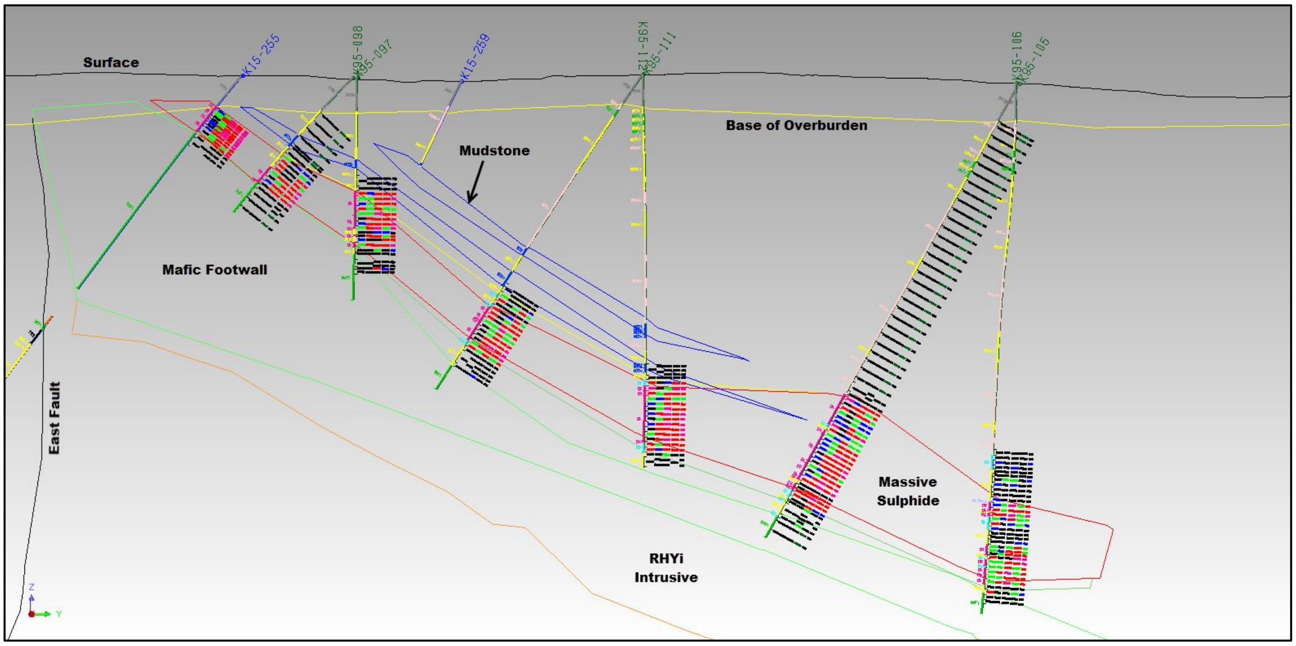


Figure 14-1: Example of interpretation of ABM mineralization and geology – Section 415,100 m E

Separate mineralization wireframes were generated for the ABM and Krakatoa zones respectively (Figure 14-2). The wireframes were also separated as either being massive sulphide or stockwork/disseminated mineralization based on logged geology.

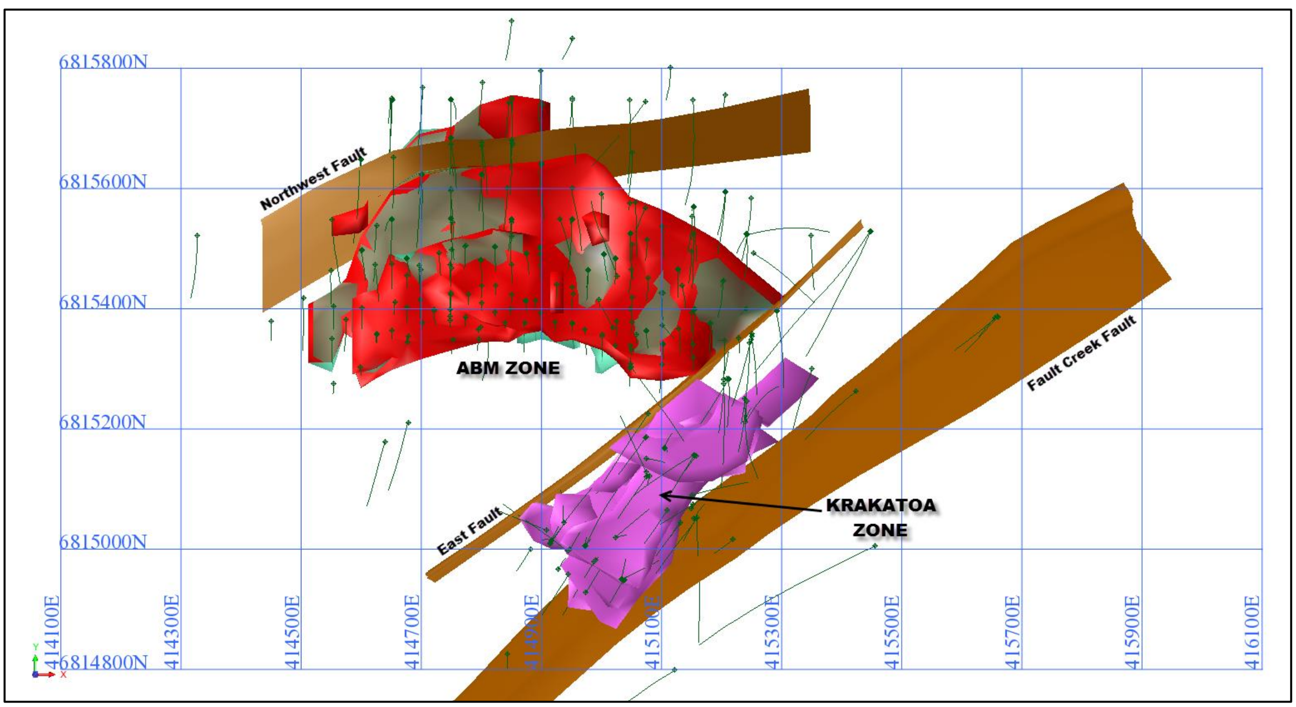


Figure 14-2: Plan view of the ABM Zone and Krakatoa Zone mineralization wireframes with fault surfaces

At BMC's request, a dilution skin encompassing all the mineralization wireframes was also created for mine planning purposes. A minimum 3 m downhole intersection was used for wireframing the dilution skin. Separate dilution skin wireframes were generated for the ABM and Krakatoa zones respectively.

14.3.2 Lithology and Structure

Lithological and structural features were defined from logged and interpreted geology. The following features were wireframed in Surpac:

- Mafic intrusive
- Rhyolite intrusive
- Carbonaceous mudstones
- Wind Lake formation
- Northwest Fault
- East Fault
- Fault Creek Fault
- Krakatoa faults
- Overburden surface.

14.3.3 Weathering

Logging and relogging of current and historical drill core determined no significant weathering profile for the ABM deposit. As described in Section 11.4.3, EDTA analyses also indicated no significant weathering effects near surface at the ABM deposit. However, geotechnical consultants Dempers & Seymour Pty Ltd (D&S) and Tetra Tech have modelled a fractured zone below the base of overburden and D&S have applied adjustments to the pit slope profile to account for the fractured zone.

14.4 Topography

In late 2015, BMC contracted Challenger Geomatics Ltd of Whitehorse, Yukon to complete a detailed LiDAR survey over the key sectors of the KZK Project. The survey focused over the ABM deposit area and potential infrastructure sites.

The survey was undertaken in September 2015 using a Leica ALS70 LiDAR system with a stated horizontal accuracy of 35 cm and vertical accuracy of 15 cm. The coordinate system for the survey was UTM Zone 9 NAD83.

14.5 Statistical Analysis – Kriging Neighbourhood Analysis

Statistical analysis was carried out by CSA Global using Supervisor v8.4™ and GeoAccess Professional™ software packages.

14.6 Drillhole Coding

Drillhole coding is a standard procedure which ensures that the correct samples are used in classical statistical and geostatistical analyses, and grade interpolation. For this purpose, solid wireframes for each mineralized envelope were used to select drillhole samples. Samples were then selected for individual mineralized envelopes and flagged for each mineralization zone and geological domain using Surpac software.

Lithological and mineralization wireframes were used to select drillhole samples, and the data was assigned a code in the field "POD". A summary of the POD codes used to distinguish the data during geostatistical analysis and estimation is shown in Table 14-2.

Table 14-2: POD field and description for the ABM deposit

POD	Zone	Description
2	ABM	Dilution
219	Krakatoa	Dilution
4, 13, 23, 33, 43, 53, 63, 73, 83, 93, 103, 113, 123, 133, 143, 153	ABM	Stockwork
202	Krakatoa	Stockwork
8, 17, 27, 37, 47, 57, 67	ABM	Massive sulphide
208, 209, 217, 218, 228, 238, 248, 258	Krakatoa	Massive sulphide

14.7 Sample Length Analysis

Based on the drillhole coding, samples from within the resource wireframes were used to conduct a sample length analysis.

The majority of raw sample intervals are 1.5 m in length for ABM (Figure 14-3) and 1.0 m in length for Krakatoa (Figure 14-4). Composites were initially extracted at 1.0 m intervals for the ABM Zone; however, this split many >1 m historical samples, which produced an overly "smoothed" set of variogram models with very low nugget effect. Therefore 1.5 m and 1.0 m were selected as the composite lengths for ABM and Krakatoa respectively, as these lengths reflect majority of sample intervals within each of the deposits and are a suitable scale for the width of the resource wireframes.

Surpac software was then used to extract downhole composites using the "best-fit" algorithm within the mineralization intervals.

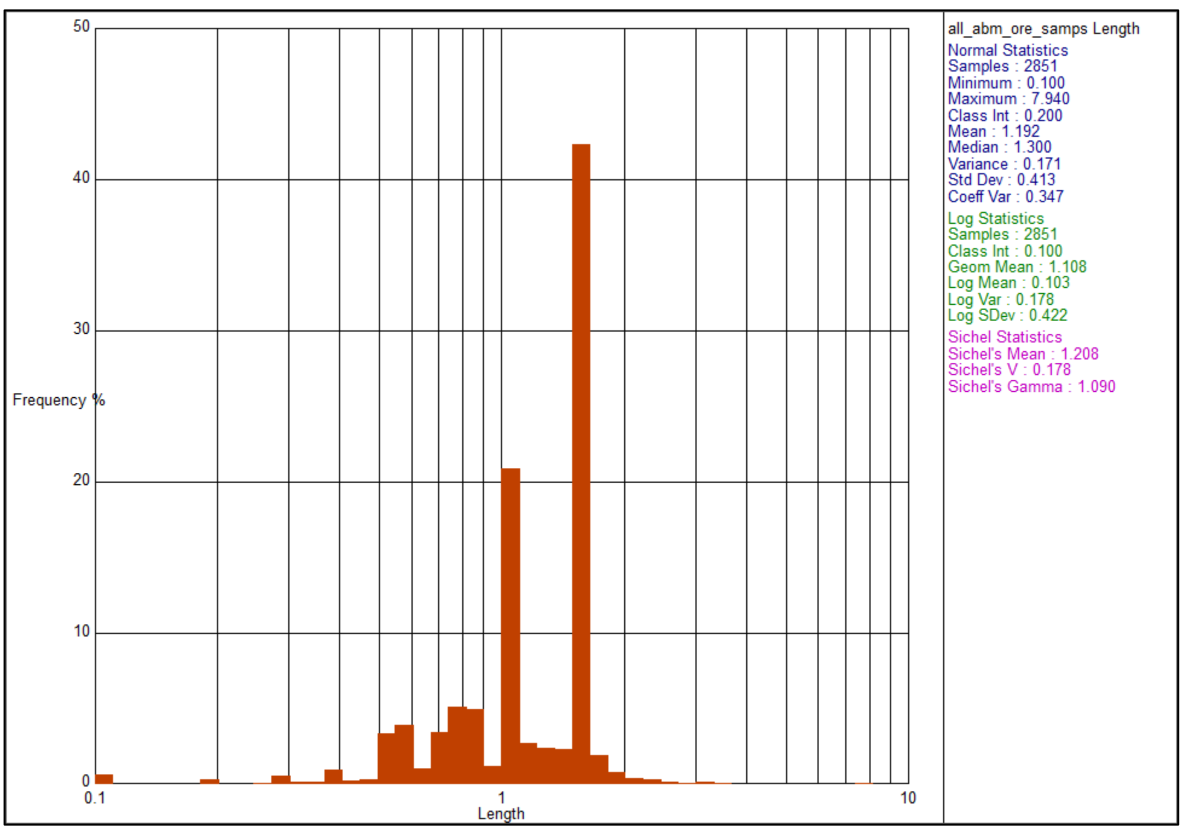


Figure 14-3: Normal histogram analysis of sample lengths in ABM database

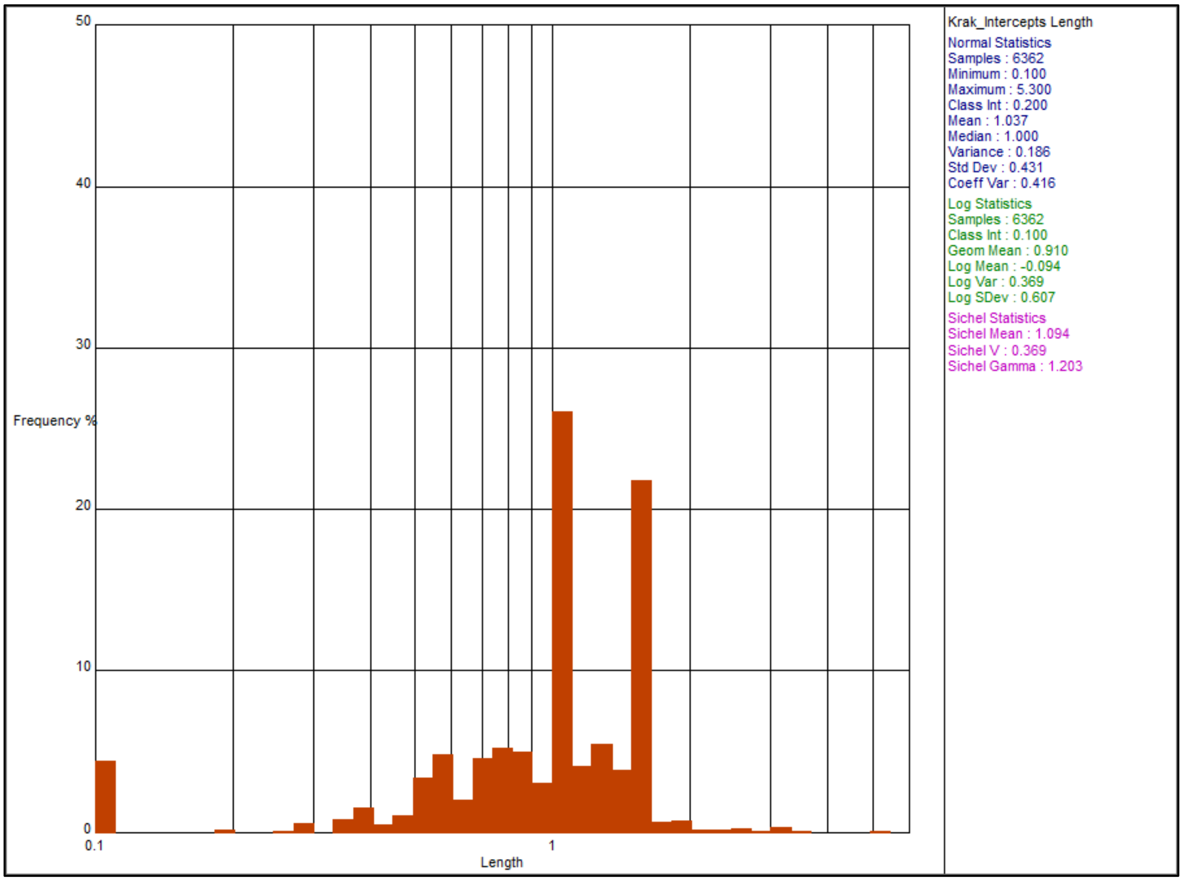


Figure 14-4: Normal histogram analysis of sample lengths in Krakatoa database

14.8 Compositing

Samples were composited at 1.5 m and 1.0 m intervals for the ABM and Krakatoa zones respectively. Basic statistical parameters were obtained for the composited data. Composites that were less or equal to 40% of the composite length were excluded from the geostatistical analysis and the estimate. This will limit any potential bias in the sample support during kriging.

14.9 Variables

Statistical analysis was carried out for the major elements Cu, Pb, Zn, Au, Ag and Fe, as well as As, Ba, Bi, Hg, S, Sb and Se. Analysis was completed for both the ABM and Krakatoa zones.

14.10 Global and Domain Statistics

For the purpose of reporting statistical analyses, the interpreted mineralized massive sulphide and stockwork domains were grouped into global domains for the ABM and Krakatoa zones. The global statistical domains are summarized in Table 14-3.

Table 14-3: Compilation of global statistical and reporting domains

Global domain	POD	Description
ABM	8, 17, 27, 37, 47, 57, 67 4, 13, 23, 33, 43, 53, 63, 73, 83, 93, 103, 113, 123, 133, 143, 153	Massive sulphide and stockwork
Krakatoa	202, 208, 217, 218, 228, 238, 248, 258	Massive sulphide and stockwork

The log histograms for the major elements are shown in Figure 14-5 and Figure 14-6. These plots are overlain with the log cumulative distribution function plots.

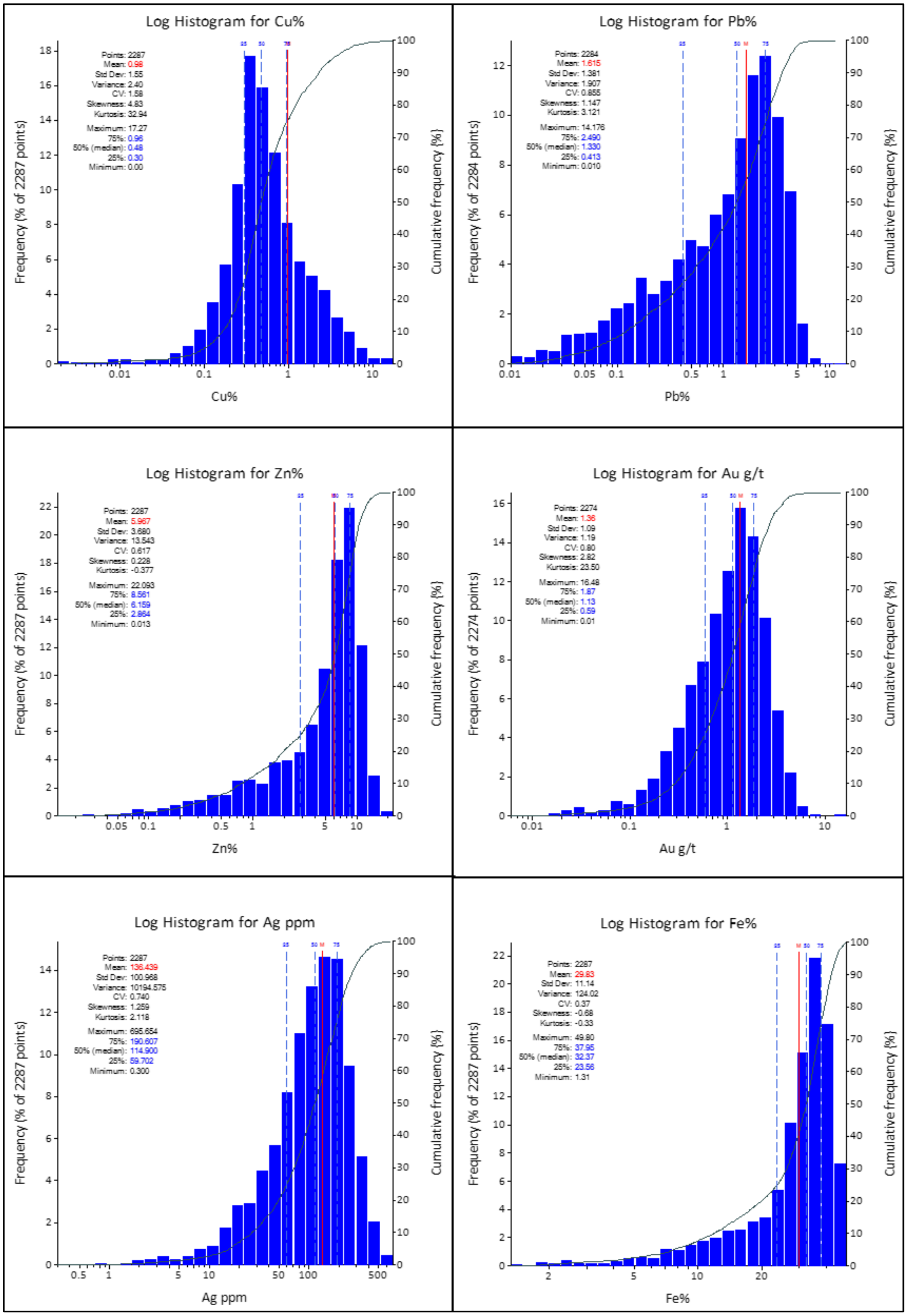


Figure 14-5: ABM Zone – global sample distribution for major elements (clustered, composited and uncut)

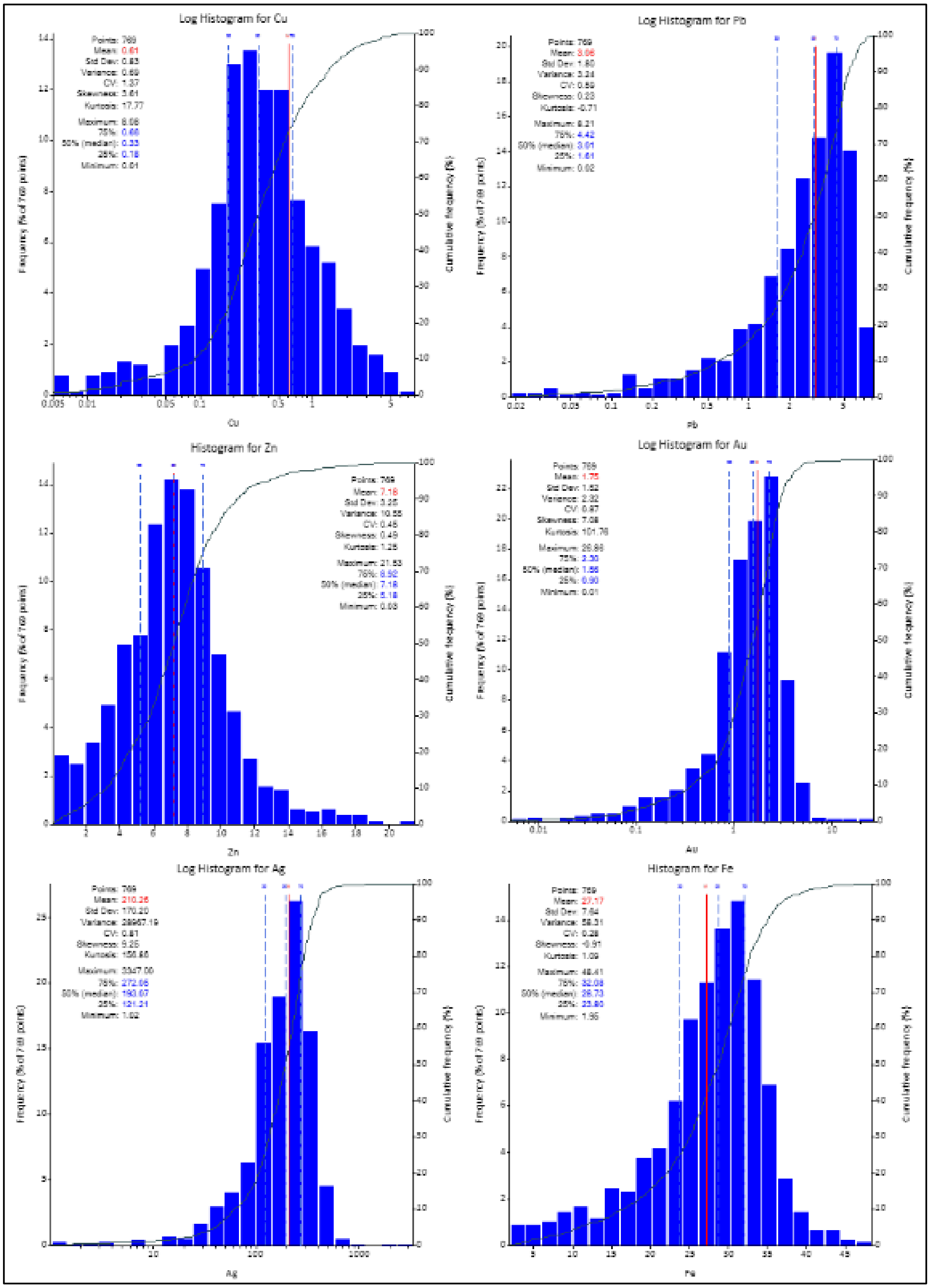


Figure 14-6: Krakatoa Zone – global sample distribution for major elements (clustered, composited and uncut)

Global statistics for the clustered, composited and un-cut major elements for the ABM and Krakatoa zones are shown in Table 14-4 and Table 14-5 respectively. Global statistics for the clustered, composited and uncut minor elements for the ABM and Krakatoa zones are shown in Table 14-6 and Table 14-7 respectively. Cu% was characterized by a higher CV (dispersion of grade around the mean grade) than the rest of the major elements.

Table 14-4: Major elements global statistics for ABM Zone

ABM	Cu (%)	Pb (%)	Zn (%)	Au (g/t)	Ag (g/t)	Fe (%)
Total samples	2,287	2,284	2,287	2,274	2,287	2,287
Minimum	0.002	0.01	0.01	0.01	0.30	1.31
Maximum	17.27	14.18	22.09	16.48	695.65	49.80
Mean	0.98	1.62	5.97	1.36	136.44	29.83
Median	0.48	1.33	6.16	1.13	114.90	32.37
Variance	2.40	1.91	13.54	1.19	10,195	124.02
Standard deviation	1.55	1.38	3.68	1.09	100.97	11.14
CV	1.58	0.86	0.62	0.80	0.74	0.37

Note: Clustered, composited and uncut.

Table 14-5: Major elements global statistics for Krakatoa Zone

Krakatoa	Cu (%)	Pb (%)	Zn (%)	Au (g/t)	Ag (g/t)	Fe (%)
Total samples	769	769	769	769	769	769
Minimum	0.005	0.02	0.03	0.005	1.02	1.95
Maximum	8.08	8.21	21.53	26.86	3,347	48.41
Mean	0.61	3.06	7.18	1.75	210	0.61
Median	0.33	3.01	7.18	1.56	193	0.33
Variance	0.69	3.24	10.55	2.32	28,967	0.69
Standard deviation	0.83	1.80	3.25	1.52	170	0.83
CV	1.37	0.59	0.45	0.87	0.81	1.37

Note: Clustered, composited and uncut.

Table 14-6: Minor elements global statistics for ABM Zone

ABM	As (ppm)	Ba (ppm)	Bi (ppm)	Hg (ppm)	S (%)	Sb (ppm)	Se (ppm)
Total samples	2,253	2,147	2,253	1,118	862	2,253	1,607
Minimum	1.00	2.00	0.01	0.01	0.04	0.05	0.50
Maximum	39,740	255,085	608	182	48.70	7,112	1,425
Mean	2,458	13,450	56	18	28.72	460	114
Median	1,384	2,700	44	10	34.90	228	0.50
Variance	9,494,791	686,884,819	2,562	532	197.31	388,538	28,690
Standard deviation	3,081	26,208	51	23	14.05	623	169
CV	1.25	1.95	0.91	1.26	0.49	1.35	1.49

Note: Clustered, composited and uncut.

Table 14-7: Minor elements global statistics for Krakatoa Zone

Krakatoa	As ppm	Ba ppm	Bi ppm	Hg ppm	S%	Sb ppm	Se ppm
Total samples	763	662	761	751	759	760	751
Minimum	11.42	100	0.083	0.098	0.2	5	1.82
Maximum	46,000	358,560	477	110	48.6	33,200	1,900
Mean	4,512	24,486	39	20	30.195	620	198
Median	2,839	1,200	24	15	32.168	372	144
Variance	27,137,516	3.38E+09	2,138	305	87.012	2,117,514	46,596
Standard deviation	5,209	58,120	46	17	9.33	1,455	216
CV	1.155	2.37	1.18	0.87	0.31	2.35	1.09

Note: Clustered, composited and uncut.

14.11 Correlations

Scatterplots were created for the global domains for the ABM deposit by plotting the clustered, composited and uncut variables against one another to assess relationships and possible correlations. Table 14-8 and Table 14-9 show the correlation matrices and Figure 14-7 and Figure 14-8 show selected scatterplots of variables with lines of regression where the correlation coefficient ('R') ≥0.70.

Review of the scatterplots show strong correlations between Ag and Au, Ag and Sb and Fe and S for the ABM Zone and between Ag and Au, Au and Sb, Ag and Sb and Fe and S for the Krakatoa Zone.

Table 14-8: Correlation matrix for ABM Zone

Indep/ Dep	Cu %	Pb %	Zn %	Au g/t	Ag ppm	Fe %	As ppm	Ba ppm	Bi ppm	Hg ppm	S %	Sb ppm	Se ppm
Cu %	1	-0.27	-0.23	0.23	0.10	0.12	-0.17	-0.16	0.60	-0.20	-0.09	-0.16	0.23
Pb %	-0.27	1	0.70	0.44	0.70	0.13	0.42	0.30	-0.23	0.44	0.36	0.50	-0.16
Zn %	-0.23	0.70	1	0.25	0.45	0.38	0.31	0.11	-0.04	0.34	0.54	0.31	-0.07
Au g/t	0.23	0.44	0.25	1	0.78	0.16	0.54	0.27	0.06	0.41	0.21	0.67	-0.10
Ag ppm	0.10	0.70	0.45	0.78	1	0.12	0.46	0.36	-0.02	0.54	0.23	0.74	-0.13
Fe %	0.12	0.13	0.38	0.16	0.12	1	0.14	-0.16	0.37	0.04	0.83	0.00	0.15
As ppm	-0.17	0.42	0.31	0.54	0.46	0.14	1	0.13	-0.14	0.41	0.31	0.53	-0.15
Ba ppm	-0.16	0.30	0.11	0.27	0.36	-0.16	0.13	1	-0.23	0.40	0.08	0.34	-0.12
Bi ppm	0.60	-0.23	-0.04	0.06	-0.02	0.37	-0.14	-0.23	1	-0.21	0.15	-0.18	0.27
Hg ppm	-0.20	0.44	0.34	0.41	0.54	0.04	0.41	0.40	-0.21	1	0.28	0.61	-0.23
S %	-0.09	0.36	0.54	0.21	0.23	0.83	0.31	0.08	0.15	0.28	1	0.17	0.32
Sb ppm	-0.16	0.50	0.31	0.67	0.74	0.00	0.53	0.34	-0.18	0.61	0.17	1	-0.18
Se ppm	0.23	-0.16	-0.07	-0.10	-0.13	0.15	-0.15	-0.12	0.27	-0.23	0.32	-0.18	1

Table 14-9: Correlation matrix for Krakatoa Zone

Indep/ Dep	Cu %	Pb %	Zn %	Au g/t	Ag ppm	Fe %	As ppm	Ba ppm	Bi ppm	Hg ppm	S %	Sb ppm	Se ppm
Cu %	1	-0.03	0.25	0.36	0.31	0.05	-0.13	-0.17	0.52	-0.03	-0.07	0.25	0.13
Pb %	-0.03	1	0.57	0.14	0.26	0.32	0.21	0.05	-0.07	0.12	0.38	-0.01	-0.04
Zn %	0.25	0.57	1	0.03	0.12	0.50	0.03	-0.23	0.53	0.22	0.55	-0.11	0.32
Au g/t	0.36	0.14	0.03	1	0.90	-0.14	0.27	0.17	-0.08	0.24	-0.04	0.88	-0.13
Ag ppm	0.31	0.26	0.12	0.90	1	-0.13	0.11	0.16	-0.07	0.30	-0.02	0.88	-0.03
Fe %	0.05	0.32	0.50	-0.14	-0.13	1	0.12	-0.27	0.27	0.07	0.85	-0.22	0.34
As ppm	-0.13	0.21	0.03	0.27	0.11	0.12	1	0.18	-0.10	0.27	0.17	0.10	-0.01
Ba ppm	-0.17	0.05	-0.23	0.17	0.16	-0.27	0.18	1	-0.28	0.03	-0.07	0.10	-0.27
Bi ppm	0.52	-0.07	0.53	-0.08	-0.07	0.27	-0.10	-0.28	1	-0.05	0.19	-0.12	0.29
Hg ppm	-0.03	0.12	0.22	0.24	0.30	0.07	0.27	0.03	-0.05	1	0.21	0.11	0.14
S %	-0.07	0.38	0.55	-0.04	-0.02	0.85	0.17	-0.07	0.19	0.21	1	-0.18	0.31
Sb ppm	0.25	-0.01	-0.11	0.88	0.88	-0.22	0.10	0.10	-0.12	0.11	-0.18	1	-0.14
Se ppm	0.13	-0.04	0.32	-0.13	-0.03	0.34	-0.01	-0.27	0.29	0.14	0.31	-0.14	1

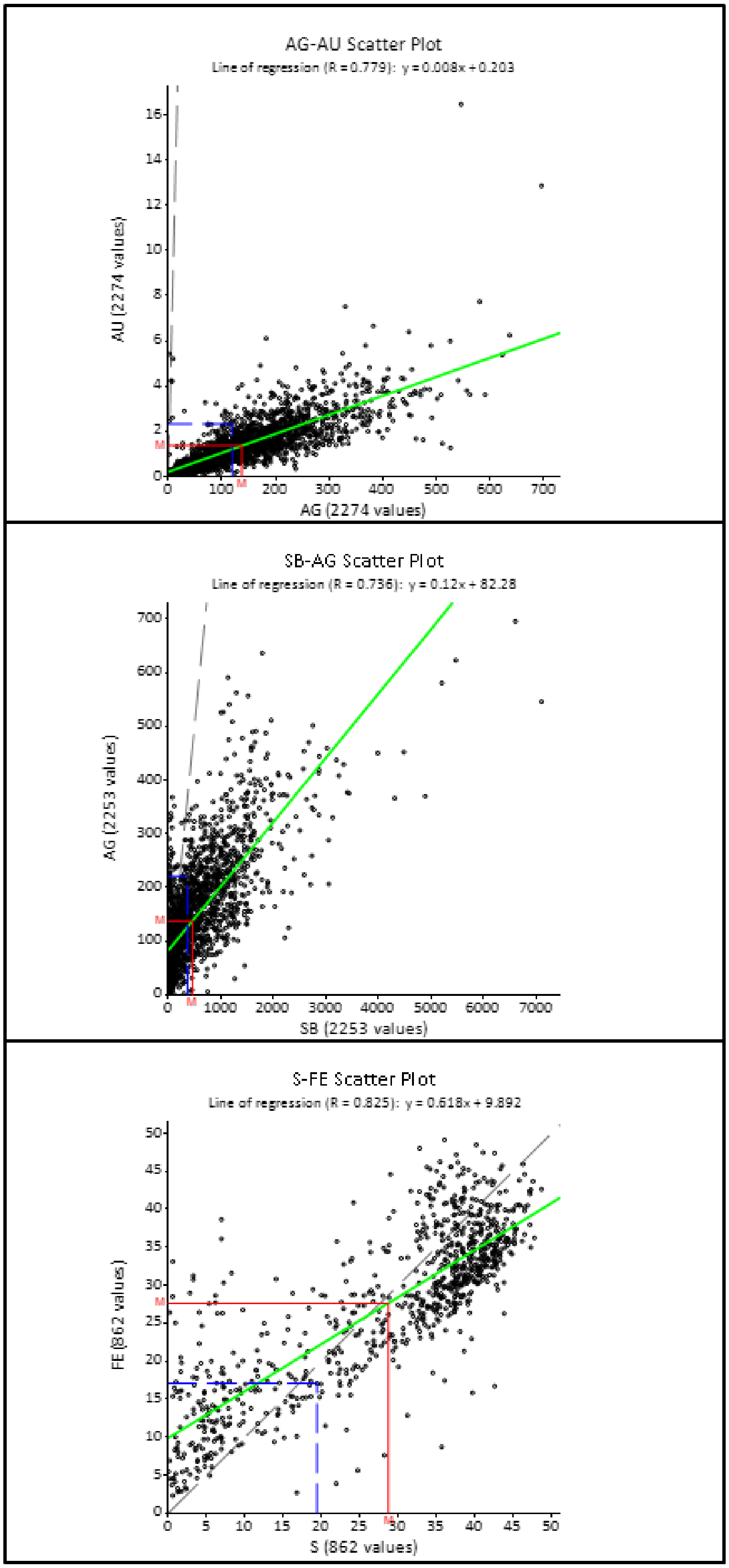


Figure 14-7: ABM Zone – scattergram correlation plots for variables with line of regression ≥0.70

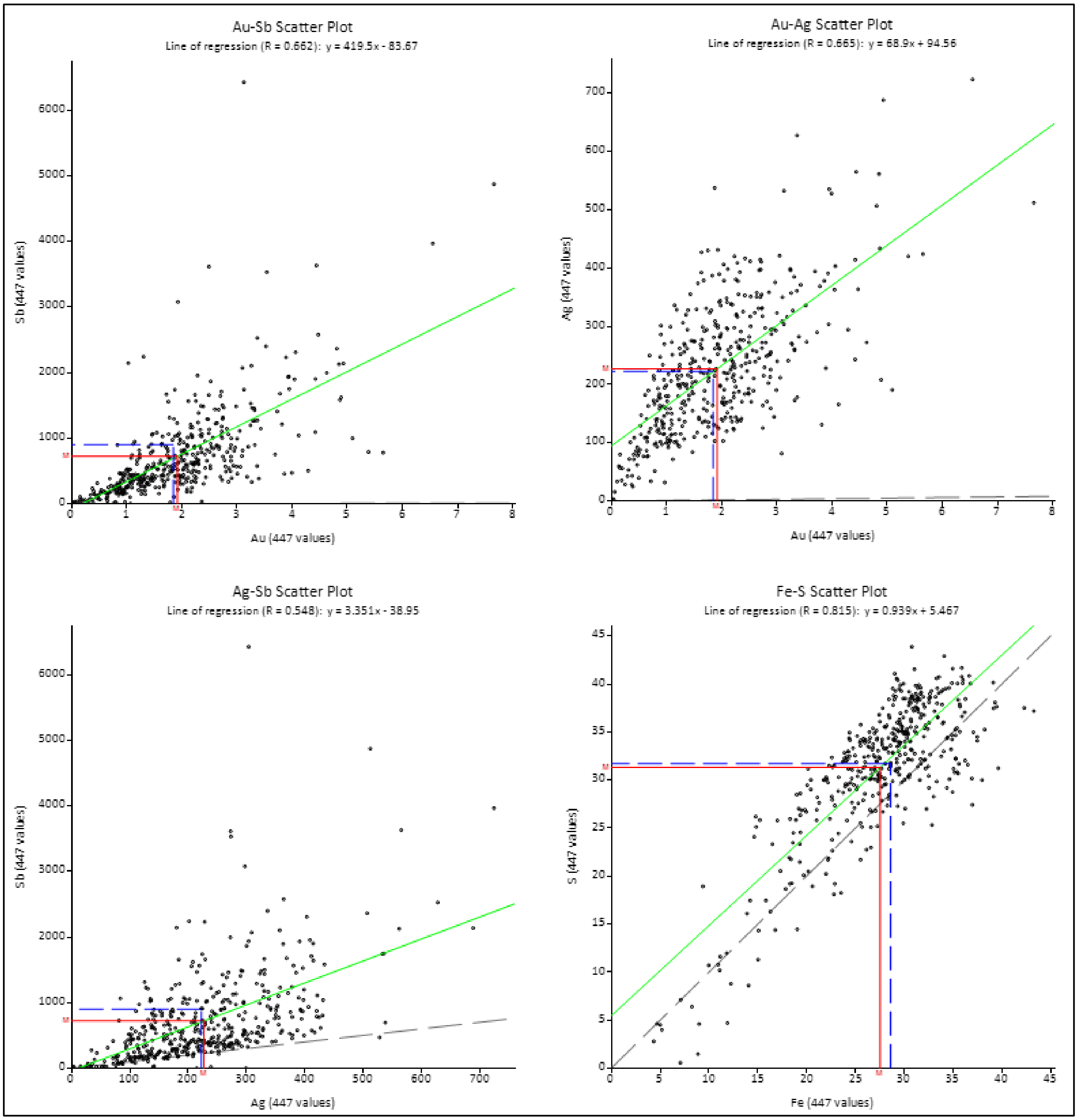


Figure 14-8: Krakatoa Zone – scattergram correlation plots for variables with line of regression ≥0.70

Cutting of Au (10 g/t), Ag (1,500 g/t) and Sb (6,000 ppm) for the Krakatoa Zone results in a relative drop in correlations for Au:Ag (0.86 to 0.74), Au:Sb (0.81 to 0.69) and Ag:Sb (0.85 to 0.65). Scatterplots showing the results of top cuts on the data are shown in Figure 14-9.

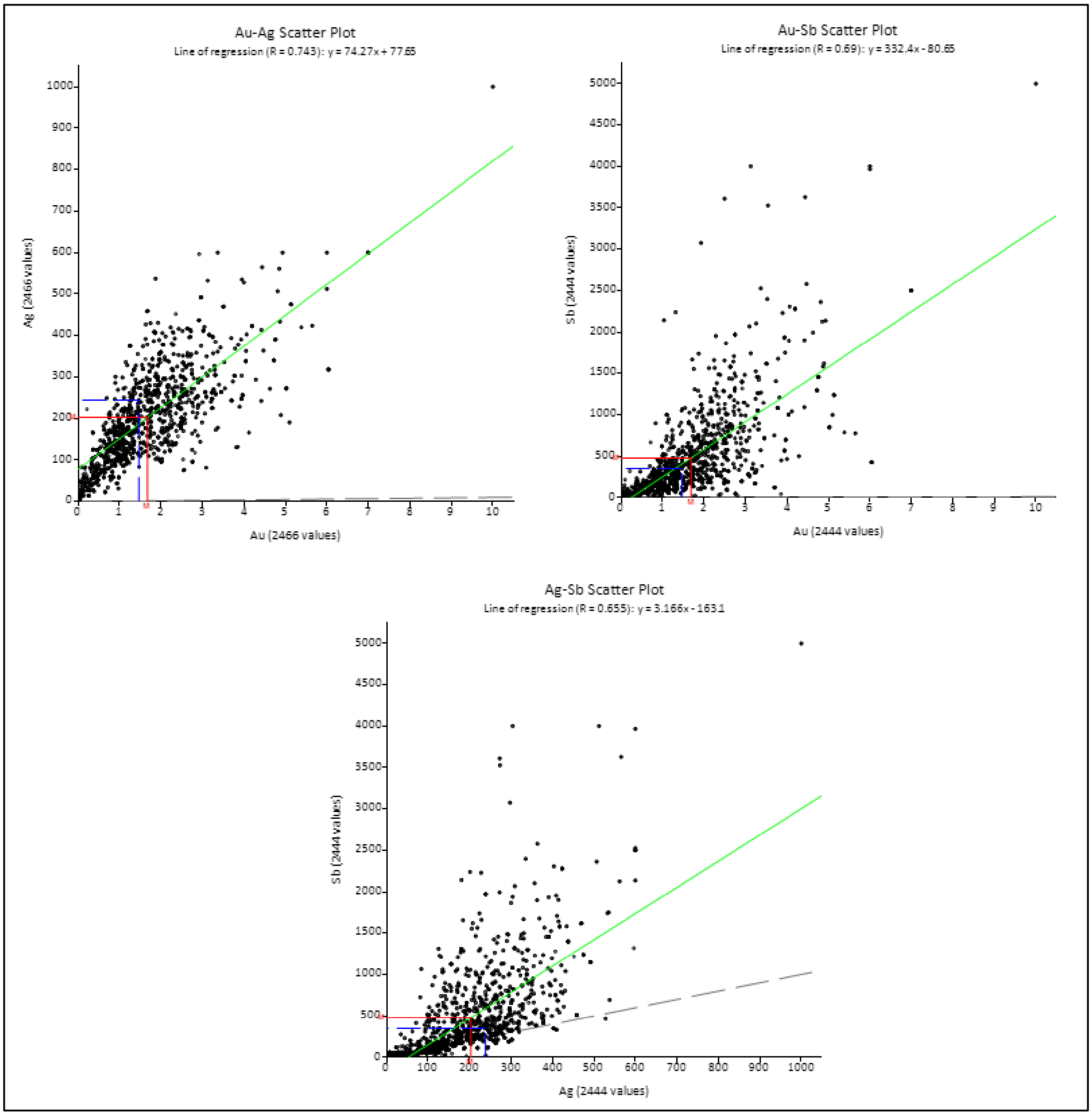


Figure 14-9: Krakatoa Zone – scattergram correlation plots for cut variables

14.12 Treatment of Outliers (Top Cuts)

A review of grade outliers was undertaken to ensure that extreme grades are treated appropriately during grade interpolation. Although extreme grade outliers within the grade populations of variables are real, they are potentially not representative of the volume they inform during estimation. If these values are not cut, they have the potential to result in significant grade over-estimation on a local basis.

Top cuts were selected following statistical review of the sample population. The cutting strategy was applied following review of the following:

- Skewness of the data
- Probability plots
- Spatial position of extreme grades.

To determine the top cuts, histograms and probability plots were reviewed for the major elements (Cu, Pb, Zn, Au, Ag and Fe) and minor elements (As, Ba, Bi, Hg, S, Sb and Se). The plots were compiled based on the 1.5 m and 1.0 m composites for each mineralized zone (POD) for the ABM and Krakatoa zones respectively (some examples are provided in Figure 14-10 to Figure 14-12).

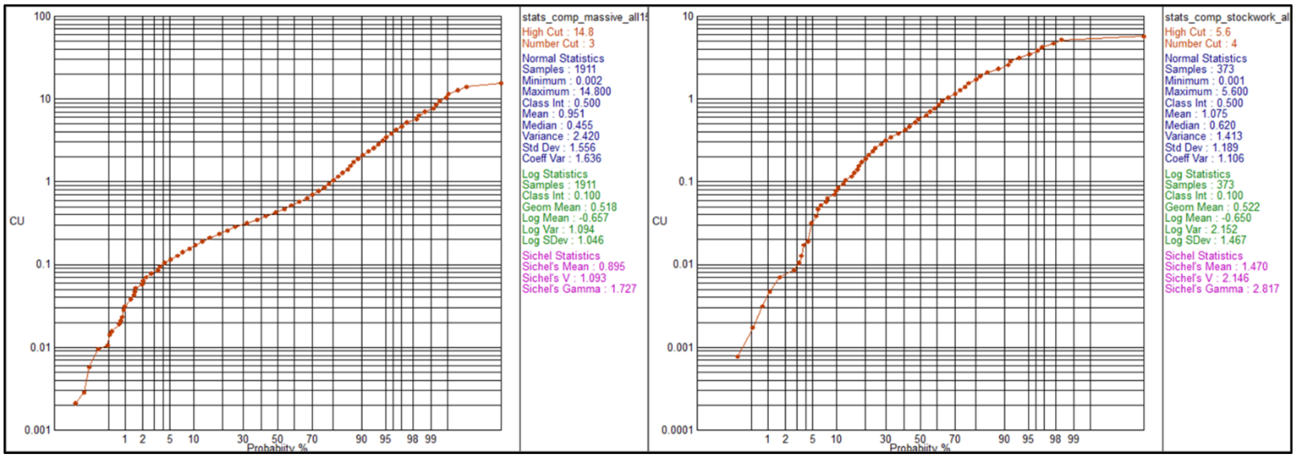


Figure 14-10: ABM Zone – log probability plots for massive and stockwork Cu

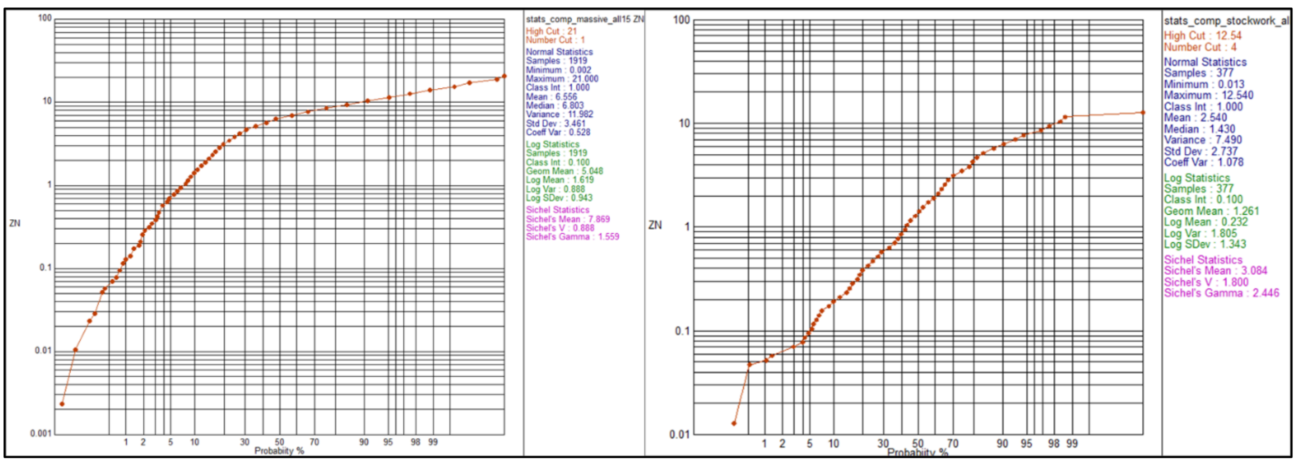


Figure 14-11: ABM Zone – log probability plots for massive and stockwork Zn

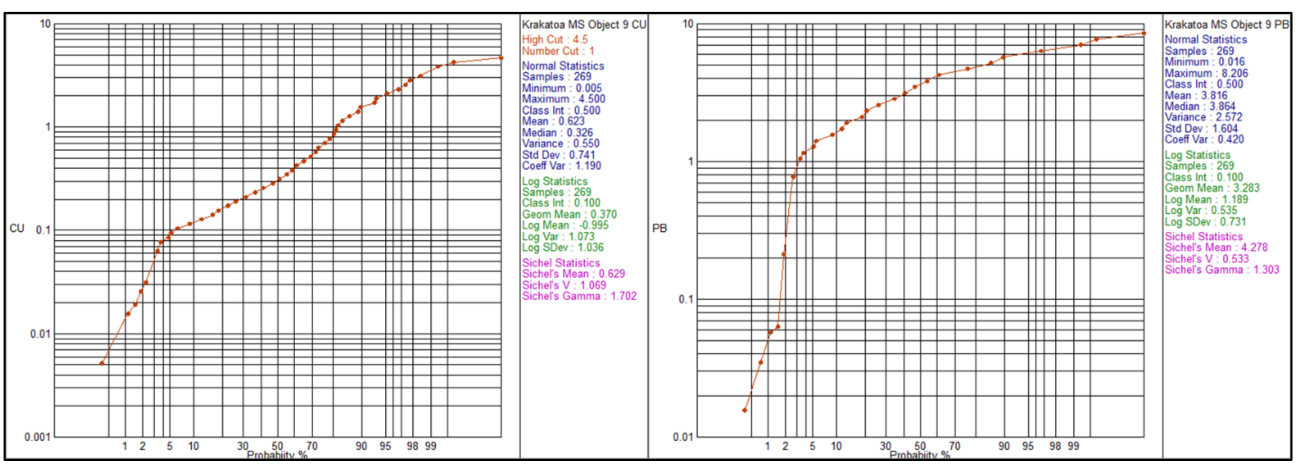


Figure 14-12: ABM Zone – log probability plots for massive Cu (left) and Pb (right)

Table 14-10 and Table 14-11 show the top cuts applied to each POD of the ABM and Krakatoa zones for all variables. Where no top cut is specified, none was applied. All samples that were greater than the top-cut value were reset to the top-cut value. The top-cut grades were applied to the composited samples.

Table 14-10: Top cuts for the ABM Zone per POD

POD	Style	Cu %	Pb %	Zn %	Au g/t	Ag g/t	Fe %	As ppm	Ba ppm	Bi ppm	Hg ppm	S %	Sb ppm	Se ppm
4	STW	5.6	5.00	12.54	-	370	-	6,000	-	-	-	-	-	-
8	MS	14.8	7.52	21.00	7.75	-	-	25,000	150,000	350	-	-	-	-
13	STW	5.6	5.00	12.54	-	370	-	-	60,000	-	-	-	100	-
17	MS	14.8	7.52	21.00	7.75	-	-	4,000	20,000	-	-	-	1,000	-
23	STW	5.6	5.00	12.54	-	370	-	-	-	-	-	-	-	-
27	MS	14.8	7.52	21.00	7.75	-	-	-	-	-	-	-	-	-
33	STW	5.6	5.00	12.54	-	370	-	1,000	-	-	-	-	30	-
37	MS	14.8	7.52	21.00	7.75	-	-	-	50,000	-	-	-	300	-
43	STW	5.6	5.00	12.54	-	370	-	-	-	-	-	-	-	-
47	MS	14.8	7.52	21.00	7.75	-	-	-	-	-	-	-	-	-
53	STW	5.6	5.00	12.54	-	370	-	-	-	-	-	-	350	-
57	MS	14.8	7.52	21.00	7.75	-	-	-	8,000	-	-	-	400	-
63	STW	5.6	5.00	12.54	-	370	-	-	-	-	-	-	-	-
67	MS	14.8	7.52	21.00	7.75	-	-	-	-	-	-	-	-	-
73	STW	5.6	5.00	12.54	-	370	-	-	-	-	-	-	-	-
83	STW	5.6	5.00	12.54	-	370	-	700	25,000	-	10	-	100	400
93	STW	5.6	5.00	12.54	-	370	-	-	-	-	-	-	-	-
103	STW	5.6	5.00	12.54	-	370	-	2,000	-	-	20	-	-	-
113	STW	5.6	5.00	12.54	-	370	-	-	65,000	-	-	-	-	-
123	STW	5.6	5.00	12.54	-	370	-	1,000	-	-	-	-	100	-
133	STW	5.6	5.00	12.54	-	370	-	-	-	-	-	-	-	-
143	STW	5.6	5.00	12.54	-	370	-	-	-	-	-	-	-	-
153	STW	5.6	5.00	12.54	-	370	-	4,000	-	150	-	-	-	-

Note: Where no top cut is defined, none was applied. STW – Stockwork mineralization, MS – Massive sulphide.

Table 14-11: Top cuts for the Krakatoa Zone per POD

POD	Style	Cu %	Pb %	Zn %	Au g/t	Ag g/t	Fe %	As ppm	Ba ppm	Bi ppm	Hg ppm	S %	Sb ppm	Se ppm
202	MS	8	-	12	-	600	-	6,000	25,000	-	-	100	5,000	-
208	MS	4.5	-	-	6	600	-	22,000	28,000	300	85	-	4,000	500
209	STW	1.8	-	-	7	600	-	25,000	90,000	-	-	-	-	1,500
217	MS	4.5	-	-	6	600	-	22,000	28,000	300	85	-	4,000	500
218	STW	1.8	-	-	7	600	-	25,000	90,000	-	-	-	-	1,500
228	STW	1.8	-	-	7	600	-	25,000	90,000	-	-	-	-	1,500
238	STW	1.8	-	-	7	600	-	25,000	90,000	-	-	-	-	1,500
248	STW	1.8	-	-	7	600	-	25,000	90,000	-	-	-	-	1,500

Note: Where no top cut is defined, none was applied. STW – Stockwork mineralization, MS – Massive sulphide.

14.13 Geostatistical Analysis – Kriging Neighbourhood Analysis

KNA was completed using Supervisor v8.4™ software, adopting the relevant variogram models for the estimation domains. KNA was completed for each of the variables, based on the combined massive sulphide dataset of the ABM Zone and the updated massive sulphide dataset of the Krakatoa Zone, respectively. Variography was attempted on stockwork mineralization; however, the data was patchy and structures poor. As such, conceptual variogram parameters based on the massive sulphide results were modelled to best reflect stockwork geometry.

The following was reviewed for each of the variables per selected domain:

- Slope of regression and kriging efficiency statistics for a well-informed block for different block sizes.
- On choosing a block size (10 m(E) x 10 m(N) x 5 m(RL), optimum minimum and maximum samples were chosen. The maximum was set at the lowest number of samples from which consistently good slope of regression and kriging effiency could be achieved. The minimum was defined as the lowest minimum from which moderate to good statistics could be derived.
- On choosing the minimum/maximum samples, search ellipse ranges were defined. The quality of the statistics was least sensitive to this parameter. The ranges chosen for Pass 1 approximated two-thirds of the range of the first structure of the variogram. For Pass 2, the ranges equated to the full range in the variogram model for the major direction.
- Negative weights were reviewed at each stage to ensure the parameters chosen were not leading to excessive negative weights (sample redundancy).
- Discretization was defined at 5 x 5 x 3 (X x Y x Z).
- Maximum number of samples allowed per each individual drillhole, per estimate, was set to three.

The KNA results show that the search parameters and block sizes selected are suitable for use in the MRE and adequately take drill spacing, geology and practicality into account.

The number of composites used for the major and potential deleterious element grade estimations in the ABM and Krakatoa zones is presented in Table 14-12 and Table 14-13 respectively.

The modelled variogram parameters together with the selected estimation panel size and number of samples was used to determine the appropriate search ellipses for the primary search pass. These are also presented in Table 14-12 and Table 14-13.

Table 14-12: Search neighbourhood parameters for the major elements for ABM and Krakatoa zones

Zone	Element	Rotation (degrees, Surpac XZY Convention)			Search range 1 (m) (SVOL1)			Search range 2 (m) (SVOL2)			Search range 3 (m) (SVOL3)			Number of Composites	
		Bearing	Plunge	Dip	Major	Semi	Minor	Major	Semi	Minor	Major	Semi	Minor	Min.	Max.
ABM	Cu cut	10	-35	0	115	1.28	11.50	175	1.35	17.50	350	1.35	17.50	6	24
	Pb cut	10	-30	0	100	1.33	10.00	150	1.36	10.00	300	1.36	10.00	6	24
	Zn cut	20	-30	0	85	1.42	8.50	130	1.53	8.67	260	1.53	8.67	6	21
	Au cut	20	-30	0	150	1.43	15.00	225	1.45	15.00	450	1.45	15.00	6	24
	Ag cut	0	-30	0	125	1.67	8.33	185	1.68	9.25	370	1.68	9.25	6	27
	Fe cut	0	-30	0	55	0.61	3.67	80	0.62	4.00	160	0.62	4.00	6	21
Krakatoa	Cu cut	12.24	8.74	1.2	52.14	7.2	1	79	7.2	1	158	7.2	1	6	20
	Pb cut	-9.6	39.02	1.56	51.48	3.9	1	78	3.9	1	156	3.9	1	4	24
	Zn cut	-42.15	39.32	1.79	186.78	16.65	2	283	16.65	2	566	16.65	2	4	26
	Au cut	-11.31	-33.34	1.29	71.94	7.78	1	109	7.78	1	218	7.78	1	4	24
	Ag cut	-45.19	44.81	3.5	85.14	1.74	1	129	1.74	1	258	1.74	1	4	24
	Fe cut	-15.19	-13.17	1.12	101.64	5.5	2	154	5.5	2	308	5.5	2	4	24

Table 14-13: Search neighbourhood parameters for the deleterious elements for ABM and Krakatoa Zones

Zone	Element	Rotation (degrees, Surpac XZY Convention)			Search range 1 (m) (SVOL1)			Search range 2 (m) (SVOL2)			Search range 3 (m) (SVOL3)			Number of Composites	
		Bearing	Plunge	Dip	Major	Semi	Minor	Major	Semi	Minor	Major	Semi	Minor	Min.	Max.
ABM	As cut	90	0	30	145	1.45	14.5	220	1.45	14.5	435	1.45	14.5	6	21
	Ba cut	90	0	30	90	1.125	9	140	1.125	9	270	1.125	9	6	21
	Bi cut	90	0	30	85	1.545	8.500	125	1.545	8.500	255	1.545	8.500	6	21
	Hg cut	100	0	30	100	1.053	10.000	155	1.053	10.000	300	1.053	10.000	6	21
	S cut	90	0	30	60	0.522	6.000	85	0.522	6.000	180	0.522	6.000	6	21
	Sb cut	90	0	30	130	1.300	13.000	195	1.300	13.000	390	1.300	13.000	6	21
	Se cut	90	0	30	95	1.267	9.500	145	1.267	9.500	250	1.267	9.500	6	21
Krakatoa	As cut	320.36	-22.5	-45.9	72.6	1.2	5.78	110	1.2	5.78	220	1.2	5.78	6	26
	Ba cut	165	0	30	36.96	0.49	2.94	56	0.49	2.94	112	0.49	2.94	4	24
	Bi cut	302.3	24.4	-32.73	61.38	2.44	4.04	93	2.44	4.04	186	2.44	4.04	4	22
	Hg cut	117.5	21.47	13.12	39.6	1	4	60	1	4	120	1	4	4	24
	S cut	129.96	0.87	4.92	22.57	1.26	2.85	34.2	1.26	2.85	68.4	1.26	2.85	4	20
	Sb cut	339.02	-44.14	9.85	49.5	2.21	2.59	75	2.21	2.59	150	2.21	2.59	4	26
	Se cut	275.77	13.57	-6.46	42.9	1	4.33	65	1	4.33	130	1	4.33	4	24

The search ranges in the ABM Zone are larger than that of the Krakatoa Zone. This is a reflection of the continuity shown in the data analysis and variography. Smoothing of grades in the areas of closely spaced drillhole data will be reduced by limiting the maximum number of samples used in the estimate.

Initial estimation runs indicated that for some of the smaller domains, the minimum number of samples was required to be reduced in order to adequately perform the estimation. In addition, search volumes for the third estimation pass were also increased in some cases to allow the estimation of all blocks in some of the smaller domains.

14.14 Block Modelling

A Surpac block model was created to encompass the full extent of the ABM deposit. A list of block model parameters is displayed in Table 14-14 and a list of block model attributes is displayed in Table 14-15.

The block model used a parent cell size of 10 m(E) x 10 m(N) x 5 m(RL) with standard sub-celling to 5 m(E) x 5 m(N) x 2.5 m(RL) to maintain the resolution of the mineralized lenses. The northing parent cell size was selected based on approximately half of the average drill section spacing in better drilled areas of the deposit. The model cell dimensions in other directions were selected to provide sufficient resolution to the block model in the across-strike and down-dip directions.

Table 14-14: Block model parameters – ABM deposit

Axis	Extent (m)		Block size (m)	Maximum sub-celling (m)
	Minimum	Maximum		
Easting	414,200	415,700	10	5
Northing	6,814,600	6,816,000	10	5
RL	1,000	1,700	5	2.5

Table 14-15: Block model attributes – ABM deposit

Attribute	Description
cu_uncut	Uncut Cu (copper) grade in percent (%)
pb_uncut	Uncut Pb (lead) grade in percent (%)
zn_uncut	Uncut Zn (zinc) grade in percent (%)
au_uncut	Uncut Au (gold) grade in parts per million (ppm)
ag_uncut	Uncut Ag (silver) grade in parts per million (ppm)
fe_uncut	Uncut Fe (iron) grade in percent (%)
cu_cut	Cut Cu grade in percent (%)
pb_cut	Cut Pb grade in percent (%)
zn_cut	Cut Zn grade in percent (%)
au_cut	Cut Au grade in parts per million (ppm)
ag_cut	Cut Ag grade in parts per million (ppm)
fe_cut	Cut Fe grade in percent (%)
class	measured, indicated, inferred, unclassified
class_code	1=measured, 2=indicated, 3=inferred, 4=unclassified
lithology	mafic, felsic, massive sulphide, air
type	fresh, overburden, air
pod	Wireframe object number
bd	bulk density in t/m^3
bdpass	Bulk Density estimation pass
min_dis_cu_uncut	Minimum Distance Cu
ave_dis_cu_uncut	Average Distance Cu
num_sam_cu_uncut	Number of Informing Samples Cu
bv_cu_uncut	Block Variance Cu
ke_cu_uncut	Kriging Efficiency Cu
kv_cu_uncut	Kriging Variance Cu
lag_cu_uncut	Lagrange Multiplier Cu
slope_cu_uncut	Slope of Regression Cu
negwt_cu_uncut	Sum of Negative Weights Cu
min_dis_pb_uncut	Minimum Distance Pb
ave_dis_pb_uncut	Average Distance Pb
num_sam_pb_uncut	Number of Informing Samples Pb
bv_pb_uncut	Block Variance Pb
ke_pb_uncut	Kriging Efficiency Pb
kv_pb_uncut	Kriging Variance Pb
lag_pb_uncut	Lagrange Multiplier Pb
slope_pb_uncut	Slope of Regression Pb
negwt_pb_uncut	Sum of Negative Weights Pb
min_dis_zn_uncut	Minimum Distance Zn
ave_dis_zn_uncut	Average Distance Zn
num_sam_zn_uncut	Number of Informing Samples Zn
bv_zn_uncut	Block Variance Zn
ke_zn_uncut	Kriging Efficiency Zn
kv_zn_uncut	Kriging Variance Zn
lag_zn_uncut	Lagrange Multiplier Zn

Attribute	Description
slope_zn_uncut	Slope of Regression Zn
negwt_zn_uncut	Sum of Negative Weights Zn
min_dis_au_uncut	Minimum Distance Au
ave_dis_au_uncut	Average Distance Au
num_sam_au_uncut	Number of Informing Samples Au
bv_au_uncut	Block Variance Au
ke_au_uncut	Kriging Efficiency Au
kv_au_uncut	Kriging Variance Au
lag_au_uncut	Lagrange Multiplier Au
slope_au_uncut	Slope of Regression Au
negwt_au_uncut	Sum of Negative Weights Au
min_dis_ag_uncut	Minimum Distance Ag
ave_dis_ag_uncut	Average Distance Ag
num_sam_ag_uncut	Number of Informing Samples Ag
bv_ag_uncut	Block Variance Ag
ke_ag_uncut	Kriging Efficiency Ag
kv_ag_uncut	Kriging Variance Ag
lag_ag_uncut	Lagrange Multiplier Ag
slope_ag_uncut	Slope of Regression Ag
negwt_ag_uncut	Sum of Negative Weights Ag
min_dis_fe_uncut	Minimum Distance Fe
ave_dis_fe_uncut	Average Distance Fe
num_sam_fe_uncut	Number of Informing Samples Fe
bv_fe_uncut	Block Variance Fe
ke_fe_uncut	Kriging Efficiency Fe
kv_fe_uncut	Kriging Variance Fe
lag_fe_uncut	Lagrange Multiplier Fe
slope_fe_uncut	Slope of Regression Fe
negwt_fe_uncut	Sum of Negative Weights Fe
min_dis_cu_cut	Minimum Distance Cu
ave_dis_cu_cut	Average Distance Cu
num_sam_cu_cut	Number of Informing Samples Cu
bv_cu_cut	Block Variance Cu
ke_cu_cut	Kriging Efficiency Cu
kv_cu_cut	Kriging Variance Cu
lag_cu_cut	Lagrange Multiplier Cu
slope_cu_cut	Slope of Regression Cu
negwt_cu_cut	Sum of Negative Weights Cu
min_dis_pb_cut	Minimum Distance Pb
ave_dis_pb_cut	Average Distance Pb
num_sam_pb_cut	Number of Informing Samples Pb
bv_pb_cut	Block Variance Pb
ke_pb_cut	Kriging Efficiency Pb
kv_pb_cut	Kriging Variance Pb
lag_pb_cut	Lagrange Multiplier Pb

Attribute	Description
slope_pb_cut	Slope of Regression Pb
negwt_pb_cut	Sum of Negative Weights Pb
min_dis_zn_cut	Minimum Distance Zn
ave_dis_zn_cut	Average Distance Zn
num_sam_zn_cut	Number of Informing Samples Zn
bv_zn_cut	Block Variance Zn
ke_zn_cut	Kriging Efficiency Zn
kv_zn_cut	Kriging Variance Zn
lag_zn_cut	Lagrange Multiplier Zn
slope_pb_cut	Slope of Regression Zn
negwt_pb_cut	Sum of Negative Weights Zn
min_dis_au_cut	Minimum Distance Au
ave_dis_au_cut	Average Distance Au
num_sam_au_cut	Number of Informing Samples Au
bv_au_cut	Block Variance Au
ke_au_cut	Kriging Efficiency Au
kv_au_cut	Kriging Variance Au
lag_au_cut	Lagrange Multiplier Au
slope_au_cut	Slope of Regression Au
negwt_au_cut	Sum of Negative Weights Au
min_dis_ag_cut	Minimum Distance Ag
ave_dis_ag_cut	Average Distance Ag
num_sam_ag_cut	Number of Informing Samples Ag
bv_ag_cut	Block Variance Ag
ke_ag_cut	Kriging Efficiency Ag
kv_ag_cut	Kriging Variance Ag
lag_ag_cut	Lagrange Multiplier Ag
slope_ag_cut	Slope of Regression Ag
negwt_ag_cut	Sum of Negative Weights Ag
min_dis_fe_cut	Minimum Distance Fe
ave_dis_fe_cut	Average Distance Fe
num_sam_fe_cut	Number of Informing Samples Fe
bv_fe_cut	Block Variance Fe
ke_fe_cut	Kriging Efficiency Fe
kv_fe_cut	Kriging Variance Fe
lag_fe_cut	Lagrange Multiplier Fe
slope_fe_cut	Slope of Regression Fe
negwt_fe_cut	Sum of Negative Weights Fe
pass	Estimation pass
ard	Acid Rock Drainage domains
zone	Waste, ABM, or Krakatoa
as_uncut	Uncut As (arsenic) grade in parts per million (ppm)
ba_uncut	Uncut Ba (barium) grade in parts per million (ppm)
bi_uncut	Uncut Bi (bismuth) grade in parts per million (ppm)
hg_uncut	Uncut Hg (mercury) grade in parts per million (ppm)

Attribute	Description
s_uncut	Uncut S (sulphur) grade in percent (%)
sb_uncut	Uncut Sb (antimony) grade in parts per million (ppm)
se_uncut	Uncut Se (selenium) grade in parts per million (ppm)
as_cut	Cut As grade in parts per million (ppm)
ba_cut	Cut Ba grade in parts per million (ppm)
bi_cut	Cut Bi grade in parts per million (ppm)
hg_cut	Cut Hg grade in parts per million (ppm)
s_cut	Cut S grade in percent (%)
sb_cut	Cut Sb grade in parts per million (ppm)
se_cut	Cut Se grade in parts per million (ppm)

A comparison of the wireframe volumes to the block model volume for each of the resource zones is shown in Table 14-16 below. The difference between the wireframe volumes and the block model volumes is within the margin of error in 29 of the 32 domains (approximately 98% of the total volume) and demonstrates that the resolution of the block model sub-celling is satisfactory.

Table 14-16: Volume comparison between mineralization wireframes and block model pods – ABM and Krakatoa zones

Zone	POD	Wireframe volume	Block model volume	Difference (%)
ABM	4	149,750	148,563	99%
	8	2,868,565	2,846,688	99%
	13	65,575	64,188	98%
	17	6,771	6,750	100%
	23	16,824	17,063	101%
	27	8,347	7,750	93%
	33	59,574	59,250	99%
	37	82,136	82,125	100%
	43	15,547	15,563	100%
	47	34,806	35,063	101%
	53	37,912	38,188	101%
	57	25,760	25,688	100%
	63	42,169	44,125	105%
	67	3,913	4,063	104%
	73	18,980	19,500	103%
	83	125,652	126,250	100%
	93	6,012	6,188	103%
	103	21,005	20,938	100%
	113	19,251	20,313	106%
	123	26,080	25,938	99%
	133	4,278	4,313	101%
	143	14,826	14,625	99%
	153	85,466	85,813	100%
	Subtotal	**3,739,199**	**3,718,945**	**99%**

Zone	POD	Wireframe volume	Block model volume	Difference (%)
Krakatoa	202	82,805	84,250	102%
	217	91,590	90,813	99%
	218	150,364	150,063	100%
	208	610,402	608,750	100%
	209	34,130	34,250	100%
	228	59,671	61,438	103%
	238	6,081	6,063	100%
	248	9,261	9,188	99%
	258	8,677	8,563	99%
	Subtotal	**1,052,981**	**1,053,378**	**100%**
TOTAL		**4,792,180**	**4,772,373**	**100%**

14.15 Grade Interpolation

For all except five of the mineralized zones in the ABM deposit, the wireframe objects were used as hard boundaries in grade interpolation. That is, only grades inside each wireframe object were used to interpolate the blocks inside the object. This process reflects field observations around the mineralization contacts. For the other five mineralized zones (objects 17, 27, 43, 67, 93), semi-soft boundaries were introduced, whereby samples from a neighbouring large domain were used in the estimation of the smaller domain, but not vice-versa. Semi-soft boundaries were required for these five zones due to lack of supporting data to reliably inform the estimate for each domain. All mineralized zones at the Krakatoa deposit were estimated using hard boundaries.

OK was selected for grade interpolation in the mineralized zones, whilst ID3 was used in the estimation of the dilution skin. OK was selected to allow a degree of smoothing within the model based on the measured variability from the variograms. It is considered by the Qualified Person to be appropriate for this style of deposit. ID3 was chosen over ID2 for the dilution skin as it further restricted the influence of individual high-grade samples, approximating a nearest neighbour approach.

An orientated “ellipsoid” search was used to select data for interpolation. An “anisotropic in the plane” ellipse (different major and semi-major distances) was oriented according to the rotations derived from the variography. Estimation parameters at ABM were calculated using all data, as the domains at ABM are deemed to be similar enough to be treated as a single domain for statistical purposes. In the Krakatoa area, this was thought not to be the case, and estimation parameters were calculated using the data from the largest domain only, which was subsequently applied to all other domains.

A three-pass estimation search was used to complete estimation for Cu, Pb, Zn, Au, Ag, Fe, As, Ba, Bi, Hg, S, Sb and Se. Approximately 99% of the blocks were informed in the first two estimation passes for the mineralization estimate. A third expanded estimation pass was used to inform remaining un-estimated blocks.

14.16 Bulk Density Assignment

For the mineralized material, including that which falls within the dilution skin, a combination of methods was utilized to assign bulk density. A clean dataset of bulk density values was constructed based on a hierarchy of confidence, for interpolation into the block model. The bulk density value for the ABM Zone was determined according to the following priorities:

1. Clean measured bulk density value was used where available.

2. Pycnometer specific gravity was used where there was no measured bulk density. This only accounts for two samples in the dataset.
3. Bulk densities were calculated using multiple regression, using sulphur data where available, optimized for the highest coefficient of determination.
4. Where S data were absent, bulk densities were calculated using weighted Fe-Cu-Pb-Zn data with the simple exponential regression (1.0*Cu%)+(1.81*Pb%)+(0.97*Zn%)+(1.20*Fe%). This was completed for the cleaned bulk densities for zone 8 and for samples having a bulk density <2.75 g/cm^3 in zones 5, 6 and 7.

For Krakatoa data, measured bulk densities were available for all samples within the mineralization wireframes.

Bulk density was estimated into the block model using OK for the mineralized zones, and ID3 for the dilution skin. Estimation parameters were duplicated from those used in the estimation of Fe; this ensured that the relationship between bulk density and Fe was maintained.

Approximately 96% of the blocks were informed in the first two estimation passes for the bulk density estimate.

The swath plots shown in Figure 14-13 and Figure 14-14 demonstrate that the estimated bulk densities in the model correspond well with the input samples for the main zone at ABM Zone and Krakatoa Zone respectively.

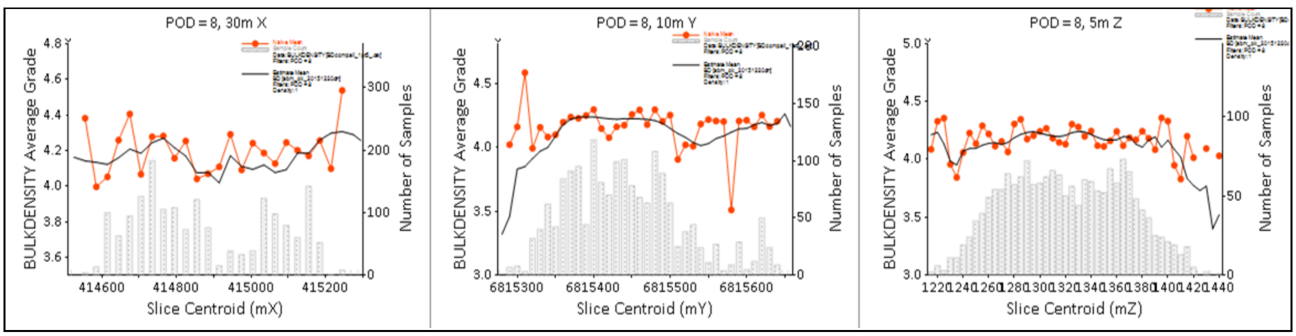


Figure 14-13: Swath plot by 30 m easting, 10 m northing, and 5 m bench, for main zone at ABM (pod 8) – bulk density

Note: The drop off in estimated bulk density in relation to sample values seen south of 6,815,350 mN and above 1,420 mRL is related to assigned bulk densities in the overburden portion of Pod 8, which do not have corresponding bulk density samples.

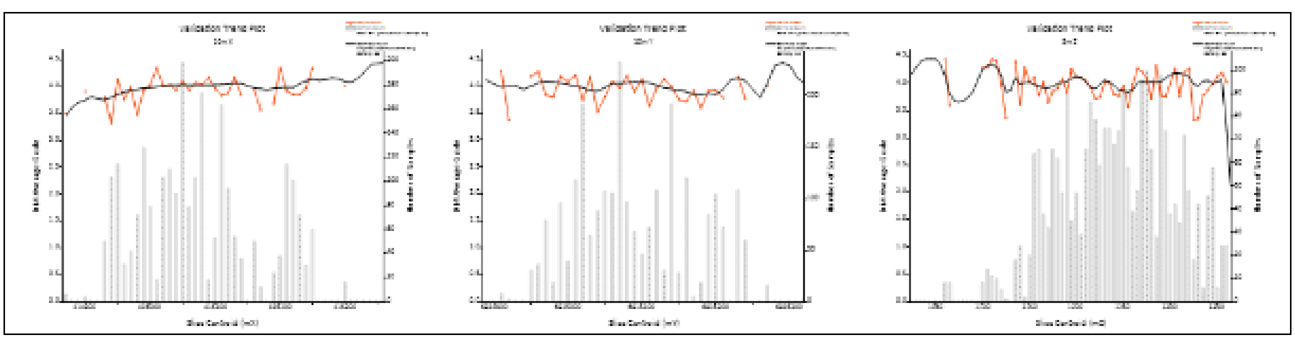


Figure 14-14: Swath plot by 30 m easting, 10 m northing, and 5 m bench, for the Krakatoa Zone bulk density

The bulk density values used for the ABM MRE are displayed in Table 14-17.

Table 14-17: Bulk density values applied to the ABM MRE

Material type	Bulk density (t/m^3)	Description
Air	0.00	Above topographic surface
Overburden	2.00	Topographic surface to base of overburden
Felsic volcanics	2.76	Assigned directly to host rock based on measured average
Mafic intrusive (MAFi)	2.80	Assigned directly to mafic wireframes and based on measured average
Rhyolite intrusive (RHYi)	2.68	Assigned directly to RHYi wireframes and based on measured average
Carbonaceous mudstone (MDS)	2.74	Assigned directly to mudstone wireframes and based on measured average
Wind Lake formation	2.74	Assigned directly to Wind Lake formation wireframes and based on measured average
ABM – Stockwork	3.44	Estimated, mean value
Krakatoa – Stockwork	3.86	Estimated, mean value
ABM – Massive Sulphide	4.19	Estimated, mean value
Krakatoa – Massive Sulphide	4.09	Estimated, mean value

14.17 Mineral Resource Classification

The resource estimate is prepared in accordance with CIM Definition Standards – for Mineral Resources and Mineral Reserves (CIM Council, 2014), adopted by the CIM Council on 10 May 2014 where:

An Inferred Mineral Resource as defined by the CIM Standing Committee is *“that part of a Mineral Resource for which quantity and grade or quality are estimated on the basis of limited geological evidence and sampling. Geological evidence is sufficient to imply but not verify geological and grade or quality continuity*.

An Inferred Mineral Resource has a lower level of confidence than that applying to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.”

An Indicated Mineral Resource has a higher level of confidence than that applying to an Inferred Mineral Resource. It may be converted to a Probable Mineral Reserve. An Indicated Mineral Resource as defined by the CIM Standing Committee is *“that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics are estimated with sufficient confidence to allow the application of Modifying Factors in sufficient detail to support mine planning and evaluation of the economic viability of the deposit.*

Geological evidence is derived from adequately detailed and reliable exploration, sampling and testing and is sufficient to assume geological and grade or quality continuity between points of observation. An Indicated Mineral Resource has a lower level of confidence than that applying to a Measured Mineral Resource and may only be converted to a Probable Mineral Reserve.” and,

A Measured Mineral Resource has a higher level of confidence than that applying to either an Indicated Mineral Resource or an Inferred Mineral Resource. It may be converted to a Proven Mineral Reserve or to a Probable Mineral Reserve. A Measured Mineral Resource, as defined by the CIM Standing Committee is *“that part of a Mineral Resource for which quantity, grade or quality, densities, shape, and physical characteristics are estimated with confidence sufficient to allow the application of Modifying Factors to support detailed mine planning and final evaluation of the economic viability of the deposit.*

Geological evidence is derived from detailed and reliable exploration, sampling and testing and is sufficient to confirm geological and grade or quality continuity between points of observation.

A Measured Mineral Resource has a higher level of confidence than that applying to either an Indicated Mineral Resource or an Inferred Mineral Resource. It may be converted to a Proven Mineral Reserve or to a Probable Mineral Reserve."

Mineral Resources that are not Mineral Reserves do not account for mineability, selectivity, mining loss and dilution and do not have demonstrated economic viability. These MREs include Inferred Mineral Resources that are normally considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as Mineral Reserves. 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.

Classification, or assigning a level of confidence to Mineral Resources, is undertaken in strict adherence to the CIM Definition Standards for Mineral Resources and Mineral Reserves (CIM Council, 2014). The ABM MRE was prepared by, or under the supervision of Aaron Green, CSA Global Principal Resource Geologist and Qualified Person for the reporting of Mineral Resources as defined by NI 43-101.

14.17.1 Reasonable Prospects for Economic Extraction

CIM Definition Standards for Mineral Resources and Mineral Reserves, adopted by the CIM Council on 10 May 2014 require that resources have "reasonable prospects for economic extraction". This generally implies that the quantity and grade estimates meet certain economic thresholds and that the mineral resources are reported at an appropriate cut-off grade taking into account possible extraction scenarios and processing recoveries.

To assist in defining reasonable prospects of economic extraction the in-ground value of each block was calculated using estimated factors for: assumed metal prices, metallurgical recoveries, smelter terms (including payable factors, concentrate costs and refining charges) and government royalties. These factors were provided by BMC. No penalties were included. Key factors determining the NSR were:

- Metal price assumptions were: US$3.50/lb copper, US$1.50/lb zinc, US$1.05/lb lead, US$1,300/oz gold and US$20/oz silver.
- An exchange rate of US$0.75 = CAD$1.00.
- Three separate concentrates recovered – copper, lead and zinc with precious metals (gold and silver) reporting to all concentrates at varying recoveries from 15% to 40%.
- Metal recovery assumptions were: 92% for copper, 90% for zinc, 70% for lead, 75% for gold (whereby 30% is recovered into copper concentrate, 30% is recovered into lead concentrate and 15% is recovered into zinc concentrate) and 85% for silver (40% into copper concentrate, 30% into lead concentrate and 15% into zinc concentrate).

Based on these assumptions the formula for the NSR on each block was calculated as:

$$NSR\ US\$/t = (52.84*Cu\_cut) + (9.56*Pb\_cut) + (19.13*Zn\_cut) + (24.41*Au\_cut) + (0.41*Ag\_cut)$$

The US dollar NSR was then converted to Canadian dollars:

$$NSR\ CAD\$/t = (NSR\ US\$/t)/0.75$$

Based on the results of the 2017 Mineral Reserve estimate prepared for the PFS (CSA Global, 2017), potential open pittable resources were reported above a cut-off NSR of CAD$25/t and potential underground resources reported above CAD$95/t.

To determine the reporting of ABM deposit Mineral Resources as either open pit or underground, a Whittle™ pit optimization was undertaken. Parameters used for the optimization included:

- Base case metal price assumptions were: US$3.50/lb copper, US$1.50/lb zinc, US$1.05/lb lead, US$1,300/oz gold and US$20/oz silver
- An exchange rate of US$0.75 = CAD$1.00
- Mining recovery of 97%
- Minimum mining width of 25 m
- Overall slope angle of 50°
- Total processing costs (fresh) of CAD$30.60/t
- Plant throughput of 2 Mt/a.

For the ABM Zone, only material reporting inside the selected pit shell (Revenue Factor = 1.00) has been reported above the NSR cut-off of CAD$25/t. For the Krakatoa Zone, mineralized material inside the pit shell has been reported above the NSR cut-off of CAD$25/t, whilst the remainder has been designated as "underground" resource and reported above a cut-off NSR of CAD$95/t (Figure 14-15).

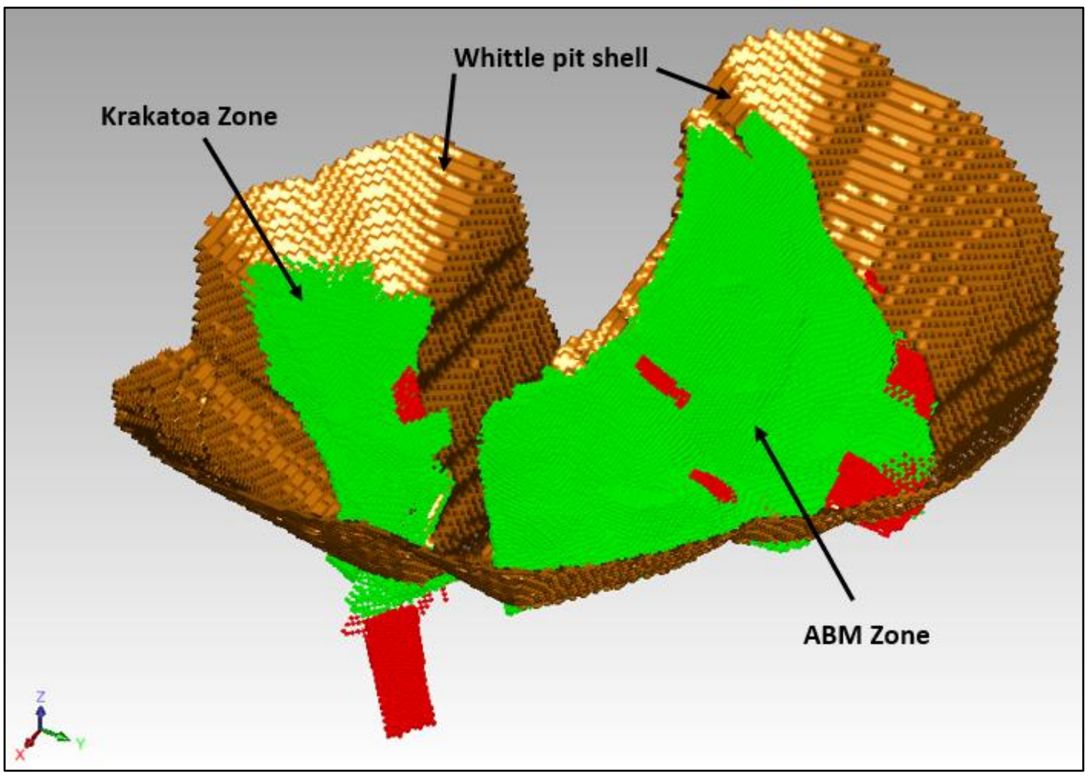


Figure 14-15: ABM deposit Mineral Resource classification inside optimized pit shell looking southwest (red = Inferred, green = Indicated)

14.17.2 Resource Classification Parameters

The ABM deposit (ABM and Krakatoa zones) MRE is classified in accordance with CIM Definition Standards for Mineral Resources and Mineral Reserves, adopted by the CIM Council on 10 May 2014. The classification level is based upon an assessment of geological understanding of the deposit, geological and grade continuity, drillhole spacing, QC results, search and interpolation parameters, and an analysis of available density information.

The ABM deposit shows excellent continuity of mineralization within well-defined geological constraints. Drillholes are located at a nominal spacing of 50 m on 25 m north-south oriented sections extending out to 100 m on the peripheries of the deposit. The drill spacing is sufficient to allow the geology and mineralization zones to be modelled into coherent wireframes for each domain. Reasonable consistency is evident in the orientations, thickness and grades of the mineralized zones.

The 2015 BMC exploration program included re-drilling of several sections within the ABM Zone, "twinning" historical holes, relogging and resampling of historical core, and re-surveying historical drill collars. This work validated the historical work undertaken by Cominco and improved the confidence level in the historical data and, along with the additional infill drilling, has largely confirmed the continuity of the geology and known mineralization.

The Mineral Resource is classified as Indicated where, in the Qualified Person's opinion, sufficient data exists to assume geological and mineralization continuity. For areas with more limited data density and limited along-strike or down-dip continuity, there is sufficient evidence to imply but not verify geological and grade continuity and these areas are classified as Inferred.

The resource classification strategy is illustrated in Figure 14-16.

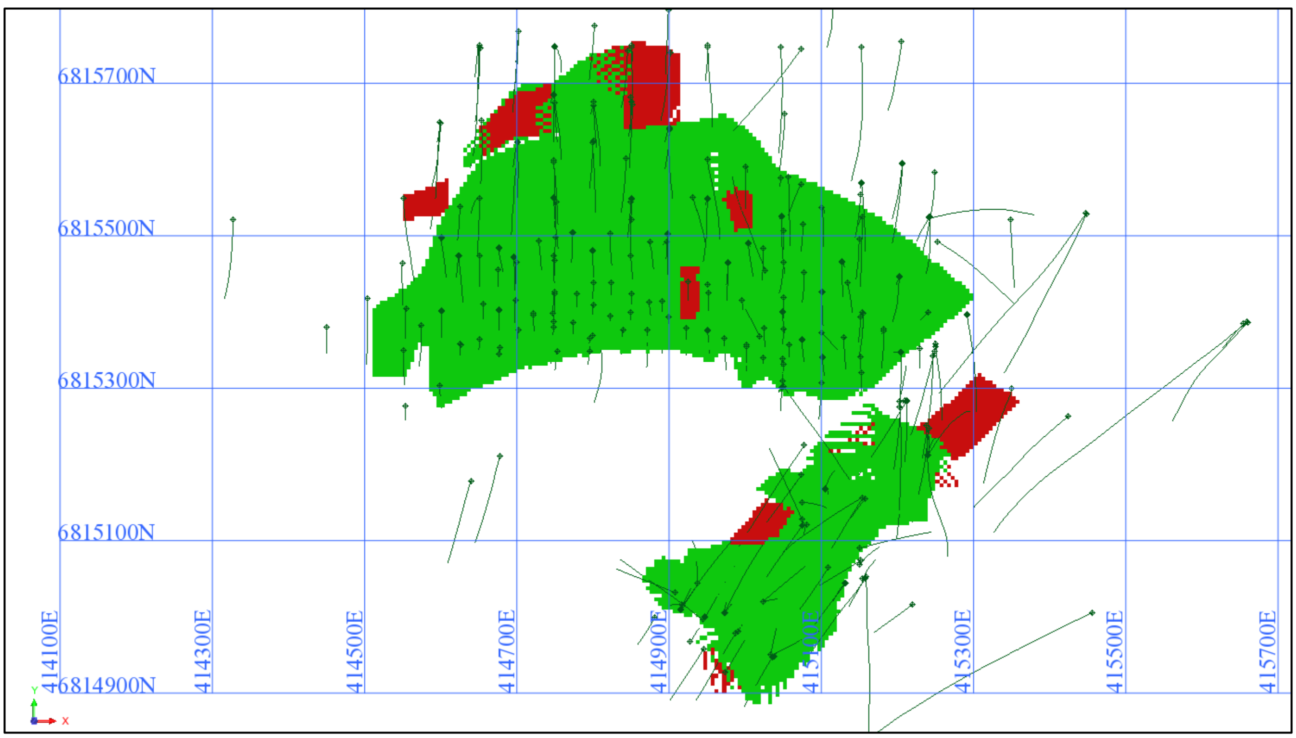


Figure 14-16: ABM deposit Mineral Resource classification in plan view (red = Inferred, green = Indicated)

14.18 Mineral Resource Reporting

Resources are reported in adherence to NI 43-101 Standards of Disclosure for Mineral Projects (Canadian Securities Administrators, 2011), and to the CIM Definition Standards on Minerals Resources and Reserves (CIM Council, 2014).

14.18.1 Results

The ABM deposit MRE is reported in Table 14-18 (open pit) and Table 14-19 (underground).

Table 14-18: ABM deposit MRE – open pittable (at NSR cut-off grade of CAD$25/t)

Zone	Category	Tonnes (Mt)	NSR (CAD$ /t)	Cu (%)	Pb (%)	Zn (%)	Au (g/t)	Ag (g/t)	Cu metal (kt)	Pb metal (kt)	Zn metal (kt)	Au metal (koz)	Ag metal (Moz)
ABM	Indicated	14.6	358	1.0	1.6	6.1	1.3	132	140.9	229.1	886.6	614.0	62.1
	Inferred	0.3	334	1.5	1.5	4.5	1.1	115	4.7	4.9	14.4	10.9	1.2
Krakatoa	Indicated	3.5	443	0.6	3.2	7.2	1.8	213	21.4	113.2	255.5	204.0	24.3
	Inferred	0.1	347	0.6	2.3	6.3	1.3	142	0.1	2.1	5.9	3.8	0.4

Table 14-19: ABM deposit MRE – underground (at NSR cut-off grade of CAD$95/t)

Zone	Category	Tonnes (Mt)	NSR (CAD$ /t)	Cu (%)	Pb (%)	Zn (%)	Au (g/t)	Ag (g/t)	Cu metal (kt)	Pb metal (kt)	Zn metal (kt)	Au metal (koz)	Ag metal (Moz)
Krakatoa	Indicated	0.2	397	1.0	2.0	6.1	1.7	170	1.7	3.5	10.5	9.2	0.9
	Inferred	0.4	447	0.8	1.6	9.5	1.2	165	3.2	6.3	37.5	14.9	2.1

Notes:

- *The Mineral Resources in this disclosure were estimated by Aaron Green, MAIG.*
- *The Effective Date of this Mineral Resource is 31 May 2017.*
- *Numbers have been rounded to reflect the precision of an Indicated and Inferred MRE.*
- *Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability but are required to have reasonable prospects for eventual economic extraction.*
- *The Indicated Mineral Resources are inclusive of those Mineral Resources modified to produce the Mineral Reserves. Inferred Mineral Resources are, by definition, always additional to Mineral Reserves.*
- *The Mineral Resources were estimated using current CIM standards, definitions and guidelines.*
- *The ABM database was audited in its entirety and contains a total of 335 diamond drillholes defining the ABM deposit for 55,782 m of drilling. 241 assayed drillholes intersect the interpreted mineralization zones. There are also 8,393 bulk density samples from the ABM deposit in the database, including 837 samples used for quality control.*
- *QAQC protocols were carried out to assess the quality of the drilling assay results and the confidence that can be placed in the assay data. The QAQC data available for the ABM deposit demonstrate the analytical data are of sufficient quality to be used in estimating Mineral Resources.*
- *The ABM Zone was sampled using diamond drillholes at nominal 50 m spacing on 25 m north-south oriented sections extending out to 100 m on the peripheries of the deposit. The Krakatoa Zone is sampled targeting pierce points of 25–60 m in the central portion of the deposit to 100 m on the peripheries.*
- *A total of 34 mineral domains were modelled (10 at Krakatoa Zone and 24 at ABM Zone) including two "dilution skin" domains., Assays were regularized within each domain to 1.5 m and 1.0 m intervals for ABM and Krakatoa zones respectively. Grade capping was applied to all grades estimated based on statistical analysis by domain. KNA was completed using Supervisor v8.4™ software, adopting the relevant variogram models for the estimation domains. KNA was completed for each of the variables, based on the combined massive sulphide dataset of the ABM Zone and the updated massive sulphide dataset of the Krakatoa Zone, respectively.*
- *OK was selected for grade interpolation in the mineralized zones, whilst ID3 was used in the estimation of the dilution skin. A three-pass estimation search was used to complete estimation for Cu, Pb, Zn, Au, Ag, Fe, As, Ba, Bi, Hg, S, Sb and Se.*
- *Fixed density values were assigned to the block models for each regolith and lithological unit ranging from 2.00 t/m^3 for overburden to 2.80 t/m^3 for the mafic intrusive rock. For the mineralized zones, a tiered approach to the selection of a preferred bulk density value was adopted, and then the bulk density was interpolated into the block model using OK for the mineralized zones and ID3 for the dilution skin. The average bulk densities determined for the ABM stockwork and massive sulphide mineralization were 3.44 t/m^3 and 4.19 t/m^3 respectively, while the average bulk density values for the Krakatoa Zone were 3.86 t/m^3 and 4.09 t/m^3 respectively.*
- *The Mineral Resource is classified as Indicated where, in the Qualified Person's opinion, sufficient data exists to assume geological and mineralization continuity (generally 50 m spaced holes on 25 m spaced sections). For areas with more limited data density and limited along-strike or down-dip continuity, there is sufficient evidence to imply but not verify geological and grade continuity and these areas are classified as Inferred.*

- *The in-ground NSR values were calculated using assumed metal prices, metallurgical recoveries, smelter terms (including payable factors, concentrate costs and refining charges) and government royalties. No penalties were included. Metal price assumptions were: US$3.50/lb copper, US$1.50/lb zinc, US$1.05/lb lead, US$1,300/oz gold and US$20/oz silver, and an exchange rate of US$0.75 = CAD$1.00. Metal recovery assumptions92% for copper, 90% for zinc, 70% for lead, 75% for gold (whereby 30% is recovered into copper concentrate, 30% is recovered into lead concentrate and 15% is recovered into zinc concentrate) and 85% for silver (40% into copper concentrate, 30% into lead concentrate and 15% into zinc concentrate). Based on these assumptions, the formula for the NSR on each block was calculated as: NSR US$/t = (52.84*cu_cut) + (9.56*pb_cut) + (19.13*zn_cut) + (24.41*au_cut) + (0.41*ag_cut).*
- *The US dollar NSR was converted to Canadian dollars using the formula: NSR CAD$/t = (NSR US$/t)/0.75.*
- *Potential open pittable resources were reported above a cut-off NSR of CAD$25/t and potential underground resources reported above CAD$95/t.*
- *To determine the reporting of ABM deposit Mineral Resources as either open pit or underground, a Whittle™ pit optimization was undertaken. Parameters used for the optimization included base case metal price assumptions of: US$3.50/lb copper, US$1.50/lb zinc, US$1.05/lb lead, US$1,300/oz gold and US$20/oz silver, an exchange rate of US$0.75 = CAD$1.00, a mining recovery of 97%, an overall pit wall slope angle of 50°, total processing costs (fresh) of CAD$30.60/t and plant throughput of 2 Mt/a. For the ABM Zone, only material reporting inside the selected pit shell (Revenue Factor = 1.00) has been reported above the NSR cut-off of CAD$25/t. For the Krakatoa Zone, mineralized material inside the pit shell has been reported above the NSR cut-off of CAD$25/t, whilst the remainder has been designated as "underground" resource and reported above a cut-off NSR of CAD$95/t.*
- *The optimal transition from open pit to underground for the Krakatoa Zone has not been considered when reporting the Mineral Resource. Key modifying factors in determining this transition have been factored into estimating of the Mineral Reserve.*

Grade-tonnage tables have been generated for Cu, Pb, Zn, Au and Ag. The global ABM deposit grade tonnage curves for Cu, Pb and Zn are shown in Figure 14-17.

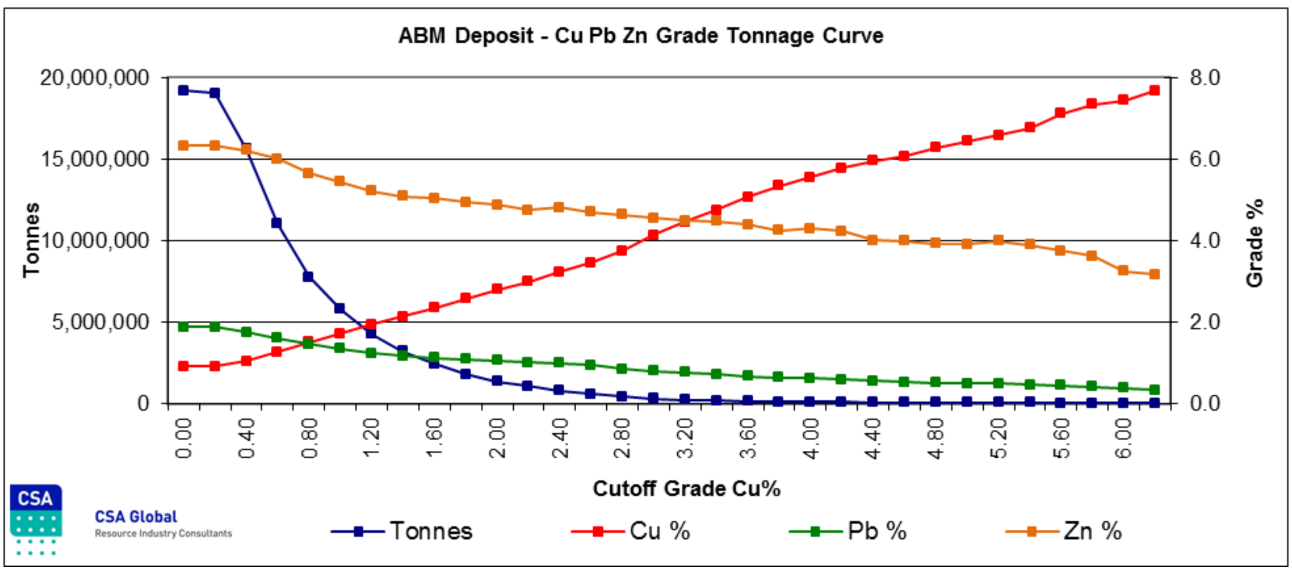


Figure 14-17: ABM deposit global grade-tonnage curve for Cu%, Pb% and Zn%

Figure 14-18 to Figure 14-23 show the block model for the ABM and Krakatoa Zones coloured according to Zn, Pb, Cu, Au, Ag and Fe respectively.

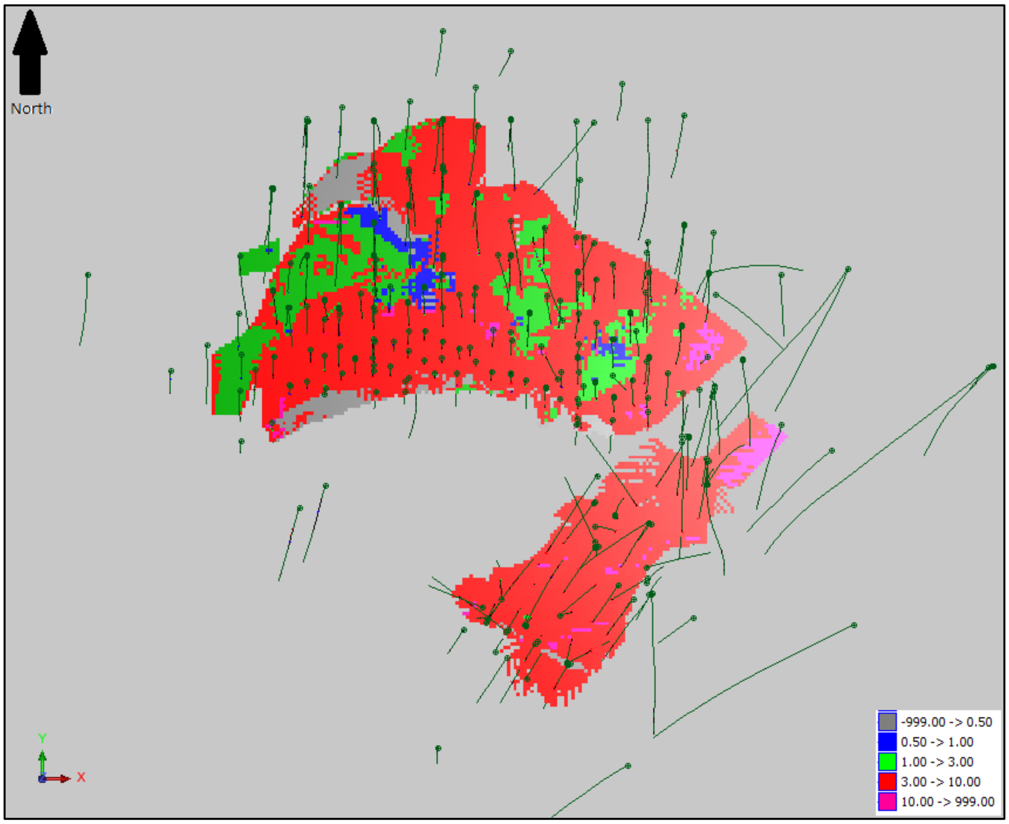


Figure 14-18: ABM-Krakatoa block model showing Zn grades (%) (plan view)

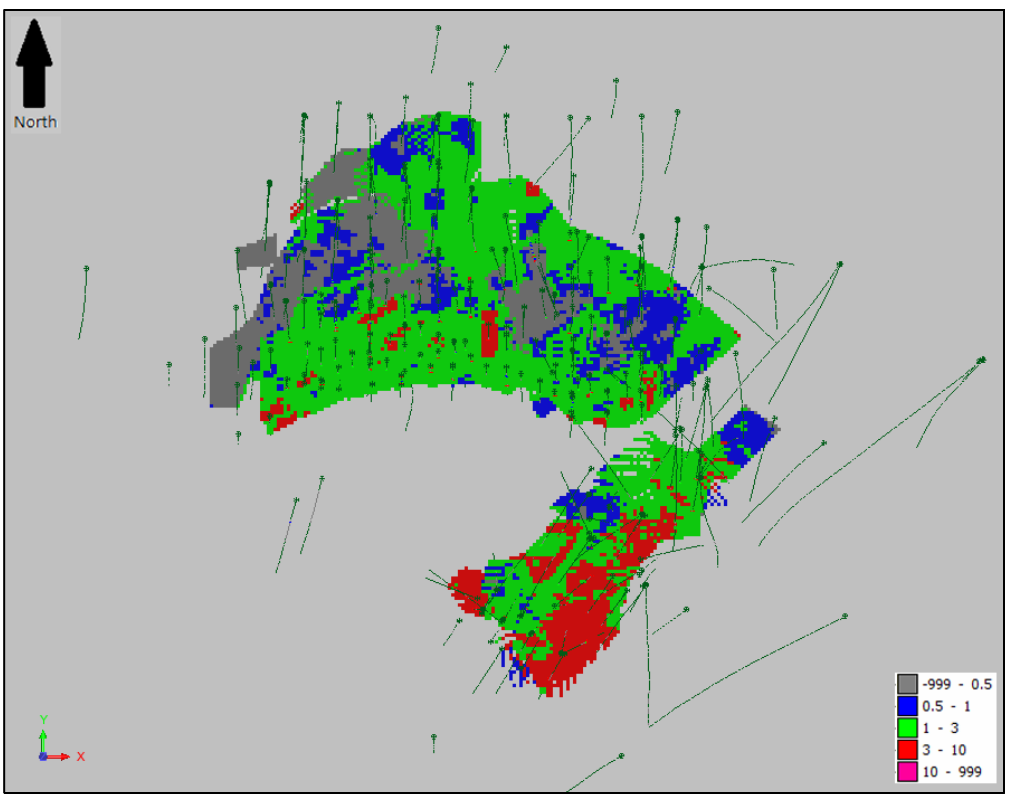


Figure 14-19: ABM-Krakatoa block model showing Pb grades (%) (plan view)

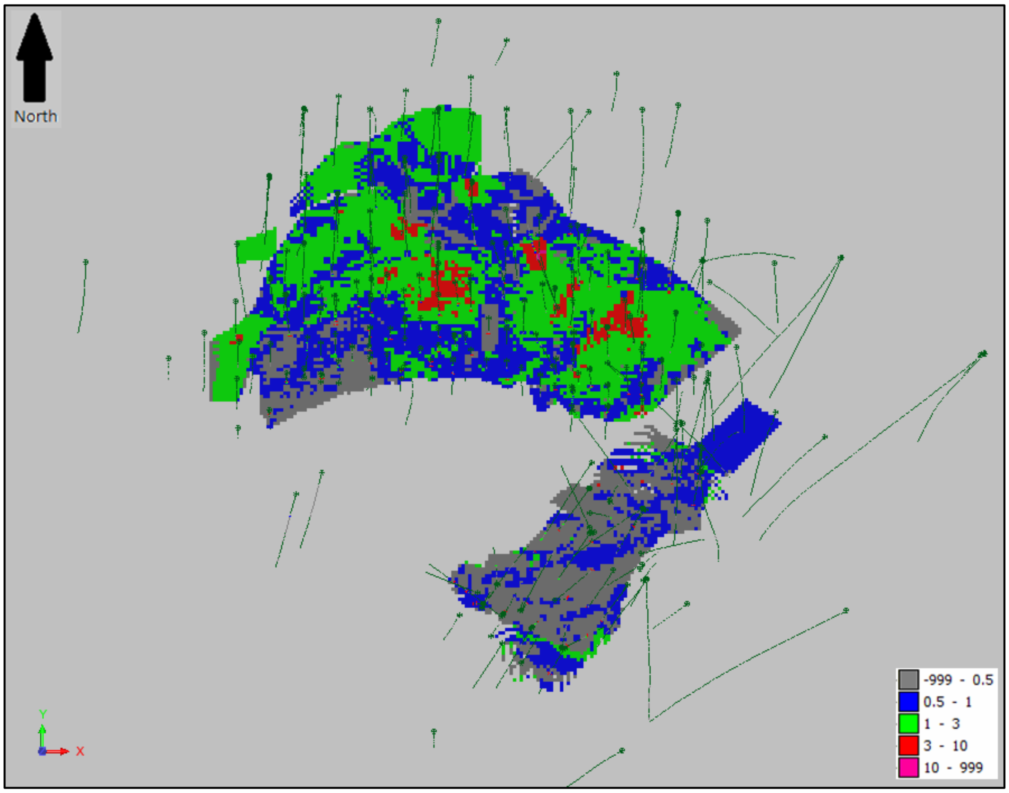


Figure 14-20: ABM-Krakatoa block model showing Cu grades (%) (plan view)

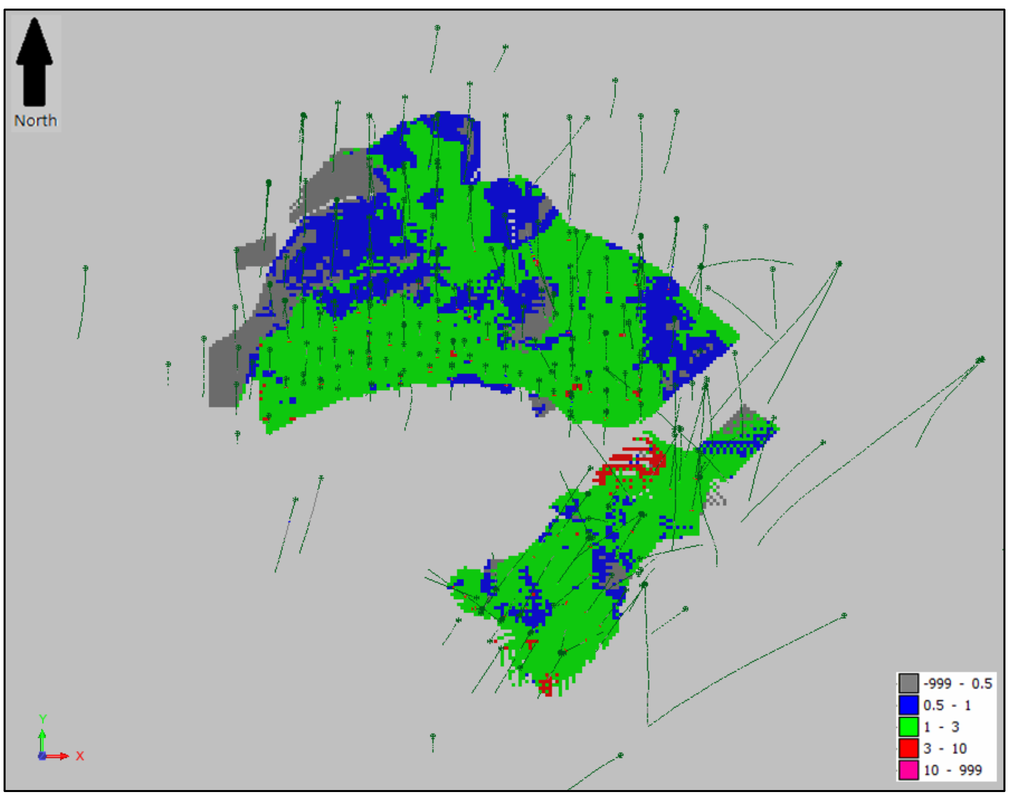


Figure 14-21: ABM-Krakatoa block model showing Au grades (g/t) (plan view)

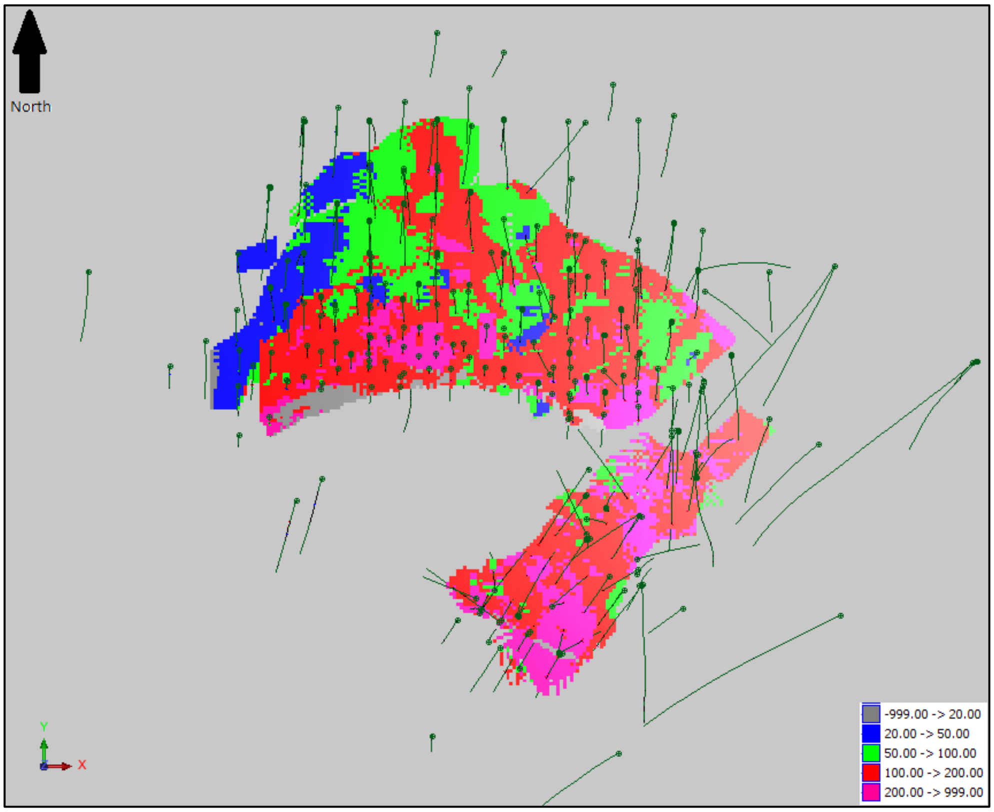


Figure 14-22: ABM-Krakatoa block model showing Ag grades (g/t) (plan view)

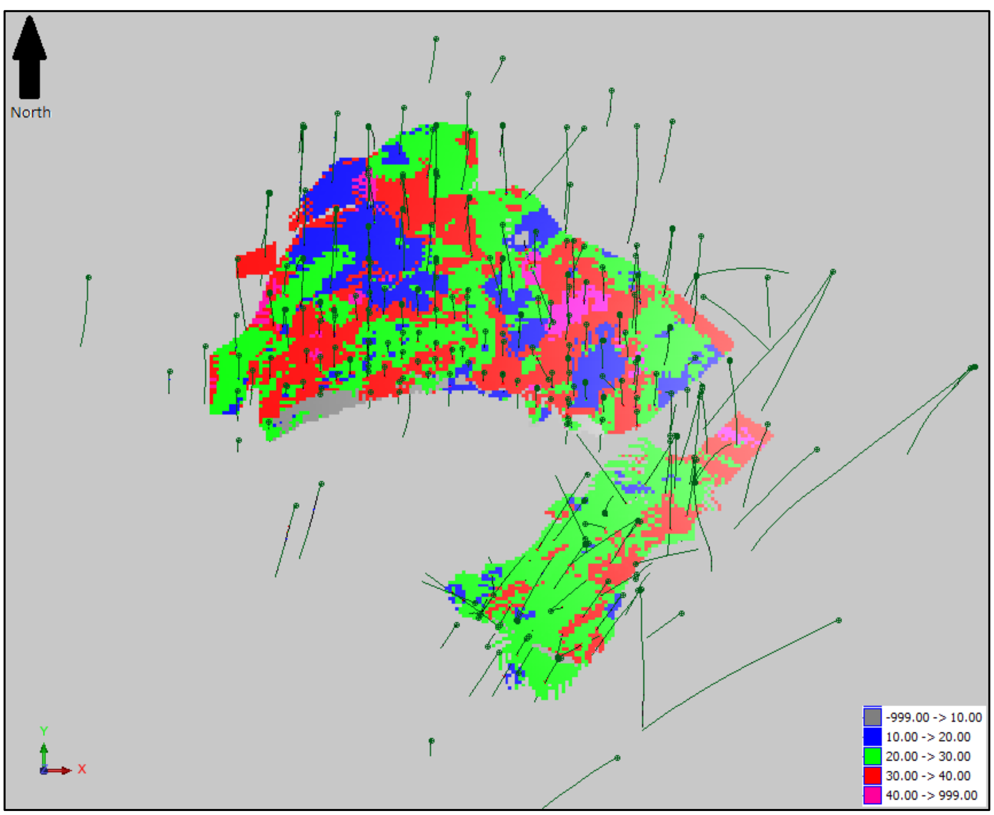


Figure 14-23: ABM-Krakatoa block model showing Fe grades (%) (plan view)

A Mineral Resource has not been reported for the GP4F deposit.

14.18.2 Factors that May Affect the Mineral Resource

CSA Global is not aware of any known environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant issues that could potentially affect the MRE. The reported Mineral Resource may be affected by future mining studies, processing, environmental, permitting, taxation, socio-economic and other factors.

Additional technical factors which may affect the MRE include:

- Geological interpretation (revision of mineralization models, modeling of internal waste domains, modelling of dykes and structural offsets such as faults and shear zones)
- Changes to the technical inputs used to estimate the various elements (e.g. bulk density estimation and grade estimation methodology)
- Metal price and valuation assumptions
- Changes to geotechnical and mining assumptions, including the minimum mining width; or the application of alternative mining methods such as open pit mining
- Changes to process plant recovery estimates if the metallurgical recovery in certain domains is lesser or greater than currently assumed.

14.18.3 Comparison with Previous Estimates

The reported Mineral Resource is comparable with the previous MRE reported by CSA Global in January 2016 in terms of both tonnage and grade (Table 6-1). However, the key differences were the upgrading of Inferred material to Indicated at Krakatoa Zone with infill drilling (2.1 Mt to 3.7 Mt) at slightly lower average grades, and the decrease in overall tonnage at the Krakatoa Zone (5.1 Mt to 4.2 Mt) as a result of drilling to the west (towards the East Fault) failing to intersect significant mineralization in the Main lens. No significant change was reported to the ABM Zone resource.

With respect to more historical estimates undertaken by Cominco (1994, 1995 and 1998), and Teck (2001 and 2007), comparing the ABM Zone only, the reported MRE is comparable in size and grade with minimal differences. Differences can be attributed to changes in estimation techniques, bulk density values, minor adjustments to resource wireframes with increased drilling (both extensional and infill) and prior reporting of resources as "mineral inventory" following mining evaluation studies. On a global basis, the 2016 CSA Global models report significantly more tonnes at higher average grades due to the incorporation of the Krakatoa Zone.

Unfortunately, a more thorough investigation of the differences between the MRE outlined in this document and historical estimates is not possible at this stage, given the lack of detailed documentation of the historical estimates.

14.19 Risk

The drilling, surveying, sampling and analytical methods, and QA processes implemented by BMC during the exploration and resource drilling campaigns are suitable and adequate for the style of deposit under consideration.

The ABM deposit remains open (i.e. Krakatoa down-dip extent) and infill and extension drilling are required to fully define the mineralization extents and upgrade the current Mineral Resource classifications.

OK is an appropriate interpolation method for the ABM deposit at the current level of advancement of the KZK Project, in the light of data currently available. KNA tests undertaken by CSA Global confirm reliable block estimates were achieved sufficiently to enable the resources to be classified as Indicated and Inferred.

There are no known environmental, permitting, legal, title, taxation, social-economic, marketing, political factors that could materially affect the MRE.

14.20 Audits and Reviews

An internal audit was completed by CSA Global which verified the technical inputs, methodology, parameters and results of the estimate.

No external audits have been undertaken of the CSA Global MRE for the ABM deposit.

15 Mineral Reserve Estimates

Mineral Reserves are reported in adherence to NI 43-101 Standards of Disclosure for Mineral Projects (Canadian Securities Administrators, 2011), and to the CIM Definition Standards on Minerals Resources and Reserves (CIM Council, 2014).

The Mineral Reserve estimate for the KZK Project is detailed in Table 15-1. All reserves are classified as "Probable Mineral Reserve", as no Measured Resources have been defined for the Project. Inferred Mineral Resources were not considered in this Mineral Reserve estimate.

Table 15-1: KZK Project Mineral Reserve estimate

Zone/Mine	Category	Ore (Mt)	Cu (%)	Pb (%)	Zn (%)	Au (g/t)	Ag (g/t)
ABM Open Pit	Probable	13.4	0.9	1.5	5.9	1.3	131
Krakatoa Open Pit	Probable	0.6	0.4	3.1	6.3	1.9	246
Total Open Pit	**Probable**	**14.0**	**0.9**	**1.6**	**5.9**	**1.3**	**136**
Krakatoa Underground	Probable	1.7	0.4	2.3	5.0	1.3	147
Total KZK Project	**Probable**	**15.7**	**0.9**	**1.7**	**5.8**	**1.3**	**138**

Notes:

- *The Mineral Reserves in this disclosure were estimated by Karl van Olden, QP.*
- *The effective date of this Mineral Reserve is 30 June 2019.*
- *Numbers have been rounded to reflect the precision of the estimates.*
- *The Mineral Reserves were estimated using current CIM standards, definitions and guidelines (CIM Council, 2014).*
- *Open pit Mineral Reserves are reported within a practical design for an open pit using a Net Smelter Return (NSR) cutoff of CAD$29.30/tonne.*
- *Practical open pit designs were informed by economic mining envelopes determined using Whittle pit optimization software.*
- *Overall slope angles for the designed open pit range from 27° to 47°.*
- *Underground Mineral Reserves are based on an underhand longhole stoping with cemented paste fill mining method and reported to a NSR cut-off of CAD$173/tonne.*
- *Mineral Reserves incorporate estimates of mining dilution and mining recovery.*
- *Mineral Reserves have been calculated based on 2018 long term metal prices of US$3.08/lb for copper, US$0.94/lb for lead, US$1.10/lb for zinc, US$1,310/oz for gold and US$18.42/oz for silver and an exchange rate of CAD$1.00:US$0.792.*
- *Processing recoveries have been calculated for each of the defined metallurgical domains. Average processing recoveries are 73.8% for copper, 73.5% for lead, 85.9% for zinc, 64.9% for gold and 86.0% for silver.*
- *NSR calculations represent the net revenue to Mine Gate after accounting for concentrate treatment and refining charges, concentrate penalties, concentrate transport costs, government royalties and payments to First Nations.*

The NSR method was used to determine economic mineralization for the KZK Project. NSR values have been calculated from October 29, 2018 long-term consensus metal prices (current at the time of commencement of open pit and underground mine design work) published by the Canadian Imperial Bank of Commerce (CIBC) and provided by BMC (Table 15-2). This included an exchange rate of US$0.792:CAD$1.00.

Table 15-2: Commodity prices used in Mineral Reserve estimate

Commodity	Unit	Metal price ($US)
Copper	$/lb	$3.08
Lead	$/lb	$0.94
Zinc	$/lb	$1.10
Gold	$/oz	$1,310.00
Silver	$/oz	$18.42

The Mineral Reserve estimate includes material extracted from the designed open pit and underground excavations that is sourced from diluted Indicated Mineral Resources only and has a block value greater than the NSR cut-off for the relevant type of mining. All open pit Mineral Reserves are reported to a cut-off NSR value of CAD$29.30/t, while underground Reserves are reported to a cut-off NSR value of CAD$172.83/t.

All tonnes and grades have been adjusted for planned and unplanned mining dilution and ore loss. Dilution was applied at zero grade.

Reporting and modelling of financial results was completed in June 2019 using updated long-term consensus metal prices at June 30, 2019 of US$3.15/lb copper, US$1.10/lb zinc, US$0.95/lb lead, US$1,321/oz gold, US$18.09/oz silver and an exchange rate of CAD$1.00:US$0.78. The Mineral Reserve estimate was reviewed under the revised metal price and exchange rate settings and no adjustments to the calculated Mineral Reserves were considered necessary. The modelling considers all capital, operating and selling costs as defined in the DFS.

Sensitivity analysis were conducted as part of the Economic Analysis of the project (Section 22.3). The analysis considered the effect of varying mining and metallurgical parameters as well as capital costs. None of the parameters evaluated resulted in a negative cash flow.

A Quartz Mining Licence (QML) and Type A Water Licence are, amongst other permits, required for the Project to move into production. Both licences are still to be granted and are subject to completion of a Screening Report under the Yukon Environmental and Socio-economic Assessment Act (YESAA) (Section 20.1). As of the Effective Date of this Technical Report, assessment of the Project was still in progress. The Qualified Person is not aware of any matter that may prevent the Screening Process being completed and the aforementioned licences being granted.

The Qualified Person is not aware of any other relevant factors that could materially affect this Mineral Reserve estimate.

16 Mining Methods

16.1 Introduction

CSA Global has completed a mining study as part of the Kudz Ze Kayah DFS (CSA Global, 2019a). The KZK Project is located in Yukon, Canada. The KZK Project addresses the development of the ABM deposit, which comprises two defined Zones, the ABM Zone and the Krakatoa Zone. The mining study describes open pit mining of the ABM Zone and a combined open pit and underground mining approach for the Krakatoa Zone.

16.1.1 General Arrangement

The general arrangement of the mine site is illustrated in Figure 16-1 and shows the location of the processing plant, administration buildings, infrastructure, waste and tailings disposal sites relative to the open pit mining areas. The site layout and infrastructure are detailed in Section 18.

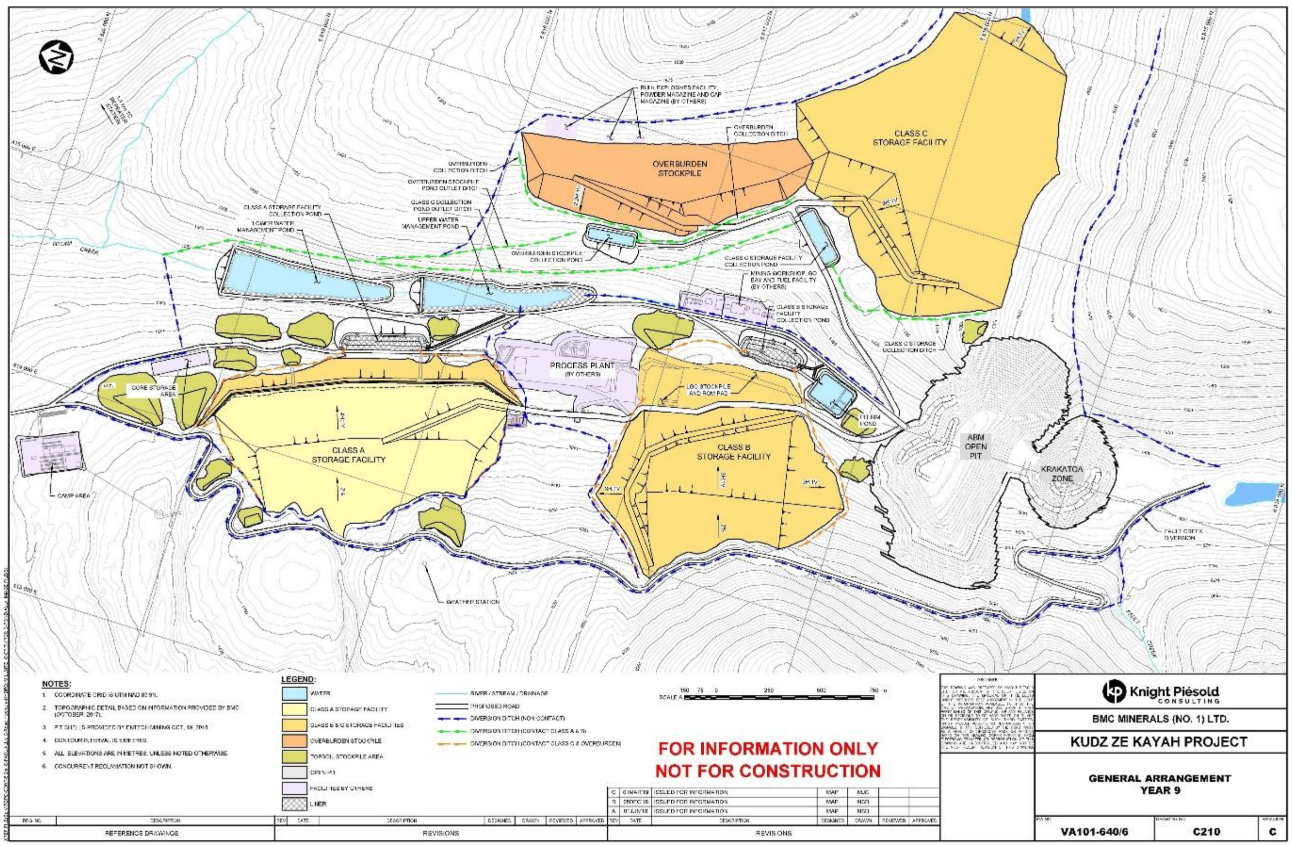


Figure 16-1: General arrangement of the KZK Project

16.2 Mine Plan

The ABM pit forms the largest portion of the mine plan and has a production life of nearly nine years following construction. The total production from Krakatoa open pit and underground is scheduled to fall within the time frame of the ABM pit production. The mining schedule is summarized in Table 16-1.

Table 16-1: KZK Project mine schedule

Source	Unit	Total	2021	2022	2023	2024	2025	2026	2027	2028	2029
Open pit (ABM and Krakatoa)											
Ore mined	**Mt**	**14.0**	**0.2**	**1.8**	**2.1**	**1.8**	**2.1**	**2.2**	**1.4**	**1.5**	**0.8**
Ore grade (Zn)	% Zn	5.9	6.9	6.4	6.4	6.4	5.3	5.6	6.0	5.4	5.6
Ore grade (Cu)	% Cu	0.9	0.5	0.6	0.9	1.2	1.2	1.0	0.9	0.8	0.7
Ore grade (Pb)	% Pb	1.6	2.2	1.9	1.7	1.6	1.3	1.4	1.4	1.5	2.3
Ore grade (Au)	g/t Au	1.3	1.7	1.6	1.4	1.3	1.2	1.2	1.0	1.2	1.6
Ore grade (Ag)	g/t Ag	136	177	162	145	142	119	122	112	129	177
Ore grade (Fe)	% Fe	30	27	29	31	32	32	29	30	29	28
Ore grade (As)	ppm As	2,530	3,541	3,134	3,276	2,517	2,231	2,322	1,625	1,860	3,232
Ore grade (Hg)	ppm Hg	18	20	23	23	21	18	15	10	16	18
Ore grade (Sb)	ppm Sb	464	817	703	627	475	376	403	279	299	417
Ore grade (Se)	ppm Se	208	132	164	203	202	184	186	259	286	233
Waste	Mt	138	7	14	31	32	25	10	10	6	2
Strip ratio	t:t	9.9	31.1	8.1	14.8	17.8	11.7	4.5	7.4	3.8	3.0
Underground (Krakatoa only)											
Ore mined	**Mt**	**1.7**	**-**	**-**	**-**	**-**	**-**	**0.3**	**0.6**	**0.5**	**0.3**
Ore grade (Zn)	% Zn	5.0	-	-	-	-	-	4.9	5.2	5.2	4.6
Ore grade (Cu)	% Cu	0.4	-	-	-	-	-	0.4	0.4	0.5	0.4
Ore grade (Pb)	% Pb	2.3	-	-	-	-	-	2.6	2.5	2.2	1.7
Ore grade (Au)	g/t Au	1.3	-	-	-	-	-	1.3	1.2	1.3	1.2
Ore grade (Ag)	g/t Ag	147	-	-	-	-	-	142	149	153	135
Ore grade (Fe)	% Fe	19	-	-	-	-	-	19	19	19	18
Ore grade (As)	ppm As	2,978	-	-	-	-	-	3,406	2,970	3,044	2,433
Ore grade (Hg)	ppm Hg	15	-	-	-	-	-	11	14	17	14
Ore grade (Sb)	ppm Sb	456	-	-	-	-	-	495	434	474	428
Ore grade (Se)	ppm Se	108	-	-	-	-	-	99	116	111	99
Waste	Mt	0.6	-	-	-	-	0.1	0.3	0.1	0.0	0.0
TOTAL											
Ore mined	**Mt**	**15.7**	**0.2**	**1.8**	**2.1**	**1.8**	**2.1**	**2.5**	**2.0**	**2.1**	**1.1**
Ore grade (Zn)	% Zn	5.8	6.9	6.4	6.4	6.4	5.3	5.5	5.7	5.4	5.3
Ore grade (Cu)	% Cu	0.9	0.5	0.6	0.9	1.2	1.2	0.9	0.8	0.7	0.6
Ore grade (Pb)	% Pb	1.7	2.2	1.9	1.7	1.6	1.3	1.6	1.8	1.7	2.2
Ore grade (Au)	g/t Au	1.3	1.7	1.6	1.4	1.3	1.2	1.2	1.1	1.2	1.5
Ore grade (Ag)	g/t Ag	138	177	162	145	142	119	125	123	135	165
Ore grade (Fe)	% Fe	29	27	29	31	32	32	28	27	26	25
Ore grade (As)	ppm As	2,579	3,541	3,134	3,276	2,517	2,231	2,455	2,018	2,168	3,009
Ore grade (Hg)	ppm Hg	18	20	23	23	21	18	14	11	16	17
Ore grade (Sb)	ppm Sb	463	817	703	627	475	376	414	324	344	420
Ore grade (Se)	ppm Se	197	132	164	203	202	184	175	217	241	196
Waste	Mt	139	7	14	31	32	25	10	11	6	2

A summary of the waste rock produced by open pit and underground mining activities over the LOM is presented in Table 16-2. Waste rock is classified as Class A, B or C according to potential for acid generation and metal leaching, as described in Section 18.1.2. The proportions of Class A, B and C waste rock that BMC expects to produce over the LOM are 8%, 28% and 65% respectively.

Table 16-2: Waste rock produced by class

	Units	Total	2021	2022	2023	2024	2025	2026	2027	2028	2029
Class A	Mt	9.6	0.2	1.1	1.7	1.0	1.5	1.5	0.8	1.2	0.7
Class B	Mt	34.7	0.6	5.2	5.0	6.1	6.2	3.0	3.9	3.2	1.4
Class C	Mt	81.5	4.1	4.1	23.6	23.8	17.0	4.5	2.8	1.4	0.2
Overburden	Mt	13.2	2.0	4.1	1.1	1.1	0.5	1.2	3.1	0.0	-
Total Waste	**Mt**	**139.0**	**6.9**	**14.4**	**31.4**	**32.0**	**25.3**	**10.3**	**10.6**	**5.7**	**2.3**

16.3 Revenue Parameters

The Resource model was populated with an NSR field to represent the revenue received from each unit of metal at the mine gate, hence net of all downstream costs. The NSR (*Free on Board (FOB) Mine Gate and Royalties by unit of metal in ROM Feed*) was calculated on a block by block basis for copper, lead, zinc, gold and silver based on the following:

- Metal prices:
  - Copper – US$3.08/lb
  - Lead – US$0.94/lb
  - Zinc – US$1.10/lb
  - Gold – US$1,310/oz
  - Silver – US$18.42/oz.
- Diluted block grade for copper, lead, zinc, gold and silver.
- Processing recovery to concentrate of all revenue elements for each of the three concentrate products.
- Grade of any contaminant elements based on concentrates produced from testwork.
- Metallurgical domain (+1340 RL, Met 2-4, Met 5-7, Met 8 and Krakatoa).

The NSR field represents the net revenue to Mine Gate after accounting for the following deductions:

- Treatment and refining charges.
- Penalties due to contaminant elements (if present).
- Transport costs:
  - Road transport
  - Port charges
  - Shipping.
- Royalties:
  - Yukon Government.
- First Nations payments.

The resultant NSR values per unit of metal are described in Table 16-3.

Table 16-3: NSR factors applied to grades in block model for each metallurgical domain

Element	Unit	+1340 RL	Met 2-4	Met 5-7	Met 8	Krakatoa
Copper	CAD$ / %	38.64	51.32	41.72	54.96	35.98
Lead	CAD$ / %	10.62	7.36	9.88	6.66	12.15
Zinc	CAD$ / %	12.66	12.65	13.31	11.98	13.38
Gold	CAD$ / g	25.51	25.53	28.77	29.74	26.88
Silver	CAD$ / g	0.47	0.52	0.50	0.35	0.55

16.4 Open Pit Mining

The open pit mining study for the ABM and Krakatoa zones consisted of a Whittle™ optimization, stage sequence selection and mine design. The stages were designed and subsequently scheduled in MineSched™ software to produce a mining and production schedule that was then used in the cost model and financial analysis.

16.4.1 Geotechnical

The open pit optimization and design has been guided by the geotechnical analysis by Dempers and Seymour(D&S) (Dempers & Seymour, 2019a).

Raw data for the project comprised geotechnical logs of exploration and geotechnical drillholes logged by D&S. This data was supplemented with geotechnical logs from core photographs and structural interpretation of televiewer surveys by D&S. In total, the database consists of 7,378 m of geotechnical data covering the project area.

A geotechnical structural model was developed using geotechnically logged core, televiewer data and core photos. A total of 29 significant geotechnical features were identified and modelled in 3D by D&S. The features have been classified according to orientation (i.e. east-west, northeast and north-northeast).

A 3D Mining Rock Mass Model was constructed based on lithological wireframes provided by BMC, geotechnical logs and significant geotechnical features.

A range of analyses were conducted by D&S to determine stable pit slope parameters:

- Rock bridge analysis
- Structural stability analysis
- Limit equilibrium and stress reduction analyses.

The Mining Rock Mass Model was interrogated in section and in plan to assess the variability of the input parameters and the effect of the overall slope angle for the proposed pit design. As a result of this review, the project area has been divided into seven domains. D&S provided specific design parameters for the pits related to these individual geotechnical domains and features within the open pit designs. The geotechnical domains and slope sections later evaluated using limit equilibrium, are shown in Figure 16-2.

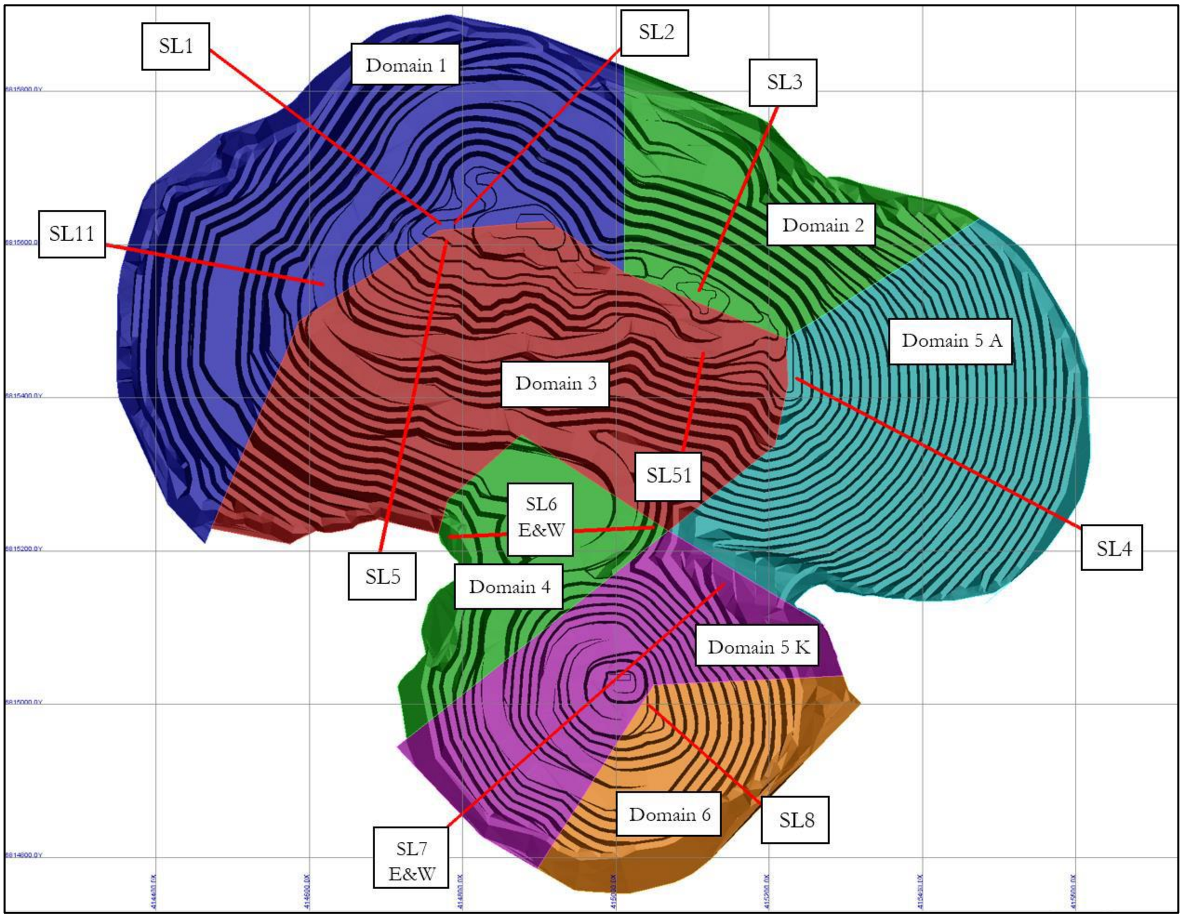


Figure 16-2: Geotechnical slope zones for ABM and Krakatoa pits

The open pit optimization evaluated several slope sets based on D&S recommendations, but primarily allowing for haul roads to be placed on the hanging wall or the foot wall of the main mineralization.

A complete set of slope profiles implemented in the final pit optimizations are shown in Table 16-4 for the ABM and Krakatoa pits. The overall slope angle (OSA) pit slopes are in degrees, the azimuth is the direction to which individual slopes apply.

Table 16-4: Whittle optimization slope profile "Model H" – west FW access V3Z (ABM pit)

Zone code	Profile	North (0°)	East (90°)	South (180°)	West (270°)
1	Domain 1 ABM	44.5	46.7	42.9	42.9
2	Domain 2 East	46.7	38.3	34.0	46.5
3	Domain 3 ABM	44.5	38.3	27.5	42.9
4	Domain 4 K	34.0	43.2	40.7	43.2
5	Domain 5 ABM	39.0	38.3	38.3	47.0
6	Domain 6 K	35.0	34.3	26.3	34.3
7	Default	45.0	45.0	45.0	45.0
8	Krakatoa (D5K)	36.3	36.3	27.6	27.6
9	Overburden	25.0	25.0	25.0	25.0
10	ABM West Fault	46.1	47.0	34.0	34.0
11	Domain 2 West	47.0	38.3	27.5	42.9

Source: CSA Global, 2019a

Pit designs were completed using the recommended pit slope. The final DFS pit design (abm_pd_stage_4_20190408) was evaluated by D&S for overall slope stability using limit equilibrium analyses. The resulting Factors of Safety are equal to or exceed 1.2 and are within accepted guidelines for open pit slope stability.

As recommended by D&S, a slope monitoring program will be implemented at the start of mining to monitor and minimize the adverse effects of instability on slopes. This will entail tension crack mapping, regular pit slope survey and groundwater monitoring.

16.4.2 Whittle™ Optimization

A Whittle™ optimization process was undertaken for both the ABM and Krakatoa zones. The resultant selected ABM pit shells, by tonnes of waste and ore, including cash flows (undiscounted and discounted), are shown in Figure 16-3.

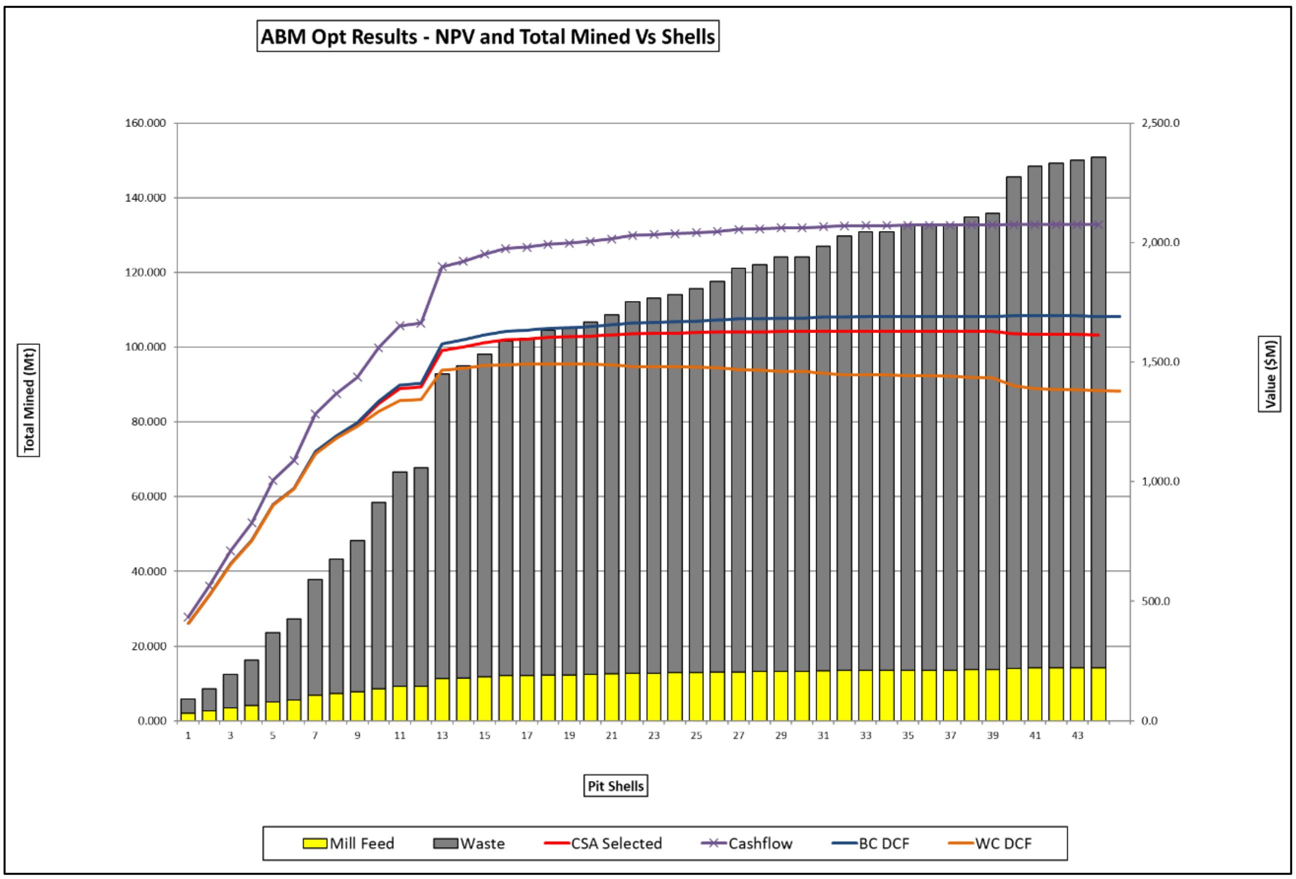


Figure 16-3: ABM optimization results – NPV and mined material

Pit shell 33 was selected as the final pit shell for the ABM open pit. Pit shells 2, 6 and 12 were selected to guide the design for the interim stages (i.e. pushbacks) of the pit. The key properties of the ABM pit stages (as defined during the Whittle™ optimization process) for the selected shells, are shown cumulatively in Table 16-5.

Table 16-5: Selected pit shells cumulative properties – ABM pit

Shell no.	Revenue factor	Pit shell depth	Total mined (Mt)	Waste mined (Mt)	Strip ratio	ROM feed (Mt)	Grade Cu%	Grade Pb %	Grade Zn%	Grade Au g/t	Grade Ag g/t	CSA Global NPV (CAD $M)	Opex (CAD$/t)
2	0.22	245	8.6	5.9	2.2	2.7	0.7	1.9	6.6	1.6	162	526	48.72
6	0.30	290	27.2	21.6	3.9	5.6	0.8	1.8	6.5	1.5	149	972	56.13
12	0.42	325	67.7	58.4	6.3	9.2	1.0	1.7	6.1	1.4	144	1,395	67.31
33	0.84	350	130.8	117.3	8.7	13.5	0.9	1.5	5.9	1.3	130	1,630	78.20

The Krakatoa pit will be mined as a single stage, as the pit is small and mined near the end of the project life. Pit shell 13 was selected as a candidate for the pit design (Table 16-6). A portion of the waste material in the Krakatoa pit will be mined early to create a lay down and portal staging area for the planned underground operation. The remainder of the pit is planned to be mined toward the end of the project and concurrently with underground mining.

Table 16-6: Selected pit shell properties – Krakatoa

Shell no.	Revenue factor	Pit shell depth	Total mined (Mt)	Waste mined (Mt)	Strip ratio	ROM feed (Mt)	Grade Cu%	Grade Pb %	Grade Zn%	Grade Au g/t	Grade Ag g/t	CSA Global NPV (CAD $M)	Opex (CAD$/t)
13	1.00	300	14.6	14.0	23.5	0.6	0.4	3.1	6.3	1.9	246	94	138.01

16.4.3 Mine Design

The ABM Zone dips to the northeast. The deposit, on the surface, is confined by the valley running approximately north to south, and the pit will mine both sides of the valley hills. The main ramp entrance was placed on the north side of the ABM pit, at the lowest point possible, to minimize the amount of pioneering road works leading to the Process Plant, stockpiles and other infrastructure.

The main pit ramp follows the western side of the ABM pit for the following reasons:

- It allows for a shorter ramp accessing the Krakatoa pit and the underground mine portals area.
- The pit slopes are shallower on the footwall side of the deposit and therefore a more economical location for the final ABM pit ramps, including switchbacks.
- The eastern side of the valley at the ABM pit is higher than the western side, leading to increased waste stripping to accommodate a ramp. The weaker Wind Lake formation is also located on this side of the pit.

Complex fault systems exist in the vicinity of the mineralized zones (Figure 16-4). The impact of the faults on pit wall stability has been acknowledged. The more parallel and closer to the fault a wall is, the more significant impact it would pose to the affected section of the pit wall. This condition has been identified by D&S in relation to the west wall of the ABM pit and the North-Northeast faults. As a consequence, the impact of the north-northeast fault was reduced by ensuring the pit wall steps behind the fault once the wall approached within 10 m of the fault. In addition, a 40 m wide geotechnical bench at 1,400 mRL was included in the design to reduce the OSA.

The complexity of the fault system is also shown in Figure 16-5, in a level section view at 1,340 mRL.

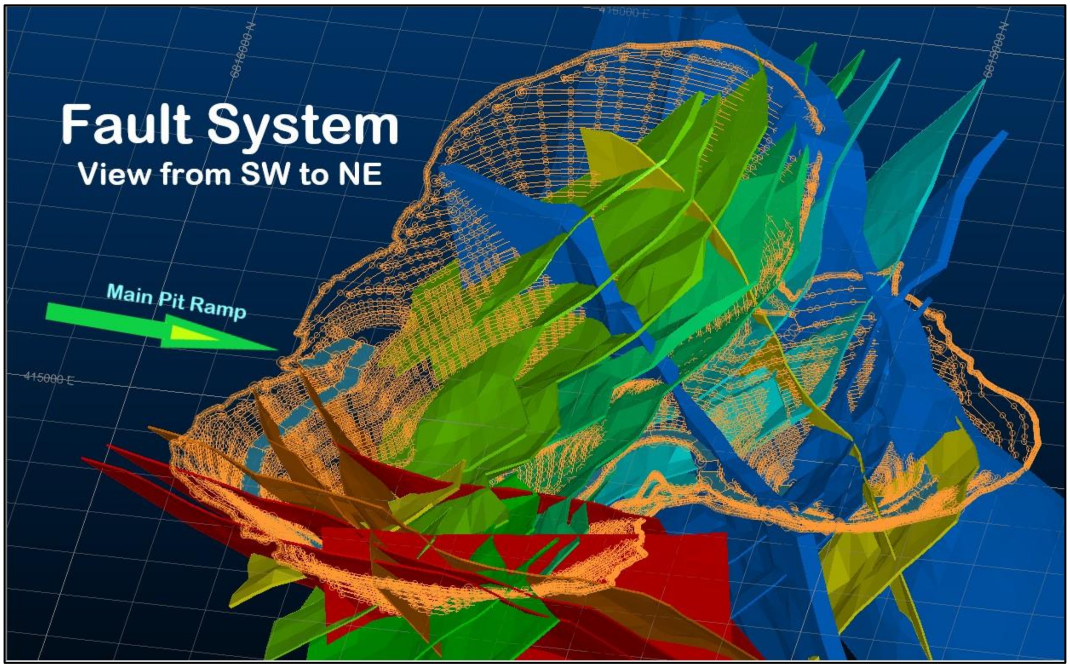


Figure 16-4: Fault system with pit outline (isometric view)

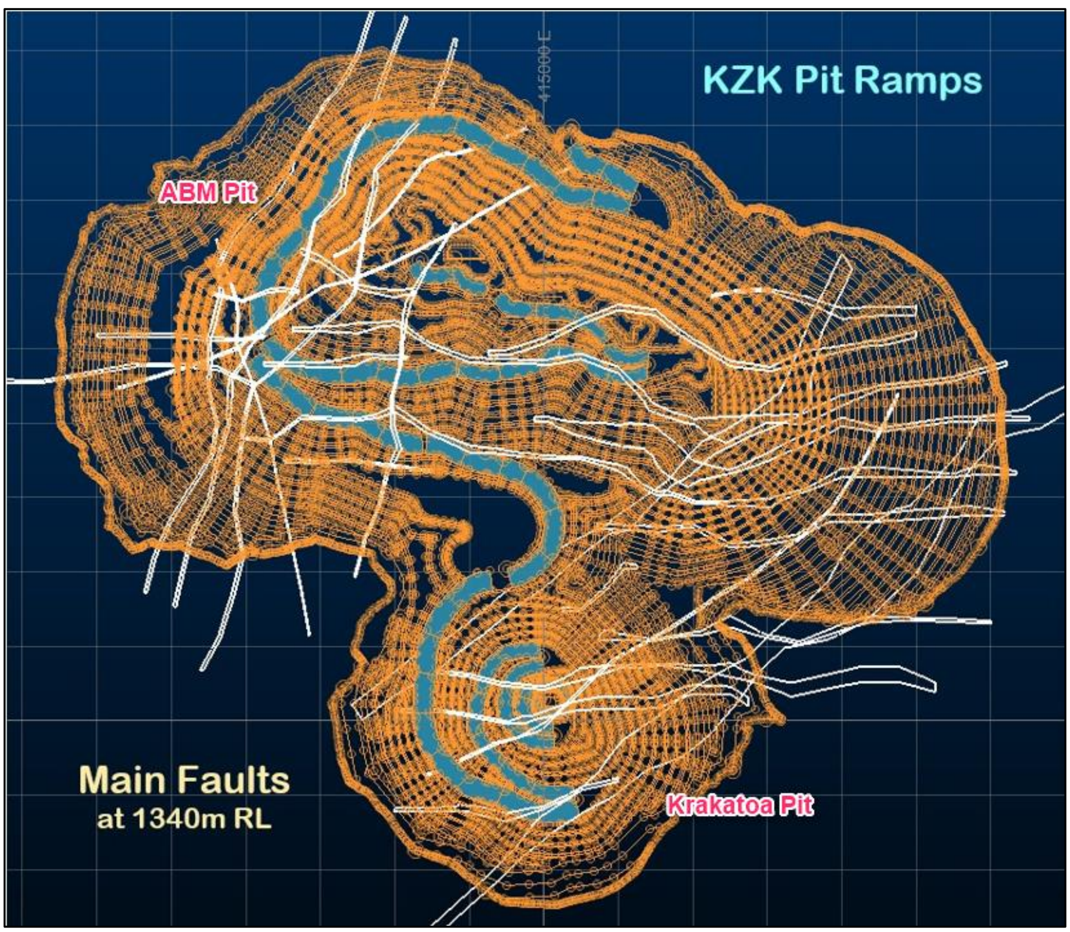


Figure 16-5: Fault system (plan view)

An 8 m-wide perimeter bench will be established where the pit design intersects the base of the overburden. The perimeter bench does not have a fixed elevation; rather, it follows the surface of the transition zone. The bench face angle through the overburden layer is related to its thickness as assessed by D&S.

The haul road designs are based on a fleet of 140 tonne haul trucks, for which 27 m-wide ramps are designed for a double-lane road, including a safety berm on the pit crest side and a drainage ditch on the pit wall side. Single-lane ramps were designed to be 20 m wide.

Mining Dilution and Recovery

Mining dilution and mining recovery were estimated by coding mining dilution and ore loss distances (Table 16-7) into the block model for each area of the deposit, defined by the domains surrounding each individual block. Some blocks are surrounded by both hangingwall and footwall, others by only one or the other, and some blocks are only in contact with other ore blocks.

Table 16-7: Mining dilution and mining recovery distances as applied within the block model

Deposit area exposed to mining dilution and ore loss	Dilution distance (m)	Ore loss distance (m)
ABM both FW and HW	1.50	0.50
ABM FW or HW	0.75	0.25
ABM internal	0.00	0.00
Krakatoa both FW and HW	2.00	0.80
Krakatoa FW or HW	1.00	0.40
Krakatoa internal	0.00	0.00

Table 16-8 shows the average mining dilution and recovery for each metallurgical domain.

Table 16-8: Mining dilution and ore loss estimated for each metallurgical domain

Metallurgical domain	Average mining dilution (%)	Average mining recovery (%)
+1340 RL (ABM Domain 1)	6.0	97.6
Met2-4 (ABM Domain 2)	2.0	99.1
Met5-7 (ABM Domain 3)	3.9	98.4
Met8 (ABM Domain 4)	9.9	96.6
Krakatoa (Krakatoa Domain 5)	13.1	94.4

16.4.4 Ore and Waste Cut-Off

An NSR was estimated for each block classified as Indicated Mineral Resource within the block model. The NSR estimated the total block revenue for each economic metal after the application of processing recoveries, operating costs incurred after the mine gate, government royalties and First Nations payments, as detailed in Section 16.3. All Indicated Mineral Resources with an NSR value greater than CAD$29.30/t were classified as ore within the open pit mining and production schedule.

16.4.5 Open Pit Mine Schedule

The Process Plant has a maximum ROM throughput capacity of 2.2 Mt/a. The mining schedule reflects a processing plant production ramp-up for eight months after commissioning, before reaching steady state throughput.

The Process Plant has a constraint on the mass of zinc fed to the zinc flotation circuit. To account for this the production schedule limits the maximum ROM throughput capacity to 357 t of contained zinc metal per day.

This is achieved by lowering the processing rate during times where the contained zinc in ROM feed exceeds this threshold.

The mining sequence aims for low strip ratio and high yield early in the schedule, followed by an even strip ratio that meets the Process Plant requirements. ABM pit stages 1A and 1B are prioritized first to meet the construction volume requirements. ABM pit stages 2, 3, 4, and the Krakatoa stage are mined sequentially, with appropriate lags where required, to maintain waste stripping and ore release that meets the target processing requirements.

The mining schedule maintains total material movements that match the nominated mining fleet. Each excavator shift added to the mining schedule has the capacity to mine at a rate of 750 kt/month. Total mining movement has been capped at 3 Mt/month, meeting the maximum capacity of the proposed mining fleet.

Mining vertical advance rates have been limited to two benches per month. The only time this mining rate occurs is when narrow benches are mined on the upper hill slopes. There is a maximum of three active mining locations as well as a maximum of three active mining benches. The underground mining portal is available upon the completion of the ABM pit stage 1C. The schedule completes mining of ABM pit stage 1C in March 2023. The production schedule includes underground ROM ore being produced and fed from March 2026 to October 2029. Underground ROM ore produced has been incorporated with the open pit ROM production to a monthly resolution. The open pit mining and production schedule was prepared on the basis of reducing the open pit ROM processing throughput by the quantity of underground ROM tonnes produced each month.

The LOM mining schedule is detailed in Table 16-1.

Figure 16-6 shows the tonnes of material movement for each open pit stage, throughout the mine life. Figure 16-7 shows the tonnes of material movement by waste tonnes and ore tonnes. Mining commences in March 2021, with pre-production mining targeting the waste classifications required for construction purposes. ABM pit stages 1 and 2 are prioritized for the first 22 months of mining for both pre-production and initial low waste stripping mining to meet the production targets. The large jump in total material movement from the higher waste stripping pits, ABM stage 3 and ABM stage 4, are deferred until February 2023 within the mining schedule, 14 months after the commencement of Process Plant ore commissioning. The total mining rate progressively reduces from September 2025 as less waste movement is required to maintain an adequate feed rate of ore to the ROM pad.

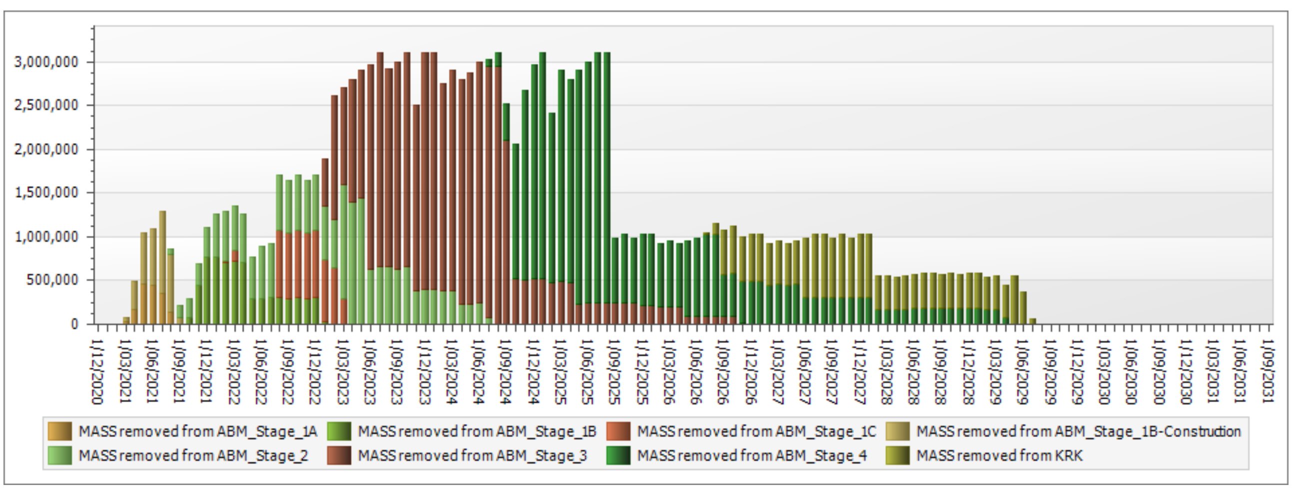


Figure 16-6: Open pit mining material movement tonnes by pit stage

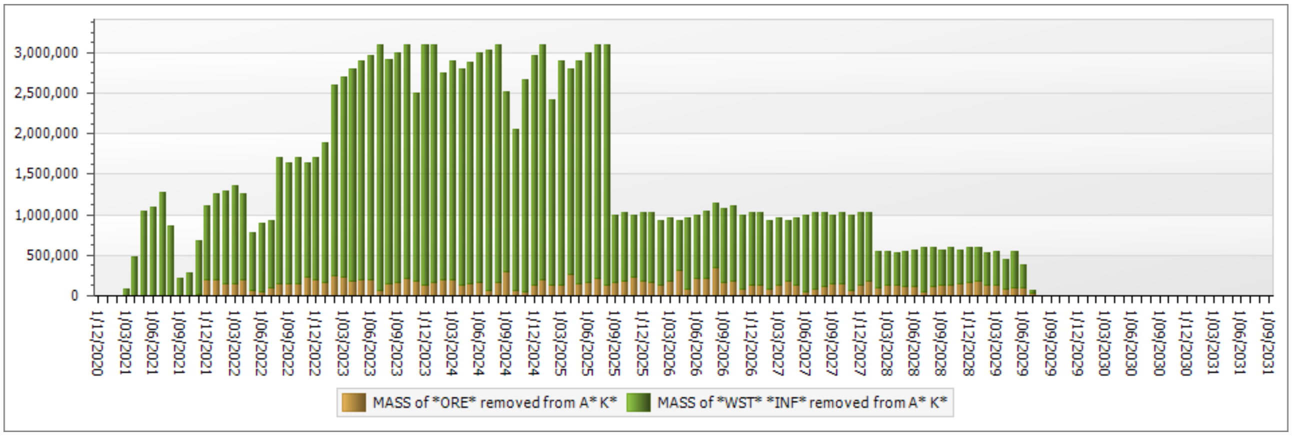


Figure 16-7: Open pit material movement by waste tonnes (green) and ore tonnes (orange)

16.4.6 Open Pit Mining Equipment

The KZK open pit operations will apply conventional open pit drill and blast mining techniques. The material will be mined using excavators and off-highway haul trucks. The drill, blast, load and haul operations will be supported by a fleet of ancillary equipment such as dozers, graders, water carts and front-end loaders.

Two sizes of excavator have been selected for the ABM and Krakatoa open pits:

- Hitachi EX3600 (or equivalent) – 360 t operating weight
- Hitachi EX1900 (or equivalent) – 190 t operating weight.

The smaller 190 t excavator is scheduled for production in the early stages of the mine life where a single unit is planned for the first 24 months of the mine life for mining the narrow, early stages on the inclined flanks of the pits. The 360-t class excavator will be the primary loading tool for open pit production.

A 140-t truck has been selected for the KZK Project. The mine plan has a maximum of 20 trucks mobilized to site during the peak material movement period. Trucks have been scheduled on an as-needed basis and where necessary trucks will be mobilized and demobilized as required.

A summary of the equipment requirements for the open pit mine is detailed in Table 16-9. Make and models are provided for reference purposes only and may be varied depending on the final binding agreement with the mining contractor.

Table 16-9: Open pit mine equipment schedule

Equipment class	Make/Model (example only)	2021	2022	2023	2024	2025	2026	2027	2028	2029
Excavator, 360 t	Hitachi EX3600	1	1	2	2	2	1	1	1	1
Excavator, 190 t	Hitachi EX1900	1	1	1	0	0	0	0	0	0
Truck, 140 t	Komatsu HD1500	8	10	20	20	20	8	8	8	4
Truck, 40 t	Caterpillar 740	4	4	4	4	4	4	4	4	4
Dozer, 70 t	Caterpillar D10	2	2	2	2	2	1	1	1	1
Dozer, 50 t	Caterpillar D9	2	2	2	2	2	1	1	1	1
Dozer, 25 t	Caterpillar D6	1	1	1	1	1	1	1	1	1
Wheel Dozer	Caterpillar 834	0	1	1	1	1	0	0	0	0
Wheel Loader, 100 t	Caterpillar 992	1	1	1	1	1	1	1	1	1
Wheel Loader, 30t	Caterpillar 980	1	1	1	1	1	1	1	1	1
Compactor	Caterpillar CP56	1	1	1	1	1	1	1	1	1
Blast Hole Drill Rig (DHH)	Epiroc DM45	1	2	2	2	2	1	1	1	1

Equipment class	Make/Model (example only)	2021	2022	2023	2024	2025	2026	2027	2028	2029
Blast Hole Drill Rig (THH)	Sandvik DX700	1	1	1	1	1	1	1	1	1
Grader	Caterpillar 16M	2	2	2	2	2	1	1	1	1
Water Cart	Water Cart	2	2	2	2	2	1	1	1	1
Service Truck	Service Truck	2	2	2	2	2	1	1	1	1
IT Loader	IT Loader	1	1	1	1	1	1	1	1	1
Rockbreaker	Rockbreaker	1	1	1	1	1	1	1	1	1
Light Vehicle	Light Vehicle	10	15	15	15	15	15	10	10	10

16.4.7 Open Pit Mining Labour

The work schedule assumes a 24-hours per day, 7-days per week and 365 days per year mining operation. Operations and maintenance personnel will work two 12-hour shifts per day. All mining personnel will work on a two week in/one week out rotation. A summary of the open pit mining labour schedule is provided in Table 16-10.

Table 16-10: Open pit mining labour schedule

Position	2021	2022	2023	2024	2025	2026	2027	2028	2029
Excavator Operator	6	6	6	6	6	3	3	3	3
Dozer Operator	6	6	6	6	6	3	3	3	3
Truck Operator	36	42	72	72	72	36	36	36	24
Wheel Loader Operator	6	6	6	6	6	6	6	6	6
Compactor Operator	3	3	3	3	3	3	3	3	3
Drill Operator	6	6	6	6	6	3	3	3	3
Grader Operator	6	6	6	6	6	3	3	3	3
Miscellaneous Equipment Operator	6	6	6	6	6	6	6	6	6
General Labourer	3	3	3	3	3	3	3	3	3
Blasting Services Operator	4	4	4	4	4	4	4	4	4
Subtotal Operators	**82**	**88**	**118**	**118**	**118**	**70**	**70**	**70**	**58**
Maintenance Superintendent	1	1	1	1	1	1	1	1	1
Maintenance Supervisor	2	3	3	3	3	2	2	2	2
Heavy Duty Mechanic/Welder	17	19	28	28	28	15	15	15	12
Serviceman	6	6	6	6	6	3	3	3	3
Maintenance Labourer	5	6	9	9	9	5	5	5	4
Maintenance Planner	1	2	2	2	2	1	1	1	1
Purchaser and Maintenance Admin	2	4	4	4	4	2	2	2	2
Subtotal Maintenance	**34**	**41**	**53**	**53**	**53**	**29**	**29**	**29**	**25**
Project Manager	1	1	1	1	1	1	1	1	1
Superintendent	1	1	1	1	1	1	1	1	1
General Foreman	2	4	4	4	4	2	2	2	2
Foreman	3	9	9	9	9	3	3	3	3
Senior Project Engineer	2	2	2	2	2	1	1	1	1
Project Engineer	2	6	6	6	6	2	2	2	2
Safety Advisor	2	4	4	4	4	2	2	2	2
Site Administrator	1	2	2	2	2	1	1	1	1
QAQC Advisor	1	2	2	2	2	1	1	1	1
Heavy Equipment Trainer	2	3	3	3	3	1	1	1	1
Supervisor Blasting Services	2	2	2	2	2	2	2	2	2
Subtotal Mining Contractor Staff	**19**	**36**	**36**	**36**	**36**	**17**	**17**	**17**	**17**

Position	2021	2022	2023	2024	2025	2026	2027	2028	2029
Mining Manager	1	1	1	1	1	1	1	1	1
Open Pit Foreman	1	1	1	1	1	1	1	1	1
Chief Mining Engineer	1	1	1	1	1	1	1	1	1
Mine Engineer	1	1	1	1	1	1	1	1	1
Chief Geologist	1	1	1	1	1	1	1	1	1
Geologist	2	2	2	2	2	2	2	2	2
Senior Geotechnical Engineer	1	1	1	1	1	1	1	1	1
Geotechnical Engineer	0	1	1	1	1	1	1	1	1
Senior Surveyor	1	1	1	1	1	1	1	1	1
Surveyor	1	1	1	1	1	1	1	1	1
Technician, Geology	2	2	2	2	2	2	2	2	2
Technician, Mining/Survey	0	1	1	1	1	1	1	1	1
Clerk	1	1	1	1	1	1	1	1	1
Safety and Training	1	1	1	1	1	1	1	1	1
Subtotal BMC Staff	**14**	**16**	**16**	**16**	**16**	**16**	**16**	**16**	**16**
TOTAL	**149**	**181**	**223**	**223**	**223**	**132**	**132**	**132**	**116**

16.5 Underground Mining

16.5.1 Underground Mining Summary

The KZK Project comprises two mineralized zones, ABM and Krakatoa. Underground mining has only been considered for the Krakatoa Zone as the majority of the ABM Zone is extracted using open pit mining methods. Insufficient mineralization has been defined, and complex geotechnical conditions exist below the current ABM pit design to justify underground mining in this area at present.

The primary mining method planned is underhand longhole stoping with cemented paste fill for footwall and hangingwall mining areas. Development of the underground mine is planned to commence in November 2024, once the ABM Stage 1 open pit has progressed to the 1,355 mRL bench and excavated the portal area. Underground operations are scheduled for completion in October 2029. Figure 16-8 presents a long section of the Krakatoa underground mine.

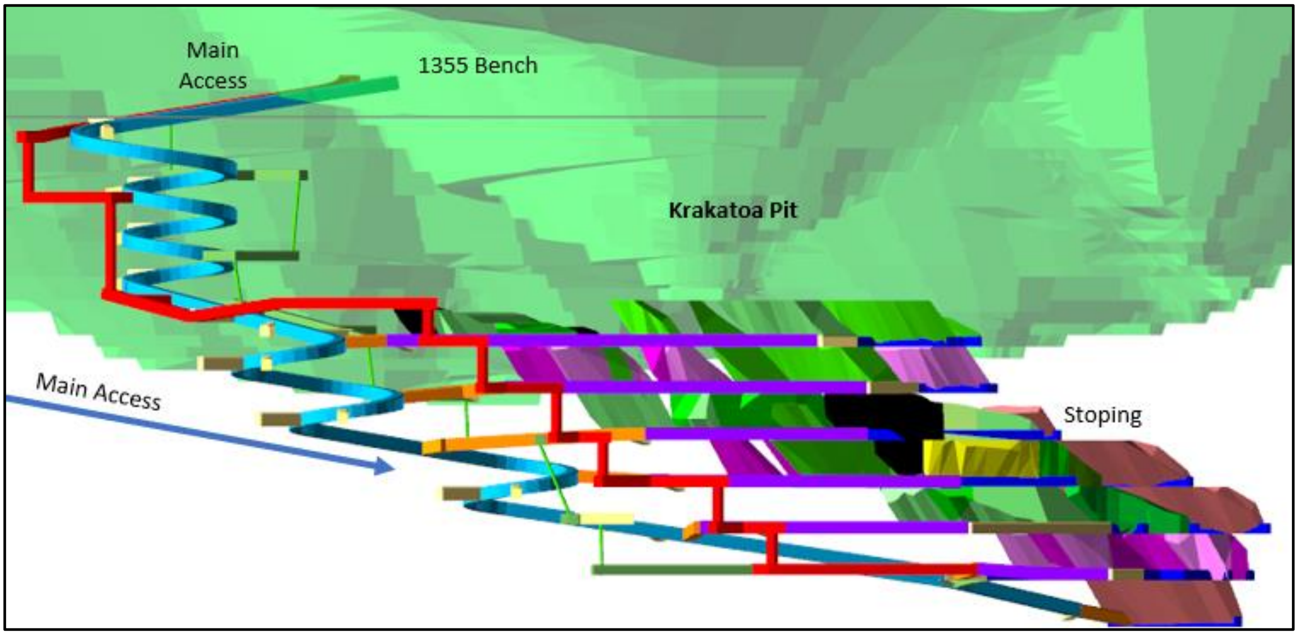


Figure 16-8: Long section of the Krakatoa underground mine (looking west)

16.5.2 Geotechnical Assessment

The underground geotechnical study was undertaken by D&S during 2018 and 2019. Data was collected for the Krakatoa underground study and incorporated into the overall KZK geotechnical database and Mining Rock Mass Model that covers the ABM and Krakatoa open pits.

Eleven mine scale geotechnical structural features were identified by D&S (2019b) that impact the planned underground mining zone and are presented in Table 16-11. The fault structures impacting the underground development and stope designs are of variable thickness and are generally greater than 3 m and up to 30 m in width. All the structures have greater than 300 m strike and dip extents and extend from the base of the overburden through the proposed underground mining zone.

Table 16-11: Geotechnically significant structures impacting the Krakatoa underground (D&S, 2019b)

Structural label	Structure ID	DTM/Str filename	Dip°/ dip direction	Thickness (m)	Strike extent (m)	Dip extent (m)	Structure/zone characteristic
NE1	GT01	kzk_gt_ne_01_trimmed	75 / SE (145)	approx. 3	800	300	Fault, partly gouge
NE2	GT02	kzk_gt_ne_02_trimmed	75 / SE (145)	approx. 10 to 30	900	300	Fault, partly gouge
NNE1	GT05	kzk_gt_nne_05_trimmed	65 / ESE (105)	>3 to 10	800	400	Fault, partly gouge, micro fracture
NNE2	GT06	kzk_gt_nne_06_trimmed	60-65 / ESE (110)	>3 to 10	800	500	Fault, partly gouge, micro fracture
NNE3	GT07	kzk_gt_nne_07_trimmed	55-65 / ESE (110)	>3 to 10	800	500	Fault, partly gouge, micro fracture
NNE7	GT11	kzk_gt_nne_11_trimmed	50-55 / ESE (115)	approx. 2	400	500	Fault, partly gouge, micro fracture
EW1	GT12	kzk_gt_ew_12_trimmed	50 / N	>3 to 20	300	300	Strongly foliated, fractured, interlayered gouge

Structural label	Structure ID	DTM/Str filename	Dip°/ dip direction	Thickness (m)	Strike extent (m)	Dip extent (m)	Structure/zone characteristic
EW2	GT13	kzk_gt_ew_13_trimmed	40-50 / N	>3 to 10	600	600	Strongly foliated, fractured, interlayered gouge
EW17	GT28	kzk_gt_ew_28_trimmed	50-55 / N	approx. 4	300	300	Strongly foliated, fractured, interlayered gouge
East Fault		east_fault_20171110	70-80 / SE (150)	approx. 30	900	700	Fault, partly gouge – these structures are modified from the original model
Creek Fault		creek_fault_20171110	75-90 / NW (315)	15 to 80	2000	600	

Figure 16-9 presents the 1,225 mRL level relative to the main faults influencing mining on that level.

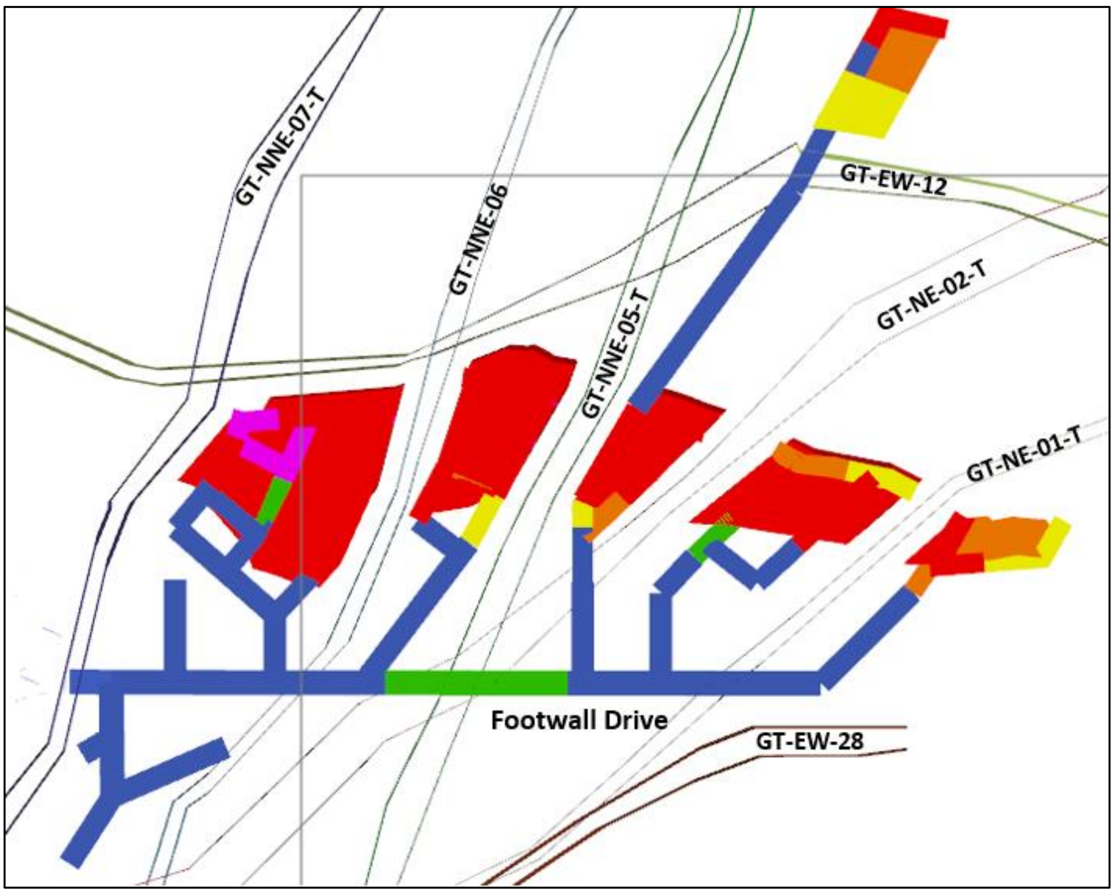


Figure 16-9: 1,225 mRL level showing main influencing faults

Rock mass domains were established based on the Norwegian Geotechnical Institute Q-Index. For each of these, ground condition characteristics and recommended support regimes where determined (Dempers & Seymour, 2019b) and are summarized in Table 16-12.

Table 16-12: Generic rock support by rock mass domain (Dempers & Seymour, 2019b)

Rock mass domain	Q-range	Ground conditions	Recommended support regimes
D1	<0.1	Exceptionally and Extremely Poor	FRS 120 mm with fibrecrete arches, screen and MD bolts with yielding cable bolts – develop with short rounds and spiling bars
D2	0.1–1	Very Poor	FRS 100 mm with MD bolts and yielding cable bolts – develop with short rounds and spiling rounds
D3	1–4	Poor	FRS 75/50mm on backs/walls, with grouted DCP bolts
D4	4–10	Fair	Screen and grouted DCP bolts
D5	>10	Good	

Note: FRS – fibre reinforced shotcrete, DCP – double corrosion protected, MD – mechanical dynamic.
Note that the DCP bolt is not currently available in Canada and the CT-bolt is recommended to be used in its place.

16.5.3 Hydrogeology

Hydrogeological investigations and groundwater modelling (Section 20.2.4) indicate a discharge rate of 13 L/s from the Creek and East faults, which are bounding structures to the underground mining area. Combined with groundwater inflows from the general rock mass through the mine workings an overall dewatering requirement of up to 50 L/s in the first few years of underground mine life is modelled, decreasing to a longer-term average of around 20 L/s, as estimated by Tetra Tech (2019). Groundwater inflows may impact the underground mining conditions and so the ground support recommendations assume saturated conditions.

16.5.4 Crown Pillar Mining

The planned sequence of mining will see the upper levels of the underground mine extracted and paste filled, and the open pit mined down towards this paste pillar later in the project. The following recommendations are provided from the Geotechnical report (Dempers & Seymour, 2019b):

- Stope voids, as well as any access development within 20 m of the final pit walls are to be tight filled with cemented paste fill
- Any stope voids or development within 15 m of the pit floor are to be probed drilled from the pit and treated as voids during pit production
- A monitoring program for the pit walls will be required to monitor and assist the planning and implementation of any mitigation strategies, where large wall movements are detected, or the potential indicated by the monitoring.

16.5.5 Cut-Off Grade Value

The NSR was used to determine whether the mineralized material met the following criteria:

- ICOG – Incremental Cut-off Grade: Measured and Indicated Mineral Resource that covers underground production mining, load and haul, direct processing, and general and administration (G&A) costs.
- FCCOG – Fully Costed Cut-off Grade: Measured and Indicated Mineral Resource that covers all operating costs associated with mining including development mining and outstanding contractor costs such as mobilization and demobilization, indirect processing, dayworks, BMC Management costs (all other costs that have not been capitalized).

The estimated cut-off values for each criterion is provided in Table 16-13.

Table 16-13: Cut-off value calculations

Parameter	CAD$/total underground ore tonne
Mining – LHOS with Cemented Pastefill	$50.70
Treatment – Direct	$16.03
Treatment – Indirect	$5.94
G&A	$10.78
ICOG (Production Mining, Direct Mill, L&H, Paste Fill)	**$66.72**
Mobilization, Demobilization, Development, Ground Support, Dayworks, BMC, G&A	$100.16
FCCOG	**$172.83**

The mining costs applied to the cut-off value calculations are sourced from the RFQ process undertaken for the KZK DFS and provided by a reputable mining contractor. Processing and G&A were provided from the DFS cost model.

The fully costed cut-off value was applied to the NSR and used to determine the lateral and depth extents of the mine and includes all site operating costs, and capitals costs associated with the underground mining.

An incremental cut-off value was applied to the NSR and used to identify potentially economic material once development was in place, that covers the costs associated with stoping, load and haul, and direct processing costs.

16.5.6 Mine Access

Access to the underground mine will be via a single ramp with portal at the 1,355 mRL bench from the saddle area within stage 1 of the ABM open pit. Access to the underground portal area will become available in the first half 2023.

Underground trucks will haul ore and waste out of the mine and dump adjacent to the portal, where the open pit load and haul fleet will re-handle the material to the ROM pad or waste storage facility as appropriate.

A second ramp is proposed for ventilation of the underground workings as there is a significant layer of overburden material overlying the competent bedrock. This makes raise boring or conventional raising difficult and uneconomic, given the likelihood that shaft sinking would be required through the overburden before commencing vertical development. The second ramp will also act as the second means of egress from the underground workings if the main ramp becomes blocked for some reason. The dual ramp configuration is shown in Figure 16-10 and Figure 16-11.

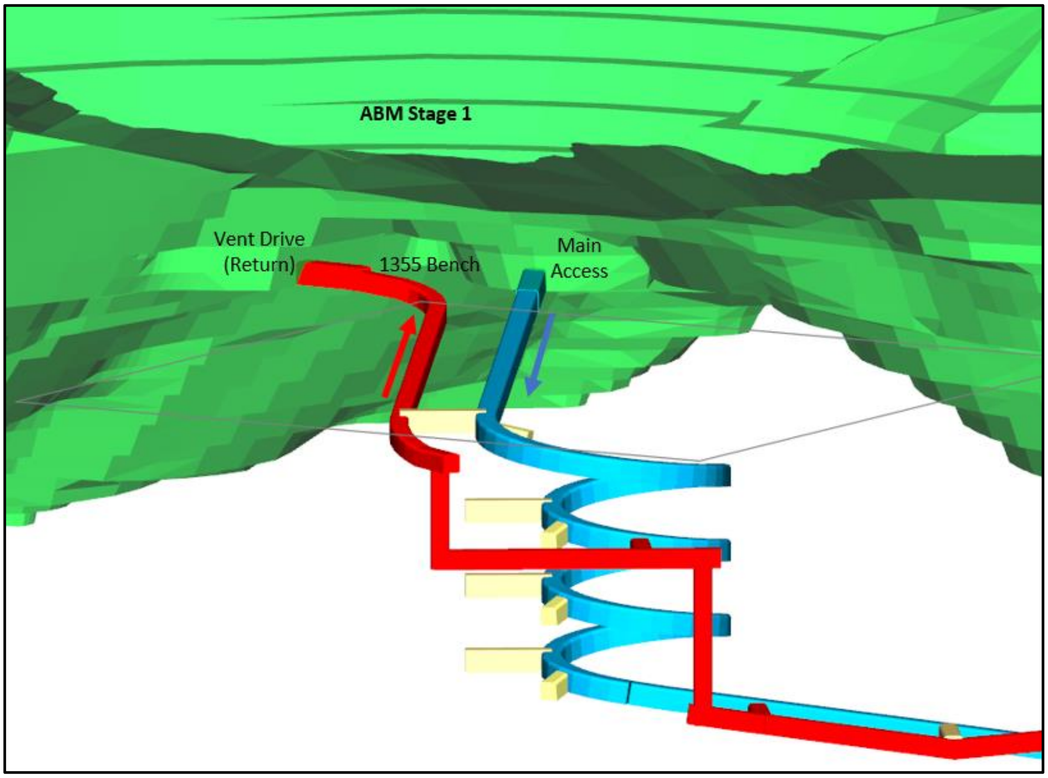


Figure 16-10: Main access and ventilation ramp, viewed looking northeast

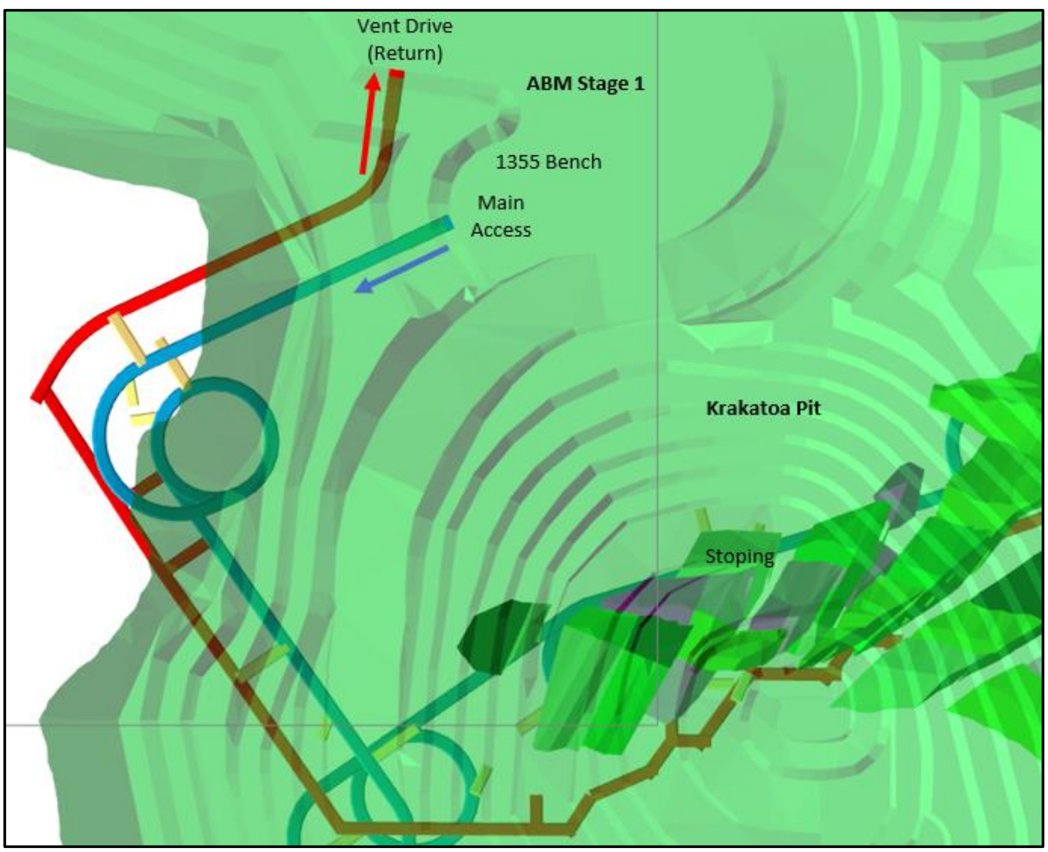


Figure 16-11: Main access and ventilation ramp, plan view

Development profiles used at Krakatoa are presented in Table 16-14.

Table 16-14: Development profiles

Development type	Profile no.	Width (m)	Height (m)
Ramp	PL01	5.5	5.8
Level Access, Footwall Drift	PL02	5.5	5.5
Stockpile	PL03	5.0	6.0
Crosscut, Return Air Drifts	Pl04	5.0	5.0
Ore Drifts, Escapeway Drift	PL05	4.5	4.5
Sump	PL06	5.0	4.5
Infrastructure Development	PL07	8.0	5.0
Initial Ramp (first 20 m from Portal)	PL08	6.1	6.1

Development will use standard mechanized underground mining equipment and methods. Utilities including compressed air, water and dewatering pump lines will be installed as headings progress and electrical cables and paste backfill lines will be installed as required. Air and water pipelines will be 4” diameter polylines in the decline. For dewatering, an 8” sched 40 pipe is planned in the decline to provide enough capacity to manage estimated underground inflows as presented in the water study (Tetra Tech, 2019).

Underground conditions for rock mass domain D1, D2 and D3 are expected to range from poorly consolidated through to blocky. These types of ground conditions are expected to be wet, based on interpretation of the groundwater regime, and will remain so until they have been dewatered by progressive mining and/or borehole extraction (Dempers & Seymour, 2019b).

If excavated ground within domains D1, D2 and D3 are not adequately supported, then it may be susceptible to time dependent loosening and deterioration, making fibrecrete based support the most suitable short-term solution. Table 16-15 provides detailed support designs recommended by D&S for the Krakatoa underground mine.

Table 16-15: Detailed ground support design for Krakatoa underground (Dempers & Seymour, 2019b)

Drift type	Height (m)	Width (m)	Profile	Support domain	Ground support profile	Surface support	FRS m³/m advance	Mesh m²/m advance	Bolt length (m)	Bolts/m advance	Bolts/ ring	Bolt pattern	Cable bolts/m advance
Decline	5.8	5.5	PL-01	D1	GSP1	FRS 120 mm and mesh to floor	2.6[A]	16.2	3.0	11.8[M *]	13	1.3 x 1.1	2.0[Y6]
				D2	GSP2	FRS 100 mm to floor	1.6	-		8.1[M *]	12 / 11	1.3 x 1.3	3.0[C1]
				D3	GSP3	FRS 75/50 mm to 1 m off floor	0.9	-		5.7[D]	9 / 8	1.5 x 1.5	-
				D4	GSP4	Mesh to 1.8 m from floor	-	12.6		10.0[D]	13	1.1 x 1.3	-
Access and Footwall Drifts	5.5	5.5	PL-02	D1	GSP5	FRS 120 mm to floor and mesh to ~0.5 m from floor	2.3[A]	14.7	3.0	11.5[M *]	15	1.1 x 1.3	1.67[Y5]
				D2	GSP6	FRS 100 mm to floor	1.6	-		8.1[M *]	11 / 10	1.3 x 1.3	3.0[C1]
				D3	GSP7	FRS 75/50 mm to 1 m off floor	0.8	-		5.0[D]	8 / 7	1.5 x 1.5	-
				D4	GSP8	Mesh to ~1.5 m from floor	-	12.6		10.0[D]	13	1.1 x 1.3	-
Stockpiles	6.0	5.0	PL-03	D3	GSP9	FRS 75/50 mm to 1.5 m off floor	0.8	-	3.0	5.7[D]	9 / 8	1.5 x 1.5	-
				D4	GSP10	Mesh to 2.3 m off floor	-	10.5		8.5[D]	11	1.3 x 1.1	-
Crosscuts and Return Air Drifts	5.0	5.0	PL-04	D1	GSP11	FRS 120 mm to floor and mesh to ~0.7 m from floor	2.0[AS]	12.6	2.4	10.0[M *]	13	1.1 x 1.3	1.67[Y5]
				D2	GSP12	FRS 100 mm to floor	1.4	-		7.3[M *]	10 / 9	1.3 x 1.3	3.0[C1]
				D3	GSP13	FRS 75/50 mm to 1 m off floor	0.8	-		5.0[D]	8 / 7	1.5 x 1.5	-
				D4	GSP14	Mesh to 1 m off floor	-	12.6		8.5[D]	13	1.1 x 1.3	-
Ore and Escapeway Drifts	4.5	4.5	PL-05	D1	GSP15	FRS 120 mm to floor and mesh to ~0.6 m from floor	1.8[AS]	12.6	2.1	10.0[M *]	13	1.1 x 1.3	1.3[Y4]
				D2	GSP16	FRS 100 mm to floor	1.3	-		6.5[M *]	9 / 8	1.3 x 1.3	2.33[C]
				D3	GSP17	FRS 75/50 mm to 1 m off floor	0.7	-		5.0[D]	8 / 7	1.5 x 1.5	-
				D4	GSP18	Mesh to grade (~1.2 m from floor)	-	10.5		8.5[D]	11	1.1 x 1.3	-

Drift type	Height (m)	Width (m)	Profile	Support domain	Ground support profile	Surface support	FRS m³/m advance	Mesh m²/m advance	Bolt length (m)	Bolts/m advance	Bolts/ ring	Bolt pattern	Cable bolts/m advance
Sump	4.5	5.0	PL-06	D1	GSP19	FRS 120 mm to floor and mesh to ~0.4 m from floor	1.9[AS]	12.6	2.1	10.0 [M] *	13	1.1 x 1.3	1.3[Y4]
				D2	GSP20	FRS 100 mm to floor	1.3	-		7.3[M] *	10 / 9	1.3 x 1.3	3.0[C1]
				D3	GSP21	FRS 75/50 mm to 1 m off floor	0.7	-		5.0 [D]	8 / 7	1.5 x 1.5	-
				D4	GSP22	Mesh to grade (~1.2 m from floor)	-	10.5		8.5 [D]	11	1.1 x 1.3	-
Infrastructure Development	5.0	8.0	PL-07	D3	GSP23	FRS 75/50 mm to 1 m off floor	0.99	-	3.0	9.6[D]	13 / 12	1.3 x 1.3	3.0[C]
				D4	GSP24	Mesh to grade (~1.2 m from floor)	-	14.7	3.0	11.5 [D]	15	1.1 x 1.3	3.0[C]
Initial Declines (from Portal)	6.1	6.1	PL-08	D3 / D4	GSP25	FRS 120 mm to floor and mesh to ~0.9 m from floor	2.8[A]	16.2	3.0	11.8 [D] *	13	1.3 x 1.1	2.0[C6]

Notes:
[A] With double fibrecrete arches at 3 m spacing or [AS] single fibrecrete arches.
** Allow for additional spiling bolts at 1 m spacing around profile (friction bolts).*
[C6] Twin strand bulbed cable bolts installed in-cycle; 6 bolts 2.5 m spaced in rings 3 m apart between FRS arches.
[C] Twin strand bulbed cable bolts installed in-cycle; 3 m x 1.5 m ring spacing, staggered rows of 4-3-4 bolts ([C1] rows of 5-4-5).
[Yx] Yielding cable bolts installed in-cycle; 3 m x 3 m ring spacing; where x = 4, 5 or 6 bolt rings between FRS arches.
Bolt types - [D] DCP bolts, full column post-grouted solid bolts; except [M] Mechanical Dynamic bolts.
Bolt pattern given as bolt spacings in: Bolt Ring x Ring spacing.

16.5.7 Underground Production

Mineable stopes were manually designed based on orebody thickness and orientation, considering geotechnical limitations determined by Dempers & Seymour (Dempers & Seymour, 2019b), the proposed stoping method and the practical limits of the proposed equipment. The stopes were reviewed for economic viability by comparing estimated NSR value to the estimated cut-off requirements, after applying estimated mining factors for dilution and recovery. Those stopes not meeting economic criteria were removed from the stoping inventory.

The mining method selected for underground mining at Krakatoa is underhand longhole stoping with cemented paste fill. A representative mining sequence of this mining method is presented in Figure 16-12. The method chosen is a top down mining sequence and respects the geotechnical constraints of the ground conditions. Level spacing is 20 m with 4.5 mW x 4.5 mH ore drifts. Longhole stoping commences with development of an upper and lower sill drift to the limits of the stoping block. A slot rise is then drilled up to the level above and blasted to create a void space to allow for mining the rest of the stoping panel. Once the panel is fully excavated, the void will be filled with paste backfill to allow mining the next stope in the sequence, with 100% extraction of the economic material.

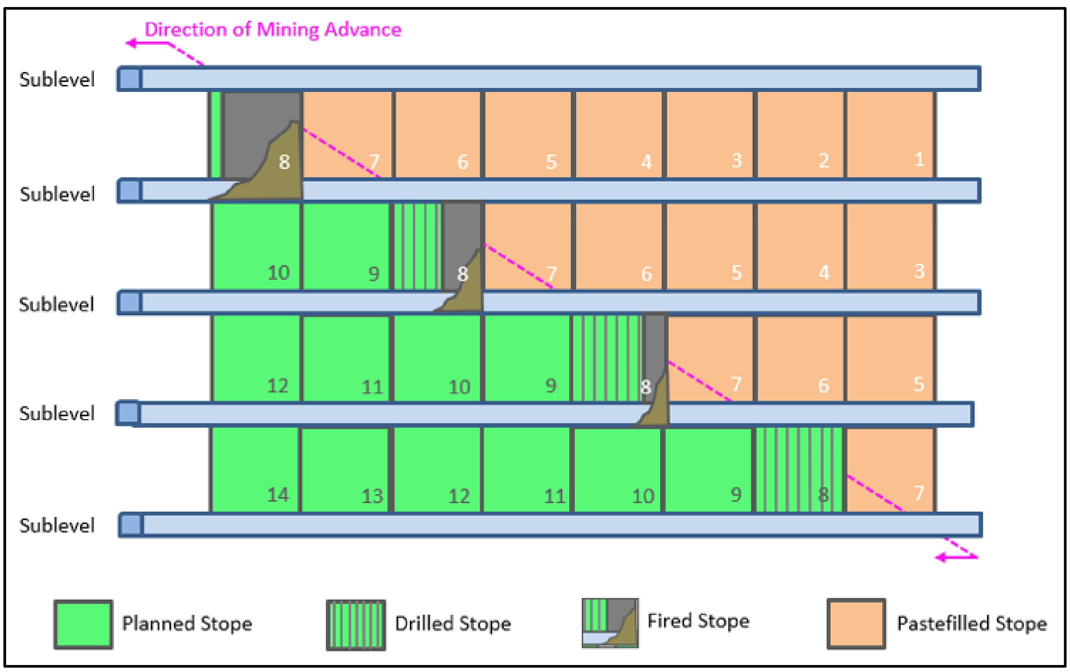


Figure 16-12: Example of stoping sequence

Stoping assumes 90% recovery from stope panels. Planned dilution is included in stope wireframing. Unplanned dilution is variable, depending on the geotechnical domain of the rock the stope is within, and is assumed to occur at zero grade. These factors are presented in Table 16-16.

Table 16-16: Stoping dilution by domain

Stoping dilution					
Domain	D1	D2	D3	D4	D5
Dilution	50%	35%	25%	20%	15%
Mining recovery	90%	90%	90%	90%	90%

16.5.8 Paste Backfill

Outotec (2018) has completed the test work program necessary to confirm the technical feasibility of paste the backfill plant to a DFS level of accuracy. Paste backfill will be placed in mined stopes to fill voids and assist with ground control. The backfill will be hydraulically placed (piped) as opposed to transported by truck.

Analysis of horizontal and vertical exposure while mining under and beside a paste filled stope was calculated and presented in Table 16-17. This high strength paste has been allowed for in the cost estimate and will be used for the lower half of all stopes to be undercut by the stope below. For the remainder of the stope, and all other stopes that will not be undercut, regular strength paste is planned at an addition rate of 60 kg/m^3 binder.

Table 16-17: Exposure analysis summary (Outotec, 2018)

Description	Man entry	28-day hydration		Estimated dilution (m^3)
		Estimated strength required (kPa)	Estimated contained binder (kg/m^3)	
15 m horizontal exposure	No man entry	1,950	145	1,000
5–15 m vertical exposure (from lateral longhole stoping	No man entry	150 - 240	30 - 40	0

The paste fill plant will be located on surface adjacent to the Process Plant. Three agitator trucks will transport the paste to the portal before dumping into a hopper and piped underground to the 1,265 mRL cuddy. Paste is then piped along the Escapeway Ramp to the 1,245 mRL level. At this point, the paste line will split. One line will lead onto the 1,245 mRL level and the other will continue down the Main Ramp to the lower areas of the mine, servicing each level it passes.

Schedule 40 pipe has been selected for distributing the paste. Permanent paste reticulation has been allowed for in the Main Ramp, level access and footwall drifts. For all stopes, 50 m of temporary HDPE line is provided per stope for paste to reticulate through the level from the footwall drift.

Cemented paste fill requirement within the schedule has been capped at 700 m^3 per day.

Paste fill will be contained within the stope by constructing a fill bulkhead in the access as close to the stope brow as possible. The curing time after each paste filled stope is scheduled at a minimum of 14 days lag to ensure required paste fill strength of 300 kPa is attained before mining the adjacent stope.

16.5.9 Underground Mine Schedule

The underground mine is scheduled to commence November 2024 after the ABM stage 1 open pit has established the portal area on the 1,355 mRL bench. The underground schedule will finish 60 months later in October 2029. The underground mine will produce at a variable rate over its life, peaking at 584,000 tonnes of ore in 2027.

The underground mine schedule was developed using Deswik software. A base case schedule was created linking all development and stoping related tasks in line with the geotechnical recommendations for the mining methods applied. These included the following:

- All longhole stopes being mined top down (underhand) and retreating to the accessing crosscut
- Lead/lag between sublevels (vertically) to prevent any safety risks from premature undercutting of the above stope.

The schedule is designed to ensure a smooth ramp up to production and, where possible, minimize variations in development rates and production, to avoid additional project costs due to underutilization of the contractor's equipment.

Development rates are scheduled according to the geotechnical domain of the rock that the development occurs within (Table 16-18), and considers the ground conditions, round length and required ground support.

Table 16-18: Deswik schedule task rates

Task	Units	Rate
Lateral Development Domain 1	m/month	N/A
Lateral Development Domain 2	m/month	45
Lateral Development Domain 3	m/month	75
Lateral Development Domain 4	m/month	90
Lateral Development Domain 5	m/day	N/A
Stope Mucking Rate	t/month	22,000
Escapeway Raise	m/day	3.5
Boxhole Slot	m/day	3.0
Return Air Rise (Raisebore then D&B Strip)	m/day	2.5
Production Drilling (89 mm)	drm/day	200
Cemented Paste Fill	m^3/day	700

The principal aim of the mine schedule is to build enough detail into sequencing the mine and applying operational constraints such that the schedule represents what is reasonably achievable in practice, thereby giving confidence the production objectives can be met.

The annual summary of the underground mining schedule is detailed in Table 16-19 and graphically in Figure 16-13. The tonnes and grade presented are inclusive of mining dilution and recovery.

Table 16-19: LOM underground mining annual summary

Mining summary	Units	LOM	Year					
			2024	2025	2026	2027	2028	2029
Capital Lateral Development	m	4,013	343	1,382	1,699	589	-	-
Operating Lateral Development	m	5,823	-	-	3,424	2,172	152	75
Total Lateral Development	**m**	**9,836**	**343**	**1,382**	**5,122**	**2,761**	**152**	**75**
Total Vertical Development	m	2,985	-	151	521	1,071	860	381
Production Drilling 89 mm	m	215,409	-	-	28,653	75,692	76,991	34,074
Cemented Paste Fill	m^3	440,143	-	-	50,141	142,121	153,706	94,175
Development Ore Tonnes	Mt	0.2	-	-	0.1	0.1	0.0	0.0
Longhole Stope Ore Tonnes	Mt	1.5	-	-	0.2	0.5	0.5	0.3
Total Ore Tonnes	**Mt**	**1.7**	**-**	**-**	**0.3**	**0.6**	**0.5**	**0.3**
Total Waste Tonnes	Mt	0.6	0.0	0.1	0.3	0.1	0.0	0.0
Total Mined Tonnes	**Mt**	**2.4**	**0.0**	**0.1**	**0.6**	**0.7**	**0.6**	**0.3**
Ore Grades								
Zn (%) Ore	%	5.0	-	-	4.9	5.2	5.2	4.6
Cu (%) Ore	%	0.4	-	-	0.4	0.4	0.5	0.4
Pb (%) Ore	%	2.3	-	-	2.6	2.5	2.2	1.7
Au (g/t) Ore	g/t	1.3	-	-	1.3	1.2	1.3	1.2
Ag (g/t) Ore	g/t	147	-	-	142	149	153	135

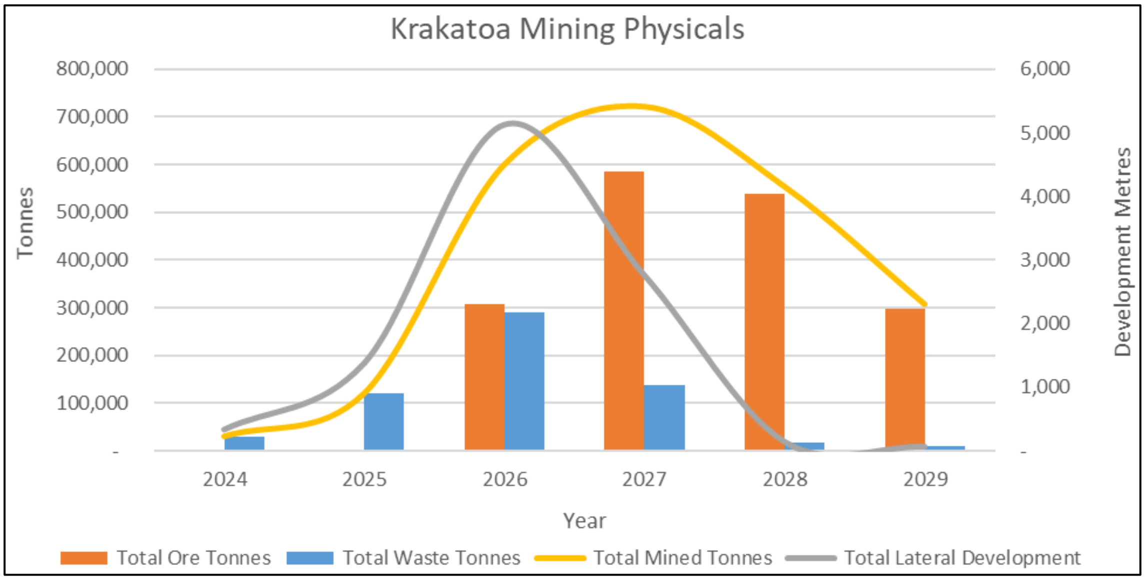


Figure 16-13: Krakatoa mining physicals

16.5.10 Portal Infrastructure

Infrastructure located at the Main Ramp portal is shown in Table 16-20.

Table 16-20: Portal infrastructure

Item	Description
Basic shelter/hut	Fitted with the following: • Tag board • VHF radio • Mine plans and basic emergency procedures for use during emergency/rescue operations • Lighting • Primary ventilation indicator, showing primary fan status • Stench gas cylinders and release point in case of emergency.
Intake heating equipment	Heater, fan, mixing box and fuel storage for mine air heating unit.
Ore and waste re-handle areas	Area provided for ore and waste delivered from underground, before re-handling to surface haulage fleet for transfer to ROM.
Traffic and miscellaneous signage	As per site traffic management plan, appropriate signage delineating traffic flows.
Parking	Light vehicle and heavy vehicle parking.

16.5.11 Water

The underground dewatering system will consist of two pumps located at or near the 1,205 mRL with WTX3 pumps fed from a settling sump just above in the Decline. These pumps, at maximum performance, can pump up to 28 L/s each, at a total head of up to 180 m allowing water to be transferred directly to the Pit Rim Pond. Alternatively, if water can be re-handled more cost effectively with larger pit pumps, the WTX3 pumps could be moved lower in the mine. A single 8-inch dewatering line is planned in the Main Ramp for managing the expected water flow.

The contractor will provide an additional two travelling pumps that can be used to centralize the water back to the pump station. Given that the water ingress locations are not firmly known and the relatively short mine life, helical rotor pumps will provide a flexible solution to remove water from the mine as they can be easily relocated to the source of the water and connected to the Main Ramp pump-line.

Minor sumps around the mine will have 8 kW submersible pumps (or similar) as needed to meet the water removal requirements. This water will be pumped in "daisy chain' configuration to the travelling helical rotor pumps provided by the contractor then up to the main pump station.

16.5.12 Primary Ventilation

Fresh air will be drawn into the mine via the Main Ramp and return to surface by a dedicated rise system. This dedicated return air system comprises inter-linking return air drifts and connecting vertical development, returning to surface via a second portal access to the pit (Figure 16-10 and Figure 16-11). Three primary exhaust fans are to be mounted in a bulkhead positioned nominally 20 m inside the return air portal located on the 1,355 mRL bench. The bulkhead will also be fitted with a man door to provide secondary egress.

These primary fans selected are a single-stage 132 kW Clemcorp CC1400 axial fan. The wall is installed downwind, allowing inspection and maintenance from the pit entry side. Each fan is fitted with an inlet cone, self-closing fan isolation doors, wall adaptor duct, flexible joint, mounting feet and short evase.

The Yukon climate requires that the air entering the portals be heated during the coldest part of the year. This will be achieved with a portal fan which heats air using diesel fired burners and then injects the air into the intake airway using a 75 kW fan.

The capacity of the primary ventilation system has been designed to meet the required ventilation standards, based on the operating equipment. A computer simulation (Ventsim™) analysis of the primary ventilation circuit confirmed that sufficient airflow will be provided by the ventilation network.

16.5.13 Auxiliary Ventilation

Axial fans, suspended from the backs, will fulfil secondary ventilation requirements. The fans will be of appropriate size, nominally with motor in multiples of 55 kW, 90 kW or 110 kW. This will provide fans sizes of 55 kW, 90 kW, 110 kW, 180 kW or 220 kW, depending on the ventilation requirement on that level, with sufficient volumes to supply all working headings. Typically, 1,400 mm ducting will be run off the Main Ramp onto the level and reduce to 1,220 mm ducting leading into the footwall drift, with T-pieces branching into active crosscuts (up to five) which will be only around 150 m from the footwall drift. Where multiple headings branch from the same ducting in the footwall drift, inactive headings will require tie-offs to ensure adequate flow to active work areas.

A single access drift from the footwall zone to the hangingwall zone is designed and will require a dedicated duct from the Main Ramp and will have around a 350–400 m maximum length. A Sandvik 517 loader will be typical of the loader used and requires approximately 20 m^3/s. Even with pressure loss and leakage, the axial fans will be sufficient to push this volume of air to the hangingwall heading on each level.

16.5.14 Compressed Air

Compressed air will be required for shotcrete application, refuge chambers, charging, production drilling and various miscellaneous work. A compressor will be situated outside the Main Portal initially to commence underground mining. The compressor will be moved down lower into the mine once a suitable location becomes available. The recommended location is the 1,305 mRL return air drift, as fresh air and access to the Main Ramp services is readily available, as is the potential to create return flow over the compressor directly into the return air system should a fire occur. The RFQ to mining contractors specified a 200 kW compressor for the underground mine.

16.5.15 Electrical Power

The Power Station complex (Section 18.4) will supply power to the underground mine. In Year 3 of the project distribution of power to the underground mine area will be facilitated through the installation of direct buried supply cable, installed on the western side of the valley, adjacent to the pit. High voltage (HV) power will be supplied to the underground mine at 13.8 kV, with 90 mm^2 HV cable extending between substations throughout the mine. Substations will typically be 2 MVA capacity and distribute low voltage (LV) power at 1,000 V, generally via 70 mm^2 cable between the substation and distribution boxes positioned near active working areas. LV will then be reticulated to working areas via 35 mm^2 cable to starter boxes for development drills, pumps and secondary ventilation. Substations will be positioned approximately 100 m vertically throughout the mine. Disused stockpiles will serve as permanent locations for substations.

16.5.16 Emergency Facilities

Refuge chambers will be installed in various locations to provide safe refuge for mining personnel when required. All refuge chambers will be fitted with radio communications, drinking water and breathing air sufficient for a minimum of 36 hours of refuge.

Portable four-person refuge chambers will also be provided by the mining contractor for use in single entry headings where the escape route is obstructed by heavy mobile machinery operating. Twelve-person refuge chambers will be installed in disused Main Ramp stockpiles, at various locations, to minimize the distance required to travel in an emergency. Walking distances to the nearest refuge chamber are not expected to be greater than 750 m.

In addition, a dedicated second means of egress (escape route) will be established. This will enable egress from the mine in the case that the main access (the ramp) becomes inaccessible. Escapeways raises are designed between 70° and 80° and are to be excavated using a raise bore. The total length of raise boring is estimated to be 192 m for the mine. Escapeways are planned to be Safescape Laddertube.

16.5.17 Underground Mining Equipment

Mine equipment has been selected based on a combination of industry experience, contractor tender and first principles. The equipment types have been determined following a review of the mine design and schedule and discussion with various contractors through the RFQ process. This represents the equipment and labour necessary to perform the following:

- Excavate lateral and ramp development in both ore and waste
- Install all ground support including rock bolting and surface support
- Maintain the underground road surfaces
- Drill, charge and muck (including remote mucking) of all stoping material
- Haul all muck to designated stockpiles on surface
- Install all underground services for development and production
- Install pipework and manage all activities underground related to paste filling.

A summary of the equipment requirements for the underground mine is detailed in Table 16-21. Equipment make and model are provided for reference purposes only and may be varied depending on the final binding agreement with the mining contractor.

Table 16-21: Underground mine equipment schedule

Equipment class	Make/Model (example only)	2021	2022	2023	2024	2025	2026	2027	2028	2029
2 boom jumbo drill	Sandvik DD421	0	0	0	1	1	2	2	1	1
Production drill	Sandvik DL311	0	0	0	0	1	1	2	2	2
LHD (17-t capacity)	Sandvik LH517i	0	0	0	1	1	2	2	2	2
Truck (45-t capacity)	Sandvik TH545i	0	0	0	1	1	3	3	3	2
Charge rig	Normet Charmec 605	0	0	0	1	1	1	1	1	1
Service vehicle	Volvo ITC L90F	0	0	0	1	1	1	1	1	1
Grader	Cat 12K	0	0	0	1	1	1	1	1	1
Fibrecrete sprayer	Normet Spraymec SF-050	0	0	0	1	1	1	1	1	1
Fibrecrete transmixer	Normet Utimec LF600	0	0	0	1	1	2	2	1	1
Surface agitator truck	Surface agitator truck	0	0	0	0	0	3	3	3	3
Light vehicle	Light vehicle	0	0	0	13	13	13	13	13	13

16.5.18 Underground Mining Labour

The work schedule assumes a 24 hours per day, 7 days per week and 365 days per year mining operation. Operations and maintenance personnel will work two 12-hour shifts per day. All mining personnel will work on a two week in/one week out rotation. A summary of the underground mining labour schedule is provided in Table 16-22.

Table 16-22: Underground mining labour schedule

Position	2021	2022	2023	2024	2025	2026	2027	2028	2029
Jumbo Operator	0	0	0	3	3	6	6	0	0
Loader Operator	0	0	0	3	3	6	6	6	6
Truck Operator	0	0	0	3	3	9	9	9	6
Production Driller	0	0	0	0	3	3	6	6	6
Charge Up	0	0	0	0	3	3	3	3	0
Nozzleman	0	0	0	3	3	3	3	3	3
Transmixer Operator	0	0	0	1	3	6	6	3	3
Grader Operator	0	0	0	0	1	1	1	1	0
Service Crew	0	0	0	0	1.5	1.5	1.5	1.5	1.5
Paste Crew	0	0	0	0	1.5	1.5	1.5	1.5	1.5
IT Operator	0	0	0	0	2	2	2	2	2
Magazine Keeper	0	0	0	1	3	3	3	3	3
Storeman	0	0	0	2	2	2	2	2	2
Subtotal Operators	**0**	**0**	**0**	**16**	**32**	**47**	**50**	**41**	**34**
Mechanical Foreman	0	0	0	1	1	1	1	1	1
Electrical Foreman	0	0	0	1	1	1	1	1	1
Electrical Tradesperson	0	0	0	2	3	3	3	3	3
Workshop Assistant	0	0	0	2	3	3	3	3	3
Maintenance Planner	0	0	0	1	1	1	1	1	0
Fitter	0	0	0	3	3	3	3	3	0
Intermediate Fitter	0	0	0	3	3	3	3	3	3
Workshop Assistant	0	0	0	2	3	3	3	3	3
Auto Electrician	0	0	0	1	3	3	3	3	3
Electrician	0	0	0	2	3	3	3	3	3
Boilermaker	0	0	0	1	1	1	1	1	1
Subtotal Maintenance	**0**	**0**	**0**	**19**	**25**	**25**	**25**	**25**	**21**

Position	2021	2022	2023	2024	2025	2026	2027	2028	2029
Project Manager	0	0	0	1	1	1	1	1	1
Administration Assistant	0	0	0	1	1	1	1	1	0
Health and Safety Manager	0	0	0	1	1	1	1	1	1
Trainer	0	0	0	3	3	3	3	3	3
Project Controls	0	0	0	2	2	2	2	2	1
Accounts Clerk	0	0	0	1	1	1	1	1	1
Mining Engineer	0	0	0	0	3	3	3	3	3
Subtotal Contractor Staff	**0**	**0**	**0**	**9**	**12**	**12**	**12**	**12**	**10**
Underground Superintendent	0	0	0	1	1	1	1	1	1
Mine Engineer	0	0	0	1	1	1	1	1	1
Geologist	0	0	0	2	2	2	2	2	2
Surveyor	0	0	0	1	1	1	1	1	1
Surface Agitator Operator	0	0	0	0	0	9	9	9	9
Subtotal BMC Staff	**0**	**0**	**0**	**5**	**5**	**14**	**14**	**14**	**14**
TOTAL	**0**	**0**	**0**	**49**	**74**	**95**	**95**	**89**	**77**

17 Recovery Methods

The Kudz Ze Kayah Process Plant and associated facilities has been designed to process ROM ore at a rate of 2.0 Mt/a to produce separate copper, lead, and zinc concentrates and tailings; however, the plant will be capable of processing at 270 t/h based on average ore comminution properties and average plant feed grades. The process rate will be varied depending on the grade of the ore.

The process flowsheet consists of following key stages:

- Crushing, stockpiling and grinding of the ore.
- Pre-float, rougher and cleaner flotation of copper, including regrind of copper rougher concentrate.
- Sequential pre-float, rougher and cleaner flotation of lead, including regrind of lead rougher concentrate.
- Sequential pre-float, rougher and cleaner flotation of zinc, including regrind of zinc rougher concentrate.
- Thickening, filtration, and stockpiling on site of copper, lead, and zinc flotation concentrates. Copper and zinc concentrates will be loaded in bulk onto trucks for transport to port, while lead concentrate will be loaded into sealable containers before transport by truck to port.
- Dewatering of flotation tailings by thickening and pressure filtration.
- Transportation of filtered flotation tailings to the Class A Waste Storage Facility for disposal.
- The overall process flow diagram is provided in Figure 17-1.

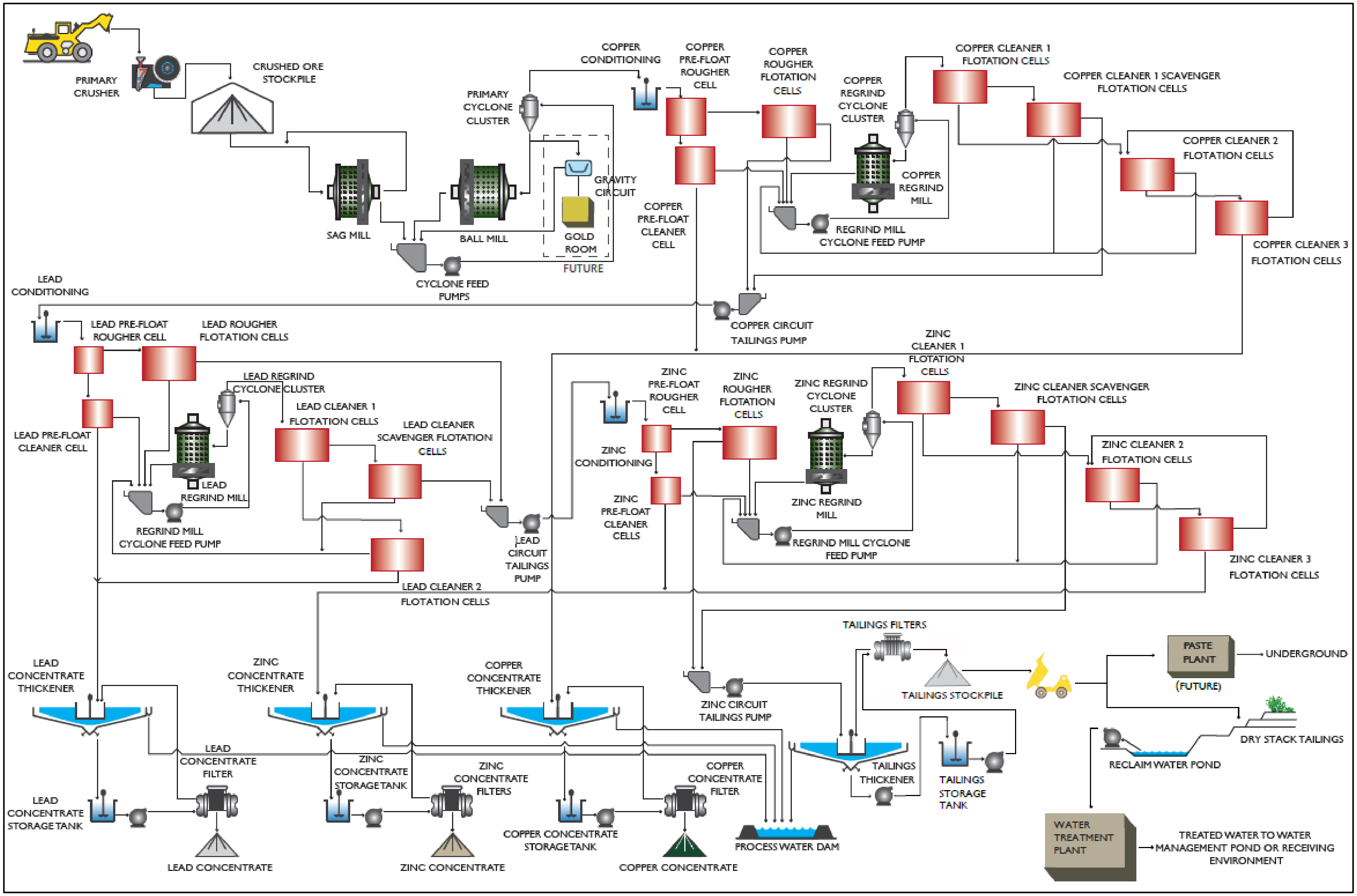


Figure 17-1: Overall process flow diagram

17.1 Process Design Criteria

A process design criteria (PDC) and mass balance detail was developed on the basis of annual ore production. The PDC considered major flows and availability of processes within the facility.

Three cases were considered in the design, based on the grade distribution from the PFS (CSA Global, 2017) processing schedule and comminution data:

- Design Case: 2.0 Mt/a at 95th percentile of copper and lead grades, 99th percentile of zinc grade (85th percentile considered to provide inadequate design contingency) and 80th to 85th percentile of comminution data, as noted below.
- Average Case: 2.2 Mt/a at average copper, lead and zinc grades and average comminution data.
- Alternate Design Case: Design Case, but with the production of lower-grade concentrates.

A summary of the key process design criteria is given in Table 17-1, showing the performance of the plant for the Design Case and the Average Case, being the two key scenarios. Operating parameters across all three cases were considered in equipment sizing in the flowsheet.

The plant will have a design availability of 93% (after ramp-up) which was considered appropriate for a plant of this type and in this location. To support this, all essential pumps will have operational standby units and there is provision for non-essential pumps to have standbys installed as a risk mitigation if shown to be required.

Selection and sizing of the crushing and grinding circuits was based on a conservative assessment of all test work and historical data available. The comminution parameters used in the Design Case are:

- The 80th percentile value of the available data set of Bond Ball Mill Work indices
- The 85th percentile value of the available data set of Bond Rod Mill Work indices
- The maximum A x b value from the SMC Test results.

Ore will be blended from ROM stockpiles with a front-end loader (FEL) to feed the ROM bin.

Table 17-1: PDC summary

Criteria		Units	Design	Average	Source
Ore throughput		Mt/a	2.0	2.2	BMC
		t/h	245.5	270	Calc.
Plant availability	Design	%	93.0	93.0	BMC, Minnovo
Head grades		% Cu	1.12	0.82	BMC PFS mine schedule
		% Pb	2.33	1.62	BMC PFS mine schedule
		% Zn	6.52	5.52	BMC PFS mine schedule
Concentrate grades	Copper	% Cu	25	24	Testwork
	Lead	% Pb	52	50	Testwork
	Zinc	% Zn	52	50	Testwork
Recovery		% Cu	84	82	Testwork
		% Pb	76	72	Testwork
		% Zn	90	88	Testwork
Physical characteristics	BWI	kWh/t	12.8	11.8	Testwork
	RWI	kWh/t	10.4	8.9	Testwork
	AI	g	-	0.115	Testwork
	Axb	-	68.7	94.9	Testwork
	DWI	kW/m³	6.05	4.40	Testwork

Criteria		Units	Design	Average	Source
Grind size	P_{80}	µm	70	70	Testwork
Gravity gold recovery			Future	Future	Minnovo
Copper circuit					
Pre-float and rougher lab residence time		minutes	8.5	8.5	Testwork
Pre-float cleaner lab residence time		minutes	1.0	1.0	Testwork
Concentrate regrind size P_{80}		µm	30	30	Testwork
Cleaner 1 lab residence time		minutes	4.5	4.5	Testwork
Cleaner 1 scavenger lab residence time		minutes	3.0	3.0	Assumed
Cleaner 2 lab residence time		minutes	4.0	4.0	Testwork
Cleaner 3 lab residence time		minutes	4.0	4.0	As Cleaner 2
Flotation residence time scale-up factors:					
- Rougher, cleaner 1 scavenger		-	2.5	2.5	Minnovo
- Pre-float cleaner		-	3.0	3.0	Minnovo
- Cleaner 1, cleaner 2, cleaner 3		-	3.0	3.0	BMC
Concentrate thickener solids flux		t/m²/h	0.20	0.20	Minnovo
Concentrate thickener u/f solids concentration		% w/w	65	65	Testwork
Filter type		-	Pressure	Pressure	Typical
Concentrate filter cake moisture – target		% w/w	9.6	9.6	Testwork
Lead circuit					
Pre-float and rougher lab residence time		minutes	7.0	7.0	Testwork
Pre-float cleaner lab residence time		minutes	1.0	1.0	Testwork
Concentrate regrind size P_{80}		µm	30	30	Testwork
Cleaner 1 lab residence time		minutes	7.0	7.0	Testwork
Cleaner 1 scavenger lab residence time		minutes	3.0	3.0	Assumed
Cleaner 2 lab residence time		minutes	4.0	4.0	Testwork
Flotation residence time scale-up factors:					
- Rougher, cleaner 1 scavenger		-	2.5	2.5	Minnovo
- Pre-float cleaner		-	3.0	3.0	Minnovo
- Cleaner 1, Cleaner 2		-	3.0	3.0	BMC
Concentrate thickener solids flux		t/m²/h	0.20	0.20	Minnovo
Concentrate thickener u/f solids concentration		% w/w	65	65	Testwork
Filter type		-	Pressure	Pressure	Typical
Concentrate filter cake moisture – target		% w/w	7.5	7.5	Testwork
Zinc circuit					
Pre-float and rougher lab residence time		minutes	9.0	9.0	Testwork
Pre-float cleaner lab residence time		minutes	2.4	2.4	Testwork
Concentrate regrind size P_{80}		µm	35	35	Testwork
Cleaner 1 lab residence time		minutes	6.0	6.0	Testwork
Cleaner 1 scavenger lab residence time		minutes	3.0	3.0	Assumed
Cleaner 2 lab residence time		minutes	6.0	6.0	Testwork
Cleaner 3 lab residence time		minutes	6.0	6.0	As Cleaner 2
Flotation residence time scale-up factors:					
Rougher, cleaner 1 scavenger		-	2.5	2.5	Minnovo
Pre-float cleaner		-	3.0	3.0	Minnovo
Cleaner 1, cleaner 2, cleaner 3		-	3.0	3.0	BMC
Concentrate thickener solids flux		t/m²/h	0.20	0.20	Minnovo
Con thickener u/f solids concentration		% w/w	65	65	Testwork
Filter type		-	Pressure	Pressure	Typical
Concentrate filter cake moisture – target		% w/w	9.5	9.5	Testwork

Criteria	Units	Design	Average	Source
Concentrate storage				
Copper	tonnes	5,000	5,000	BMC
Lead	-	Containers	Containers	BMC
Zinc	tonnes	10,000	10,000	BMC
Off spec concentrate	tonnes	700 to 1,000	700 to 1,000	BMC
Tailings thickener solids flux	$t/m^2/h$	1.0	1.0	Testwork
Tailings thickener u/f solids concentration	% w/w	60	60	Testwork
Tailings filter type	-	Pressure	Pressure	Testwork
Tailings filter cake moisture – target	% w/w	13	13	Testwork
Paste feed solids type	-	Filter cake	Filter cake	BMC
Paste binder addition (dry solids basis)	%	4.0	4.0	Testwork
Reagent addition:				
Lime	kg/t	1.31	1.31	Testwork review
SMBS	g/t	648	648	Testwork review
Collector A3894	g/t	31	31	Testwork review
Sodium cyanide	g/t	121	121	Testwork review
Zinc sulphate	g/t	374	374	Testwork review
Collector 3418A	g/t	9	9	Testwork review
Copper sulphate	g/t	634	634	Testwork review
Collector A208	g/t	77	77	Testwork review
Frother	g/t	41	41	Testwork review
Flocculant	g/t	20	20	Testwork review

17.2 Process Description

17.2.1 General

An overview of the processing facilities is shown in Figure 17-2. The facilities comprise the following main areas:

- Crushing
- Transfer station
- Coarse ore stockpile
- Grinding – SAG and Ball mills
- Flotation
- Regrind – vertical mills
- Concentrate thickeners and filters
- Concentrate storage and loadout
- Tailings thickening and filtration
- Reagents – storage, mixing and distribution
- General plant services
- Process building
- Assay laboratory.

17.2.2 Equipment Selection

Equipment selection has been in accordance with the following criteria:

- All equipment will be new
- Proven performance in similar application
- Ability to operate continuously 24 hours per day
- Be readily capable of performing the specified duty with minimum maintenance
- Incorporate the best materials and practices in line with modern engineering concepts to ensure maximum serviceability of the equipment in operation
- Provide for the replacement of wearing parts with the least possible downtime
- Provide features which will reduce the cost of maintenance and operation and improve accessibility for maintenance purposes.

Preference has been given to the use of standard proprietary items of equipment and components. The number and sizes used have been kept to a minimum to reduce the size of spares inventories.

17.2.3 Crushing

A reinforced concrete steel bridge, with retaining wall, extends from the ROM pad to the ROM bin dump point. This bridge provides access for the FEL. A Caterpillar 988 FEL or equivalent will be used to feed the crusher from the ROM pad. A 900 mm grizzly for scalping oversize material is installed on the ROM Bin which will have 20 minutes retention time (135 t).

ROM ore is withdrawn from the bin by an apron feeder and fed to a vibrating grizzly for removal of fines. The vibrating grizzly will operate ahead of the primary crusher to remove fines from the crusher feed. The Design Case throughput rate for the crusher was estimated to be 446 t/h, operating 13.5 hours per day.

Provision was made for tramp metal magnet on the Primary Crusher Discharge Conveyor, as well as a metal detector on the Coarse Ore Stockpile Feed Conveyor.

Insertable bag type dust collectors are provided to collect and filter out fugitive dust emissions from the materials handling circuit.

Normal crusher feeding practice will entail the crusher operator manning the FEL and loading the ROM bin. Video monitors in the FEL cab will allow the operator to monitor the status of the crusher during loading operations. An emergency stop switch located in the FEL cab will be capable of shutting down all crushing operations remotely from the cab.

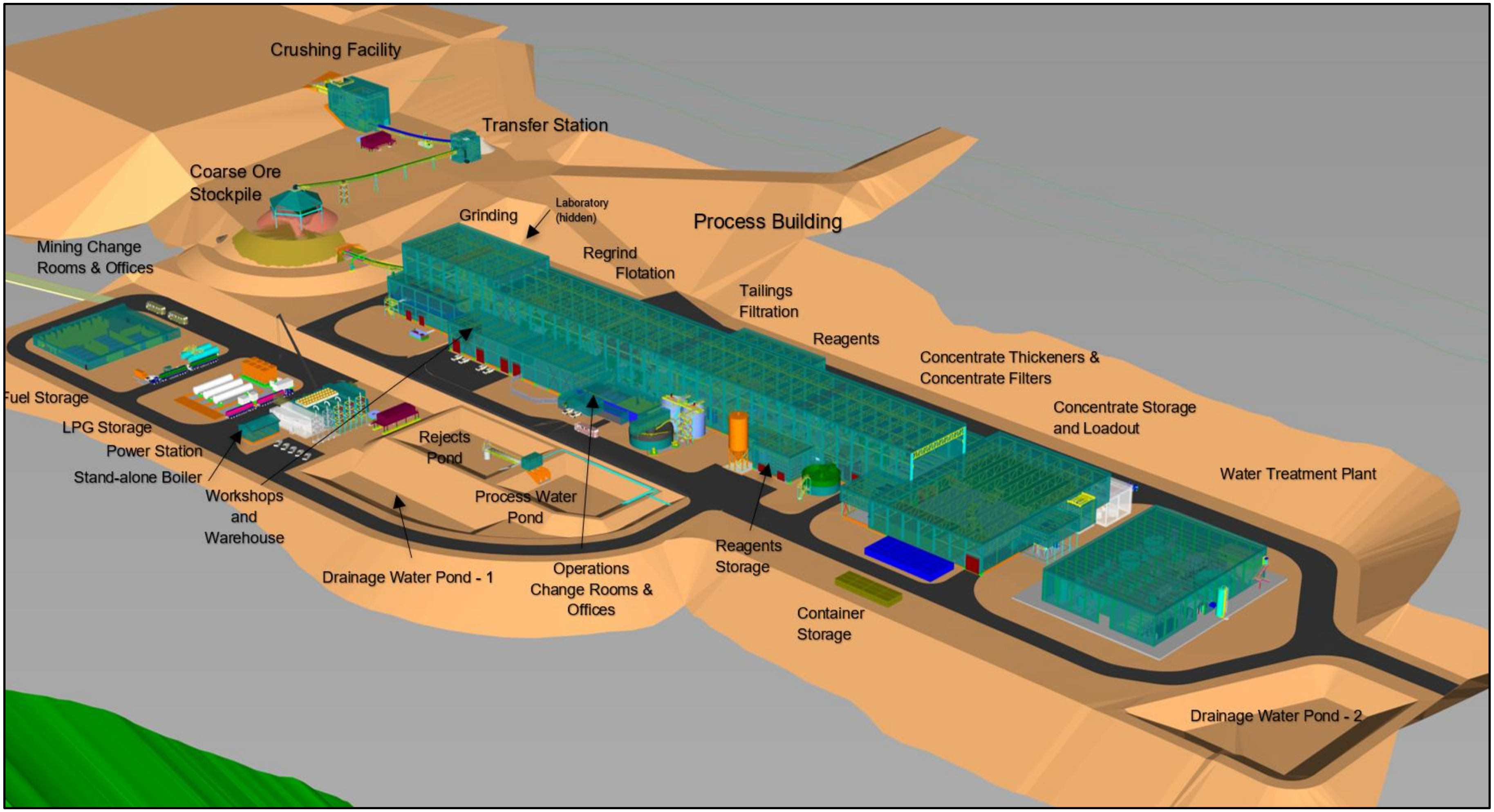


Figure 17-2: Processing facilities overview

17.2.4 Transfer Station

The Transfer Station is a braced steel structure that supports the head end of the Primary Crusher Discharge Conveyor and the tail end of the Coarse Ore Stockpile Feed Conveyor. A pneumatically actuated flop-gate can divert crushed rock into a bunker if required to produce crushed waste rock for the production of road base or blast hole stemming. It can also be used as a short-term stockpile for crushed ore in the event of a downstream stoppage. Tramp metal collected by the Primary Crusher Discharge Conveyor will also be collected in a skip bin at this facility.

17.2.5 Coarse Ore Stockpile

The Coarse Ore Stockpile facility is of the conventional open stockpile type, receiving crushed ore via the 1,200 mm-wide Coarse Ore Stockpile Feed Conveyor. Ore reclaim is by passive slot type Reclaim Hopper via two in-line apron type reclaim feeders discharging onto the 1,200 mm-wide SAG Mill Feed Conveyor.

The Coarse Ore Stockpile has a target live capacity of 12 hours of SAG Mill new feed. The total stockpile capacity (including dead capacity) will be approximately two days. A dozer can be used to reclaim the “dead” stockpile material to provide emergency feed to the grinding circuit.

Two apron feeders have been selected to reclaim ore from the stockpile, each able to deliver 100% of the design SAG Mill feed rate.

17.2.6 Grinding – SAG and Ball Mills

The SAG Mill and Ball Mill grinding circuit are located at the southern end of the Process Building where the ore is fed from the Coarse Ore Stockpile via the SAG Mill Feed Conveyor. The grinding circuit will comprise an open circuit SAG Mill and a Ball Mill in closed circuit with cyclones.

The SAG Mill will be a 5.79 m diameter x 5.49 m effective grinding length (EGL), grate discharge, steel-lined mill, driven by a 2.7 MW single pinion drive. The SAG Mill will discharge over a 12 mm aperture trommel screen. Trommel oversize (pebbles and steel scats) will be returned to the SAG Mill Feed Conveyor by three recycle conveyors in series.

The Ball Mill be a 4.72 m diameter x 7.32 m EGL, overflow discharge, rubber-lined mill driven by a 2.7 MW single pinion drive. The mill will be driven by a fixed speed drive and will operate at nominally 75% of critical speed. The Ball Mill will discharge over a 10 mm aperture trommel screen. The oversize will discharge into a bunker for disposal as waste. The undersize will gravitate into the mill discharge hopper.

The SAG Mill trommel screen undersize will also gravitate to the mill discharge hopper, where the combined mill discharge will be diluted with process water and pumped via duty/stand-by pumps to a hydrocyclone cluster for classification. The overflow from the cluster will flow by gravity to the flotation feed trash screen, to remove any trash prior to flotation. A portion of the cyclone underflow can be directed to the SAG Mill to facilitate balancing of the grinding in the SAG Mill and Ball Mill as the SAG Mill will generally have spare capacity on most ore types.

A common liner handler will be used to facilitate removal and installation of the SAG Mill and Ball Mill liners during planned mill relines.

An overhead gantry crane will be used for recharging grinding media as well as maintenance of the mills and cyclones.

Provision has been made within the Grinding and Classification area to receive a gold recovery facility if required in the future.

17.2.7 Flotation

The flotation circuit is sequential, with copper, lead, and zinc recovered in that order. All flotation cells are mechanically agitated, forced air conventional cells. In addition to the flotation cells noted below, the plant layout has allowed space for an additional flotation cells to be installed within the plant footprint should future ore sources warrant this.

Copper

Trash screen underflow slurry is pumped to the first of two 40 m^3 agitated copper conditioning tanks, where flotation reagents are added. Conditioned slurry flows to the first of six 38 m^3 rougher flotation cells in series. The first rougher cell will operate as a pre-float rougher which is then pumped to a single 8 m^3 pre-float cleaner cell and the concentrate pumped to the copper final concentrate hopper. Pre-float cleaner tailings flows to the copper regrind mill. Tails from the pre-float rougher passes to the remaining rougher cells. Concentrate from the rougher cells is pumped to the copper regrind mill.

The regrind mill treats a number of streams from in the copper circuit: the pre-float tails; rougher concentrate; and cleaner tails.

Regrind mill product flows to the cleaner circuit which includes three cleaner stages, with a cleaner scavenger operating after the first cleaner. The copper first cleaner consists of four 8 m^3 cells in series and two 8 m^3 copper cleaner-scavenger cells in series. The copper second cleaner consists of four 4.3 m^3 cells in series. The copper third cleaner consists of three 4.3 m^3 cells in series. Third cleaner concentrate is pumped to the copper final concentrate hopper, where it combines with the copper pre-float cleaner concentrate.

Rougher and scavenger tailings flow to the copper flotation tailings hopper feeding the lead circuit.

Lead

Combined copper flotation tailings are pumped to the first of two 40 m^3 agitated lead conditioning tanks, where flotation reagents are added. Conditioned slurry flows to the first of five 38 m^3 rougher flotation cells in series. The first rougher cell will operate as a pre-float rougher. Pre-float rougher concentrate is pumped to a single 8 m^3 pre-float cleaner, where the concentrate flows to the lead final concentrate hopper. Pre-float cleaner tailings is pumped to the lead rougher concentrate hopper. Rougher concentrate from the other four rougher cells is pumped to the lead regrind mill.

The regrind mill treats a number of streams from in the lead circuit: the pre-float tails; rougher concentrate; and cleaner tails.

Regrind mill product flows to the lead cleaner circuit, which includes two cleaner stages with a cleaner-scavenger operating after the first cleaner. The lead first cleaner consists of four 16 m^3 cells in series and two 16 m^3 lead cleaner-scavenger cells in series. The lead second cleaner consists of three 4.3 m^3 cells in series. Second cleaner concentrate flows to the lead final concentrate hopper where it combines with the lead pre-float cleaner concentrate.

Rougher and scavenger tailings flow to the lead flotation tailings hopper feeding the lead circuit.

Zinc

Combined lead flotation tailings are pumped to the first of two 40 m³ agitated zinc rougher conditioning tanks, where flotation reagents are added. Conditioned slurry flows to the first of six 38 m³ zinc rougher flotation cells in series. The first rougher cell will operate as a pre-float rougher. Pre-float rougher concentrate is pumped to two 16 m³ pre-float cleaners in series, where the concentrate is pumped to the zinc final concentrate hopper.

The regrind mill treats a number of streams from in the zinc circuit: the pre-float tails; rougher concentrate; and cleaner tails.

Regrind mill product flows to the zinc cleaner circuit which includes three cleaner stages, with a cleaner-scavenger operating after the first cleaner. The zinc first cleaner consists of three 38 m³ cells in series and two 38 m³ cleaner-scavenger cells in series. The zinc second cleaner consists of three 16 m³ cells in series. The zinc third cleaner consists of three 16 m³ cells in series. Zinc third cleaner concentrate flows to the zinc final concentrate hopper, where it combines with the zinc pre-float cleaner concentrate.

Rougher and scavenger tailings flow to the zinc flotation tailings hopper feeding the tails thickener.

17.2.8 Regrind – Vertical Mills

Regrinding the copper, lead and zinc rougher concentrates to P_{80} of 30 µm, 30 µm and 35 µm respectively has been included based on testwork. A single Metso SMD 355 kW unit was selected for each of the copper, lead, and zinc regrind duties. The plant layout has allowed space for an additional regrind mill should future expansion be required.

17.2.9 Thickener and Concentrate Filters

High-rate thickeners have been selected for concentrate thickening. The copper concentrate duty requires a 9 m diameter thickener. The same diameter thickener has been selected for the lead concentrate thickener to allow commonality of spares. The zinc concentrate duty requires a larger thickener of 14 m diameter.

Thickened concentrates will be pumped from the underflow to agitated filter feed tanks. There is one 220 m³ filter feed tank for each of copper and lead concentrates, and two 300 m³ filter feed tanks for zinc concentrate.

Vertical plate pressure filters have been selected for concentrate filtration, based on the relatively fine concentrate regrind sizes and the need to consistently achieve the TML of the concentrates. Filters have been specified with membrane squeeze and air blow capability, to ensure the target moistures are achieved. A common filter size was selected for all three concentrate filtration duties to minimize spares. The zinc duty requires two filters operating in parallel to cater for its higher production rate. The recommended (common size) pressure filter has 85 m² of filtration area.

The concentrate filters will be installed onto an elevated steel platform. Discharge from the filters will be via chute and into the bunkers located within the Concentrate Handling and Load-out section of the Process Building. The filter discharge chutes be fitted with doors for separation of the heating, ventilation and air conditioning (HVAC) for the main Process Plant building.

17.2.10 Concentrate Storage and Loadout

Filtered concentrate (filter cake) will be discharged into individual reinforced concrete bunkers directly below each filter in the Concentrate Storage and Loadout Shed (Figure 17-3). The copper and zinc filter cake will be removed by FEL and stacked in reinforced concrete bunkers inside the shed. Lead concentrate will be loaded directly into sealed transportable containers. Concentrate will be dewatered to a nominal 9% moisture content.

The storage capacity of copper and zinc concentrate in the bunkers is 5,000 t and 10,000 t respectively. There is also provision for storage of off-spec concentrate in a separate bunker of 700 to 1,000t capacity (depending on concentrate type).

The bottom section of the building walls will be made of reinforced concrete panels and walls against which the bulk concentrate will be retained. The walls have been designed to allow the FEL to reclaim against them and the loads from building columns.

Copper and zinc concentrate will be bulk loaded into trucks and lead concentrate will be loaded into sealed containers for transport by road to the port of Stewart.

The Concentrate Storage and Loadout Shed has been designed to receive a Double-B type truck with bogey for the transport of concentrate to the port. Loading will be by FEL while the truck is on the weighbridge.

Lead concentrate will be loaded from the bunker under the lead filter into empty containers located within a segregated loading facility. The containers will be loaded onto the truck by a container lifter for transport to port. The loading facility will also contain a container washing system.

Space is allowed in the layout for a future truck wash facility if operational needs require it.

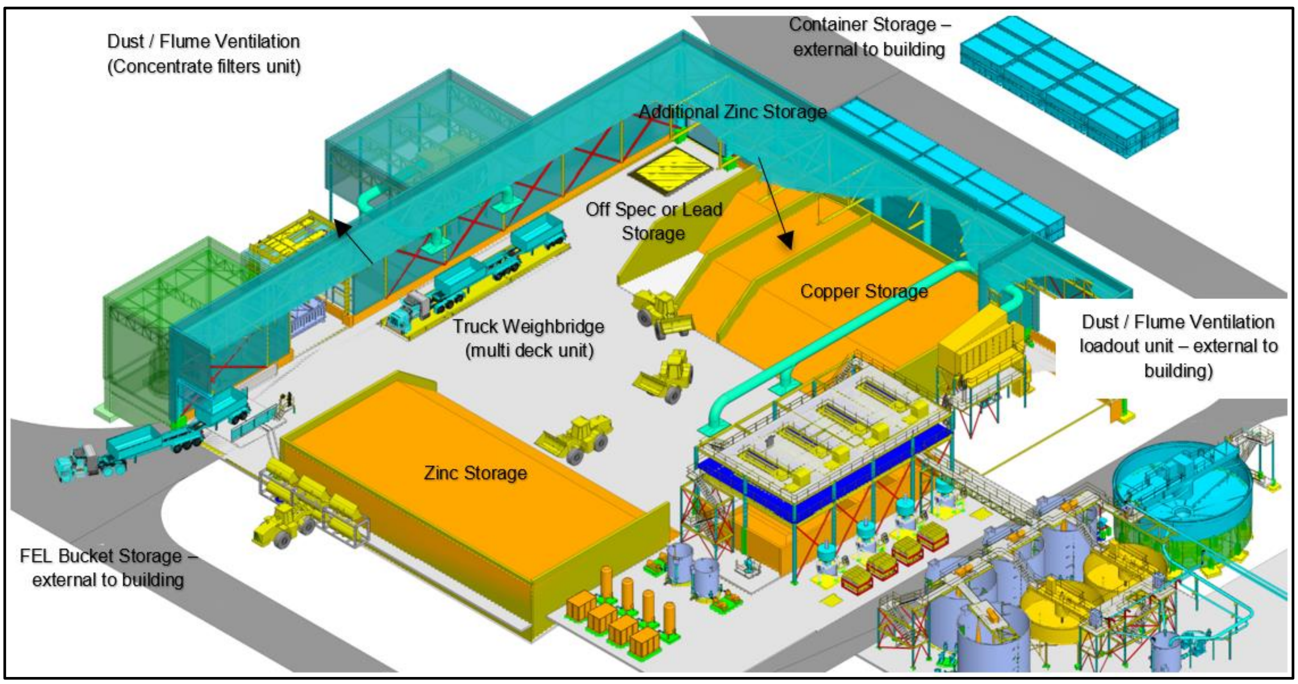


Figure 17-3: Concentrate storage and loadout shed

17.2.11 Tailings Thickening and Filtration

A high-rate thickener will be used to thicken the flotation tailings stream prior to pressure filtration. The tailings duty requires a 20 m diameter thickener. The tailings thickener sizing has also considered reclaimed tailings slurry pumped from the reject pond.

Thickener underflow is pumped to two 750 m³ filter feed tanks that will operate in parallel to provide surge volume for the thickened tailings slurry. The tanks have been designed with a total live residence time of eight hours, being four hours per tank. A waste stream from the Water Treatment Plant (WTP) will also be pumped to the Tailings Filter.

Two vertical plate pressure filters, each having 467 m² filtration area, have been selected to operate in parallel for tailings filtration. Selection of duty-duty filters minimizes the impact of the filter downtime on slurry surge requirements, as at least one filter will be operating at any time.

At those times when a single tailings filter is operating and the level of the filter feed tanks is high, excess tailings slurry can be temporarily diverted to the HDPE lined Rejects Pond, which has a 12-hour capacity. This allows the plant to continue operating at normal solids feed rate in the expectation that the second tailings filter will shortly return to service. Tailings slurry reclaimed from the Rejects Pond will be pumped back to the tailings thickener feed box.

The Tailings Filters will be installed onto an elevated steel platform within the Process Building. The platform will be provided with a suspended concrete floor. The bottom section of the building walls will comprise of reinforced concrete panels and walls against which the bulk filtered tailings will be retained. The walls will be designed to allow the FEL to reclaim against them.

The tailings filter area has been designed to receive 50-t articulated dump trucks. One truck can be loaded at any one time. Loading will be by FEL.

17.2.12 Reagents – Storage, Mixing and Distribution

Majority of the reagent storage and mixing facilities are contained within the Process Building, apart from Lime Slaking and Lead Circuit Depressant 1 (sodium cyanide), which will be contained within separate building structures annexed to the Process Building. Sodium cyanide mixing equipment requires an enclosed building and bunding and will be annexed to the Process Building. All reagent mixing and storage tanks located indoors will be provided with a roof and vented either naturally or through forced extraction systems outdoors as nominated.

Reagents will be delivered to site in several forms. Dry bulk deliveries including quicklime and binder for paste backfill (Year 3 onward) will be delivered to site in bulk and transferred to dry storage silos. Bulk bag solid deliveries (750 kg to 1,000 kg) will be in dry solid form in bulk bags. Bulk liquid deliveries will be as 100% strength liquid in 1 m³ intermediate bulk containers, which will be used as storage vessels on site by connecting at least two intermediate bulk containers into a pipe manifold to the dosing pumps. Steel and ceramic grinding media will be delivered to site in 1 tonne bulk bags and stored outside.

Dry bulk delivered reagents will have up to seven days onsite storage. Copper circuit depressant will have up to 14 days storage onsite. All other reagents will have up to 30 days storage onsite. Grinding media storage is unrestricted.

Lime will be mixed using raw water in an automated lime slaking system (Vertimill™). Milk of lime slurry will be dosed to the grinding and flotation circuits from a ring main using actuated on-off valves.

Bag breaking systems will be elevated and installed on steel structures supported by reinforced concrete pedestals. Placement of the reagent package into the bag splitters will be by motorized monorail hoist. The reagents will be mixed with raw water and placed in a storage tank, where it will be distributed to the Process Plant.

Reagent distribution to the dosing points will be by pressure pipeline with the exception of the pH modifier (lime) which will be by ring main.

17.2.13 General Plant Services

Compressed Air

Plant and instrument air will be provided via two screw type air compressors. The air will pass through a dryer before use. The dried air will then be fed to the Plant Air Receiver, from which it will be distributed to smaller air receivers such as those for instrumentation, SAG Mill and Ball Mill.

Five duty blowers and one stand-by blower will be installed to supply low pressure air to the flotation cells.

Two screw type air compressors will produce compressed air that will feed the copper, lead, and zinc filter air receivers. Compressed air will be reticulated from the air receivers to each concentrate filter.

A single dedicated screw type air compressor will produce compressed to the tailing filters air receiver.

Process Control

The plant will be appropriately automated to reduce the need for continuous or frequent operator intervention. Moderate levels of process and engineering data collection and equipment monitoring will be provided.

The plant will have a central control room from which the status of major electrical and mechanical equipment can be monitored, and process control loops can be monitored and adjusted. The Plant Control Room will be networked to the programmable logic controllers (PLCs) and operate a Supervisory Control and Data Acquisition (SCADA) system that provides an interface to the PLCs for control and monitoring of the plant. The SCADA system is configured to provide outputs to alarms, control the function of process equipment, and provide logging and trending facilities to assist in analysis of plant operations.

Video monitors and remote controls in the FEL cab will allow the Primary Crusher operator to monitor the status of the crusher during loading operations.

A particle size analyzer will provide grind size data on the Ball Mill cyclone overflow, and copper, lead and zinc regrind mill cyclone overflow streams.

Flotation feed concentrates and tailings streams as well as copper, lead and zinc regrind cyclone overflow streams are sampled and pumped to the on-line sample analyser area. Sampling and assay of the selected streams will be continuous by the on-line sample analyser with data returned to the Control Room.

Field instruments provide inputs to a set of PLCs. Process control cubicles are located in the motor control centres, and contain the PLC hardware, power supplies, and input/output cards for instrument monitoring and loop control.

17.2.14 Process Building

The main operating area of the Process Building is approximately 340 m long and 33 m wide. The Concentrate Storage and Loadout is approximately 54 m long and 78 m wide. The building will be 28 m at its highest point, above the grinding mills.

The pad for the building will be excavated by cut to fill methods with key foundations located in rock.

The Process Building will be a pre-engineered building, fully clad with profiled insulated sheet metal (sandwich panel type) on the roof and all sides. Wall panels will be reverse lined at the concentrate storage area to assist with housekeeping and lead concentrate clean-up. To eliminate ice and snow shed as well as icicle formation from the building rooves, a flat roof design will be incorporated with heated parapets around the perimeter and internal downspouts to avoid freeze up.

The design isolates the main operating area from potential fugitive dust, and diesel emissions of the concentrate storage and handling operations. Only the main operating area will be provided with HVAC supplied by heat recovery from the power station. For the concentrate storage and handling facility, a dedicated dust collection system will be provided.

A service way, accessible to mobile equipment, is located adjacent to the grinding area for grinding media recharge and maintenance activities. A second service way is located adjacent to the concentrate thickening and filtration equipment.

The main operating area is designed as a portalized structure such that internal columns are eliminated to enable unencumbered OHT crane access. A total of four cranes operate within the main operating area of the building including: one 25 t Safe Lifting Limit (SLL) with an auxiliary 2.5 t SLL servicing the horizontal and vertical mills; and three 15 t SLL overhead gantry-type cranes with an auxiliary 1.5 t SLL servicing the flotation area, tailings filters, thickeners and concentrate filters within the low bay of the building. A fifth 10 t SLL overhead gantry crane will service the Concentrate filters, within the concentrate filter facility and its operation will be independent of the other cranes. Motorized hoists have also been provided where access is denied to these cranes due to overhead structures.

A transportable type Plant Control Room will be installed inside the building, elevated to enable the Operator to view the flotation floor. A lean-to workshop is located the main operating area. Site administration, metallurgical offices and first aid will adjoin the building. Three Electrical Rooms will be located adjacent to the Concentrate building, one at each of the following facilities; Grinding, Flotation and Thickeners.

17.2.15 Assay Laboratory

A laboratory will be built on site for day-to-day analytical requirements. The laboratory will process around 300 samples per day. A laboratory services provider will be appointed to carryout design, equipping and operation of the laboratory. The laboratory will provide analytical services for the following sampling requirements:

- Open pit grade control sample preparation only (assaying at offsite laboratory)
- Underground grade control
- Metallurgical investigation work
- Process plant on-line sample analyzer checks
- Copper, lead and zinc concentrates
- Waste rock acid generation potential.

Given that open pit grade control samples will be generated on a campaign basis, open pit grade control samples will be conducted at an offsite laboratory. Fire assaying for gold will also be completed at an offsite laboratory, considering that only base metal grades will be used for day-to-day operational control of the plant, and gold assays will only be used for reconciliation purposes. Exploration and environmental analytical testwork will also be conducted at offsite laboratories.

17.3 Process Plant Utilities and Reagents

17.3.1 Power Requirements

The power consumption required for processing was estimated to be approximately 44 kWh/t and an estimated 692 GWh will be required for the Process Plant over the LOM.

17.3.2 Process Water Requirements

Average process water demand from the Process Water Dam was estimated to be 757 m^3/h (including bleed water to WTP) and the dam will have capacity for two hours (live) residence time.

The plant water balance is maintained by returning WTP treated water (68 m^3/h) to the Process Water Dam and through additional raw water input (18 m^3/h) from the Lower Water Management Pond. Make-up water from these two sources was estimated to be approximately 86 m^3/h in total. When the paste plant is in operation an additional 21 m^3/h will be required from the Lower Water Management Pond.

17.3.3 Reagents and Consumables

Projected requirements for reagents and consumables are given in Table 17-2.

Table 17-2: Annual requirements of process plant reagents and consumables

Reagent/Consumable	Units	Consumption rate
Reagents		
Quicklime	t/yr	2,873
SMBS	t/yr	1,426
NaCN	t/yr	266
$ZnSO_4$	t/yr	823
$CuSO_4$	t/yr	1,395
DF469	t/yr	68
3418A	t/yr	20
DF262	t/yr	169
Frother	t/yr	135
Flocculant	t/yr	59
Consumables		
Grinding media steel (total)	t/yr	1,078
Grinding media ceramic	t/yr	80
Liners	t/yr	105
Filter cloths (concentrate)	No./yr	2,048
Filter cloths (tailings)	No./yr	2,880

18 Project Infrastructure

The general arrangement of the KZK Project is shown in Figure 18-1.

Key items of infrastructure include:

- Open pit and underground mines.
- Processing facility and associated ROM and low-grade stockpile facilities.
- Paste backfill plant.
- Three waste storage facilities for tailings and waste rock. Waste rock will be placed in different storage facilities based on the assessed potential for generation of acidic drainage and metal leaching.
- Overburden and topsoil stockpiles that will be used for site reclamation during operations and closure.
- Water management infrastructure, including a Pit Rim Pond for mine dewatering, collection ponds, water management ponds and surface water diversion ditches.
- Camp facilities.
- General mine infrastructure including explosives facilities, workshops, fuel facilities and core storage area.

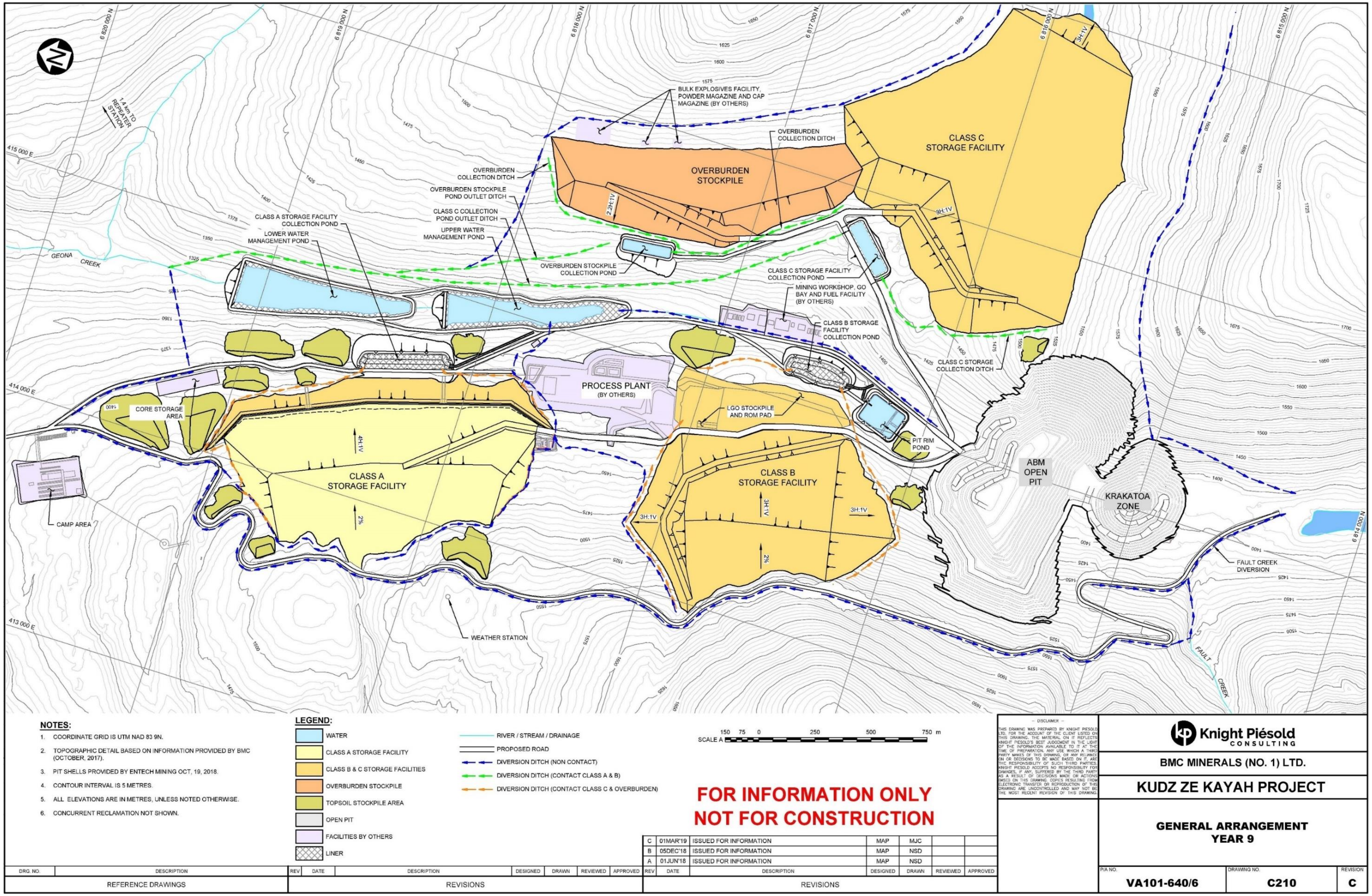


Figure 18-1: KZK Project general arrangement

18.1 Waste Storage Facilities

18.1.1 Tailings Storage Facilities

Tailings are defined as the fraction of processed ore that is produced at the processing plant that is not considered to be economically valuable. Tailings are managed as a waste material and require a geotechnically and geochemically stable storage option that will manage the tailings throughout the mine life and after closure.

After consideration of the various tailings storage methodologies, together with potential storage sites, filtered tailings storage was selected as the preferred storage method. This requires the tailings to be dewatered to produce a filter cake which will then be transported by truck to the Class A Storage Facility where they will be comingled and stored together with Class A waste rock.

18.1.2 Waste Storage Facilities

BMC will pursue every practical opportunity that presents itself over the life of the Project to use mined waste and tailings as fill within the mined underground voids; however, the waste storage facilities have been designed to cater for the maximum production of each material type from the mining and processing operation.

Waste rock will be classified as either Class A, B or C, based on its acid drainage and metal leaching potential, when placed in a waste storage facility over short-term and long-term periods. The identification of the waste rock "Class" will be based on laboratory analysis for sulphur and carbonate content to estimate the acid and neutralization potential. Separate facilities will be constructed to store each Class of waste rock.

Class A waste rock is defined as potentially acid generating (PAG) and/or metal leaching in the short term (i.e. within the life of the operation). This material will be contained in a storage facility with controlled drainage during operations and will be encapsulated and reclaimed after cessation of mining to minimize contact with oxygen and water.

Class B waste rock is defined as PAG and/or with metal leaching potential over the longer term (after cessation of mining activities). Storage of this material will require controlled drainage during operations. Encapsulation will be required after cessation of mining as part of the reclamation plan.

Class C waste rock is defined as material that is non-reactive or potentially acid consuming and will have low metal leaching potential. Therefore, specific ARD management strategies are not required. This material is suitable for construction purposes around the site as well as capping material required for reclamation during closure.

The mine waste rock production schedule is given in Table 16-2.

Class A Waste Storage Facility

The Class A Storage Facility is located on the western hillside of Geona Creek, north of the processing plant (Figure 18-1). It is designed to manage filtered tailings and waste rock material that is classified as strongly PAG which will be acid generating and/or metal leaching in the short term (i.e. within the life of the operation). During operations, tailings from the Process Plant and Class A waste rock excavated from the pit will be co-mingled, placed in thin lifts and compacted.

The footprint of the Class A Waste Storage Facility will be cleared of trees and topsoil stockpiled to expose the relatively thin layer of glacial till overburden and weathered bedrock and constructed. At commencement of operations, the Class A Waste Storage Facility will have a size of approximately 700 m x 400 m, with periodic increases to an ultimate size of approximately 1,550 m x 800 m at the end of operations. The final slope of the facility will be constructed with an overall slope of 4H:1V.

The design of the facility incorporates basin and cover layers, each comprising a layer of low permeability glacial till material, an 80 mil textured HDPE geomembrane and a final layer of crushed sand and gravel placed above the geomembrane for protection from construction vehicle traffic and to collect and convey seepage within the facility. A buttress will be constructed on the downstream slope of the Class A Waste Storage Facility using Class C material, improving the overall geotechnical stability of the facility. A continuous drainage layer placed above the HDPE geomembrane liner will provide a pathway for seepage beneath the tailings material and will be graded to collect and convey seepage flows to two internal sumps at the base of the facility, at the upstream toe of the buttress. Seepage will be pumped from the sumps to the Class A collection pond where it will be pumped to the WTP.

On closure, the facility will be capped with a layer of a minimum of three meters of Class C material for frost and erosion protection, and to improve long-term geotechnical stability. Overburden and topsoil will be spread above the Class C rock and the facility will be revegetated to mimic pre-mining conditions on the hillside.

The Class A Waste Storage Facility has been designed with a capacity of approximately 14 Mm^3. This will be more than sufficient for storage of the estimated quantities of tailings (approximately 8 Mm^3) and Class A waste rock (approximately 6 Mm^3). The design considered potential variations in actual volumes that may be encountered during operations, as more accurate methods are developed for the identification and management of the different classes of waste rock.

Class B Waste Storage Facility

The Class B Storage Facility is located on the western hillside of Geona Creek, north of the open pit and south of the process plant area (Figure 18-1). It is designed to contain waste rock classified as weakly PAG. In addition, the ROM pad and Low-Grade Ore (LGO) stockpile are incorporated into the design of the Class B Storage Facility.

The footprint of the Class B Waste Storage Facility will be cleared of trees and topsoil stockpiled, exposing the relatively thin layer of glacial till overburden and weathered bedrock. At commencement of operations, the Class B Waste Storage Facility will have a size of approximately 850 m x 250 m, with periodic increases to an ultimate size of approximately 1,150 m x 700 m at the end of operations. The final slope of the facility will be constructed at an overall slope of 3H:1V.

As Class B waste rock will have the potential for acid generation and/or metal leaching over the longer term (after cessation of mining activities), Class B material requires encapsulation to limit contact with oxygen and water. Therefore, the facility is designed with the same composite liner and cover system as the Class A Storage Facility. It is anticipated reclamation will begin upon the completion of mine operations. In addition to the closure layer, the Class B facility will be covered with a 3 m to 8.5 m layer of Class C waste rock for stability, as well as frost and erosion protection. Overburden and topsoil will be spread above the Class C rock and the facility will be revegetated to mimic pre-mining conditions on the hillside.

The Class B Waste Storage Facility has been designed to have a capacity of approximately 21 Mm^3, which is more than sufficient for storage of the estimated quantity of Class B waste rock. Similar to the Class A Waste Storage Facility, additional capacity has been allowed in the design to consider variations in Class B waste rock volumes that may occur as better understanding is gained in the identification of the different classes of waste rock.

Class C Waste Storage Facility

The Class C Waste Storage Facility is located on the eastern hillside of Geona Creek, in a hanging valley northeast of the open pit (Figure 18-1). It is designed to store waste rock that is classified as non-reactive or potentially acid consuming and will has low metal leaching potential. Therefore, specific ARD management strategies are not required for the Class C facility.

Prior to operating the facility, the footprint of the Class C Waste Storage Facility will be cleared of trees and topsoil stockpiled, exposing the relatively thin layer of glacial till overburden and weathered bedrock.

At the end of operations, the facility will be closed. It will be contoured and revegetated to resemble slopes at a similar elevation. Any excess material on the adjacent Overburden Stockpile that is remaining at closure will be placed on the Class C Facility. The final slope of the facility will be constructed at an overall slope of 3H:1V.

The Class C Waste Storage Facility has been designed to have a capacity of approximately 44 Mm^3, which is more than sufficient for storage of all Class C waste rock. The Class C Waste Storage Facility will not be commissioned until the start of operations as prior to this the material will be used for site construction purposes. By the end of operations, the footprint of the Class C Waste Storage Facility will be approximately 1,500 m x 1300 m.

Overburden Stockpile

The Overburden Stockpile is designed to temporarily store overburden material comprising glacial till and glaciolacustrine sediments excavated from the open pit area or beneath site infrastructure. It is located along eastern side of Geona Creek, north of the Class C Waste Storage Facility (Figure 18-1). Overburden material will be selectively stored in and sourced from the stockpile and will be used during operations as a construction material as well as during closure for reclamation and select cover material for the other facilities. The stockpile will be completely removed after the end of operations during the closure phase.

The footprint of the facility will be cleared and topsoil removed. Because the stockpile is a temporary structure, it will be constructed with a maximum slope of 2.2H:1V for short-term physical stability.

Approximately 8.5 Mm^3 of overburden will need to be excavated at KZK for construction and operation of the mine. This is also expected to satisfy the overall requirement for closure covers. Any excess overburden material remaining at closure will be placed on the Class C Facility.

Topsoil Stockpiles

Topsoil will be removed from the base of the Overburden Stockpile, Class A, B and C waste storage facilities and the open pit footprint during construction. Topsoil will be used during closure and reclamation to revegetate the Class A, B and C waste storage facilities as well as the Overburden Stockpile area itself. The average in-situ topsoil thickness is approximately 0.2 m, although localized variations throughout the Project area show topsoil layers up to 0.5 m thick.

The total estimated volume of topsoil, based on the average thickness, is approximately 1.8 Mm^3, which will be placed in various localized temporary stockpiles and windrows around site. Topsoil stockpiles will be placed and contoured to a 4H:1V slope. The stockpile surfaces will be temporarily revegetated during operations to stabilize the slope surfaces and control erosion from runoff.

18.2 Water Storage and Management Facilities

Water management planning formed an integral part of the project infrastructure design. BMC's objective for water management is to provide enough water to support operating requirements, while mitigating environmental impacts to downstream receiving waters.

The water management plan involves collecting and managing site runoff from disturbed areas and maximizing the recycle of mining and process water. Surplus water will be stored on site with excess water treated (if required) prior to being released to Geona Creek or pumped to Finlayson Creek.

All water in contact with the mine facilities, including the Class A, B and C waste storage facilities, Overburden Stockpile, Open Pit and the Process Plant Site and other infrastructure will be collected in water collection ponds for sediment control prior to conveying to either the WTP or Upper Water Management Pond, as appropriate. Ultimately the water will be released into Geona Creek or Finlayson Creek when appropriate, but only when contaminants are within project water quality guidelines.

The WTP will be constructed adjacent to the Process Plant and is designed to reduce contaminants in the contact water to acceptable levels in accordance with project water quality guidelines. The WTP will receive water from the Class A Waste Storage Facility Collection Pond and the Pit Rim Pond (ABM pit dewatering pond), and site run-off from the ROM Pad and the Process Plant Facility. The WTP will also treat a portion of the process water from the Process Plant.

Erosion and sediment control management strategies will include limiting mine site disturbance to the minimum practicable extent, establishing diversion and collection ditches to manage surface water runoff, constructing sediment management devices such as collection and diversion ditches (Figure 18-1), sediment traps and sediment ponds, stabilizing disturbed land surfaces to minimize erosion, ripping of rehabilitation areas to promote infiltration, establishing temporary vegetative covers, re-establishing vegetation that is similar in structure to natural vegetation where final slopes are created and restricting access to rehabilitated areas.

Overburden dewatering in the open pit area will occur during the pre-production period to facilitate mining. The overburden dewatering design incorporates a series of trenches and sumps (Figure 20-4) that will be used to collect water for pumping to the Pit Rim Pond, where sediment will be allowed to settle out prior to water being reused or discharged.

During operations, Fault Creek will be diverted south (Figure 18-1) towards the North Lakes, temporarily interrupting flow towards the open pit area. The current design incorporates a lined diversion ditch constructed alongside the access road; however future site investigation of the area may demonstrate that alternative and/or additional measures are required to successfully divert the water in Fault Creek.

A water balance model was developed to simulate the potential effects of climate variability on surface and groundwater flows and assess the effectiveness of the proposed mine water management system. Forty six(46) different climate scenarios were simulated, and the results indicated that water may need to be held in the Lower Water Management Pond during years with lower winter flows in the receiving environment in order to meet water quality guidelines. The results also indicated that water storage in the Lower Water Management Pond during years with higher mine dewatering flows is likely, even when winter flows are relatively high. All iterations predicted that the maximum volume of water held in the Lower Water Management Pond would be less than the design pond volume.

The closure objective for site water management will be to breach diversions around the mine site and return water courses to their natural directions of flow. The Class A Collection Pond will be pumped to the WTP until such time that water quality is suitable for passive release from the ponds perpetually.

Open pit dewatering will cease, and Fault Creek will be redirected to the open pit to facilitate pit filling. All open pit benches and slopes will be reclaimed in such a manner as to prevent erosion and minimize the suspension of sediments.

18.3 Water Treatment Plant

BMC engaged Integrated Sustainability Consultants Inc. (ISC) to determine an effective water treatment strategy and carryout subsequent initial engineering design of a suitable WTP for the KZK Project. The WTP will be capable of meeting the objectives relating to the discharge of excess water to the environment (Integrated Sustainability, 2019).

Influent water streams to the WTP consist of runoff from the Class A Waste Storage Facility, process water from the Process Plant, ROM pad runoff, Process Plant site runoff, and water generated from mine dewatering, as detailed in Table 18-1. Water quality from each of these sources will be monitored to determine if treatment through the WTP is required, and for the purposes of the DFS all these streams were considered to be treated. The main contaminants identified in these streams as being potential concern for the KZK Project are Se, Al, As, Cd, Cu, Zn and Fe. The treatment system is designed to discharge into the Lower Water Management Pond prior to release to the receiving environment. Any off-spec water will be discharged to the Pit Rim Pond to be recycled to the WTP.

Table 18-1: WTP design concentrations and flow rates

Parameter	Unit	Class A and process water (Years 0 to 5[1])		Class A and process water (Years 6+[2])		LGO-ROM, mill and pit dewatering	
		Average	Maximum	Average	Maximum	Average	Maximum
Flow rate	m^3/h	86	121	97	140	183	390
Aluminium	mg/l	0.02	0.01	4.95	8.47	0.014	0.016
Arsenic[3], total	mg/l	0.01	0.01	0.12	0.21	0.0035	0.071
Cadmium	mg/l	0.00	0.01	0.53	0.90	0.00027	0.00043
Copper	mg/l	0.48	0.35	38.25	63.73	0.0013	0.0022
Iron, total	mg/l	0.04	0.03	267	470	0.56	0.56
Selenium[4]	mg/l	0.53	0.39	0.99	1.22	0.0013	0.0017
Zinc	mg/l	1.06	1.61	83	143	0.05	0.064
Sulphate	mg/l	451	623	1,778[5]	3,320[5]	119	133
Total Suspended Solids[6]	mg/l	10	180	10	180	10	180

Notes:

1. *WTP is expected to be operational within approximately the last two months of pre-production.*
2. *WTP is expected to be operational during closure, as required.*
3. *Arsenic speciation distribution is approximately 50% As(III) and 50% As(V).*
4. *Selenium used in process design was estimated to be 100% selenate Se(VI).*
5. *A simple solubility control of sulphate vs gypsum precipitation was applied to Class A water sulphate concentrations in Years 6+ as the predicted concentrations of sulphate were deemed unrealistically high.*
6. *Based on previous projects, TSS was assumed to be 10 mg/l average and 180 mg/l maximum suspended solids concentrations for the design.*

The proposed treatment system consists of a metals removal circuit and a Selen-IX™[3] process for selenium removal (BQE Water, as cited in ISC, 2019). The Selen-IX™ treatment involves an ion exchange circuit followed by an electroreduction circuit from which selenium is precipitated from brine as stable iron oxyhydroxide solid containing selenium which is suitable for non-hazardous disposal.

Operation of the WTP has been divided in two phases:

- Phase 1 (Years 0 to 5) – the WTP is to be operational two months prior to the commencement of ore processing
- Phase 2 (Years 6+) – the WTP is to be operational, as required, after closure of mining activities.

In the first five years of the KZK Project, metals removal will be conducted by ferric and sulphide precipitation (Figure 18-2). In Year 6 before the Class A Water may potentially turn acidic, the WTP will be modified to include a High Density Sludge (HDS) lime neutralization system which will treat the Class A Water (Figure 18-3). As the WTP must treat a number of different streams with variable chemistry, the system is designed to keep the different sources of water segregated across different treatment trains. By doing this, treatment can be applied selectively, and the overall cost of treatment reduced. This also allows staged deployment of the treatment system.

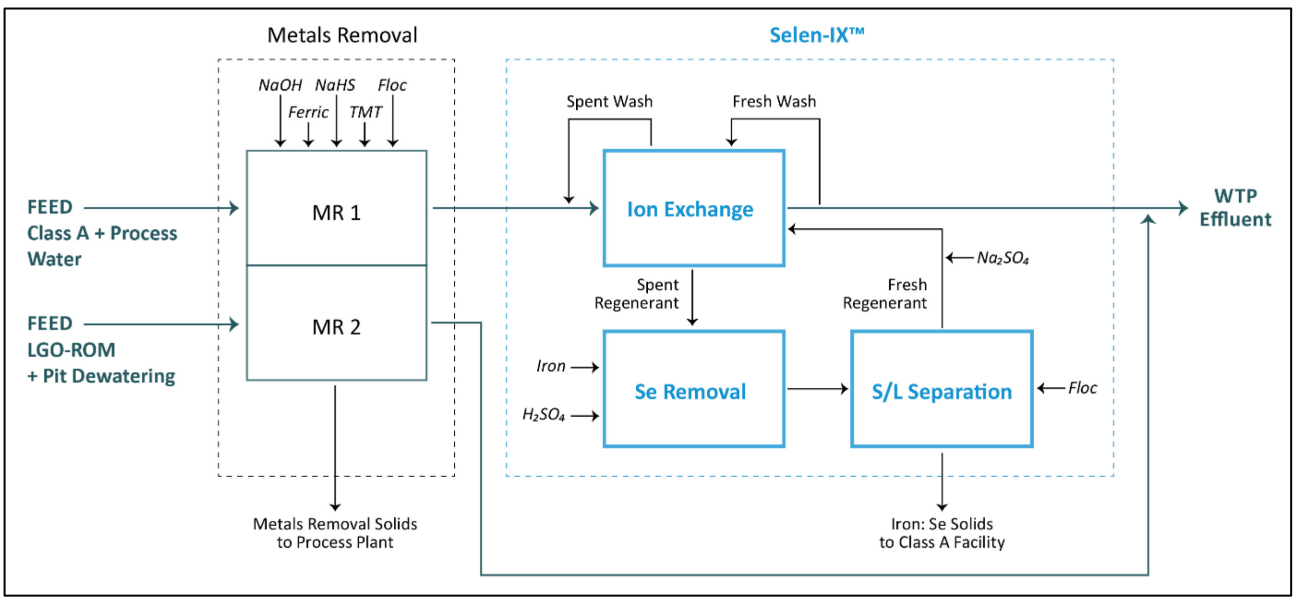


Figure 18-2: KZK WTP block flow schematic – Years 0 to 5

Source: Integrated Sustainability, 2019
Note: Selen-IX is a trademark of BQE Water as cited in ISC. (2019)

[3] Selen-IX is a trademark of BQE Water as cited in (ISC, 2019)

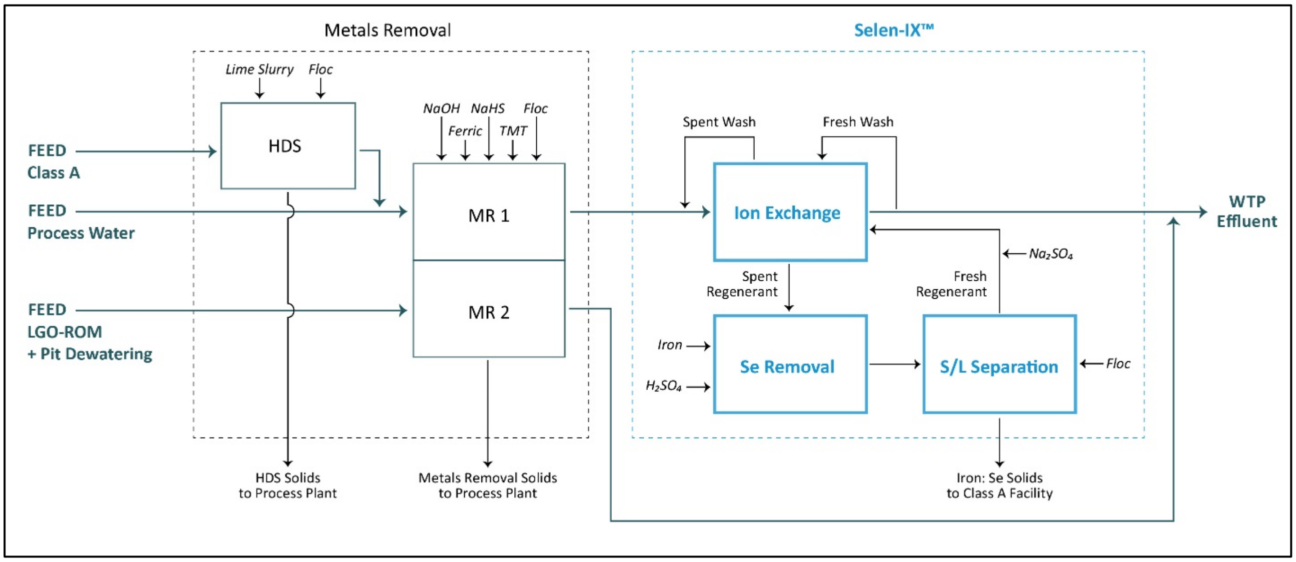


Figure 18-3: KZK WTP block flow schematic – Years 6+
Source: Integrated Sustainability, 2019
Note: Selen-IX is a trademark of BQE Water as cited in ISC. (2019)

By-products of the KZK WTP operations will be:

- A highly stable cake consisting of iron-selenium precipitates generated in the selenium removal circuit and will be disposed of in the Class A Waste Storage Facility.
- "Spent" electroreduction cell anodes that have been consumed to a size no longer efficient to use in the electrocell and will be either sold as scrap or disposed of in the Class A Waste Storage Facility.
- Thickened sludge from the metals removal and HDS clarifiers. These will be pumped to the process plant and disposed of with filter tailings from the process plant. After closure of the Process Plant, thickening within the WTP will be required.

Because of its close proximity to the Process Plant and integration with the main control room, labour required for the operation of the plant will be limited to one full-time operator per shift and shared maintenance and supervision with the main Process Plant operations.

Based on this process design, ISC developed a plant layout, 3D plant model developed to 30% (Figure 18-4), structural engineering estimates, architectural engineering preliminary drawings, mechanical equipment lists and specifications, piping engineering design, electrical engineering design, cost estimates for engineering and fabrication, a construction manning schedule, a construction cash flow schedule and an operating cost estimate.

The estimated operating and capital cost of the WTP is provided in Section 21.

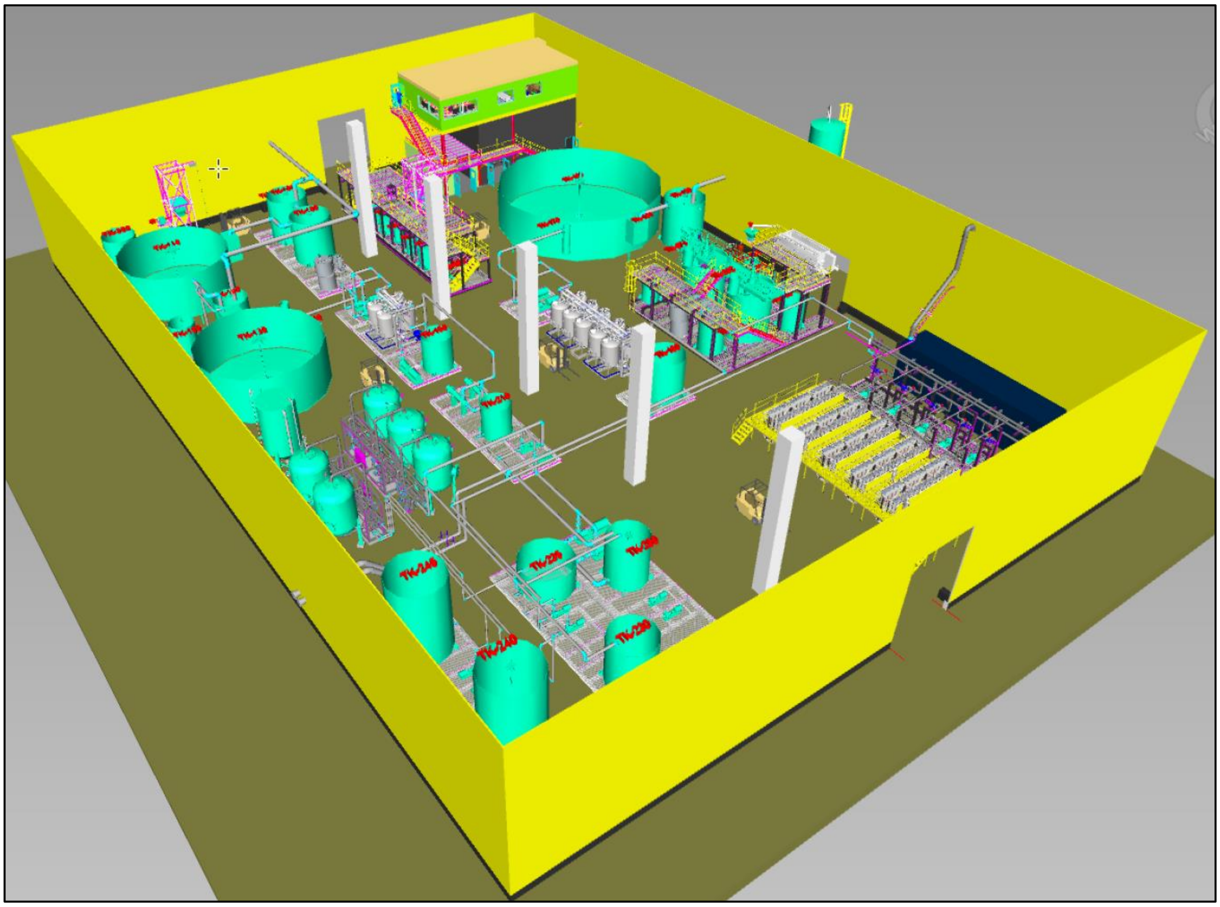


Figure 18-4: Isometric view of 3D model of the water treatment plant design
Source: Integrated Sustainability, 2019

18.4 Power Generation and Electrical Distribution

During the pre-production period, diesel-powered generators will be established for the following facilities:

- Local Generator 1 – Camp facilities including camp water and sewage system, and gatehouse.
- Local Generator 2 – Mine Pit Rim Area including: Pit Rim Pond pump, truck maintenance, wash bay, mine workshop and heavy vehicle fuel facility.

During operations, power for the mine will be generated at a Power Station complex located east of the Process Building on an adjacent terrace. Power will be generated by up to five 5.5 MW main generators in an N+1 configuration. The Power Station also includes a 750 kW diesel generator for powering main generator auxiliaries during black start operation. Power will be generated at 13.8 kV.

The generators at the Power Station will be dual-fuel (natural gas – diesel) generators. These generators are capable of running a range from 100% diesel to 99% natural gas/1% diesel.

The generators will connect to the 13.8 kV main switchgear assembly integral to the Power Station. This main switchgear will feed the process infrastructure directly without any intermediary transformer.

The Power Station will be a staged installation. For the first two years of the project's operation, the Power Station will have only three of the five main generators installed, as well as the standby black start diesel generator. In Year 3, work will begin on the underground mine and this will include a new 13.8 kV power supply cable to the mine area from the main Power Station. Consequently, at Year 3, two more generators will be added to meet load increases associated with underground mining and mine infrastructure on surface, the loads from Local Generator 2 will be transferred over to the new power supply, and Local Generator 2 will be decommissioned. The camp facilities and systems will remain connected to Local Generator 1.

Remote equipment and systems such as pumps in water collection ponds and storage facilities will not be connected to the main power distribution system. These include water management pumps, dewatering pumps, Class A and B Waste Storage Facility Collection Pond pumps, and the explosives facility. These will be powered from individual skid-mounted diesel generators.

18.5 Fuel Supply

Fuel supply and storage on the site includes LNG for the Power Station, and diesel fuel for stand-alone generators, mining equipment and the Power Station.

18.5.1 LNG Storage

The storage facility will comprise three 100 m^3 LNG tanks. Each tank is enough to operate the Power Station for 16 hours at a peak load of 15 MW. Therefore, with three tanks installed and filled, there is two days capacity for operating the Power Station at peak load. An LNG vaporizer system is used to supply natural gas fuel to the generators at the required pressure and flow rate.

LNG will be sourced from a liquefication facility and transported to site using LNG trailers in a B-train configuration to minimize transportation costs. LNG will be offloaded from the LNG tankers using LNG cryogenic pumps located at the mine site.

18.5.2 Diesel Storage

The main diesel storage facility (four tanks of 100,000 litres each) will be located adjacent to the Mine Workshop and maintenance area, with enough storage capacity for nine days of normal mining operation.

Two 113,500-litre diesel fuel tanks will be installed for the Power Station. The average storage capacity of these tanks for the operation of the Power Station is approximately three days. The tanks will be refilled by delivery tankers, or if needed, from the main diesel storage facility using open pit refuelling trucks.

18.5.3 Gasoline

It is the intention to minimize the use of gasoline powered equipment at the site. A 30,000-litre gasoline storage facility will be located near the processing plant facility to provide for fuelling pickups and other small gas engines.

18.6 Heat Recovery for HVAC

In consideration of the long heating season and high fuel costs, it has been determined economic to include full heat recovery, including jacket water heat exchangers and exhaust gas heat recovery units. The waste heat will be used by the HVAC system to heat the Process Building through a glycol loop.

It has been estimated that 12 MW of heat can be recovered from four fully operating generators and this is suitable to meet the heating requirements for the process building. However, during the first two years of operation, only three generators will be installed and operating with less available heat to transfer to the glycol heating system. For this reason, heat generation capacity will be supplemented with standalone natural gas fired boilers to the glycol heat medium distribution system.

18.7 Mining Infrastructure

18.7.1 Mine Workshop Facilities

Mine workshop facilities will be constructed for both the open pit and underground mining operations, approximately 1 km north of the open pit as shown in Figure 18-1. Initially only the open pit facilities will be constructed, with the underground facilities planned to be constructed in 2024 to align with mobilization of the underground mining contractor.

Both the open pit and underground mine workshops will be a fabric-on-frame type structure anchored to seacans, which will serve as additional working and storage areas for the workshop facilities. The mine workshop area will have sufficient space available for the mining contractors to install site offices, wash car and the like for management of the contracted mining works as well as an equipment ready line for mobile fleet.

A vehicle wash bay will also be constructed adjacent to the mine workshop facilities and will be utilized by both the open pit and underground mining contractors. It will also be a fabric on frame type structure, anchored to seacans.

Refuelling facilities for mining operations will also be constructed at the mine workshop facility and is discussed in Section 18.5.

18.7.2 Explosives

Explosives will be stored in secure, fenced facilities separate from the main activity areas, adjacent to the overburden stockpile. Bulk explosives will be stored within a bulk explosives compound. An explosive magazine will store all packaged explosive products required for the open pit and underground mining operations. A separate magazine will be installed for storage of all detonators.

The design of all storage facilities will meet government regulations and will be located according to required separation distances as regulated by the Explosives Regulatory Division of Natural Resources Canada (NRCan). Based on this, the minimum separation distance from inhabited buildings has been assessed as 960 m and the selected storage sites exceed this distance.

Bulk ammonium nitrate prill and bulk ammonium nitrate emulsion will be transported to site in 25-t bulk transport trailers and 20-t tanker trailers respectively. Bulk products will be stored in separate prill and emulsion silos within the bulk explosive compound. The bulk explosive compound will also contain a garage for the explosives loading trucks and a small office for the explosives contractor.

Packaged explosives and detonators will be delivered by approved explosives freight trucks.

Explosives will not be manufactured on site; however, the explosives trucks for the open pit and underground mines will be capable of mixing blasting agents in varying ratios to meet the specific requirements of each blast, such as the presence of wet holes and the need to vary explosive density.

18.8 Communications

Site communications will be established via microwave link to connect KZK directly to the NorthwesTel terrestrial network. The nearest existing access point is the McEvoy Tower located at 61 45’ 18”N 130 12’ 50”W approximately 36 km northeast of the project area.

Based on engineering assessments conducted to date, the link from the McEvoy Tower will require one intermediate repeater site that will relay the signal into the mine site. The site identified is approximately

3 km from the camp and was chosen to optimize link capacity, as well as providing a location that is potentially accessible in harsh conditions. This location also provides two-way coverage along the Access Road as well as into the mine site itself.

An antenna will be installed on the existing McEvoy Tower connected to a new 20 m tower with 0.6 m parabolic dish installed adjacent to the existing tower.

The intermediate site recommended will consist of an integrated system that will include a 20 m tower with 1.2 m parabolic dish facing the McEvoy Tower and 0.6 m parabolic dish facing site. Equipment will be housed in a fabricated shelter that is skid mounted to allow for easy transportation and positioning. An additional shelter has been included to house the diesel-powered DC generator, battery bank and associated controls and monitoring if required.

At site, a 10 m tower will be installed and mounted to the local communications shelter. The tower will support a 0.6 m antenna.

Site radio communications will consist of the following components:

- Base station radios with external antennas for office and other buildings as required
- Remote site repeater for road coverage
- Site headend with interconnection to underground leaky feeder radio system (Year 3 onward).

The system includes capacity for eight channels. Some of these will be utilized for data and controls.

Communications speeds are based on 20 MBps IP-Connect and 40 MBps bandwidth.

A VHF leaky feeder system will be used in the underground mine to integrate with the surface communications. The system is comprised of the head end unit along with line amplifiers, DC power supplies, splitters, and the leaky feeder cable itself.

18.9 Site Roads

The main access to the site will be via the upgraded Tote Road, from the Robert Campbell Highway to the Camp (Section 18.10). Beyond this point, several roads will be either upgraded or constructed to service the needs of the operation (Figure 18-5).

The portion of the Tote Road that extends south of the camp will be upgraded to allow larger two-way construction vehicle traffic to the Open Pit area. It also serves as access to construct the Tote Road Diversion Channel to initially divert non-contact water from Geona Valley and begin dewatering in the project area.

The upper diversion road will access the Fault Creek diversion channel and later it will serve as access to construct the upper diversion channel.

The Mill Access road will commence from an intersection adjacent to the camp and extend through to the main process plant pad. This road will carry all heavy construction traffic and later serve as the first stage of the concentrate haul route. The Mill Access road leads to the Geona Valley road via a link road.

The Geona Valley road will be constructed below the process plant site and traverses the valley floor, providing access to the upper and lower water management ponds, the mine workshops, Class B Collection Pond and Pit Rim Pond. The road terminates at the pit entrance and will be constructed to mine haul road specifications for use by open pit and underground mining fleet to access the workshops and fuel storage areas.

Perimeter roads will also be constructed on the eastern embankment of the Upper and Lower Water Management Ponds. A separate access road will be required for the Overburden Stockpile collection pond.

A number of mine haul roads will be constructed by the mining contractor fleet during pre-production for the 140-t mine dump trucks to access the main waste storage facilities and ROM pad. The roads include the following:

- Class A and B haul road
- Class C haul road
- Class C to Overburden haul road.

The Class A and B haul road will also provide access via a ramp from the filter building for tails haulage to the Class A Waste Storage Facility and for the agitator trucks to haul paste backfill to the discharge hopper in the pit during underground mining operations.

Approximately 25 km of roads will be constructed around the site. The proposed roads are shown in Figure 18-5.

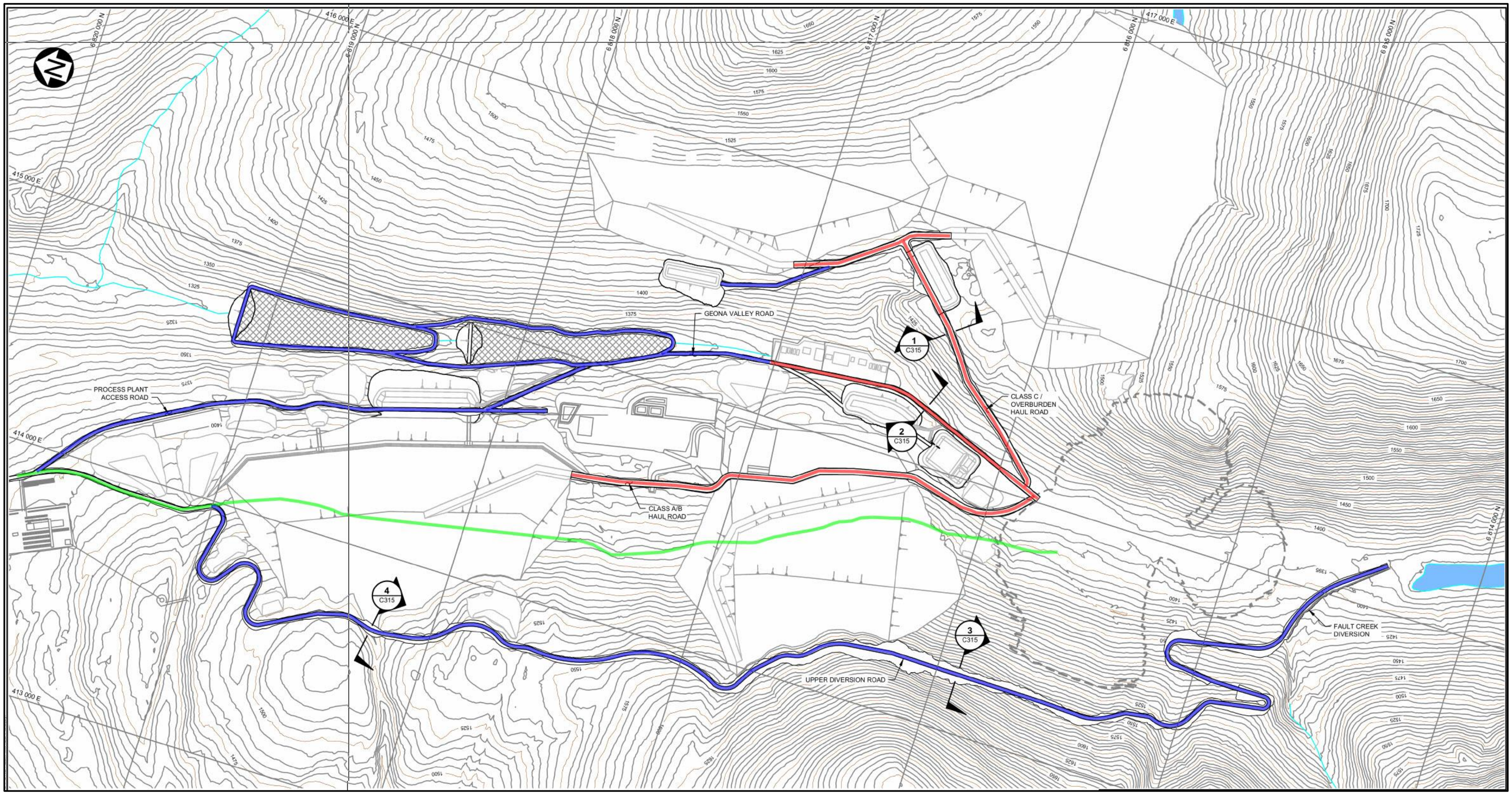


Figure 18-5: Site roads – green indicates road upgrade, blue indicates new light vehicle road, red indicates mine haul road

18.10 Access Road

The Tote Road, originally constructed in 1995, is approximately 24 km in length and extends from the Robert Campbell Highway, south to the KZK Project site, as illustrated in Figure 18-6. An Interim design to upgrade the road from a Tote Road to an Access Road for operational use was carried out by Onsite Engineering Limited (Onsite, 2016).

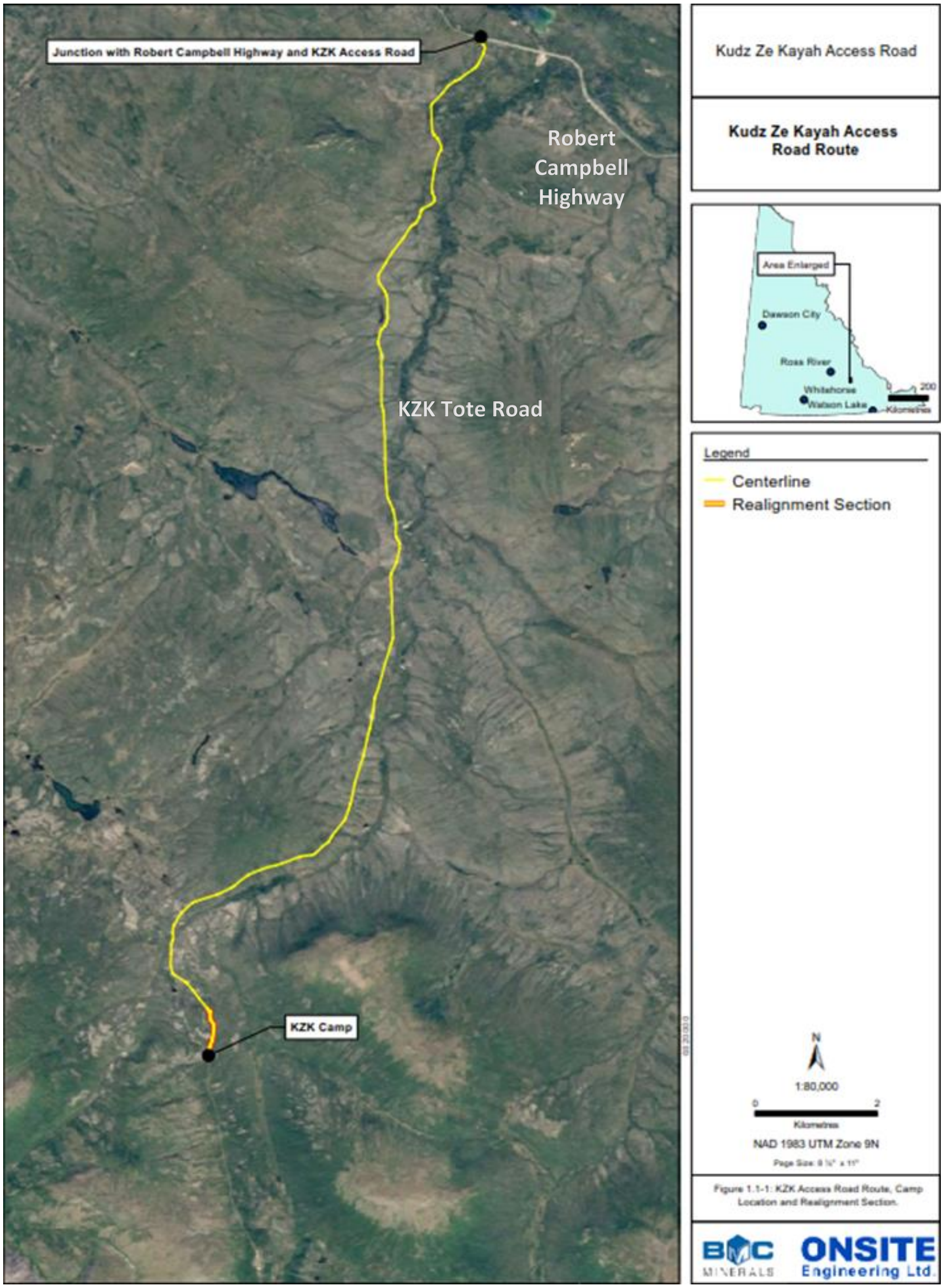


Figure 18-6: KZK Tote Road (field of view 26 km wide)

In its present state, the Tote Road is not suitable for the vehicle traffic that will be using it during the construction and production phases of the KZK Project. Upgrades and realignment of the existing road are proposed to improve road user safety, increase the travel speed, and provide all-season access.

The designed upgrade is for an all-weather, single lane road with sufficient pullouts to safely support two-way, radio-controlled traffic travelling at a speed limit of 50 km/h. The portion of the existing Tote Road between the camp and the open pit will be upgraded to allow larger two-way construction truck traffic. This will also function as the primary diversion channel in early years of operations.

A 30 m-wide right of way corridor of trees will be cleared ahead of road construction activities. In areas where cut and fill slopes extend outside of the 30 m cleared corridor, the clearing width will be increased to 3 m beyond the extent of the cut slope and/or 10 m beyond the extent of the fill slope. Clearing widths within 30 m of creeks will be reduced to 10 m or 3 m beyond cut and fill slopes, whichever is greater. Additional areas will be cleared depending on spoil and borrow site requirements.

Wherever possible, conventional cut to fill construction techniques will be implemented to minimize material movement.

The Tote Road crosses 10 streams, nine of which are culverts with the tenth being the existing steel portable bridge at Finlayson Creek. All culverts require upgrading, eight of which will require complete structure replacement. The bridge over Finlayson Creek was upgraded in 2015 to meet current environmental requirements and a load rating of British Columbia Forest Service L-100. No work is proposed for this crossing.

Access to the KZK Project site is currently controlled with a gatehouse located on the Tote Road, immediately after turning off the Robert Campbell Highway. This facility will be maintained throughout construction, operations and closure activities.

The road upgrade will be carried out in two phases. An initial upgrade will be carried out in the early stages of construction to improve access for heavy construction traffic. This will include reducing existing gradients, improving visibility on bends, constructing pull outs and some culvert replacements, amongst other works. The balance of the upgrade work will be carried out during the early stages of the Process Plant commissioning phase, when traffic movements along the road are expected to be light.

18.11 Accommodation Camp

A permanent accommodation camp will be constructed adjacent to the existing exploration camp.

The permanent camp, complete with dormitories, kitchen, mess hall, and recreation facilities will be provided for the work force during construction, and this will be partially retained for operations personnel. It is expected that an initial camp of approximately 100 single-beds will be required for early works, and that will be expanded to 348 dormitory rooms including temporary double-bunks to cater for the peak construction phase. Road access to the camp is directly off the main site Access Road, adjacent to a security boom gate. Parking will be provided for the bus fleet and a number of contractor vehicles, but not for all camp occupants.

The camp size has been determined from the resourcing schedules, based on a “motel” system being applied to room allocation (i.e. personnel will not be assigned the same room upon return to site from R&R rotations). To facilitate this approach, a heated building will be provided with lockers for storage of personal effects and safety gear between rotations.

The camp dormitory buildings are supplied as modular units erected on sleepers and interconnected by enclosed arctic breezeways. On-board services are supplied with the modules, including lighting, wiring, HVAC, plumbing, and fire detection/suppression. The common modules for kitchen, mess hall, recreation, and laundry are similarly supplied and fit for purpose.

Services for the accommodation complex will be islanded from the mine and Process Plant complex:

- Local power generation (1000kVA/600V/3P complete with emergency backup)
- Fuel storage for five days
- Potable and fire water well, storage, and treatment
- Sewerage treatment plant and disposal field.

The communications network will not be islanded. Given its proximity to the communications tower, it will be integrated with the site microwave network.

During construction, rooms will be fitted out with bunking style accommodation to cater for the increased workforce expected at that time. Support services and facilities have been sized to support the increase occupancy during this period.

Catering services at the camp will be outsourced to an experienced catering contractor. The catering staff will also be housed within the accommodation facility on the same basis as the main workforce.

18.12 Airstrip

The Finlayson airstrip is the closest public airstrip to the KZK Project, located approximately 40 km from the site, and is currently used for servicing the exploration and environmental monitoring requirements of the KZK Project. It is a gravel strip, 563 m in length, and is capable of being serviced by aircraft of up to 14-seat capacity. It is intended that this facility will be utilized as the primary airstrip for all phases of the KZK Project (construction, operations and closure). Contingency plans will be put in place due to the variability of weather which affect flight conditions in Yukon throughout the year, with alternative airstrips available at Faro and Watson Lake.

During operations, it is expected that an average of eight flights per week will be required to service personnel roster requirements. Construction workforce levels will be higher than that of operations and up to 12 flights per week could be required to service construction requirements. During the construction, it may prove more efficient to utilize larger aircraft and fly to either the Faro or Watson Lake airstrips, which will be confirmed during execution planning. This alternative is not expected to materially impact to the cost or timing of construction.

18.13 Port Facilities

The preferred port facility for export of the concentrates produced by the KZK Project is Stewart World Port (SWP) at the Port of Stewart on the northwest coast of Canada, at the head of the Portland Waterway about 80 nautical miles from the open sea. The Portland Waterway comprises the Portland Inlet and the Portland Canal.

Braemar Technical Services LLC (Braemar) assessed the SWP for the export of copper, zinc and lead concentrates (Braemar, 2018). SWP currently comprises of a concrete deck and steel pile jetty with one berth suitable for Handysize and Handymax class vessels. The berth has also been designed for Panamax vessels, subject to the installation of a mooring buoy. Advantages of SWP also include good access from project to port for trucks via existing highways, an existing town, a town bypass road, winter access for shipping, little competing demand for port access and being protected from prevailing winds by high peaks. Issues identified with SWP include the distance to the nearest pilot and tug facilities (115 nautical miles to Prince Rupert), restriction of berthing operations to daylight hours and likely loading delays from its high precipitation levels.

During operations, concentrates from the mine will be stored at a purpose-built storage facility at SWP (Figure 18-7). Copper and zinc concentrates will be bulk stored until a ship arrives, where it will then be conveyor fed to a ship loader for vessel loading. Lead concentrate will be transported from site to SWP via dual carriage sealed purpose-built half height containers. The containers will be stored in an outdoor hardstand area at SWP until vessel loading via a container rotating system and ships crane.

Copper and zinc concentrate operations are anticipated to be undertaken using a 2,500 t/h ship loader yielding a maximum loading rate of 50,000 t/day. The loading rate for the lead will be slower and is estimated to be up to 3,700 t/day.

Figure 18-7: Isometric schematic drawing of concentrate storage facility (JDS Energy & Mining, 2019)

18.14 Concentrate Haulage

Concentrate will be hauled from the KZK Project to the Port of Stewart along a 905 km southerly route utilizing a combination of the gravel site Access Road, gravelled and sealed secondary highways, and paved primary highways (Figure 18-8) (Allnorth, 2018).

Figure 18-8: Haul route from KZK Project to the port of Stewart

For majority of the year (312 days), the transportation haul will be executed based on a Non-Restricted operation with up to 100% legal axle loading. A portion of the highway system in Yukon (Highway 4 Robert Campbell) is subject to seasonal payload restrictions of 75% of legal axle loading during the spring break up period. During this period (approximately 45 days), transportation of concentrate will be executed as a Restricted operation.

Non-Restricted operations will transport copper concentrate and zinc concentrate in conventional style covered bulk ore style boxes. However, during Restricted operations, copper and zinc concentrate products will be transported with “modified” sealed containers capable of side dumping. Copper and zinc concentrate will be off loaded at SWP using a “Tipper Table”. Each product will be stored within a separate covered, open bulk pile, within the concentrate storage facility (Figure 18-7) until it is ready to be loaded onto a ship.

Year round, lead concentrate will be transported in sealed containers, stored containerized at port. When a ship arrives to transport the lead concentrate, the containers will be trucked to the berth alongside the ship then emptied directly into the ship’s hold. Containerized storage capacity at Stewart will provide up to 60 days plus an additional 14 surge days of haul production for total of 9,176 wet tonnes. A total of 389 standard containers plus 50 “modified” containers (for copper and zinc) will be utilized.

During Highway 4 Restricted operations, seven axle tractor/trailer units will be utilized to haul single sealed containers from the mine site to Watson Lake. At the Watson Lake transfer facility, a 30-t forklift will transfer sealed containers onto the nine axle Super B units for transport to Stewart. Because of the 75% restriction, operations during the Restricted period, compared to Non-Restricted, will operate at 1.8 t (4%) per load less for lead concentrate and 4.2 t (8.5%) per load less for copper concentrate and zinc concentrate.

18.15 Security

Access to site is currently controlled with a gatehouse located on the Tote Road entrance, immediately after turning off the Robert Campbell Highway. All vehicles entering the KZK Project are required to stop at this gatehouse and register before continuing into the property. This gatehouse will be maintained during construction and operations as an initial security point for access to site.

A second gatehouse will be established on the Access Road prior to arriving at the accommodation facility and will function as the key access to the operating site. A register of all vehicles and personnel visiting the site will be maintained to ensure that accurate data is available in the event of an emergency.

Security fencing and gate access will be constructed around the explosives facilities to limit access to authorized personnel. The incinerator, landfill facility and all ponds will also be stock fenced to prevent access by wildlife.

19 Market Studies and Contracts

StoneHouse Consulting Inc., an independent concentrate marketing advisory firm, completed a concentrate marketing study for the DFS (StoneHouse, 2018). Subsequent to this study, StoneHouse Consulting were requested to provide an update to the study based on the latest available information on concentrate specifications (StoneHouse, 2019). A summary of the relevant and most current findings is presented in this section. The Qualified Person has reviewed the marketing report and considers the expected terms to be a reasonable indication of KZK's product marketability and confirms that the results support the assumptions in this technical report.

Three separate concentrates will be produced during operations: copper, lead and zinc, all with varying levels of precious metal credits and deleterious elements.

19.1 Product Quality

Metallurgical testwork completed for the DFS, as detailed in Section 13, included production of typical concentrates with subsequent full elemental analysis to assess the quality of each concentrate. As described in Section 13, in the first 18 months of the project the +1340 RL metallurgical domain will be the only source of ore, until the open pit reaches sufficient depth to access other metallurgical domains and enable blending. On this basis, two concentrate qualities (Quality A and B) were estimated for each product, for the purpose of assessing marketability, as shown in Table 19-1 and Table 19-2.

Initial production (Concentrate Quality A) is based on the predicted concentrate qualities from metallurgical testwork for the +1340 RL metallurgical domain, while longer term production (Concentrate Quality B) is a composite of predicted concentrate qualities from metallurgical testwork of all metallurgical domains, after considering blending of ore to the Process Plant from available domains. Ranges of key economic and deleterious elements have also been included in Table 19-1 and Table 19-2.

Table 19-1: Concentrate qualities – Quality A

Element	Unit	Copper concentrate		Lead concentrate		Zinc concentrate	
		Quality A		Quality A		Quality A	
		Typical	Range of key elements	Typical	Range of key elements	Typical	Range of key elements
Cu	%	25.0	21.0-26.0	1.1	0.9-1.3	0.5	0.3-0.5
Pb	%	6.2	6.0-7.0	52.0	49.0-52.0	1.5	1.0-2.0
Zn	%	6.0	5.0-9.0	4.5	3.9-5.8	52.0	46.0-52.0
Au	g/t	18	15-24	20	16-21	1.2	1.0-1.3
Ag	g/t	3,500	2,500-4,500	2,100	1,800-2,500	165	150-180
As	ppm	6,000	4,800-6,000	5,650	5,350-6,100	2,450	2,400-2,580
Sb	ppm	16,500	11,500-20,000	8,000	6,700-9,500	500	-
Se	ppm	580	550-620	2,800	2,300-3,500	220	190-250
Hg	ppm	25	18-28	18	15-19	160	140-170
Cd	ppm	500	-	350	-	3,500	3,300-3,800
Fe	%	25	-	13	-	10	9-12
F	ppm	70	-	<20	-	200	100-300
Cl	ppm	<100	-	<100	-	<100	<100
Mo	ppm	90	-	80	-	80	-

Element	Unit	Copper concentrate		Lead concentrate		Zinc concentrate	
		Quality A		Quality A		Quality A	
		Typical	Range of key elements	Typical	Range of key elements	Typical	Range of key elements
Co	ppm	30	-	50	-	<20	-
S	%	33	-	25	-	33	-
SiO_2	%	0.2	-	0.3	-	0.1	-
Sr	ppm	<10	-	<10	-	<10	-
Al	ppm	200	-	200	-	200	-
Ba	ppm	900	-	600	-	550	-
Be	ppm	<5	-	<5	-	<5	-
Bi	ppm	130	-	1,000	-	60	-
Ca	ppm	1,000	-	700	-	1,700	-
Cr	ppm	100	-	500	-	300	-
K	ppm	200	-	<100	-	400	-
Li	ppm	<5	-	<5	-	<5	-
Mg	ppm	700	-	200	-	600	-
Mn	ppm	200	-	100	-	200	-
Na	ppm	80	-	100	-	<50	-
Ni	ppm	200	-	500	-	300	-
P	ppm	<100	-	<100	-	<100	-
Ti	ppm	<100	-	<100	-	<100	-
V	ppm	10	-	<10	-	<10	-
Y	ppm	<100	-	<100	-	<100	-
Si	%	0.72	-	0.12	-	0.05	-
Sn	ppm	<200	-	<100	-	<100	-
Te	ppm	0.4	-	0.4	-	<0.2	-
U	ppm	<2	-	<2	-	<2	-
Zr	ppm	20	-	<20	-	<20	-

Table 19-2: Concentrate qualities – Quality B

Element	Unit	Copper concentrate		Lead concentrate		Zinc concentrate	
		Quality B		Quality B		Quality B	
		Typical	Range of key elements	Typical	Range of key elements	Typical	Range of key elements
Cu	%	25.0	21.0-26.0	1.2	0.8-1.4	0.5	0.3-0.5
Pb	%	3.5	2.0-5.0	52.0	49.0-52.0	1.5	1.0-2.0
Zn	%	3.8	2.0-5.0	3.3	3.1-3.6	52.0	50.0-52.0
Au	g/t	15	9-22	15	10-20	1.2	1.0-1.3
Ag	g/t	2,150	1,000-3,500	2,100	1,800-2,500	150	110-180
As	ppm	3,100	2,200-4,750	8,100	5,700-12,000	1,800	1,250-2,380
Sb	ppm	4,500	1,800-7,500	5,800	4,000-7,500	250	-
Se	ppm	530	400-680	2,950	2,400-3,800	300	220-500
Hg	ppm	12	9-16	15	9-23	150	100-170
Cd	ppm	500	-	250	-	3,500	3,300-3,800
Fe	%	27	-	15	-	9.4	8.5-11.0
F	ppm	70	-	<50	-	200	100-300

Element	Unit	Copper concentrate		Lead concentrate		Zinc concentrate	
		Quality B		Quality B		Quality B	
		Typical	Range of key elements	Typical	Range of key elements	Typical	Range of key elements
Cl	ppm	<100	-	<100	-	<100	<100
Mo	ppm	90	-	120	-	80	-
Co	ppm	30	-	50	-	<20	-
S	%	33	-	27	-	33	-
SiO_2	%	0.2	-	0.3	-	0.1	-
Sr	ppm	<10	-	<10	-	10	-
Al	ppm	200	-	200	-	200	-
Ba	ppm	900	-	700	-	700	-
Be	ppm	<5	-	<5	-	<5	-
Bi	ppm	130	-	1,000	-	90	-
Ca	ppm	1,000	-	700	-	2,400	-
Cr	ppm	100	-	500	-	300	-
K	ppm	200	-	<100	-	400	-
Li	ppm	<5	-	<5	-	<5	-
Mg	ppm	700	-	800	-	1,000	-
Mn	ppm	200	-	100	-	250	-
Na	ppm	80	-	100	-	80	-
Ni	ppm	200	-	500	-	300	-
P	ppm	<100	-	<100	-	<100	-
Ti	ppm	<100	-	<100	-	<100	-
V	ppm	10	-	<10	-	<10	-
Y	ppm	<100	-	<100	-	<100	-
Si	%	0.72	-	0.12	-	0.08	-
Sn	ppm	<200	-	<100	-	<100	-
Te	ppm	0.4	-	0.2	-	<0.2	-
U	ppm	<2	-	<2	-	<2	-
Zr	ppm	20	-	<20	-	<20	-

Deleterious elements that will attract penalty costs were identified from the concentrate qualities in Table 19-1 and Table 19-2 and predictive algorithms for the recovery to concentrate of these elements have been developed, as noted in Section 13.8.

19.2 Sales Contracts

There are no existing sales contracts for any concentrates at this time.

19.3 Zinc Concentrate Marketing and Concentrate Terms

The zinc concentrate produced will be a mid-grade material with payable silver content. There are some elements that may attract penalties, especially mercury and cadmium. The cadmium level exceeds the current limit for import into China; however, this market could be accessible through blending the product with other concentrates via trading houses. Other potential markets exist in the Asia region such as Korea and Japan but will be subject to respective concentrate quality criteria. Korea Zinc has indicated that chromium is a critical element as their smelters are limited in the Cr content of the ferrous product from their fuming furnaces. European markets remain a possibility.

There are several advantages of this concentrate. With many of the world's smelters integrated with additional processing facilities, the copper and lead content of the zinc concentrate represent value added opportunities. The very low silica content of the concentrate (0.1%) will be very attractive, as many of the world's large zinc mines contain silica levels of 4% and higher, and smelters are struggling with the silica load.

The zinc concentrate will be shipped out of the port of Stewart, BC. Logistics costs will likely be cheapest to Asia which is expected to be the primary destination for the zinc product; however, European ports are also accessible from this location.

The typical offtake terms expected to apply to the Quality A and Quality B zinc concentrate are listed below:

- Zinc – Pay 85% of content, subject to minimum deduction of eight units at the LME price
- Concentrate treatment charge – Long term US$225 per dmt of concentrate delivered with no price participation
- Gold credit – Deduct 1 g/t from the content and pay for 80% of the balance
- Silver credit – Deduct 93 g/t from the content and pay for 70% of the balance.

Penalty charges applicable to deleterious elements are listed below:

- Iron – US$1.50 for each 1% above 8%
- Cadmium – US$2.00 for each 0.1% over 0.3%
- Mercury – US$1.50 for each 10 ppm over 100 ppm.

19.4 Copper Concentrate Marketing and Concentrate Terms

The copper concentrate quality for the first 18 months (Quality A concentrate) will be complex, with elevated levels of lead, zinc, antimony and arsenic. The arsenic and lead levels exceed the current Chinese import limit for copper concentrates, but since the silver and gold levels are elevated this concentrate could possibly be sold directly into China as a precious metals concentrate. Alternatively, it could be blended with product from other producers via a trading house.

The precious metals content of the Quality B concentrate is lower compared to Quality A; however, the arsenic and lead levels are comfortably below the Chinese import limit, although cadmium is still at the limit. With the lower impurities, the Quality B product has more flexibility with respect to smelter destinations. Whilst still incurring penalties for arsenic and antimony, and to a lesser extent zinc and lead, the Quality B concentrate is expected to be directly marketable to mainstream smelters in China, Korea and Japan. In addition, given the high level of silver in the Quality B concentrate, the copper concentrate could possibly be shipped into China as a silver concentrate, depending on the final grades.

Chinese smelters will value the lead, zinc and antimony content, whereas these elements would be significant penalty items at Western copper smelters.

Complex concentrates, such as KZK Quality A, are unique materials and the terms for the purchase of such concentrates are likely to be unique. The typical offtake terms expected to apply to the Quality A and Quality B copper concentrate are listed below:

- Copper – Pay 96.5% of the content, subject to a minimum deduction of 1 percentage point, at the price for LME Grade A copper. Copper refining charge of US$0.15 per payable pound for Quality A concentrate, and US$0.085 per payable pound for Quality B concentrate.

- Concentrate treatment charge – Quality A concentrate will attract a non-traditional TC, estimated to be US$150 per dmt, while Quality B concentrate will attract benchmark TC, estimated to be US$85 per dmt.
- Gold credit – The sale of Quality A concentrate will be subject to a gold payment of 90% of content and is reduced from the standard gold payment to allow for the negative impact of blending if required. The sale of Quality B concentrate will be subject to a gold payment of 96% of content. A gold refining charge of US$5 per payable ounce will apply to both concentrate qualities.
- Silver credit – For both concentrate qualities, silver grades greater than 30 g per dmt will be subject to a payment of 90% of content. A refining charge of US$0.50 per payable ounce will apply to both concentrate qualities.

Penalty charges applicable to deleterious elements are listed below:

- Zinc plus lead – US$3.00 for each 1% of lead and zinc above 3% up to 6%, plus US$6.00 for each 1% of lead and zinc above 6%
- Arsenic – US$3.00 for each 0.1% above 0.2% up to 0.5%, plus US$6.00 for each 0.1% above 0.5%
- Antimony – US$3.00 for each 0.1% above 0.2%
- Selenium – US$2.00 for each 100 ppm above 300 ppm
- Mercury – US$1.00 for each 10 ppm above 10 ppm.

19.5 Lead Concentrate Marketing and Concentrate Terms

The lead concentrate is a mid-grade concentrate with good silver and gold values.

The lead concentrate contains levels of zinc, copper and antimony, which could potentially be recovered by Chinese lead smelters as economic by-products. The Quality A lead concentrate is expected to be directly sold into China, whereas the Quality B concentrate will exceed the current arsenic import limit in China of 0.7%. This together with elevated chromium levels means the Quality B concentrate is expected to be sold directly to Western smelters, or to trading houses for blending.

The typical offtake terms expected to apply to the Quality A and Quality B lead concentrate are listed below:

- Lead – Pay 95% of the content, subject to a minimum deduction of 3 percentage points. Minimum deduction to apply at lead grades less than 60%.
- Treatment charge – US$180 per dmt.
- Gold credit – Pay 95% of gold content, subject to a minimum deduction of 1 g/t. A gold refining charge of US$10.00 per payable ounce will be applicable.
- Silver credit – Pay 95% of silver content, subject to a minimum deduction of 50 g/t. A silver refining charge of US$0.80 per payable ounce will be applicable.

Penalty charges applicable to deleterious elements are listed below:

- Arsenic; US$1.50 for each 0.1% above 0.5%.

Certain smelters may also levy a penalty charge on selenium content; however, StoneHouse (2018) recommended that a penalty for selenium in lead concentrate not be considered for the DFS.

19.6 Commodity Prices and Foreign Exchange Rate

Commodity prices used for the DFS economic model are consensus prices, established by taking the average price forecasts from a range of financial institutions between as at June 30, 2019 are presented in Table 19-3.

Table 19-3: Average price forecasts (as at June 30, 2019)

Parameter	Unit	2019	2020	2021	2022	Long term
Copper	US$/lb	$2.94	$3.03	$3.12	$3.31	$3.15
Lead	US$/lb	$0.94	$0.95	$0.95	$0.96	$0.95
Zinc	US$/lb	$1.24	$1.18	$1.15	$1.12	$1.10
Gold	US$/oz	$1,304	$1,335	$1,337	$1,331	$1,321
Silver	US$/oz	$15.74	$16.66	$17.02	$17.46	$18.09
Exchange rate	CAD/US$	0.758	0.764	0.770	0.779	0.782

20 Environmental Studies, Permitting, and Social or Community Impact

20.1 Environmental Assessment and Permitting

The KZK Project, as proposed in the DFS, will require major authorizations issued under two Territorial statutes and two Federal statutes, as set out below.

- ***Yukon Environmental and Socio-economic Assessment Act* (YESAA)** and various regulations (Federal), which mandates a public process for assessing the Project's potential socio-economic and environmental impacts. YESAA screening at the Executive Committee level is triggered by ore production capacity of greater than 1,500 t/day. KZK ore production will be approximately 6,000 t/day.
    - BMC initiated the Environmental Assessment of the KZK Project in March 2017 by submitting a Project Proposal to the Executive Committee of the Yukon Environmental and Socio-economic Assessment Board (YESAB). The Project Proposal was deemed Adequate in January 2018 and passed through to the Screening stage of assessment, a multi-stage public review process. During the Screening stage, BMC has continued to respond to information requests made by YESAB as the Screening review of the Project continues.
    - At the end of Screening, YESAB will issue a Screening Report to the Yukon Major Projects office and the Department of Fisheries and Oceans (the Decision Bodies). The Decision Bodies will review the Screening Report and will either issue a Decision Document accepting the recommendations or refer the recommendations back to the YESAB Executive Committee for reconsideration.
- ***Quartz Mining Act***, and *Mining Land Use Regulations* (Yukon Territory) are administered by the Government of Yukon's Department of Energy, Mines and Resources (EMR) and issue a Quartz Mining Licence (QML) for commercial mineral production. The QML is supported by a number of plans describing the development, operation and closure of the planned mining operation. Following issuing of a Decision Document, application can be made for the QML to enable mining activities to commence. The QML is an active licence and will continue to be developed and expanded to reflect the planned works required for development, operation and closure of the Project.
    - The amount of security bonding required to offset Governments' liability will be assessed and secured under this licence. The determination of the amount of security is made through development of the detailed, costed Reclamation and Closure Plan (RCP) for current conditions. A bond for the security is typically paid in tranches in accordance with Governments' determination of amount required to offset its' liability based on current site conditions. Security bonding provisions are reassessed through annual operational reports and biannual updates to the RCP and are adjusted (up or down) to reflect estimates of closure liabilities.
- ***Waters Act*** and *Waters Regulations* (Yukon Territory), under which the deposit of waste and the use of water for processing (greater than 100 tonnes per day) requires issuance of a Type A Water Licence. This license is issued by the Yukon Water Board which follows a quasi-judicial review process including a formal public hearing. The Water Licence Application process also requires that applications are deemed "adequate" before they are accepted for public review and licensing determination. Type A licence applications undergo a Public Hearing prior to the licence being written.
    - Current exploration activities are authorized under a Sched III Notice as well as the existing Type B Water Licence. Following issuing of the Decision Document, an amendment to the Type B Water

Licence will be secured to enable preliminary site construction activities to commence in advance of the receipt of all final permits (including the Type A Water Licence) for operations.

  - Water Licences, issued for a maximum of 25 years, will include various operational management plans, terms and conditions of water use and deposit of waste, and monitoring and reporting requirements. Although not normally utilized, the *Waters Act* provides for the posting of security should the Water Board determine that additional security beyond the amount assessed for the QML. The Water Board process commences once the YESAB has issued the Decision Document.

- ***Fisheries Act*****,** and *Metal and Diamond Mining Effluent Regulations* (Federal), under which a Fish Offsetting Plan will be developed in collaboration with the Federal Department of Fisheries and Oceans (DFO), to offset the disruption/destruction of upper Geona Creek during operations.
  - Any proposed construction measures (associated with the Fish Offsetting Plan) have been incorporated in the Project Proposal for review by YESAB, and DFO may not issue the final authorization for the plan until after the Decision Document. Community consultation about the Fish Offsetting Plan is a required component of DFO's regulatory process.
  - The *Metal and Diamond Mining Effluent Regulations* prescribe monitoring and reporting requirements during operations through the adoption of an Environmental Effects Monitoring program. There is a requirement to continue this program post closure, until the mine is designated closed by DFO upon application by BMC.

There are also numerous "minor" permits that will be required (e.g. building permits) which are secured as and when they are needed and typically do not affect the overall KZK Project development schedule and do not represent significant operational cost considerations. The numerous minor permits are relatively straightforward and will be applied for as required. Minor permit processes will occur concurrently during major permit strategic permitting.

For the purposes of preparing the DFS, milestone dates were established for receipt of key permits, as detailed in Table 20-1. These milestones will continue to be monitored post completion of the DFS to determine the impact on project delivery.

Table 20-1: Key permitting milestones

Milestone	Date
Final Screening Report (YESAB)	December 2019
Decision Document (Yukon Major Projects, DFO)	January 2020
Quartz Mining Licence (EMR)	April 2020
Type A Water Licence (Water Board)	July 2021

20.2 Environmental Studies

The environmental and socio-economic conditions in and around the project area are well characterized. Baseline environmental and socio-economic studies were initiated in 1994/1995 to support the Initial Environmental Evaluation (Cominco Ltd., 1996) submitted for regulatory review in March 1996 and approved in December 1997. Additional baseline studies were conducted in 1996 to support the Type A Water Licence Application (Licence QZ97-026, approved in December 1998 and expired in September 2018). Baseline studies (water quality and aquatic resources) were conducted every two years between 1998 and 2018, to meet the requirements of the water licence. In 2015, BMC initiated a full suite of environmental baseline studies, to support the Executive Committee Screening of the Proposed KZK mine. The fourth consecutive year of these studies was completed in March of 2019.

The following provides a brief summary of the environmental baseline conditions (based on the data collected up until March 2018).

20.2.1 Climate

A meteorological station was installed and commissioned at the KZK Project in August 2015, for the measurement of air temperature, relative humidity, wind speed and direction, barometric pressure, solar radiation and total precipitation. An evaporation pan was also installed, and data collected during the open water seasons (approximately the end of June to September) of 2016 through 2018. Manual snow surveys were conducted monthly (January, February, March and April) in 2016 through 2018.

In 2018, climate and hydrological data collected at KZK between 2015 and 2017, and regional datasets collected by Environment and Climate Change Canada, were analysed to provide long-term estimates of average and extreme hydrometeorology conditions at KZK (Knight Piésold, 2018).

All values below are given for the Kudz Ze Kayah climate station (elevation 1,542 masl):

- The long-term mean annual temperature is estimated to be -2.8°C, with minimum and maximum mean monthly temperatures estimated to be -12.9°C and 9.9°C in December and July, respectively.
- The long-term mean annual precipitation is estimated to be 520 mm. This value is less than values previously estimated for the site, but is supported by site and regional datasets, and the Baseline Watershed Model analysis, which integrates precipitation, losses (e.g. evapotranspiration) and streamflow to generate a consistent hydrologic cycle at the KZK Project site.
- Precipitation at the site is split between rain and snow, with approximately 37% of it estimated to fall as snow, on average.
- The 24-hour 100-year, 200-year, and probable maximum precipitation values are estimated to be 89 mm, 95 mm, and 274 mm, respectively.
- The 1:10-year wet annual precipitation is estimated to be 646 mm, and the 1:10-year dry annual precipitation is estimated to be 394 mm.

20.2.2 Terrain

The KZK Project occurs within the Pelly River and Pelly Mountain ecoregions. It is located within the northern foothills of the Pelly Mountains of the Yukon Plateau, on the east side of the divide between the Pelly River and the Liard River drainage basin. The topography of the area consists of rolling hills, locally with ponds and lakes occupying valley bottoms (Figure 20-1).

The project area was glaciated, and bedrock exposures typically occur only in deep ravines or on steep slopes where post-glacial erosion removed overburden. Valley bottoms are covered with till and glaciofluvial sediments that are locally overlain by alluvial fan sediments. Colluvial apron sediments are also common. The project is located within the discontinuous but widespread permafrost zone, with permafrost typically within approximately 2 m of the surface.

Figure 20-1: Landscape hosting the ABM deposit within the KZK Project area

Photo is looking northwest (in the direction of streamflow at this location). Creek flowing down the Geona Creek valley.

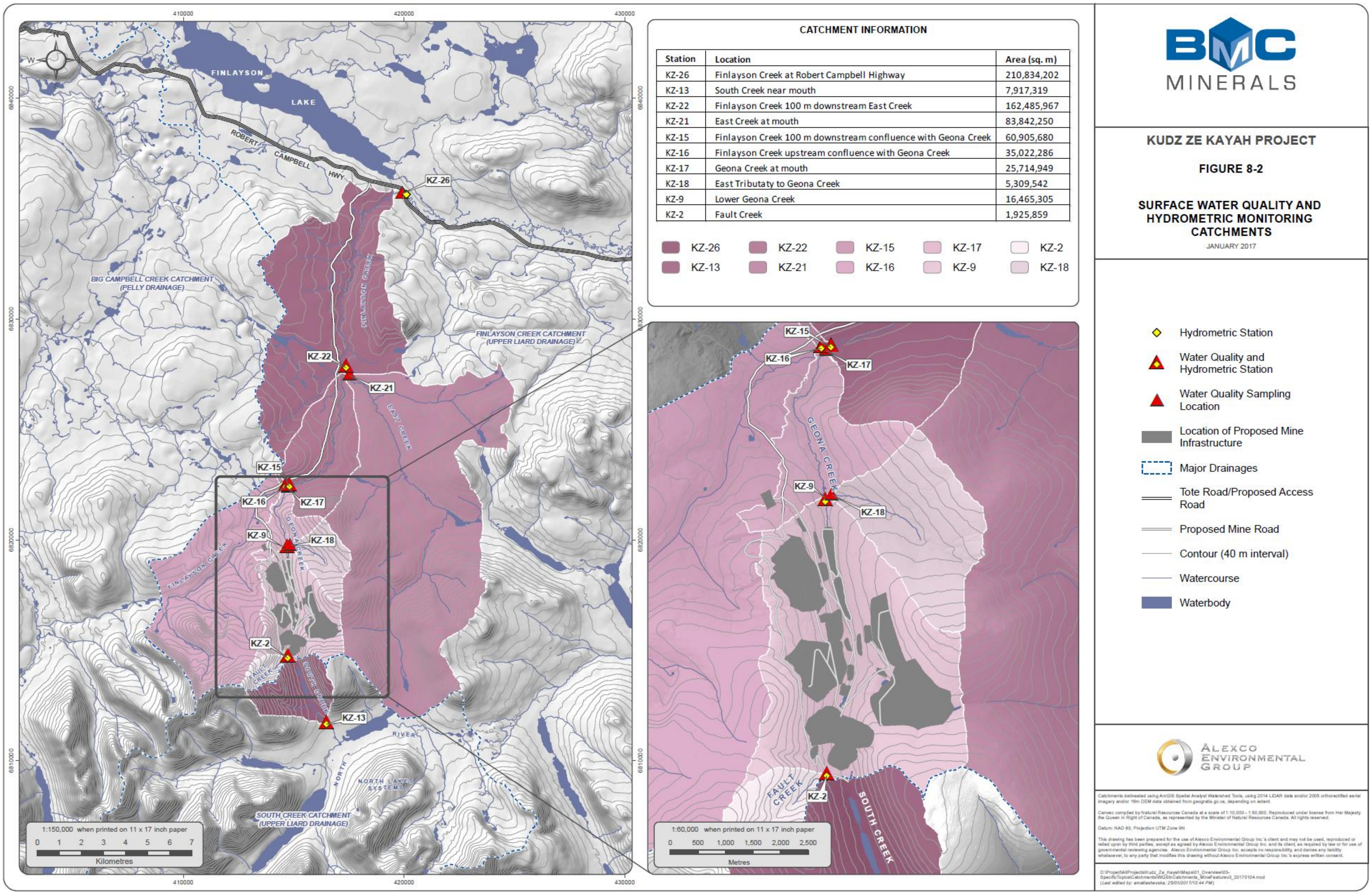
20.2.3 Hydrological Assessment

Hydrological Overview

The KZK Project lies in the Geona Creek watershed, central to which is Geona Creek, a north-flowing tributary to Finlayson Creek. Finlayson Creek meets the outflow of Finlayson Lake below the Robert Campbell Highway and flows east to eventually join the Frances River and ultimately the Mackenzie River.

The Geona Creek watershed covers approximately 26 km^2 (Figure 20-2), has a median elevation of 1,479 masl and spans from the alpine to forested areas at lower elevations. The Finlayson Creek catchment area is approximately 35 km^2 above the confluence with Geona Creek and expands to 211 km^2 where it flows under the Robert Campbell Highway, shortly before it joins the outflow of Finlayson Lake. The southern watershed divide between Geona Creek and South Creek is located immediately south of the ABM deposit and is characterized by several small lakes, locally referred to as “South Lakes”.

Fault Creek is the most significant tributary to Geona Creek in the deposit area, emptying into Geona Creek immediately south of the ABM deposit. The small Fault Creek catchment area (2 km^2, 1,708 masl median elevation), to the west of Geona Creek is steeper, with similar vegetation.



Station	Location	Area (sq. m)
KZ-26	Finlayson Creek at Robert Campbell Highway	210,834,202
KZ-13	South Creek near mouth	7,917,319
KZ-22	Finlayson Creek 100 m downstream East Creek	162,485,967
KZ-21	East Creek at mouth	83,842,250
KZ-15	Finlayson Creek 100 m downstream confluence with Geona Creek	60,905,680
KZ-16	Finlayson Creek upstream confluence with Geona Creek	35,022,286
KZ-17	Geona Creek at mouth	25,714,949
KZ-18	East Tributaty to Geona Creek	5,309,542
KZ-9	Lower Geona Creek	16,465,305
KZ-2	Fault Creek	1,925,859

Figure 20-2: Surface water catchment

Hydrology

Site discharge hydrographs are typically characterized by high spring snowmelt-driven flows, lower summer flows sustained by groundwater inflows and periodic rainfall events, followed by large autumn rainfall events. Winter flow is very low as a result of cold temperatures, freezing conditions and the gradual depletion of groundwater storage.

All years show a distinct flow peak in late May or June and another peak in September and early October. Many hydrographs also show rainfall-induced peaks in mid-summer, particularly in July. Consistent with typical hydrologic patterns, those stations with higher elevations and smaller catchments tend to experience higher unit runoff during the freshest and lower unit runoff during the summer.

Table 20-2 provides estimates for various hydrological parameters at South Creek (KZ-13), Fault Creek (KZ-2), Geona Creek below the project infrastructure (KZ-9), Geona Creek below the confluence with the tributary below KZ-9 (KZ-37), Geona Creek above the confluence (KZ-17), Finlayson Creek below the confluence with Geona Creek (KZ-15), and Finlayson Creek at the Robert Campbell Highway (KZ-26).

Table 20-2: Hydrological statistics for various catchments within the KZK Project area

	Site/Catchment						
	KZ-13	**KZ-2**	**KZ-9**	**KZ-37**	**KZ-17**	**KZ-15**	**KZ-26**
Catchment area (km^2)	7.92	1.93	16.5	21.8	25.7	60.9	211
Mean annual runoff (mm)	425	610	388	388	373	386	249
Mean annual flow (m^3/s)	0.107	0.037	0.202	0.268	0.304	0.745	1.664
Mean summer flow (m^3/s)	0.179	0.063	0.340	0.450	0.511	1.252	2.796
Mean annual low flow (monthly (m^3/s)	0.022	0.008	0.041	0.054	0.061	0.150	0.335
Mean annual low flow (daily (m^3/s)	0.013	0.003	0.028	0.037	0.044	0.104	0.358
Mean summer low flow (monthly) (m^3/s)	0.161	0.056	0.306	0.404	0.459	1.126	2.514
Mean summer low flow (daily) (m^3/s)	0.086	0.029	0.161	0.213	0.241	0.594	0.997
Mean annual flood (daily) (m^3/s)	0.526	0.339	1.063	1.406	1.63	3.731	12.288
Mean summer flood (daily) (m^3/s)	0.52	0.367	1.004	1.328	1.496	3.244	9.879

Surface Water Quality Baseline

Creeks that drain the KZK Project were circumneutral to alkaline (pH 6.7 to 8.7; median 7.7) with hardness increasing from moderately hard (South Creek and Fault Creek) to hard (Geona Creek) in the upper watershed to very hard (Finlayson Creek and East Creek) in the lower watershed. Dissolved organic carbon ranged from less than 0.5 mg/l to 17.2 mg/l, with the highest concentrations measured in Geona Creek. At all surface water stations, naturally occurring nitrogen species (nitrate, nitrite, cyanide, ammonia) were all typically below or marginally above the detection limit, with the exception of nitrate-N, which ranged from median concentrations of 0.01 mg/l in East Creek to 0.16 mg/l in Fault Creek. Nitrate-N concentrations were typically highest during the winter months; however, no concentrations exceeded the CCME threshold of 3 mg/l.

Water quality was compared against the most recently revised water quality guidelines for protection of aquatic life established by the Canadian Council of Ministers of the Environment (CCME) or BC Ministry of the Environment. Water quality guideline exceedances were observed sporadically for a number of constituents including total concentrations of aluminum, arsenic, cadmium, chromium, copper, iron, lead, mercury, selenium, and zinc. Majority of these exceedances (except selenium) coincided with freshet, when total suspended solid concentrations were highest and metal(loid)s were largely transported as particulates.

In general, more water quality guideline exceedances were noted for total metal concentrations than their dissolved counterparts, suggesting that a significant portion of the metals were particulate-bound, especially during freshet and/or other periods characterized by elevated total suspended solid levels. The fact that, dissolved metal concentrations exhibited much less frequent water quality guideline exceedances is important, since it is the dissolved fraction that is the most bioavailable.

20.2.4 Hydrogeological Assessment

Groundwater

The most recent groundwater report for the Project was published in April 2018 and includes a description of the groundwater elevations, flows, and chemistry, based on data collected in 1995 and from 2015 to 2017 (three consecutive years of data collection) (AEG, 2018a).

The current monitoring network consists of 32 bedrock and overburden wells strategically installed around the Project area that are monitored quarterly.

The principal hydrogeologic units at KZK are bedrock and overburden. The overburden consists of two subunits:

- Fine-grained lower permeability sediments composed of silts and fine sands
- Coarse-grained higher permeability sands and gravels.

In the depth range of 10–70 m below ground surface, the bedrock hydraulic conductivity generally ranges from 1×10^{-7} m/s to 1×10^{-5} m/s and does not appear to exhibit a trend of increasing or decreasing hydraulic conductivity with depth. The geometric mean of short-term tests conducted in bedrock is 1.2×10^{-6} m/s, which is similar to the results of a longer-term bedrock pumping test (1.7×10^{-6} m/s). Faults and fracture zones can be expected to have higher hydraulic conductivity than the surrounding bedrock. During calibration of the groundwater model, the hydraulic conductivity of the Fault Creek Fault was assigned to be 3.5×10^{-6} m/s based on a packer test which is interpreted to have tested the Fault Creek Fault.

For tests conducted in the fine-grained overburden, the measured hydraulic conductivities have a geometric mean of 5.2×10^{-6} m/s. Based on two field tests, including a 2015 long-term pumping test conducted by EBA, the hydraulic conductivity of the coarse-grained overburden is about 1.3×10^{-4} m/s.

Continuous groundwater level monitoring was conducted in eight monitoring wells across the site from mid-November 2015 through November 2017. With varying levels of intensity, the water levels in both bedrock and overburden wells exhibited the following seasonal trends:

- Rising water levels through the summer months (approximately May to August)
- Peak water levels reached between August and September, depending on the year
- Falling water levels through the winter months (approximately October to March)
- Lowest levels reached between April and May, depending on the year.

In most monitoring wells, the maximum-minimum water level difference ranged between 2 m and 8 m; the maximum observed difference was 14 m.

Project-wide, the groundwater field pH ranged from circumneutral to slightly alkaline (5.68 to 8.63, or an average value of 7.5) for both bedrock and overburden wells.

Water quality results were compared against the Yukon Contaminated Sites Regulation Standards, which indicated a few exceedances for dissolved cadmium and cobalt, and single exceedances of dissolved arsenic and zinc. Sulphate and fluoride concentrations were generally higher in the wells screened in bedrock compared to overburden; however, the concentrations of other anions, nutrients, and metals did not show marked differences between overburden and bedrock wells. Groundwater sampled in the proposed open pit area generally returned higher anion, nutrient, and metal concentrations than groundwater sampled elsewhere on the KZK Property. Groundwater concentrations of cadmium, iron, and zinc were elevated in the proposed open pit area relative to the rest of the KZK property, likely due to the subsurface mineralization present in this area. Additionally, sulphate concentrations were typically more elevated within the proposed pit area, likely due to the oxidation of the sulphidic minerals in the deposit.

Hydrogeological Assessment

The hydrological assessment was conducted by Tetra Tech (2019). The data collected for use in the hydrogeological assessment study included hydrogeological wells (Figure 20-3), geologic zonations, recharge rates, stream and lake data, water level target data, permafrost mapping, packer testing data collected in the footprint of the proposed open pit, and aquifer testing of the rock in and around the fault zones.

Steady-state and transient groundwater flow models were constructed and calibrated as part of this study. The steady-state model was calibrated to pre-mining water level elevations and Geona Creek base flows. The steady-state model was then used as initial conditions for the transient flow model. The transient flow model was calibrated to the long-term aquifer tests conducted as part of this study to determine values for hydraulic conductivity and storage. During calibration, a sensitivity analysis was conducted on the transient flow model to help select which parameters should be adjusted in calibration and which could be left at values derived from field observation or professional judgement.

Following the calibration process, the groundwater flow model was used to simulate the hydrological sequence associated with the nine-year excavation of the ABM pit and underground workings. Model simulations were conducted to evaluate pathways for potential contaminant migration and travel time from the pit, the waste storage facilities, and the water management ponds during mine decommissioning and closure. Post-mining model simulations assumed the underground workings are closed and do not interact with the pit lake expected to form followed by the re-diversion of Fault Creek into the pit to flood it over time, as specified in the proposed mine plan.

Due to the interconnectedness of faults, such as Fault Creek, from the underground workings to the pit, there is not expected to be significantly increased pressure heads within the plugged underground workings relative to the pit. Particle tracking was implemented to examine potential contaminant pathways from each of the site features including the pit and to estimate travel times from the pit to Geona Creek.

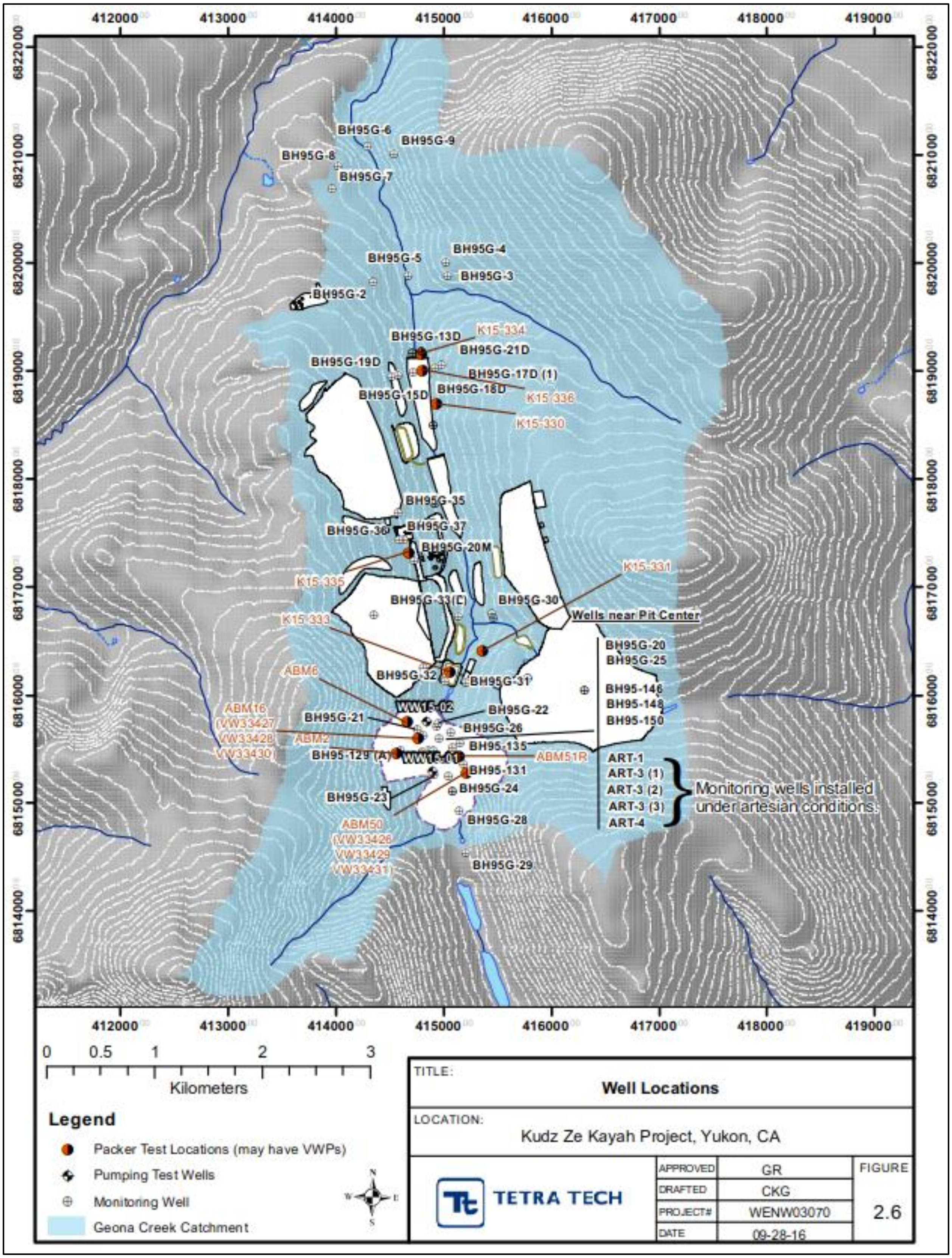


Figure 20-3: Hydrogeological well locations
Source: Tetra Tech (2019)

Based on the modelling results presented in this report, the following conclusions were arrived at:

- The groundwater flow modelling suggests that pre-mining dewatering will require four trenches to be placed and oriented orthogonally to the original trench (Figure 20-4). Although the initial month of dewatering will produce higher rates of flow (around 8,100 m^3/day, or 94 L/s), the final month should be reduced to around 2,200 m^3/day.
- Except for areas of faulting or fracturing, the pit bedrock appears to be of sufficiently low permeability to permit water seepage to be managed using face seepage drains and horizontal drains in the pit wall as necessary. Depending on the nature of the distribution of fracture sets or other prominent fault conduits intersecting the pit within the bedrock, it was suggested it may be worthwhile to install approximately 15 100-m deep dewatering wells, arrayed at 500 m spacing around the perimeter of the pit, to fully dewater the pit slope. Assuming groundwater flow occurs through a reasonably isotropic weathered bedrock with interconnected fractures, these wells may be sufficient to dewater the weathered bedrock around the pit to minimize seepage face flow. Provision was made for eight 150 m deep wells around the pit perimeter.
- Fault zones within the underground workings are expected to produce water at higher rates of discharge and require the drilling of horizontal drains to stabilize hydraulic conditions locally. These will drain an estimated 1,100 m^3/day from the entire underground workings. Due to uncertainty on the lateral extent of the fault zones that intersect the underground workings, flow from the faults may be brief as they locally drain or sustained if they are hydraulically connected to bedrock around an extensive fault-conduit feature that extends further from the pit.
- Following completion of mining and closing of the underground workings, the pit will begin to refill through the combination of redirected surface water flow from Fault Creek, and groundwater seepage as the drawdown associated with mining begins to subside and groundwater levels begin rising. The pit is expected to have filled to half its depth approximately four years after the end of operations and will fill completely to the spill elevation of 1,380 masl after approximately 16 years.
- After the pit has filled, the pit is expected to act as a lake (referred to as ABM Lake) through which streamflow enters and leaves, and which is augmented by groundwater discharge of approximately 1,400 m^3/day.
- Tracking of particles, sourced at each of the storage facilities, flow toward Geona Creek where they either immediately discharge to the stream, or travel through the overburden along the stream valley until they eventually discharge to the stream.
- Tracking of particles originating at the ABM pit lake, flow north away from the pit, following the upward hydraulic gradients in the bedrock and overburden, until they discharge to Geona Creek, approximately 1 km north of the ABM Lake.

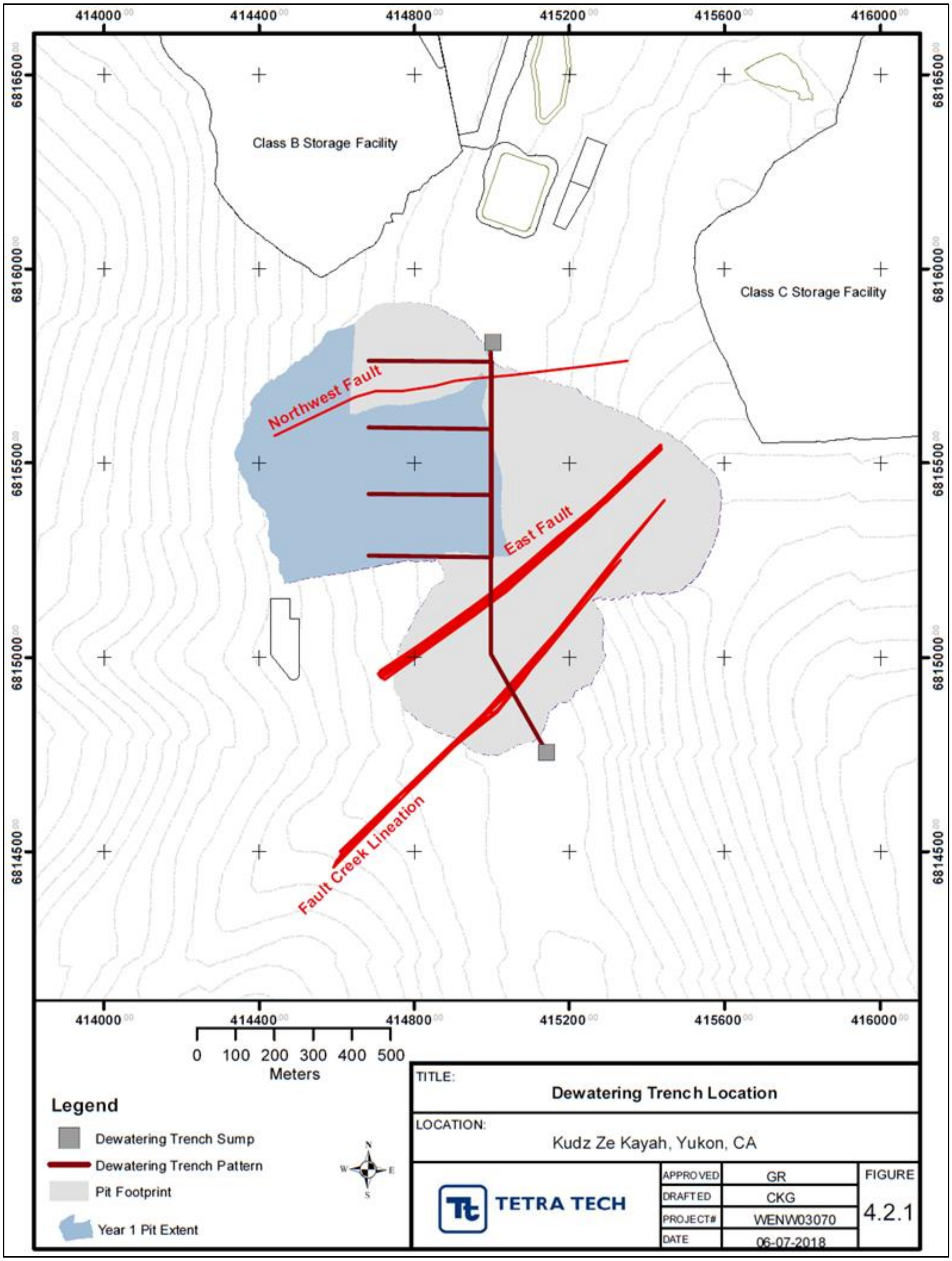


Figure 20-4: Proposed dewatering trench locations
Source: Tetra Tech (2019)

20.2.5 Surface Water Quality

The most recent surface water quality baseline report for the KZK Project was published in May 2018 (AEG, 2018b). The report summarizes the data collected:

- In 1994/1995 to support the Initial Environmental Evaluation
- In 1996 to support the water licence application
- Between 2000 and 2016 to support the water licence monitoring
- From 2015 to 2018 (three consecutive years of monthly data collection, to support the Project Proposal, which is currently being reviewed by the YESSAB Executive Committee) (AEG, 2018b).

The program currently consists of monthly water quality sampling at twelve surface water stations located on South Creek, Fault Creek, Geona Creek, Finlayson Creek, and East Creek.

All creek waters were circumneutral to mildly alkaline (pH 6.7 to 8.7; median 7.7), with hardness increasing from moderately hard (South Creek and Fault Creek) to hard (Geona Creek) in the upper watershed to very hard (Finlayson Creek and East Creek) in the lower watershed.

Water quality was compared against the most recently revised water quality guidelines for protection of aquatic life established by the Canadian Council of Ministers of the Environment (CCME) or British Columbia Ministry of the Environment (BCMoE). Comparison to the guidelines indicated sporadic exceedances for fluoride, phosphorus, aluminum, arsenic, cadmium, chromium, copper, iron, selenium, and zinc. In general, more water quality guideline exceedances were noted for total metal concentrations than their dissolved counterparts, suggesting that a significant portion of the metals were particulate bound, especially during freshet and/or other periods characterized by elevated total suspended solids levels. The much less frequent water quality guideline exceedances observed for dissolved metals is important since it is the dissolved fraction that is the most bioavailable.

Higher total metal concentrations were measured during spring freshet due to the high runoff and associated stream flow which increased the particulate content of the streams. The exceptions to this trend were concentrations of fluoride, total selenium, uranium, and hardness, which were lowest during spring freshet and summer. Selenium, uranium, and hardness concentrations rose throughout the late fall and winter months, suggesting that groundwater is an important contributor to the surface water concentrations of these constituents. Elevated concentrations of these constituents are observed in shallow groundwaters of the KZK Project area.

A higher frequency, five samples in 30 days sampling program was carried out at most monitoring locations in May to June 2017 and February to March 2018. Concentrations were broadly within the range of concentrations observed in May to June 2015 and 2016 and peak concentrations in May 2017 were comparable to historical May peak concentrations. A similar observation was made for the February to March 2018 dataset, which exhibited limited variability for most parameters and returned concentrations comparable to those observed in February to March 2016 and 2017.

20.2.6 Water Quality Modelling

A water quality model has been developed to estimate the water quality in mine discharge and receiving environment waters and to help guide water management for the KZK Project with respect to water quality (AEG, 2018e), (AEG, 2018f) and (AEG, 2018g). The model incorporates data from the groundwater and surface water quality baseline monitoring, meteorological studies, site water balance data, hydrogeological modelling results, and laboratory and field based geochemical testing of trace element leaching from site material.

Effluent water quality from the WTP considered treated concentrations for parameters reported by WTP consultant, BQE Water or applied specified factors. Water quality modelling was performed on a monthly time step through the Construction, Operations and Closure phases of the project to estimate seasonal variation.

During the Construction phase, "contaminants of potential interest" concentrations in the Geona and Finlayson Creek receiving environment were estimated to be generally comparable to, or slightly lower than baseline due to the diversion of upstream waters (i.e. Fault Creek) and dilution from discharge of the ABM Open Pit dewatering water for all scenarios. The Fault Creek diversion resulted in increased estimated selenium, cadmium and zinc concentrations in South Creek, but don't exceed preliminary water quality objectives.

During Operations, contaminants of potential interest concentrations in the water management pond discharge increased slightly each year, reflecting increases in load contribution from pit wall runoff as the ABM Open Pit is excavated. No water quality parameter concentrations were estimated to exceed their respective preliminary water quality objective exceedance limit under any precipitation scenario.

During the Active and Transition Closure phases, contaminants of potential interest concentrations in South Creek were estimated to return to baseline levels as the Fault Creek diversion is removed, allowing South Creek to revert to its pre-project flow regime. Elsewhere, contaminants of potential interest concentrations in the Geona and Finlayson Creek receiving environments increased during the Active Closure phases due to drain-down of the Class A and B storage facilities; however, water from the drain-down of the facilities will be treated by the WTP and loading from those facilities were not estimated to cause exceedances in the receiving environment. Nitrite-N marginally exceeded its preliminary water quality objectives for one and eight months in Active Closure under mean and dry year scenarios, respectively, but not in the wet year scenario. Infrequent, slight exceedances of the copper preliminary water quality objective were estimated by the model during the Transition Closure phase in Geona Creek in all precipitation scenarios. No contaminants of potential interest exceedances were estimated in Finlayson Creek or South Creek during the Active and Transition Closure phases.

During the Active and Transition Closure phases, flooding of the ABM open pit will occur due to the cessation of local groundwater dewatering and the removal of the Fault Creek diversions. It is anticipated that the ABM open pit will take 16 years to fill before the ABM Lake overflows into Geona Creek, via the constructed wetland treatment systems. Modelling suggests that the ABM Lake will contain elevated concentrations of antimony, arsenic, cadmium, copper, lead, selenium, uranium, and zinc due to solubilization of the load accumulated on the pit floor and walls; however, in situ treatment of the ABM Lake is anticipated to significantly lower the concentrations of these elements.

Outflow from ABM Lake marks the start of the Post-Closure phase, resulting in increases in some contaminants of potential interest concentrations in Geona Creek. No exceedances of any preliminary water quality objectives were estimated in Geona Creek, Finlayson Creek or South Creek during Post-Closure in the precipitation scenarios evaluated. The results from the water quality modelling indicate no significant adverse effects are anticipated as all the estimated concentrations were below the environmental assessment high magnitude threshold of two times the preliminary water quality objective.

20.2.7 Aquatic Ecosystems and Resources

The most recent aquatics and ecosystems resources baseline report for the KZK Project was published in November 2016 (AEG, 2016a) and includes a summary of the baseline studies conducted in the 1990s, 2000s and 2015 and 2016. The report presents data and observations of baseline fish and fish habitat, stream sediments, benthic invertebrates, periphyton, and chlorophyll *a* surveys. Additional supporting information describing water quality and other aquatic ecosystem physical parameters is also provided below for an overall ecosystem context. No aquatic rare, endangered or species of special concern were identified.

Sediments

Sediments over the monitoring period (i.e. from the 1990s through to 2016) have shown year-to-year variability and site-to-site variability. However, in general arsenic, cadmium, copper and zinc are elevated to varying degrees in sediments throughout the study area, indicating that these drainages lie within naturally mineralized zones.

Fish

Geona Creek and the small ponds overlying the ABM deposit generally have low abundances of fish, containing just a few slimy sculpin (*Cottus cognatus*) and arctic grayling (*Thymallus arcticus*). Somewhat further downstream, adult arctic grayling occur in Finlayson Creek. Fish sampling in South Creek, North River, and East Creek indicated the presence of Arctic grayling, slimy sculpin, and burbot. No fish were captured in Fault Creek, despite significant effort. Larger rivers and lakes in the area host grayling, lake whitefish, lake trout, burbot and dolly varden.

Benthic Invertebrates

Benthic invertebrates were collected for community composition and tissue metal analysis from Geona Creek, Finlayson Creek, North River, and East Creek during all aquatic sampling events.

Benthic invertebrate abundance and density was highest in Geona Creek and lowest in Fault Creek, and diversity was greatest in lower Finlayson Creek. Diptera (true flies) were the dominate taxon at most sample locations within the Project area. Diptera, Plecoptera or Ephemeroptera share dominance in Geona Creek downstream of the Project footprint, and in the upper portions of Finlayson Creek. The downstream reach of Finlayson Creek is the only site where *Oligochaeta* (aquatic worms) were the dominant taxon or shared dominance with Diptera. Results from 1995 to 2016 have shown fluctuations in both the density and community composition of benthic invertebrates at all sample locations, but these fluctuations are considered natural as no development has occurred in any of the drainages studied.

The mean metal concentrations documented during the 1995–2016 studies were well below the mean concentrations in the Yukon database (Laberge and Can-Nic-A-Nick, 2015; 2016), except in 2014 when metal concentrations were an order of magnitude higher than all other years. Elevated concentrations were not observed again in 2016. High metal concentrations observed in 2014 are believed to be from the laboratory testing the samples as plant tissue rather than benthic invertebrate tissue.

Periphyton

In 2015, Periphyton sampling occurred in Finlayson Creek, Geona Creek, and Fault Creek. Abundance and diversity of periphyton was highest in Geona Creek and lowest in Fault Creek, which was generally consistent with the benthic invertebrate results. *Bacillariophyta* was the dominant phylum observed at all sites. Bacillariophyceae and *Fragilariophyceae* were the dominate orders within the phylum, representing an average of 41% to 56% of cells across all sites. All other phyla generally represented less than 1%, expect for Cyanobacteria in Fault Creek (4.35%) and Chlorophyta at in lower Finlayson Creek (1.6%). *Didymosphenia geminate*, an invasive species, was observed at four locations throughout the Project area. In all cases *Didymosphenia* represented <1% of the total number of cells.

Chlorophyll a

Determining chlorophyll *a* concentrations provides a measure of algae biomass and therefore the primary productivity of a given location. Chlorophyll *a* studies began in 2015, and were not included in the Initial Environmental Evaluation in 1995, or required under the KZK Project's Water Licence.

Chlorophyll *a* concentrations in the KZK Project area are generally low which is an indication of low productivity systems. The highest concentrations were observed at the mouth of Geona Creek and the lowest at the confluence of Geona and Finlayson Creek.

20.2.8 Wildlife

Caribou

The Finlayson caribou herd (FCH) is considered the most valued wildlife species in the region because of their ecological, economic, and cultural importance to the Kaska, resident, and guided hunters. Approximately two thirds of the FCH use the Pelly Mountains south of the Robert Campbell Highway for calving, post-calving, and rutting and then move north to the Pelly River lowlands for over-wintering. Significant management and monitoring efforts have been directed to the FCH by the Yukon Government since the 1980s. In 1994 and 1995, Cominco conducted several detailed population studies to support the development of the Initial Environmental Evaluation (Norecol, Dames and Moore, 1996). Since 2015, BMC has conducted annual late winter, post-calving, and rutting surveys to assess the FCH herds seasonal distribution around the KZK Project area (AEG, 2018c).

From mid-September to mid-October, the FCH use habitats in and around the KZK Project footprint for rutting. From June to September, they use habitats on the north-eastern edge of the KZK Project for post-caving, and they over-winter in habitats surrounding the Finlayson airstrip and along parts of the Robert Campbell Highway. Calving likely occurs from early May to early June in the highlands east, west, and south of the KZK Project, but surveys have not been definitive because caribou hide well during this period.

Moose

Moose distribution and habitat use were assessed with aerial surveys conducted in 1995 and from 2015 through 2018. In 2015, the survey areas were expanded to include all of Game Management Subzone 10-07, based on discussions with the Yukon Government biologists. Surveys indicate that moose are well dispersed in the KZK Project area during the summer and early fall and congregate in the upper elevations of the KZK Project area post-rut. Moose were more frequently observed using the upland areas east, south, and west of the KZK Project site, with bull moose more frequently observed above treeline. Moose typically spend the early winter in the KZK Project area but have also been observed within the KZK Project area in late winter months.

Sheep

Environment Yukon has produced a map of wildlife key areas showing the known locations of Stone's sheep seasonal distribution in the general vicinity of the KZK Project. Of importance are the lambing areas approximately 13 km southeast near Money Creek, and directly south near Fire Lake. This population is located outside of the KZK Project boundaries; therefore, Stone's sheep were not formally surveyed in the 1990s or during the more recent baseline studies (2015 through 2018). However, incidental observations made during other wildlife studies have been recorded and mapped. The closest siting has been approximately 7 km southeast of the ABM deposit.

Bears

Nationally, grizzly bears are a species of Special Concern and listed in Schedule 3 of the federal *Species at Risk Act* (COSEWIC, 2012). No bear den sites were observed during the aerial surveys in 1995 and none were reported during other KZK Project related work in the area.

As part of the KZK Project's baseline studies, grizzly bear denning surveys were conducted in 2015 and 2016. One den was located during both the 2015 and 2016 denning surveys. The denning areas are approximately 4–5 km south and west of the KZK Project footprint. No dens have been identified within the immediate KZK Project footprint. In addition to denning surveys, grizzly sightings and tracks were recorded during other wildlife baseline studies and sightings were recorded by explorations. Baseline studies, incidental sighting, and denning surveys confirm that grizzly bears use habitat within the KZK Project footprint.

A grizzly bear cumulative effects assessment was conducted in 2018 to determine the availability and quality of grizzly habitat within the Project area, and to determine how the development of the KZK Project could affect these habitats (AEG, 2018d). Within the overall grizzly bear study area, there are extensive areas of high-quality habitat in which it is considered safe for bears to forage undisturbed, and to travel through without being affected by human activities.

Black bears were not surveyed in the Project area, but observations have been recorded in lower elevation sites along the Tote Road in 2015, 2016, and 2018 but not in 2017.

Collared Pika and Marmot

Collared pika (*Ochotona collaris*) and hoary marmot (*Marmota caligata*) were observed on multiple mountains surrounding the Project area. However, most observations were on mountains to the south and west of the proposed Project footprint. Pika were observed in habitats dominated by large talus rocks with crevices that provide cover. Nationally, the collared pika is listed as special concern (COSEWIC, 2011). Based on observations during baseline investigations, it is likely that collard pika and hoary marmots use high elevation habitats surrounding the KZK Project area.

Furbearers and Small Mammals

Furbearing animals known or suspected to use the Project area include grey wolf (*Canis lupus*), wolverine (*Gulo gulo*), red fox (*Vulpes vulpes*), Canada lynx (*Lynx canadensis*), coyote (*Canis latrans*), American marten (*Martes americana*), mink (*Mustela vison*), muskrat (*Ondatra zibethicus*), river otter (*Lontra canadensis*), and weasel (*Mustela nivalis*) though little is known about their abundance or distribution in the region. Incidental observations of these species have been reported, in addition to track observations made during the snow track surveys conducted in March 2016 and 2017.

There is evidence that beavers (*Castor canadensis*) have historically used Geona and Finlayson creeks. Surveys conducted in 2015 and 2016 did not find evidence of recent beaver activity. However, two beaver observations were reported at the headwaters of South Creek in 2017, and recent beaver activity has been observed on Geona Creek approximately 2.5 km upstream of the confluence with Finlayson Creek.

Bats

Currently, only three species of bats have been observed in Yukon, and only one was expected within the Local Study Area (LSA), the little brown bat (*Myotis lucifugus*). In 2015, a bat detector was installed near a subalpine wetland in the Geona Creek valley, and no bats were detected. Because the little brown bat may not use subalpine habitats, two detectors were installed at lower elevation wetlands in the boreal forest along the Tote Road in 2016, and both detectors recorded a *Myotis spp*. Based on the habitat and known sighting in the area both recording were assumed to be *M. lucifugus*.

Breeding Birds and Waterfowl

The KZK Project is located near the Tintina Trench, a known flyway for many migratory bird species. Wetlands and the adjacent riparian vegetation provide breeding and staging habitat for waterfowl, songbirds, and shorebirds. A variety of raptor species nest in the boreal forest in the valley bottoms and on cliff faces in alpine areas.

In 2015 and 2016, bird surveys were conducted in riparian, wetland, alpine, mixed subalpine, and boreal forest habitats. Forty-two species were recorded in 2015 and 61 in 2016. Five of these species are listed as "at risk" by COSEWIC (COSEWIC, 2007) including the Olive-sided flycatcher (*Contopus cooperi*), bank swallow (*Riparia riparia*), and barn swallow (*Hirundo rustica*) which are considered threatened, while the red-necked phalarope (*Phalaropus lobatus*), and rusty blackbird (*Euphagus carolinus*) are considered special concern. The most frequently observed species were white-crowned sparrow (*Zonotrichia leucophrys*), American tree sparrow (*Spizella arborea*) and Wilson's warbler (*Cardellina pusilla*).

A golden eagle (*Aquila chrysaetos*) nest was active within the LSA in 2015 and 2016, presumably by the same pair. A northern harrier (*Circus cyaneus*) nest was also located in the LSA in the headwater wetlands of Geona Creek. Other raptor species observed in the Project area include bald eagle (*Haliaeetus leucocephalus*) and gyrfalcon (*Falco rusticolus*). Ptarmigan (*Lagopus sp*.) were frequently observed in the high elevation habitat around the Project area. Spruce grouse (*Falcipennis canadensis*) were often seen along the Tote Road. Large sandhill crane (*Grus canadensis)* flocks fly over the LSA going north in April and south in August and September, but very few land in or near the KZK Project's footprint.

20.2.9 Archaeology and Heritage Resources

The KZK Project area has been the subject of heritage resources impact assessments beginning in 1995 (Rutherford, 1995a and 1995b), 1996 (Rutherford, 1996), 2015 (Mooney and Bennett, 2016), 2016 (Bennett 2016), 2017 (Bennett, 2018) and 2018 (Bennett, 2019). Four heritage sites have been identified within the Project area to date. These sites have been flagged as "No Work Zones" until the decision document has been received and will not be disturbed until the appropriate approvals are in place.

20.2.10 Vegetation and Soils

The most recent vegetation and soils baseline report for the Project was published in December 2016 (AEG, 2016b). The report combines historical information from surveys completed during the initial project assessment in the 1990s, and information collected during the re-initiation of KZK Project baseline surveys in 2015 and 2016 to support the Project Proposal, currently being reviewed by the YESAB Executive Committee.

The KZK Project area lies within the sub-alpine and alpine vegetation zones with boreal forest predominant in the lower parts of the property grading into shrub- and herb-dominated areas at higher elevation. Black spruce and sub-alpine fir are predominant within forest environments whereas tall shrub vegetation types (e.g. dwarf birch, dwarf willow birch) predominate higher up. At the highest elevations, vegetation types consist mostly of willow dwarf and alpine dwarf shrubs, in addition to herb vegetation types. Feather Moss dominates the understory in dense coniferous stands whereas sedge or sphagnum tussocks are common in wetlands and under black spruce.

Vegetation surveys have included: rare plants, invasive species; tissue sampling; stand density and volume estimates; and ground truthing to support the Terrestrial ecosystem mapping. Wetland surveys were also conducted.

No rare or endangered plants were identified during the targeted survey, and none were observed incidentally during other vegetation survey efforts.

During the surveys, invasive species were identified along the Tote Road, in the vicinity of the proposed open pit, and the BMC gatehouse (the junction with Robert Campbell Highway and the Tote Road). An Invasives Management Plan was developed following the 2015 survey and has been implemented every field season since 2016. This plan will continue to be used throughout the construction, operations and closure phases of the Project.

Soil and vegetation tissue were sampled and analyzed for elemental metal concentrations in 2015 and 2016. Five soil results exceeded CCME industrial soil guidelines at some of the sample sites for arsenic, copper, nickel, selenium, and zinc. Metal concentrations were naturally elevated in some vegetation tissue collected from a variety of plant species, which is typical in mineralized areas.

Timber volume and density estimates were made for forested polygons along the Tote Road. In general, the timber resources are of poor quality from a forestry perspective; the number of stems per hectare was very low.

Wetlands were also surveyed in 2015 and 2016. The wetlands are typically fens associated with riparian systems or bogs that occur in isolated kettle depressions or low angle slopes with near surface permafrost (AEG, 2016b).

20.3 Community Engagement

Teck (formerly Cominco) undertook an extensive consultation and engagement program, which informed Project design and helped develop the mitigation and management strategies for the KZK Project. BMC's consultation and engagement efforts commenced prior to purchase of the KZK Project, building on the strong and cooperative relationships between the Ross River Dena Council (RRDC) and Cominco/Teck. This has subsequently been maintained with First Nations Governments, stakeholders and interested parties during the preparation of the exploration permit application, and initiation of the environmental and socio-economic baseline studies.

BMC has initiated consultation and engagement with government agencies, First Nations, various stakeholder groups and interested parties to introduce the company and to engage and consult these parties regarding the proposed Project. BMC staff meet regularly with RRDC and Liard First Nation (LFN) leadership and officials, as well as regular community meetings and providing ongoing financial capacity support to enable First Nation participation in the project development, assessment and permitting. The engagement with First Nations is consistent with and builds upon the existing Socio-Economic Participation Agreement (SEPA). BMC has also produced a quarterly newsletter to keep the local communities abreast of the Projects developments as well establishing a project website to engage with a wider public audience.

BMC has developed a Consultation and Engagement Plan (CEP) that describes the path forward as the KZK Project moves through the new environmental assessment process, feasibility and permitting. The CEP sets out the tools, techniques and context for consulting with the entire suite of governments, agencies, boards, organizations and stakeholder groups with whom BMC will continue to engage to support assessment and eventual licensing and operation of the KZK Project. Techniques described in the CEP will ensure that assessment, licensing and operations, and closure of the proposed mine is underpinned by thorough, formal consultation. BMC consider the CEP to be a useful tool in guiding activities and approaches. In many cases, BMC believes it has gone beyond the basic expectations set out in that document.

BMC intend that the use of local and traditional knowledge provided by the First Nations will factor into the proposed mine's policies and design of monitoring programs and that this will allow BMC to avoid culturally and/or ecological significant and sensitive areas. It is understood that BMC has also made a commitment to provide support to First Nations for their involvement in planning and traditional use studies/oral history projects.

20.4 Mine Closure

The Reclamation and Closure Plan (RCP) (AEG, 2017) addresses the long-term physical and chemical stability of the site, including decommissioning of the mill and other facilities, reclamation of waste storage facilities and surface disturbances, and treatment of mining impacted waters. Additionally, a program for site management and monitoring will be implemented, both during implementation of the closure activities, and after decommissioning and reclamation measures are completed.

Three distinct closure phases, or "periods" have been identified for reclamation and closure at the end of the operational mining period (year 10): Active Closure Period (years 11 to 13), Transition Closure Period (years 14 to 26), and the Post-Closure Period (years 27 to 36).

The bulk of reclamation activities, such as demolition of facilities and placement of covers, will be conducted in the Active Closure Period. Covers for the Class A and B storage facilities will be monitored for effectiveness and compliance with design criteria. The Class A and Class B waste storage facilities collection ponds will be pumped to the WTP throughout the Active Closure Period and into the Transition Closure Period, until the waste rock covers have been established and met design criteria. The Class A and B waste storage facility collection ponds will then be passively released to the Constructed Wetland Treatment Systems if the water quality meets closure criteria. Then the WTP will be stood down unless needed to treat water for the remainder of the Transition Closure Period.

The Transition Closure Period will include the construction and commissioning of the Constructed Wetland Treatment Systems. Monitoring and site care and maintenance will continue through the remainder of the Transition Closure Period. In-situ treatment of water in the ABM open pit will also continue through the remainder of the Transition Closure Period.

The Post-Closure Period will commence when the ABM Lake water spills into Geona Creek. This period will include monitoring and site care and maintenance. One to two years will be required for the passive water treatment systems to achieve performance expectations with the new water contribution from the ABM Lake. At this point, the WTP and remaining infrastructure will be demobilized/decommissioned, and site water management will be passive.

The RCP also contains a cost estimate for implementation of the proposed closure measures as well as the long-term monitoring and maintenance of the site and is the basis for establishing the Reclamation and Closure Liability that will be required on the project.

The overall goal of the RCP is to ensure facilities are designed for closure conditions, to ensure physical and chemical stability when decommissioned with no active operation and minimum maintenance. This is achieved with clearly defined closure objectives and measures for each facility and component of the project as described below and summarized in Table 20-3.

Table 20-3: Reclamation and closure objectives

Area	Reclamation and closure objectives
Physical stability	All mine related structures and facilities are physically stable and performing in accordance with designs and can withstand severe climatic and seismic events.
Chemical stability	Release of contaminants from mine related waste materials occurs at rates that do not cause unacceptable exposure in the receiving environment.
Health and safety	Reclamation and closure implementation eliminates or minimizes existing hazards to the health and safety of the public, workers and area wildlife by achieving conditions similar to local area features.
Ecological conditions and sustainability	Reclamation and closure activities protect the aquatic, terrestrial and atmospheric environments from mine related degradation and restore environments that have been degraded by mine related activities. The mine site supports a self-sustaining biological community that achieves land use objectives.
Land use	Lands affected by mine related activities (including building sites, chemical and fuel storage sites, roads, sediment ponds, waste rock and tailings storage areas, open pit and underground mine areas) are restored to conditions that enable and optimize productive long term use of land by wildlife and traditional use by Kaska members (focused primarily on hunting). Conditions are typical of surrounding areas or provide for other land uses that meet community needs, interests, and expectations through discussions and involvement from Kaska representatives from RRDC and LFN.
Aesthetics	Restoration outcomes are visually acceptable.
Socio-economic expectations	Reclamation and closure implementation avoids or minimizes adverse socio-economic effects on local and Yukon communities, while maximizing socio-economic benefits and meeting community and regulatory expectations.
Long-term certainty	Minimize the need for long-term operations, maintenance and monitoring after reclamation activities are complete.
Financial considerations	Minimize outstanding liability and risks after reclamation activities are complete.

At the end of mine life, all buildings, offices etc. and associated infrastructure will be removed or demolished to return land to original wildlife land use with no active operation or maintenance. Buildings and infrastructure demolition debris will be buried in approved onsite landfill areas or transported off site for disposal as required. Remaining chemicals, reagents, and hydrocarbons will be removed from site. The mill and ore pad areas will be excavated where required to remove contamination and placed in the Class A storage facility. Excavated areas will be backfilled with material from the overburden stockpile and the mill and pad areas will then be rehabilitated. Non-essential roads will be rehabilitated. Key or essential roads will be identified to develop a weed control plan.

Water retention and sediment control structures, and appurtenances will be decommissioned or upgraded to ensure that drainage at, and adjacent to the site, is stable in the long term. Additionally, flows will be conveyed throughout the mine footprint, and off site in a controlled, stable fashion under a reasonable range of anticipated conditions by maintaining suitable gradients to permit flow and reduce infiltration and erosion.

Facilities will be designed to minimize contact of surface flow with mine influenced soil, with modifications to flow patterns at site to achieve enhanced stability or accommodate water quality objectives. Temporary (operational) structures, including stream crossings, and diversions, such as the Fault Creek diversion, will be removed and flow paths redirected to their original alignment.

Upon closure, the open pit will be allowed to flood as dewatering will cease and Fault Creek will be redirected to the open pit. To minimize contaminant loading from the pit, the closure measures will include batch treatment of the pit lake by adding lime and carbon, with a contingency of wetland treatment within Geona Creek. An engineered spillway will be constructed to control outflow from the pit lake. The safety of people and terrestrial animals in the pit area will be accomplished with catch berms around the highwall, at a setback from the crest. Slopes and benches will be stabilized with a high factor of safety, minimizing erosion and the suspension of sediments. The pit haul road will be left open for walking to allow access into the pit but blocked to vehicles.

During operations, waste rock will be stored in three storage facilities. The Class A Waste Storage Facility will contain tailings co-deposited with Strongly Potentially Acid Generating waste rock. The Class B Waste Storage facility will contain Weakly Potentially Acid Generating waste rock. The Class C Waste Storage facility will contain Potentially Acid Consuming waste rock.

The Class A Waste Storage Facility will be constructed with an overall slope of 4H:1V to ensure long-term stability. This facility will be reclaimed with a 0.5 m layer of protective material underlaying an HDPE geomembrane (Knight Piésold, 2019). Above the liner will be a 0.5 m layer of compacted low permeability natural till cover to assist in encapsulation and preventing water infiltration and ingress of oxygen. Approximately 5 m of Class C material will also be placed for frost protection as well as 0.15 m of topsoil for revegetation.

The Class B Waste Storage Facility will be constructed similarly to the Class A Waste Storage Facility. The Class B Waste Storage Facility will be reclaimed with the same cover structure as the Class A Waste Storage Facility. The Class C Waste Storage Facility will be reclaimed with a compacted with a minimum 30 cm layer of overburden. All three facilities will be revegetated to reduce erosion and to return the area to the current wildlife habitat. The quick establishment of vegetation may require a preliminary revegetation prescription for stabilization, with slower growing native community establishment to follow.

The Overburden Stockpile will provide foundation material for the Class A and Class B facilities and for construction of the Water Management and Seepage Collection Ponds over the mine life. Upon closure, all remaining overburden materials will be used for waste storage facility covers. The overburden stockpile area will be re-contoured to stable slopes and the area will be covered with a layer of topsoil and reseeded.

Monitoring of closure components will continue during the Active, Transition and Post-Closure phases until such a time that closure objectives have been met.

21 Capital and Operating Costs

21.1 Capital Costs

The capital cost estimate for the KZK Project was compiled by CSA Global with coordinated input from key contributors, expert in their respective fields. Key contributors to the cost estimate include:

- Allnorth Consultants Ltd (Allnorth): Process Plant and associated non-process infrastructure
- Knight Piésold Ltd (KP): Surface waste storage facilities and water management infrastructure
- Integrated Sustainability Consultants Inc (ISC): Water treatment plant (WTP)
- Alexco Environmental Group (AEG): Reclamation and closure
- Onsite Engineering (OSE): Tote Road upgrade
- JDS Energy & Mining Inc. (JDS): Port facilities.

A summary of the estimated capital costs for the life of the KZK Project are presented in Table 21-1. An estimated CAD$496 million will be required in pre-production capital to bring the mine into production, while an additional estimated CAD$264 million of sustaining capital will be required during operations and closure. A more detailed discussion of capital costs is included in the following section.

Table 21-1: Capital cost summary

Description	Pre-production (CAD$M)	Sustaining (CAD$M)	Total (CAD$M)
Process Plant	$197	$13	$211
Open Pit Mine Development and Infrastructure	$41	$4	$45
Underground Mine	$0	$81	$81
Non-process Infrastructure	$35	$10	$46
Waste Storage Facilities	$21	$34	$55
Water Storage and Management Facilities	$17	$11	$28
Water Treatment Plant	$22	$3	$25
Port Facilities	$15	$0	$15
Other Infrastructure	$6	$5	$12
Subtotal Direct Costs	**$355**	**$162**	**$516**
Closure	$0	$102	$102
Total Direct Costs	**$355**	**$264**	**$618**
Owners Costs	$16	Included	$16
Indirect Costs	$78	Included	$78
Contingency	$47	Included	$47
TOTAL CAPITAL COST (CAD$M)	**$496**	**$264**	**$760**
TOTAL CAPITAL COST (US$M)	**$381**	**$206**	**$587**

Notes:

- *Values presented are rounded to the nearest CAD$ 1 million. Totals may not sum precisely.*
- *Currency exchange rate varies over the life of mine as detailed in Table 19-3 and averages US$0.77 per CAD$1.00 during Pre-production and US$0.78 per CAD$1.00 during Operations (or the Sustaining Capital period).*
- *Costs are adjusted for asset leasing.*
- *100% equity financing is assumed.*
- *Excludes project finance interest, offtake agreements, reclamation bonding, and other financing arrangements and costs, working capital, exchange rate fluctuations, all licence fees and allowances for special incentives (schedule/safety or others).*
- *Mining of open pit waste is considered to be an operating cost, except for pre-production mining activity.*

21.1.1 Basis of Estimate

The capital cost estimate is considered accurate to within the normal limits expected for a FS as defined in the CIM Definition Standards for Mineral Resources and Mineral Reserves. The costs are considered current as of Q4 2018 and are presented in real terms; hence, escalation of costs has not been applied after this date.

All costs are provided in Canadian currency unless designated otherwise. Exchange rates used to convert vendor pricing to Canadian currency for pre-production capital costs are detailed in Table 21-2. Sustaining capital costs are all quoted in Canadian currency.

Table 21-2: Exchange rates applied in pre-production capital cost estimate

Currency	Exchange rate (per CAD$)
US Dollar	$0.77
Euro	$0.66
Australian Dollar	$1.007

A Work Breakdown Structure (WBS) was established for the initial capital cost estimate. Costs have been classified into the various WBS areas to ensure that the full cost of developing the KZK Project has been captured. The capital cost estimate includes contributions from several parties, who are specialists in their field, and was compiled by CSA Global. Responsibilities for preparation of each component of the WBS are detailed in Table 21-3.

Table 21-3: WBS summary

WBS	Description	Responsibility	Scope
2000	Mining	CSA Global	Open pit mine development and production Underground mine development and production Mine surface infrastructure
		KP	Waste storage facilities
3000	Processing	Allnorth	Process plant facility Civils Crushing and reclaim Grinding and classification Flotation and regrind Concentrate handling and storage Tailings management Reagents Process plant utilities Building
4000	Transport and Logistics	Allnorth	Concentrate Transport
		JDS	Port concentrate storage facility
5000	Infrastructure	Allnorth	Site administration facilities Accommodation facilities Power generation and distribution Bulk fuel storage Communications Site Services
		KP	Surface water management facilities Site roads
		ISC	Water treatment plant
		OSE	Tote road upgrade

WBS	Description	Responsibility	Scope
6000	Closure	AEG	Reclamation and closure plan
7000	Owners Costs	Allnorth	Owners management team Spares and inventory Mobile equipment
		CSA Global	Environmental
8000	Indirect Costs	Allnorth	EPCM Temporary facilities Temporary services and support Construction catering and accommodation Commissioning
9000	Contingency)	CSA Global	

21.1.2 Process Plant

A description of the Process Plant is provided in Section 17.

Allnorth prepared a bottom-up cost estimate for the facility. Quantities were estimated from material takeoff of 3D design models (e.g. Navisworks) and 2D design drawings. Project-specific vendor quotes were sourced for all major materials and equipment. Minor equipment costs were taken from the Allnorth database of costs. Labour requirements were estimated using typical industry norms developed by Allnorth for similar style projects. Labour costs are discussed below in more detail. Freight cost were estimated based on factored mark-up of the direct costs.

Cost for labour, plant, materials and freight were estimated for the following categories for each of the WBS areas:

- Civil
- Structural
- Platework
- Mechanical
- Piping
- Electrical
- Controls and instrumentation
- Building and architectural.

Table 21-4 summarizes the direct capital costs to construct the Process Plant facilities and is based on the purchase and installation of new equipment.

Table 21-4: Summary of Process Plant capital costs

Description	Pre-production (CAD$M)
Earthworks and Drainage	$12.0
Crushing and Reclaim	$14.7
Grinding and Classification	$21.4
Flotation and Regrind	$39.2
Concentrate Handling and Storage	$21.6
Tailings Management	$19.0
Reagents	$8.8
Process Plant Utilities	$19.4
Plant Building	$41.3
Direct Costs	**$197.4**

Construction labour costs were based on the latest May 2018 BC/Yukon open shop labour rates, with construction work carried out on the basis of a 21 days rostered on, seven days rostered off work cycle, working 12 hours per day, with overtime premium included in the labour rates. The all-found construction labour rates are detailed in Table 21-5 and include legislated employer costs, minor equipment and small tools allowance, safety equipment allowance, construction consumables, construction equipment rental, contractor's head office overheads and contractor's field supervision.

Table 21-5: Construction labour all-found hourly costs

Description	Labour cost per hour worked (CAD$ per manhour)
Site Work	$95
Concrete	$97
Steelwork	$100
Mechanical	$99
Piping	$98
Electrical	$103
Instrumentation	$105
Architectural	$99

Supplied and installed concrete costs, inclusive of reinforcement bar and cast-ins with aggregate supply from Watson Lake Concrete costs averaged CAD$1,537/$m^3$ across the different concrete applications (foundations, footings and pedestals, ring wall, slabs on grade, curbs, elevated slab on deck, and sumps). Separate provisions were made for heating and hoarding.

Structural steel costs averaged CAD$5,973/t supplied and installed, over all grades of steelwork (light, medium and heavy). Separate provisions were made for freight costs.

Contingency was estimated by Allnorth for each specific line item in the cost estimate. Where there was considered to be higher levels of confidence in the estimate, lower contingency rates were applied. The overall contingency for the Process Plant averaged 8.3%, as summarized in Section 21.1.12.

21.1.3 Open Pit Mine Development and Infrastructure

Open pit mine development and infrastructure costs were prepared by CSA Global and KP. These costs include pre-production mining costs incurred for establishment of open pit mining operations, production of waste rock for construction of site infrastructure and production of an initial ore stockpile and are included in Table 21-6.

Table 21-6: Summary of open pit mine development and infrastructure costs

Description	Pre-production (CAD$M)	Responsibility
Open Pit Dewatering	$4.4	KP/CSA Global
Mine Infrastructure	$7.5	CSA Global
Open Pit Mine Development	$29.1	CSA Global
Total Mining Capital Costs	**$41.0**	

Open pit dewatering costs include the excavation of trenches in the overburden, drilling and installation of pit perimeter dewatering bores and pumping of water. Mine infrastructure costs include costs for construction of surface explosives facilities, mine workshop and mining contractor offices and mine fuel facilities. Open pit mine development costs allow for drill, blast, load and haul of waste material either to either surface infrastructure sites or designated waste storage facilities and stockpiles as well as production of an initial ore stockpile to commence production and provision for overheads.

Contingency was estimated by KP and CSA Global for each cost area and averaged 6.9%, as summarized in Section 21.1.12.

21.1.4 Non-Process Infrastructure

Costs for non-process infrastructure were largely prepared by Allnorth, with certain minor costs included by CSA Global as indicated in Table 21-7.

Table 21-7: Summary of non-process infrastructure capital costs

Description	Pre-production (CAD$M)	Responsibility
Site Administration	$7.9	Allnorth/CSA Global
Camp	$10.4	Allnorth
Process Plant Roads	$1.9	Allnorth
Power Generation and Transmission	$8.5	Allnorth
Fuel Storage and Distribution	$1.6	Allnorth
Communications	$0.9	Allnorth
Services and Utilities	$3.0	Allnorth
Miscellaneous	$1.0	CSA Global
Total	**$35.0**	

Site administration includes construction and fitout of site office complexes, the laboratory building and equipping the Process Plant workshop, warehouse and first aid facilities. Computers, software, site telephone system and survey equipment have also been included within these costs.

Camp facilities are described in Section 18.11. The cost includes all costs associated with the staged development of the camp facilities to a 348-person camp. It includes siteworks, dormitories, dry mess, recreation facilities and supporting infrastructure.

As noted in Section 21.2.10, major capital equipment for power generation and the LNG fuel facility will be implemented on an operating lease. The components of these facilities that will not be covered by the leasing arrangements are provided for in the power generation and transmission and fuel storage and distribution costs detailed in Table 21-7. These costs include electrical distribution and motor control centres, emergency diesel generators and piping of diesel and LNG fuel to the power station.

Services and Utilities captures costs for buried services required throughout the site, site lighting and the site waste management facility.

Costs for fencing of surface water ponds and dams, generators for ponds that are not connected to the site power grid and ROM pad stabilization are included in the Miscellaneous cost category.

Contingency was estimated by Allnorth and CSA Global for each cost area and averaged 10.3%, as summarized in Section 21.1.12.

21.1.5 Waste Storage Facilities

Costs to construct the waste storage facilities were prepared by KP and cover the initial development of all surface waste storage facilities described in Section 18.1 and summarized in Table 21-8. This includes construction for the first stage of the facilities and involves preparation of the basins, including lining and drainage where applicable, construction of the Class A buttress and stockpiling of associated topsoil and other spoil. Costs for the installation of the closure covers are included in Section 21.1.15. The capital cost also includes the engineering work required to prepare detailed designs for construction.

Table 21-8: Summary of waste storage facilities capital costs

Description	Pre-production (CAD$M)
Class A Waste Storage Facilities	$10.0
Class B Waste Storage Facilities	$9.1
Class C Waste Storage Facilities	$0.0
Overburden Stockpile	$0.2
Topsoil Stockpiles	$0.2
Miscellaneous	$0.2
Engineering	$1.5
Total	**$21.2**

Construction of the Class C Waste Storage Facilities was deferred until the commencement of operations, as prior to this all Class C rock generated by pre-production mining activities will be used for construction of the various surface infrastructure components of the mine.

Contingency was estimated by KP on a line-by-line basis. Higher contingencies were allowed for earthworks. The contingency averaged 17.5% for the pre-production phase. Pre-production contingency costs are summarized in Section 21.1.12.

21.1.6 Water Storage and Management Facilities

Costs to construct the water storage and surface water management facilities were prepared by KP and cover the development of all surface water storage facilities as summarized in Table 21-9. The costs primarily allow for earthworks and associated lining of the facilities, as well as pumps and pipework required to integrate them. The capital cost also includes the engineering work required to prepare detailed designs for construction.

Table 21-9: Summary of water storage and management facilities capital costs

Description	Pre-production (CAD$M)
Lower Water Management Pond	$4.9
Upper Water Management Pond[1]	$0.0
Class A Storage Facility Collection Pond	$1.9
Class B Storage Facility Collection Pond	$0.8
Class C Storage Facility Collection Pond	$1.1
Overburden Stockpile Collection Pond	$0.5
Pit Rim Pond	$1.1
Collection and Diversion Ditches	$2.2
Mechanical Systems and Pipelines	$1.6
Finlayson Creek Discharge Pipeline[1]	$0.0
Miscellaneous	$1.5
Engineering	$1.5
Total	**$17.2**

Note: 1. Cost included as sustaining capital; item shown for completeness.

Construction of the Upper Water Management Pond and the discharge pipeline from the Lower Water Management Pond to Finlayson Creek were deferred until the first year of operations, as determined by the water balance, and are included in Sustaining Capital.

Contingency was estimated by KP on a line-by-line basis. Higher contingencies were allowed for earthworks. The contingency averaged 18.6% for the pre-production phase. Pre-production contingency costs are summarized in Section 21.1.12.

21.1.7 Water Treatment Plant

The WTP is described in Section 18.3. Costs to construct the WTP were estimated by ISC and are summarized in Table 21-10. Where applicable, construction materials and labour were aligned with the construction of the Process Plant as the two will be constructed concurrently. Infrastructure and site development allow for the concrete civil works, the WTP building and non-modular services infrastructure. The expansion of the WTP to include lime treatment for Class A Waste Storage Facilities water once it turns acidic is included in the sustaining capital costs summarized in Section 21.1.14.

Table 21-10: Summary of WTP capital costs

	Pre-production (CAD$M)
Infrastructure and Site Development	$9.9
Modules	$2.3
Metals Removal Circuit	$1.8
Selenium Removal Circuit	$1.9
Eluate Treatment Circuit	$2.5
Reagent and Utilities	$1.0
Freight	$0.8
Direct Costs	**$20.1**
EPCM	$1.4
Fabrication and Implementation	$0.1
Commissioning	$0.3
TOTAL	**$21.9**

A contingency of 25% has been allowed for the pre-production construction work and is summarized in Section 21.1.12.

21.1.8 Port Facilities

Costs to construction the concentrate storage facilities at Stewart World Port were estimated by JDS and are summarized in Table 21-11. In addition to the costs estimated by JDS, additional provisions were made for supply of structural backfill for construction and the rotating container handler for lead concentrate containers, as noted in Table 21-11. All other capital infrastructure at the port for the storage, handling and loading of concentrate onto ships will be provided by Stewart World Port and is included in the operating costs detailed in Section 21.2.7.

Table 21-11: Summary of port facility capital costs

	Pre-production (CAD$M)
Concrete	$5.7
Architectural (includes building)	$4.5
Mechanical	$1.4
Civil	$0.3
Piping	$0.0
Electrical and Instrumentation	$0.3
Direct Costs	**$12.2**
EPCM	$0.8
Indirects	$0.9
Subtotal Concentrate Storage Facilities	**$13.9**
Supply of Structure Backfill	$0.3
Rotating Container Handler	$0.5
TOTAL PORT FACILITIES COSTS	**$14.7**

Concentrate road transport will be outsourced and all capital costs associated with establishing the transport operation will be amortized over the contract.

JDS estimated the contingency for the port facilities of 10% and is included in the summary of contingency costs in Section 21.1.12.

21.1.9 Other Infrastructure

Four cost areas have been consolidated into the Other Infrastructure category in Table 21-1 in Section 21.1. These are summarized in Table 21-12.

Costs to upgrade the existing Tote Road to an Access Road suitable for project development were prepared by OSE. The total cost of implementing the Access Road upgrade has been estimated to be CAD$2.8 million.

Construction work will be spread over two seasons, with limited initial works undertaken immediately to address the most critical sections of the road to facilitate movement of heavy equipment and materials for construction. The remainder of the work will be completed prior to commencement of operations.

Costs to construct roads to the Class A, B and C waste storage facilities and the overburden stockpile are provided under Mine Roads, while all other site roads are covered under Site Roads.

Table 21-12: Summary of other infrastructure capital costs

	Pre-production (CAD$M)	Responsibility
Access Road	$2.8	OSE
Mine Roads	$0.5	KP
Site Roads	$1.7	KP
Fish Offset Measures	$1.5	CSA Global
Total	**$6.5**	

Contingency was estimated by the responsible parties for each cost area and averaged 13.3%, as summarized in Section 21.1.12.

21.1.10 Owners Costs

Owners costs have been prepared by Allnorth and CSA Global. These are summarized in Table 21-13 below, together with the party responsible for each area.

Environmental management and monitoring costs during the pre-production period were prepared by BMC and reviewed by CSA Global.

Table 21-13: Summary of owner's costs

	Pre-production (CAD$M)	Responsibility
Owner's Team	$4.5	Allnorth
Training of Process Operators	$0.7	Allnorth
Environmental Monitoring	$1.9	CSA Global
Spares and Inventory	$9.1	Allnorth/CSA Global
Total	**$16.3**	

Spares and inventory include provisions of CAD$4.7 million for major equipment capitalized parts, CAD$3.1 million for minor equipment capitalized parts, CAD$1.2 million for first fills and CAD$0.2 million for mine rescue equipment.

Owner's team staffing is summarized in Table 21-14. Personnel will be progressively mobilized to site as site activities increase. The Mining Manager, Chief Mining Engineer and Chief Geologist will be mobilized to site the month preceding commencement of mining activity. The Processing Manager will be mobilized to site six months prior to commencement of processing for familiarization of the Process Plant, planning for commissioning and training of Process Plant operators. Process Plant operators will be employed two months prior to commissioning for training and plant familiarization.

Owner's team labour costs have been benchmarked against data available for comparable northern mining operations and project. Salary loading for owner's team labour includes:

- Four weeks annual leave per year;
- Statutory payments for Canadian Pension Plan, Employment Insurance and Yukon Workers Compensation Board;
- Statutory holiday allowance of 10 days per year; and
- Flexible benefits package of $5,000 per year for the employee to use for health insurance and other medical benefits.

Table 21-14: Owner's team staffing

Position	No. of personnel
Project Director	1
Site General Manager	1
Construction Manager	1
Surveyor	2
OHS & ES Officer	1
Site Nurse	1
Environmental Officer	2
Site Clerk	2
Gatehouse Security	2
Mining Manager	1
Chief Mining Engineer	1
Chief Geologist	1
Processing Manager	1
Total	**17**

Contingency was estimated for each cost area and averaged 10.0%, as summarized in Section 21.1.12.

21.1.11 Indirect Costs

Indirect costs have been prepared by Allnorth, KP and CSA Global. These are summarized in Table 21-15.

Table 21-15: Summary of indirect costs

	Pre-production (CAD$M)	Responsibility
EPCM	$37.2	Allnorth
Temporary Facilities/Site Establishment	$26.3	Allnorth/KP/CSA Global
Temporary Services and Support	$1.6	Allnorth
Construction Camp and Catering Services	$8.3	Allnorth
Winter Construction Costs	$3.6	Allnorth
Commissioning	$1.1	Allnorth
Total	**$78.0**	

EPCM costs estimated by Allnorth are based on an assessment of the manhours required for each discipline. The estimate makes provision for the EPCM contractors services such as design, drawings, specifications, work scopes, procurement, expediting, inspection, site supervision, management, scheduling, cost control, accounting, monitoring, reporting, commissioning and associated expenses.

Temporary facilities and site establishment costs include provisions for temporary services and facilities to establish and support the construction work being carried out. These include:

- Contractor field assessment, mobilization, and demobilization
- Material receiving, storage, inspection, and protection
- IT and communication services
- Engineering and management field services such as offices, lighting, sanitary facilities, safety equipment, communications equipment, cleaning and maintenance
- Contractor field services and trailers
- Contractor indirect office staffs

- Contractor services and costs such as fuel, electrical power, water supply, sanitary facilities
- Site services such as temporary roads, parking, laydown areas, fencing, yard lighting
- Daily transportation between camp and site
- Air transportation of construction crew and staff between Whitehorse Airport and Finlayson Airstrip plus land transportation to camp
- Site ambulance services
- Construction equipment such as site FELs, grader, gravel truck, backhoes (included in composite Labour rate)
- Dewatering pumps and hoses
- Scaffolding
- Large construction cranes
- BMC's mobile equipment to commence operations.

Allnorth estimated the capital costs required to purchase the mobile equipment that BMC will require to operate the mine. As noted in Section 21.2.10, majority of this equipment is planned to be leased and is not considered a pre-production capital cost. The items of equipment that are not leased were included in Indirect Costs. The total cost of this equipment is $0.2 million and includes lighting towers, portable heaters, compressors and portable toilets.

Temporary services and support allow for the provision of outside consultants during construction works for the processing plant, including survey, geotechnical, quality assurance, testing for civil, concrete, piping and electrical and instrumentation.

Construction camp and catering services include the costs for accommodation and messing facilities for all personnel on sit during the construction works.

Winter construction costs have been identified separately should the plant be constructed during a different season and these costs may not be incurred. Winter construction costs have been estimated to be CAD$1.5 million for concrete heating and hoarding and CAD$2.1 million for impacts on labour productivity.

Commissioning costs include provisions for vendor assistance and standby of construction crews during commissioning.

Contingency was estimated for each cost area and averaged 10.0%, as summarized in Section 21.1.12.

21.1.12 Contingency

Contingency provisions for each area of the pre-production capital cost estimate have been noted in the preceding sections. These costs are summarized in Table 21-16 for convenience. The overall contingency provision for the pre-production capital is 10.5%.

The underground mine comprises part of the sustaining capital cost. As noted in Section 21.1.14, contingency is inherent in the sustaining capital costs and is not reported separately.

Table 21-16: Pre-production contingency cost summary

	Pre-production (CAD$M)	Pre-production (%)
Open Pit Mine	$2.8	6.9%
Process Plant	$16.4	8.3%
Non-process Infrastructure	$3.6	10.3%
Waste Storage Facilities	$3.7	17.5%
Water Management Facilities	$3.2	18.6%
Water Treatment Plant	$5.5	25.0%
Port Facilities	$1.5	10.0%
Other Infrastructure	$0.9	13.3%
Owners Costs	$1.6	10.0%
Indirect Costs	$7.8	10.0%
Total Contingency (CAD$M)	**$47.0**	**10.5%**

21.1.13 Working Capital

Working capital is based on 60 days Accounts Receivable plus Inventory minus 30 days Accounts Payable. Accounts Receivable is based on 10% of two MAMA (month after month of arrival).

Working capital requirements commence in September 2019. During steady state production, the working capital requirements typically range between CAD$13 million and CAD$19 million.

The feasibility cash flow model does not include project financing facilities (including working capital facilities) and their impacts.

21.1.14 Sustaining Capital

Sustaining capital costs have been prepared by the appropriate report authors and are summarized in Table 21-17 by primary operating cost centres. Contingency has been included in sustaining capital cost estimates and is not reported separately. The largest sustaining capital costs are the development of the underground mine, construction of the paste backfill plant, construction of water management infrastructure and expansion of the various waste storage facilities.

Replacement of mobile equipment at the end of serviceable life is included in sustaining capital cost estimates.

Table 21-17: Summary of sustaining capital costs

	Cost (CAD$M)									
	Total	**2021**	**2022**	**2023**	**2024**	**2025**	**2026**	**2027**	**2028**	**2029**
Waste Storage Facilities	$34.1	$2.1	$12.1	$0.0	$11.0	$0.0	$4.8	$0.0	$4.1	$0.0
Surface Water Management	$11.3	$1.5	$7.5	$0.1	$1.0	$0.1	$0.5	$0.1	$0.5	$0.0
Pit Dewatering Infrastructure	$3.6	$0.3	$1.8	$1.4	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0
Site Roads	$3.3	$0.0	$0.1	$0.0	$3.2	$0.0	$0.0	$0.0	$0.0	$0.0
ROM Surface Preparation	$1.9	$0.0	$1.0	$0.9	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0
Vehicle Replacement	$0.3	$0.0	$0.0	$0.0	$0.0	$0.3	$0.0	$0.0	$0.0	$0.0
Subtotal – Open Pit Mining	**$54.6**	**$3.9**	**$22.5**	**$2.4**	**$15.3**	**$0.4**	**$5.3**	**$0.1**	**$4.7**	**$0.0**
Mining Capital Development	$77.4	$0.0	$0.0	$0.0	$16.6	$23.8	$28.4	$8.6	$0.0	$0.0
Paste Fill Plant	$9.9	$0.0	$0.0	$0.0	$5.4	$4.5	$0.0	$0.0	$0.0	$0.0
Mine Infrastructure	$3.4	$0.0	$0.0	$0.0	$0.8	$1.5	$0.8	$0.1	$0.2	$0.0
Power Generation and Reticulation	$8.9	$0.0	$0.3	$8.6	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0
Subtotal – Underground Mining	**$99.6**	**$0.0**	**$0.3**	**$8.6**	**$22.8**	**$29.8**	**$29.2**	**$8.7**	**$0.2**	**$0.0**
Mobile Equipment Replacement	$3.3	$0.0	$0.0	$0.0	$0.0	$0.4	$2.9	$0.1	$0.0	$0.0
Water Treatment Plant HDS Circuit	$2.8	$0.0	$0.0	$0.0	$0.0	$0.2	$2.7	$0.0	$0.0	$0.0
Subtotal – Processing	**$6.2**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.5**	**$5.5**	**$0.1**	**$0.0**	**$0.0**
Mobile Equipment Replacement	$1.2	$0.0	$0.0	$0.0	$0.0	$0.4	$0.6	$0.2	$0.0	$0.0
Miscellaneous	$0.4	$0.0	$0.4	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0
Subtotal – Administration	**$1.6**	**$0.0**	**$0.4**	**$0.0**	**$0.0**	**$0.4**	**$0.6**	**$0.2**	**$0.0**	**$0.0**
TOTAL	**$161.9**	**$3.9**	**$23.1**	**$11.0**	**$38.1**	**$31.2**	**$40.6**	**$9.1**	**$4.8**	**$0.0**

21.1.15 Closure Costs

Mine closure is discussed in Section 20.4. Closure costs for the program of works have been summarized in Table 21-18. Given the duration of the passive closure phase, costs incurred beyond the last period of the financial model (some 10 years) were discounted at a rate of 7% per annum to a Net Present Cost at the end of the model.

The closure cost estimate has been prepared by AEG (Alexco Environmental Group, 2019) using third party rates (YG, 2017). Indirect costs have been estimated to be 20% of the direct costs. A contingency allowance of 20% was added to the total cost of closure implementation and post-closure costs in consideration of the level of design to date and the time to closure.

For the purposes of assessing financial provisions for bonding based on Yukon Government guidelines (YG, 2013), Interim Care, Maintenance and Monitoring costs will be included to consider the scenario of temporary mine closure. The cost of this provision has been estimated to be CAD$10.4 million, inclusive of 20%

contingency and has not been included in the DFS cost estimate as it is to consider a hypothetical temporary closure scenario.

Table 21-18: Summary of closure costs

	Closure cost (CAD$M)
Closure Implementation	
G&A, Closure Planning	$6.5
Open Pit	$1.6
Waste Rock and Tailings Storage Facilities	$45.2
Surface Facilities	$4.3
Water and Solutions Management	$5.3
Other	$1.1
Indirects	$12.8
Subtotal Closure Implementation	**$76.8**
Post-Closure Costs (undiscounted)	$13.6
Contingency	$17.0
TOTAL CLOSURE COSTS (undiscounted)	**$107.4**
TOTAL CLOSURE COSTS (final years discounted)	**$101.6**

It has been assumed that the processing plant and other non-process facility infrastructure will be salvaged and sold into the second-hand market where appropriate, with a salvage value of US$25 million being realized during the first year of closure.

21.2 Operating Cost

The operating cost estimate for the KZK Project was compiled by CSA Global with coordinated input from key contributors, experts in their respective fields. Key contributors to the cost estimate include:

- CSA Global (CSA): Mining, site administration and royalties;
- Allnorth Consultants Ltd (Allnorth): Process Plant and associated non-process infrastructure and road transportation of concentrates
- Knight Piésold Ltd (KP): Surface waste storage facilities and water management infrastructure
- Integrated Sustainability Consultants Inc (ISC): Water treatment plant (WTP)
- Stewart World Port (SWP): Port facilities
- StoneHouse Consulting Inc. (StoneHouse): Concentrate marketing
- Braemar Technical Services LLC (Braemar): Ocean freight.

The operating costs over the life of the KZK Project are detailed in Table 21-19. Over the life of the KZK Project, total operating costs are expected to be in the order of CAD$2,886 million.

Table 21-19: KZK Project – annual operating cost summary

	Cost (CAD$M)											
	Total	**2019**	**2020**	**2021**	**2022**	**2023**	**2024**	**2025**	**2026**	**2027**	**2028**	**2029**
Open Pit Mining	$620	$0	$0	$4	$64	$123	$128	$112	$60	$59	$45	$24
Underground Mining	$159	$0	$0	$0	$0	$0	$0	$0	$46	$54	$36	$22
Processing	$330	$0	$0	$1	$38	$41	$41	$45	$43	$44	$45	$31
Water Treatment	$16	$0	$0	$0	$1	$2	$2	$2	$2	$3	$3	$2
Administration	$167	$0	$0	$2	$20	$22	$22	$22	$21	$21	$20	$17
Road Transport	$354	$0	$0	$0	$39	$48	$51	$48	$49	$47	$45	$27
Sea Transport and Port Operations	$212	$0	$0	$0	$22	$28	$31	$28	$30	$27	$28	$17
Equipment Leases	$78	$0	$0	$2	$10	$10	$10	$14	$14	$11	$5	$3
First Nations	$50	$1	$1	$4	$6	$6	$6	$6	$6	$6	$6	$3
Royalties	$221	$0	$0	$0	$22	$27	$33	$25	$31	$22	$36	$27
TC/RC and Penalties	$679	$0	$0	$0	$79	$96	$99	$86	$94	$84	$88	$53
Total	**$2,886**	**$1**	**$1**	**$13**	**$302**	**$402**	**$423**	**$388**	**$396**	**$377**	**$356**	**$226**

Table 21-20 shows operating costs over the life of the KZK Project on a unit cost of ore processed basis. Over the LOM, unit operating costs are expected to be CAD$184 per tonne of ore processed, with operating costs ranging between approximately CAD$160 and CAD$215 per tonne of ore processed.

Table 21-20: KZK Project – annual operating unit cost summary

	Unit cost (CAD$/tonne ore processed)								
	LOM average	**2019-2021**	**2022**	**2023**	**2024**	**2025**	**2026**	**2027**	**2028**	**2029**
Ore processed (Mt)		0.0	1.8	2.0	2.0	2.2	2.1	2.1	2.2	1.4
Open Pit Mining	$39.42	$0.00	$34.86	$61.93	$65.50	$51.08	$28.67	$28.41	$20.24	$17.77
Underground Mining	$10.10	$0.00	$0.00	$0.00	$0.00	$0.00	$22.08	$25.97	$16.53	$16.06
Processing	$20.96	$0.00	$20.83	$20.73	$20.78	$20.61	$20.63	$20.91	$20.30	$23.07
Water Treatment	$1.04	$0.00	$0.81	$0.90	$0.92	$0.80	$0.87	$1.22	$1.17	$1.78
Administration	$10.60	$0.00	$11.04	$11.10	$11.22	$9.93	$10.04	$9.92	$9.21	$12.46
Road Transport	$22.50	$0.00	$21.22	$24.20	$25.76	$21.87	$23.59	$22.32	$20.53	$19.85
Sea Transport and Port Operations	$13.48	$0.00	$12.16	$14.19	$16.00	$12.81	$14.35	$13.05	$12.59	$12.42
Equipment Leases	$4.96	$0.00	$5.26	$4.84	$5.29	$6.36	$6.51	$5.06	$2.08	$2.58
First Nations	$3.18	$0.00	$3.20	$2.96	$2.99	$2.68	$2.80	$2.81	$2.66	$2.17
Royalties	$14.09	$0.00	$11.96	$13.35	$16.70	$11.61	$14.63	$10.37	$16.32	$19.55
TC/RC and Penalties	$43.21	$0.00	$43.00	$48.17	$50.52	$39.33	$44.99	$40.30	$39.76	$39.28
Total	**$183.53**	**$0.00**	**$164.34**	**$202.37**	**$215.69**	**$177.06**	**$189.15**	**$180.34**	**$161.40**	**$166.99**

Information on each operating cost centre follows.

21.2.1 Basis of Estimate

The operating cost estimate is considered accurate to within the normal limits expected for a FS as defined in the CIM Definition Standards for Mineral Resources and Mineral Reserves. The costs are considered current as of Q4 2018 and are presented in real terms; hence, escalation of costs has not been applied after this date.

All costs are provided in Canadian currency unless otherwise noted.

21.2.2 Open Pit Mining

The operating mining cost estimates for the open pit mining are based on submissions from mining contractors to a request for quotation (RFQ) from BMC. Contractor costs were exclusive of diesel fuel, accommodation and roster flights. The most appropriate submission was selected to form the basis of the estimate.

The explosive costs are based on supply under contract by an explosives company. The contract includes the establishment of fixed infrastructure for storage of ammonium nitrate and emulsion explosives and the provision of magazines for packaged explosives and detonators.

BMC will incur costs to operate the open pit that are not attributable to contractors. These are accounted for as BMC owner's costs. The owner's costs account for:

- BMC labour costs
- Owner's Plant and equipment:
- Material and services (including diesel supply to all activities and dewatering).

Diesel consumption was estimated by factoring material movement and explosive consumption, and including nominal provisions for owner's equipment and other miscellaneous requirements. Fuel prices were varied based on forecast and are presented in Table 21-42.

Costs for accommodation and roster flights are included in Section 21.2.9.

The open pit operating costs are summarized in Table 21-21.

Table 21-21: Open pit mining costs

Category	LOM cost (CAD$M)	LOM unit cost (CAD$/t)
OPEN PIT MINING CONTRACTOR COSTS		
Mobilization costs	$8.1	$0.05/total t mined
Demobilization costs	$7.9	$0.05/total t mined
Site preparation costs	$2.3	$0.02/total t mined
Drilling costs	$36.2	$0.24/total t mined
ABM pit waste load and haul costs	$213.5	$1.75/waste t mined
ABM pit ore load and haul costs	$22.4	$1.67/ore t mined
Krakatoa pit waste load and haul costs	$33.2	$2.01/waste t mined
Krakatoa pit ore load and haul costs	$1.2	$2.01/ore t mined
Ex-pit haulage costs	$60.9	$0.44/waste t mined
Re-handle underground material and transport costs	$6.0	$0.04/total t mined
Tailings re-handle costs	$38.9	$0.26/total t mined
Crushed material costs	$0.7	$0.005/total t mined
Daywork	$8.3	$0.05/total t mined
TOTAL OPEN PIT MINING CONTRACTOR COSTS	**$439.8**	**$2.89/total t mined**
TOTAL EXPLOSIVE CONTRACTOR OPEN PIT COSTS	**$60.7**	**$0.40/total t mined**
BMC OWNER COSTS		
BMC labour costs	$16.2	$0.11/total t mined
Plant and equipment costs	$2.9	$0.02/total t mined
Materials and services costs (including diesel)	$124.7	$0.82/total t mined
Contractors and consultants costs	$1.8	$0.01/total t mined
TOTAL BMC OWNER COSTS	**$145.5**	**$0.95/total t mined**
TOTAL OPEN PIT MINING COSTS	**$645.9**	**$4.24/total t mined**
CAPITALIZED PRE-PRODUCTION MINING COST ADJUSTMENT	($26.1)	$0.17/total t mined
OPEN PIT MINING OPERATING COSTS	**$619.8**	**$4.07/total t mined**

BMC staff for management and supervision of the open pit mining operation are summarized in Table 21-22. Labour costs have been benchmarked against data available for comparable northern mining operations and projects. The basis for salary loading is described in Section 21.2.9.

Table 21-22: BMC open pit mining personnel

Position	Roster type	Personnel	Loaded cost (CAD$/year)
Mining Manager	9/5	1	$223,000
Open Pit Foreman	9/5	1	$159,000
Chief Mining Engineer	2/1	1	$165,000
Mine Engineer	2/1	1	$119,000
Senior Geotechnical Engineer	2/1	1	$148,000
Geotechnical Engineer	2/1	1	$119,000
Chief Geologist	2/1	1	$148,000
Geologist	2/1	2	$107,000
Senior Surveyor	2/1	1	$101,000
Surveyor	2/1	1	$93,000
Technician – Geology	2/1	2	$90,000
Technician – Mining/Survey	2/1	1	$90,000
Clerk	9/5	1	$84,000
Safety and Training	9/5	1	$107,000
Total			**$1,951,000**

21.2.3 Underground Mining

The cost base for the Krakatoa underground project is based on submissions from mining contractors to an RFQ from BMC, where four companies submitted pricing for the construction of the underground mine. The RFQ was based on an earlier mine design than the final design used in the DFS. For completeness, the final DFS design was requoted with a Tier 1 contractor (Table 21-23).

Table 21-23: Underground contractor mining costs

Underground Mining Contractor Costs	LOM cost (CAD$M)	LOM unit cost (CAD$/t underground ore)
Mobilization	$5.3	$3.06
Demobilization	$1.2	$0.72
Development cost to drill, blast and excavate	$72.6	$41.97
Ground support	$32.2	$18.64
Production drilling and blasting	$17.8	$10.29
Production mucking	$18.3	$10.56
Paste fill activities	$3.0	$1.76
Other construction activities	$6.8	$3.92
Load and haul	$18.8	$10.88
Daywork	$3.5	$2.04
Total	**$179.5**	**$103.83**

Note: Costs include capital development.

BMC owner costs are presented in Table 21-24 and account for the costs directly under BMC management's responsibility. Plant and equipment costs include owner's mobile equipment maintenance, paste fill plant maintenance and technical equipment and software. Material and services costs include diesel and electrical power, cemented paste fill binder and miscellaneous consumables. Contractors and consultants costs include stope definition exploration drilling and provisions for use of consultants throughout the life of the underground mine.

Costs for accommodation and roster flights are included in Section 21.2.9

Table 21-24: BMC owner costs

Cost	LOM cost (CAD$M)	LOM unit cost (CAD$/t underground ore)
BMC labour costs	$6.5	$3.79
Plant and equipment costs	$2.2	$1.27
Material and services costs	$45.0	$26.04
Contractors and consultants' costs	$2.9	$1.66
Total	**$56.6**	**$32.76**

BMC staff for management and supervision of the underground mining operation are summarized in Table 21-25. Labour costs have been benchmarked against data available for comparable northern mining operations and projects. The basis for salary loading is described in Section 21.2.9.

Table 21-25: BMC underground mining personnel

Position	Roster type	Personnel	Loaded cost (CAD$/year)
Underground Superintendent	2/1	1	$171,000
Mine Engineer	2/1	1	$119,000
Geologist	2/1	2	$107,000
Surveyor	2/1	1	$93,000
Surface Agi Truck Operator	2/1	9	$113,000
Total			**$1,615,000**

As noted in Section 21.1.14, development of the underground mine has been capitalized. This includes all excavations relating to infrastructure including: the Main Ramp, Return Air Drifts, Escapeway Drifts, Footwall Access Drifts, Stockpiles and Sumps. A portion of fixed costs associated with underground mining were allocated to capital by pro-rata of development tonnes. The distribution of capital and operating costs over the life of the mine is summarized in Table 21-26.

Table 21-26: LOM underground mining capital and operating costs

Cost	LOM cost (CAD$M)	LOM unit cost (CAD$/t underground ore)
Underground mining contractor costs	$179.5	$103.83
BMC owner costs	$56.6	$32.76
Total underground mining costs	**$236.2**	**$136.59**
Capitalized mine development costs adjustment	($77.4)	($44.75)
Operating costs	$158.8	$91.84

21.2.4 Processing

Processing operating costs were estimated by Allnorth and include all site related operating costs associated with processing ore from the ROM pad to produce copper, lead and zinc concentrates, and filtered tailings. LOM processing costs are summarized in Table 21-27 and are discussed in more detail below.

Table 21-27: Processing operating cost summary

Cost centre	LOM cost (CAD$M)	LOM unit cost (CAD$/t)
Labour	$66.7	$4.24
Power	$91.6	$5.82
Maintenance Materials	$39.1	$2.48
Reagents and Consumables	$118.4	$7.53
Miscellaneous	$13.8	$0.88
Total	**$329.6**	**$20.96**

Labour

Labour numbers and costs for the Process Plant facility are detailed in Table 21-28. Senior management will work a nine-day on/five-day off roster, with all other labour working a two-week on/one-week off roster. Maintenance personnel on a two-week on/one-week off roster will typically work on dayshift only, with a callout system implemented for any critical maintenance requirements on nightshift.

Labour costs have been benchmarked against data available for comparable northern mining operations and projects. The basis for salary loading is described in Section 21.2.9.

Table 21-28: Processing labour summary

Position	Roster type	Personnel	Loaded cost (CAD$/year)
Treatment Plant Manager	9/5	1	$171,000
Plant Superintendent	9/5	1	$142,000
Safety and Training	9/5	1	$113,000
Shift Supervisor	2/1	3	$130,000
Plant Operator	2/1	24	$130,000
Mill Day Crew	2/1	3	$130,000
Clerk	9/5	1	$84,000
Senior Metallurgist	2/1	1	$136,000
Metallurgist	2/1	2	$119,000
Water Treatment Plant Operator	2/1	3	$130,000
Maintenance Manager	9/5	1	$171,000
Maintenance Planner	9/5	1	$119,000
Maintenance Supervisor (Mechanical)	9/5	1	$125,000
Maintenance Supervisor (Electrical)	9/5	1	$125,000
Fitter	2/1	6	$159,000
Boilermaker	2/1	2	$159,000
Electrician	2/1	3	$159,000
Trades Assistant/Apprentice	2/1	2	$113,000
Instrument Technician	2/1	1	$159,000
Refrigeration/HVAC	2/1	2	$159,000
LV Mechanic	2/1	1	$159,000
Total			**$8,342,000**

Power

Power costs for the Process Plant facility are detailed in Table 21-29. Power consumption has been estimated for all electrical equipment, based on the installed power with typical power draw and service factors. Power consumption for the SAG Mill and Ball Mill has been based on the expected pinion power on average ore hardness determined from the comminution testwork, with allowance for drive losses. Power consumption for the regrind mills has been based on the expected power determined from the design regrind work indices. Unit power costs are detailed in Section 21.2.8 and average CAD$0.133/kWh over the life of the Project.

Table 21-29: Process Plant power costs

Process Plant area	Installed power (kW)	Consumed power (kWh/t)	LOM power cost (CAD$M)
Crushing	456	0.7	$1.6
Ore Storage	54	0.1	$0.2
Grinding	6,856	19.1	$39.9
Flotation	2,913	7.1	$14.9
Regrind	1,627	3.4	$7.1
Tailings Thickening	400	0.5	$1.0
Tailings Filtration	1,087	1.5	$3.2
Concentrate Dewatering	1,086	1.4	$2.8
Concentrate Storage	715	1.5	$3.2
Reagents	229	0.4	$0.8
Services: Water	1,048	1.6	$3.4
Services: Water Treatment	229	0.4	$0.7
Services: Air	1,755	4.3	$8.9
Services: Other	3	0.0	$0.0
Plant HVAC	812	1.7	$3.9
Total Power	**19,268**	**43.8**	**$91.6**

Maintenance Materials

Maintenance costs for the processing plant include irregular, non-scheduled, and minor spares/parts/consumables and minor capital equipment replacement and/or modification required to keep the plant functional and fit for purpose.

Regular, scheduled, and major spares and consumables, including grinding media, crusher liners, mill liners, and filter cloths, are not included in the Maintenance estimate. These items are included in the Reagents and Consumables estimate.

Maintenance costs have been estimated for each plant area, as a percentage of the direct installed capital cost (maintenance factor), based on data from similar operations, as shown in Table 21-30.

Table 21-30: Process Plant maintenance costs

Process Plant area	Maintenance costs (CAD$/t)	LOM maintenance cost (CAD$M)
Crushing and Ore Storage	$0.26	$4.2
Grinding	$0.38	$6.0
Flotation and Regrind	$0.71	$11.1
Tailings	$0.35	$5.4
Concentrate	$0.39	$6.1
Reagents	$0.16	$2.5
Services	$0.09	$1.4
Plant Buildings	$0.09	$1.5
Plant Infrastructure	$0.05	$0.8
Total	**$2.48**	**$39.1**

Reagents and Consumables

Reagents and Consumables include all plant reagents and all regular, scheduled, or major plant spares and consumables, including grinding media, crusher liners, mill liners, and filter cloths. Irregular, non-scheduled, or minor spares and consumables are not included in the Reagents and Consumables estimate. These items are included in the Maintenance estimate.

Reagent consumptions have been derived from the DFS metallurgical test work program. Crusher and mill liner wear was estimated based on operations with similar ore characteristics. Steel grinding media wear was calculated using the ore abrasion index and established Bond media wear equations.

Reagents and consumables costs have been summarized in Table 21-31.

Table 21-31: Process Plant reagents and consumables costs

Reagent/Consumable	Unit cost (CAD$/t)	Consumption rate (kg/t)	LOM reagents and consumables cost (CAD$M)
Quicklime	$340	1.306	$7.0
SMBS	$1,000	0.648	$10.2
NaCN	$4,500	0.121	$8.6
$ZnSO_4$	$1,813	0.374	$10.7
$CuSO_4$	$3,438	0.634	$34.3
DF469	$6,750	0.031	$3.3
3418A	$14,375	0.009	$2.0
DF262	$4,000	0.077	$4.8
Frother	$5,000	0.062	$4.8
Flocculant	$3,843	0.027	$1.6
Grinding media (total)			$18.8
Liners (total)			$10.2
Filter cloths (total)			$1.9
Total			**$118.4**

Miscellaneous Costs

Miscellaneous costs allow for the items summarized in Table 21-32.

Table 21-32: Process Plant miscellaneous costs

Item	LOM miscellaneous cost (CAD$M)
External testwork	$2.0
Consultants	$1.6
Mobile equipment	$1.3
Contract cranage	$1.2
General contract labour	$1.2
Plant loaders	$6.5
Total	**$13.8**

21.2.5 Water Treatment Plant

Operating costs for the WTP were estimated by ISC and the LOM water treatment operating cost is summarized in Table 21-33. The underlying operating inputs for water treatment are tabulated in Table 21-34 (Metals Removal Circuit) and Table 21-35 Selenium Removal Circuit). Variable operating costs were estimated on a cost per cubic metre of water treated basis, with the volumes of water treated on a monthly basis detailed in Section 18.3. Labour fixed costs for the WTP are included in Table 21-28.

Table 21-33: Water treatment plant operating cost summary

Circuit	Cost centre	LOM cost (CAD$M)	LOM unit cost (CAD$/t)
Metals Removal	Reagents	$3.9	$0.25
	Maintenance	$0.3	$0.02
	Power	$1.2	$0.08
Selenium Removal	Reagents	$7.9	$0.50
	Maintenance	$0.7	$0.04
	Power	$2.3	$0.15
Total		**$16.3**	**$1.04**

Table 21-34: Metals Removal Circuit operating costs[1]

Reagents and consumables	Consumable unit cost (CAD$/t)	Water treated per tonne of consumable (Ml/t)	Unit cost of water treated (CAD$/Ml)
Flocculant	$6,550	871.23	$7.5
Ferric Sulphate	$1,980	66.25	$29.9
TMT	$3,500	2,544	$1.4
NaHS	$1,350	1,060	$1.3
Caustic	$1,350	29.17	$46.3
Hydrated lime	*$850*	*2.59*	*$328.1*
Maintenance	**Maintenance cost (CAD$/day)**	**Design volume water treated (ML/day)**	**Unit cost of water treated (CAD$/Ml)**
Metals Removal Circuit	$125	12.72	$9.8
High-Density Sludge Circuit	*$118*	*12.72*	*$9.3*
Power	**Average LOM power cost[2] (CAD$/kWh)**	**Power consumption (kWh/Ml)**	**Unit cost of water treated (CAD$/Ml)**
Metals Removal Circuit	$0.133	461.79	$61.2
High-Density Sludge Circuit	*$0.133*	*41.19*	*$5.5*
AVERAGE VARIABLE COST FOR METALS REMOVAL CIRCUIT			**$157**
AVERAGE VARIABLE COST FOR METALS REMOVAL WITH HIGH-DENSITY SLUDGE			***$500***

Notes:
1. *Costs identified in italics are incurred once water from the Class A Waste Storage Facilities is predicted to become acidic requiring operation of a High-Density Sludge Circuit and are in addition to other costs.*
2. *Power cost varies over the life of the Project (Section 21.2.8). LOM average cost used for presentation of data.*

Table 21-35: Selenium Removal Circuit operating costs

Reagents and consumables (units)	Consumable unit cost (CAD$/unit)	Water treated per unit of consumable (Ml/unit)	Unit cost of water treated (CAD$/Ml)
Flocculant (t)	$6,550	6,376	$1.0
Ferric Sulphate (t)	$1,980	533.3	$3.7
Sodium Sulphate (t)	$1,980	2.49	$795.5
Sulphuric Acid (t)	$1,050	11.59	$90.6
Resin (m^3)	$4,300	305.5	$14.1
Steel Anodes (t)	$1,153	2.85	$404.9
Maintenance	**Maintenance cost (CAD$/day)**	**Design volume water treated (ML/day)**	**Unit cost of water treated (CAD$/Ml)**
Selenium Removal Circuit	$365	3.36	$108.6
Power	**Average LOM power cost[1] (CAD$/kWh)**	**Power consumption (kWh/Ml)**	**Unit cost of water treated (CAD$/Ml)**
Selenium Removal Circuit	$0.133	2,887.5	$382.7
AVERAGE VARIABLE COST FOR SELENIUM REMOVAL CIRCUIT			**$1,801**

Note: 1. Power cost varies over the life of the Project (Section 21.2.8). LOM average cost used for presentation of data.

21.2.6 Road Transport of Concentrates

Costs for road transportation of concentrates from the KZK Project to Stewart World Port were estimated by Allnorth. Allnorth prepared the estimate on a first-principals basis as well as engaged with transportation suppliers for budget pricing. A summary of the unit costs established by Allnorth for haulage of concentrate is detailed in Table 21-36, which should be read in conjunction with Section 18.14 regarding "Restricted" and "Non-Restricted" transport periods.

Table 21-36: Concentrate road transportation unit costs

Concentrate type	Non-Restricted period unit cost (CAD$/wmt)	Restricted period unit cost (CAD$/wmt)	Average blended unit cost (CAD$/wmt)
Days in period	320	45	365
Copper/Zinc concentrate	$130.54	$183.96	$137.13
Lead concentrate	$151.54	$194.28	$156.81

The total cost of road transportation of concentrate over the life of the project is summarized in Table 21-37.

Table 21-37: Total concentrate road transportation costs

Concentrate type	Concentrate transported (wmt)	LOM cost (CAD$M)
Copper	441,000	$60.5
Lead	414,000	$65.0
Zinc	1,665,000	$228.4
Total	**2,521,000**	**$353.8**

21.2.7 Sea Transport and Port Operations

Budget port operational costs were provided by Stewart World Port (SWP, 2019) for the handling of the three concentrate products, as summarized in Table 21-38. Unit costs are inclusive of all storage, loading, wharfage and terminal fees.

Table 21-38: Port operational costs

Concentrate type	Concentrate shipped (wmt)	Unit cost (CAD$/wmt)	LOM cost (CAD$M)
Copper	441,000	$17.50	$7.7
Lead	414,000	$25.00	$10.4
Zinc	1,665,000	$17.50	$29.1
Total	**2,521,000**		**$47.2**

Sea transport operating costs were provided by Braemar (Braemar, 2018) for shipping between Stewart and East Asian destinations, as summarized in Table 21-39.

Table 21-39: Sea transport unit costs

Load port	Discharge port	Bulk freight rate (US$/wmt)		
		5,500 t	11,000 t	16,500 t
Stewart World Port	North China	$63.00	$42.00	$38.00
	South China	$68.00	$45.00	$41.00
	Korea	$60.00	$40.00	$37.00
	Japan	$63.00	$42.00	$38.00

Zinc concentrates will be shipped in 11,000 wmt lots, while copper and lead concentrates will be shipped in 5,500 wmt lots. Once in production, a minimum of 11,000 wmt of concentrate will be shipped every month. In many months two shipments of 11,000 wmt will be made. Ordering of ships for concentrate transportation will be managed such that 5,500 wmt lots of copper (or lead) concentrate will be shipped at the same time as an 11,000 wmt lot of zinc concentrate or a 5,500 wmt of lead (or copper) concentrate. In this manner unit rates for shipping 11,000 wmt parcels will be maintained as a minimum and in many months 16,500 wmt freight rates should be realized.

The DFS has assumed that all sea freight will attract the 11,000 wmt freight rates. In consideration that all four discharge port locations are potential receiving locations for KZK concentrates, the mid range unit cost of US$42/wmt has been allowed for all shipped concentrate. Shipping costs over the life of the mine are summarized in Table 21-40. Insurance costs and referee charges of US$5/dmt and US$5/dmt respectively have been allowed for.

Table 21-40: Sea freight costs

Concentrate type	Concentrate transported (wmt)	LOM cost (USD$M)	LOM cost (CAD$M)
Copper	441,000	$18.5	$23.7
Lead	414,000	$17.4	$22.3
Zinc	1,665,000	$69.9	$89.4
Subtotal	**2,521,000**	**$105.9**	**$135.4**
Insurance and Referee Charges		$22.9	$29.3
TOTAL		**$128.8**	**$164.7**

21.2.8 Power Generation and Fuel

The cost of electrical power is distributed to each operating cost centre on a cost per kWh consumed basis. The underlying inputs for power generation costs are detailed in Table 21-41, and were estimated by Allnorth. On current fuel price projections, it is expected that LNG will be the primary fuel used for power generation. The dual fuel generators can operate at up to 99% LNG/1% diesel mix. The DFS has assumed that 95% LNG/5% diesel will be the average fuel blend over the life of the project.

Table 21-41: Power generation input costs

Input	Value
Natural Gas Consumption Rate	7,711 kJ/kWh
Diesel Consumption Rate	0.235 l/kWh
Lubricant Consumption Rate	0.0006 l/kWh
Natural Gas Unit Cost	Variable, see Table 21-42
Diesel Unit Cost	Variable, see Table 21-42
Lubricant Unit Cost	CAD$3.17/l
Maintenance Cost	CAD$17,022/month/generator

LNG and diesel fuel costs used in the DFS vary each year in accordance with commodity price forecasts (Table 21-42). LNG commodity price forecasts were sourced from GLJ Petroleum Consultants Ltd (2019). Edmonton Par oil pricing was derived from West Texas Intermediate oil forecasts (BMC, 2019a).

Table 21-42: Fuel unit costs

Year	LNG commodity AECO[1] (CAD$/GJ)	LNG delivered to site (CAD$/GJ)	Edmonton Par oil price (CAD$/bbl)	Diesel, power generation (CAD$/l)	Diesel, off-road Use (CAD$/l)
2021	$2.53	$14.47	$73.9	$0.980	$1.020
2022	$2.75	$14.69	$75.9	$0.995	$1.035
2023	$2.98	$14.92	$76.7	$1.001	$1.041
2024	$3.06	$15.00	$76.7	$1.001	$1.041
2025	$3.17	$15.11	$76.7	$1.001	$1.041
2026	$3.23	$15.17	$76.7	$1.001	$1.041
2027	$3.30	$15.24	$76.7	$1.001	$1.041
2028	$3.36	$15.30	$76.7	$1.001	$1.041

Note: 1. The AECO trading hub was used as the reference natural gas benchmark for supply of LNG to the KZK Project. The AECO trading hub is Canada's largest natural gas trading hub and serves as a benchmark for Alberta wholesale natural gas transactions.

Average annual unit power costs over the life of the project are summarized in Table 21-43.

Table 21-43: Average annual power generation unit costs

Year	Unit power cost (CAD$/kWh)
2021	$0.143
2022	$0.128
2023	$0.129
2024	$0.131
2025	$0.133
2026	$0.134
2027	$0.134
2028	$0.134
2029	$0.137
LOM average	$0.133

21.2.9 General and Administration

A summary of the annual site administration costs is shown in Table 21-44.

Table 21-44: G&A costs

Administration cost	LOM cost (CAD$M)
Labour	$25.2
Vehicles	$1.0
Power and Heating	$32.9
Administration	$9.9
Health and Safety	$0.9
Transportation	$18.3
Human Relations/Public Relations	$1.9
Environment	$0.9
Site Services	$25.3
Accommodation	$38.8
Offsite Concentrate Marketing	$14.7
Total	**$169.6**
Capitalized Pre-Production Costs	$2.9
Administration Operating Costs	**$166.7**

Five vehicles are included in the administration cost area for management, stores, environment and gatehouse security.

Power and heating costs include providing power to offices, Process Plant workshop, warehouse, laboratory, water management pumps and the camp. It also includes LNG fuel when heat from supplementary boilers is required.

Administration costs cover all costs for the day to day administration of the operation including legal, insurances, permits and payroll. Health and safety costs include safety and medical supplies and safety incentive payments. Transportation costs include charter flights, bus transportation and freight. Human relations costs include costs for recruitment and apprentice training. Environmental costs include all costs for ongoing environmental monitoring and compliance. Site services costs include provision of site assaying services and Access Road maintenance. Accommodation costs cover the cost for provision of accommodation to all personnel on site. Offsite concentrate marketing allows for marketing of concentrates and logistics planning of shipping to final customers.

Labour

Labour numbers and costs for mine administration are detailed in Table 21-45. Senior management will work a nine-day on/five-day off roster, with all other labour working a two-week on/one-week off roster.

Table 21-45: General and Administration labour summary

Position	Roster type	Personnel	Loaded cost (CAD$/year)
General Manager	9/5	1	$264,000
OHS&T Manager	9/5	1	$148,000
Commercial & Administration Manager	9/5	1	$211,000
Environment and Community Manager	9/5	1	$148,000
IT Officer/Business Systems	9/5	1	$101,000
Site Accountant	9/5	1	$142,000
Accounts Payable Clerk	9/5	1	$90,000
Payroll Clerk	9/5	1	$90,000
Administration Clerk	9/5	1	$84,000
Store Manager and Procurement Supervisor	9/5	2	$130,000
Purchasing Officer	9/5	1	$119,000
Stores Personnel	9/5	3	$96,000
Gatehouse Security	2/1	3	$96,000
Environmental Officer	2/1	1	$101,000
Environmental Technician	2/1	2	$90,000
Community Liaison/Mentor	9/5	2	$101,000
Site Nurse/ERT	2/1	2	$107,000
Emergency Services Officer/ERT	2/1	2	$107,000
Annual Cost			**$3,144,000**

Labour costs have been benchmarked against data available for comparable northern mining operations and projects. Salary loading for all BMC staff includes:

- Four weeks annual leave per year
- Statutory payments for Canadian Pension Plan, Employment Insurance and Yukon Workers Compensation Board
- Statutory holiday allowance of 10 days per year

- Flexible benefits package of $5,000 per year for the employee to use for health insurance and other medical benefits.

21.2.10 Equipment Leases

Certain capital equipment assets for the establishment of the operation are planned to be provided under an operating lease arrangement. These assets include the power generation plant and associated LNG fuel storage equipment, heavy mobile equipment for the processing facility and light vehicles for all departments to commence operations. Leasing terms were sourced from a Canadian financing provider and are summarized in Table 21-46. At the conclusion of the leasing period, ownership of the assets will be transferred to BMC for a nominal payment of CAD$1. Replacement equipment required beyond this period are included in sustaining capital.

Table 21-46: Operating lease costs

Operating lease	Lease term	LOM lease cost (CAD$M)
Power Generation	72 months	$66.3
LNG Fuel Storage	72 months	$6.0
Processing Heavy Vehicle	60 to 72 months	$4.4
Light Vehicles	48 months	$1.3
Total		**$77.9**

21.2.11 First Nations

Payments to First Nation communities have been estimated based on expected costs for administration and profit sharing over the LOM and total CAD$50 million.

21.2.12 Royalties

No vendor or third-party royalties are applicable to the mine plan presented in this feasibility study.

Yukon mining royalties under the *Quartz Mining Act* are payable to the Yukon Government annually. The *Quartz Mining Act* royalty is a net profits royalty, based on annual mineral production and sales after deduction of eligible expenses and allowances. Deductible, eligible expenses include:

- On-site production costs, including related exploration and development costs
- Offsite costs for preparing and transporting mineral concentrates
- Reclamation costs
- Development Allowance (amortizing eligible pre-production exploration and development costs to bring the mine into production)
- Depreciation Allowance (depreciation of original capital costs of eligible assets on 15% per annum straight line basis until costs are fully deducted)
- Community and Economic Development Allowance (qualifying expenditures in community and economic developments are pooled and deducted at the lesser of the remaining undeducted balance, 15% of the amounts claimed in the year as Deductions, Development Allowance and Depreciation Allowance, or 20% of the value of output of a mine after other deductions in the year).

The royalty is applied in accordance with Table 21-47. The royalty is applied on an incremental cumulative basis until the mine's calculated net profit is accounted for.

Table 21-47: Quartz Mining Act royalty rates

Royalty lower band (CAD$)	Royalty upper band (CAD$)	Royalty applicable to band
$0	$10,000	0%
$10,000	$1,000,000	3%
$1,000,000	$5,000,000	5%
$5,000,000	$10,000,000	6%
$10,000,000	$15,000,000	7%
$15,000,000	$20,000,000	8%
$20,000,000	$25,000,000	9%
$25,000,000	$30,000,000	10%
$30,000,000	$35,000,000	11%
$35,000,000 and above		12%

Source: Quartz Mining Royalty Regulation (OIC 2010/91), Energy Mines and Resources, Yukon Government.

LOM royalties payable under the *Quartz Mining Act* were estimated to be CAD$221 million.

22 Economic Analysis

22.1 Economic Analysis Summary

Economic analysis of the project as presented in this Technical Report demonstrates that the KZK Project is commercially viable given the Base Case economic results presented in Table 22-1. These Base Case results were estimated using the assumptions detailed in Table 22-2. The Base Case LOM metal price assumptions and treatment charges used (which are in real dollar terms), include industry consensus metal prices and concentrate charges derived from long-term forecasts. Base Case metal production is shown in Table 22-3 and annualized Base Case cash flows (in US$) are shown in Table 22-4.

Table 22-1: Economic result – Base Case

Financial metric	Unit	Pre-tax	Post-tax
Free Cash Flow	US$M	$1,245	$901
NPV @ 7% pa[1] (June 30, 2019 valuation)	US$M	-	$527
IRR (June 30, 2019 valuation)	%	-	40%
Payback period[2]	years	-	1.9
EBITDA (steady state average per year)[3]	US$M	$245	-

Note:
1. Real after-tax discount rate.
2. From start of production December 2021.
3. Excludes Year 1 and final year of production.

Table 22-2: Project assumption – Base Case

Commodity	Unit	LOM average price ($/lb)	LOM average price ($/t)	Parameter	Value
Copper	US$/lb	$3.16	$6,960	CAD:US$ Pre-production	0.76 to 0.77
Zinc	US$/lb	$1.10	$2,435	CAD:US$ Long term	0.78
Lead	US$/lb	$0.95	$2,104	NPV discount rate	7%
Gold	US$/oz	$1,322			
Silver	US$/oz	$18.02			

The Base Case economic results are based on metal production as shown in Table 22-3.

Table 22-3: Base Case metal production

	Copper		Zinc		Lead		Gold	Silver
	'000 t	M lb	'000 t	M lb	'000 t	M lb	'000 oz	M oz
Year 1 production	5.6	12	81.0	179	22.5	50	48.5	6.6
Steady state average per annum[1]	14.4	32	106.8	235	25.3	56	56.5	7.8
LOM production	100.2	221	786.3	1,733	195.4	431	432.0	59.8

Note: 1. Excludes Year 1 and final year of production.

Table 22-4: Base Case cash flow (US$M)

	Total	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031 to 2035
Revenue														
Copper revenue	670	0	0	0	43	75	117	133	100	75	75	51	0	0
Zinc revenue	1,619	0	0	0	181	225	235	203	225	214	214	122	0	0
Lead revenue	387	0	0	0	45	52	52	36	57	52	57	38	0	0
Gold in concentrate revenue	478	0	0	0	55	64	68	61	69	51	65	45	0	0
Silver in concentrate revenue	909	0	0	0	101	118	129	104	129	107	130	92	0	0
Total gross revenue	**4,064**	**0**	**0**	**0**	**426**	**534**	**601**	**538**	**580**	**498**	**540**	**348**	**0**	**0**
Selling costs														
TC/RCs	500	0	0	0	56	69	73	63	70	64	66	40	0	0
Transport & port handling	314	0	0	0	34	43	45	42	44	41	40	24	0	0
Ocean freight	129	0	0	0	13	17	19	17	18	17	17	10	0	0
Penalties	31	0	0	0	5	6	5	4	4	2	3	2	0	0
Total selling cost	**974**	**0**	**0**	**0**	**109**	**135**	**142**	**127**	**136**	**124**	**126**	**76**	**0**	**0**
Net revenue	**3,090**	**0**	**0**	**0**	**317**	**399**	**459**	**411**	**444**	**375**	**414**	**272**	**0**	**0**
Operating costs														
Open pit mining costs	485	0	0	3	50	96	101	88	47	46	35	19	0	0
Underground mining costs	124	0	0	0	0	0	0	0	36	42	29	17	0	0
Processing costs	270	0	0	1	31	34	33	37	35	36	37	26	0	0
G&A costs	130	0	0	1	16	17	17	17	16	16	16	13	0	0
First Nations operating costs	39	1	2	5	5	5	5	5	5	5	5	0	0	0
Equipment leases	61	0	0	2	8	8	8	11	11	8	4	3	0	0
Total operating costs	**1,109**	**1**	**2**	**12**	**109**	**159**	**164**	**157**	**150**	**154**	**125**	**78**	**0**	**0**
Cash flow from operations	1,981	-1	-2	-12	208	240	295	254	294	220	290	194	0	0
Other														
Cash income tax	344	0	0	0	0	46	51	53	63	46	63	46	-11	-13
Yukon royalty cash tax	173	0	0	0	0	17	21	26	20	24	17	28	21	0
Working capital	0	-1	-10	-5	6	-6	-1	2	0	2	2	13	-4	2
Construction capital	381	2	101	277	1	0	0	0	0	0	0	0	0	0
Sustaining capital	127	0	0	3	18	9	30	24	32	7	4	0	0	0
Closure	79	0	0	0	0	0	0	0	0	0	0	0	19	60
Asset Terminal Value	-25	0	0	0	0	0	0	0	0	0	0	-25	0	0
Total other	**1,080**	**1**	**90**	**275**	**25**	**65**	**101**	**105**	**115**	**79**	**86**	**62**	**25**	**49**
Project free cash flow	901	-2	-92	-287	183	174	194	149	179	141	204	131	-25	-49

Annual C1 Costs, net of by-product and selling costs will range between US$(0.60) and US$0.00 per pound of payable zinc, with the LOM average being US$(0.25) per pound payable zinc, as shown in Table 22-5. As silver is the other key revenue contributor for the project, C1 and All-In Sustaining Costs (AISC) are also presented on a cost per ounce of payable silver basis. C1 and AISC are defined in Section 29.3.

Table 22-5: KZK Project – annual cash cost summary

	Unit	Total	2021	2022	2023	2024	2025	2026	2027	2028	2029
C1, net of by-product credits and selling cost	US$/lb Zn	**$(0.25)**	$0.00	$(0.27)	$(0.07)	$(0.28)	$(0.28)	$(0.34)	$(0.03)	$(0.39)	$(0.60)
AISC, net of by-product credits and selling cost	US$/lb Zn	**$(0.06)**	$0.00	$(0.19)	$0.05	$(0.04)	$0.00	$(0.08)	$0.13	$(0.28)	$(0.34)
C1, net of by-product credits and selling cost	US$/oz Ag	**$(21.43)**	$0.00	$(21.39)	$(18.68)	$(23.37)	$(26.17)	$(22.98)	$(19.25)	$(22.29)	$(20.10)
AISC, net of by-product credits and selling cost	US$/oz Ag	**$(15.78)**	$0.00	$(19.25)	$(14.75)	$(16.27)	$(17.46)	$(15.76)	$(13.98)	$(19.40)	$(14.54)

Annual project free cash flow for the Base Case is shown in Figure 22-1.

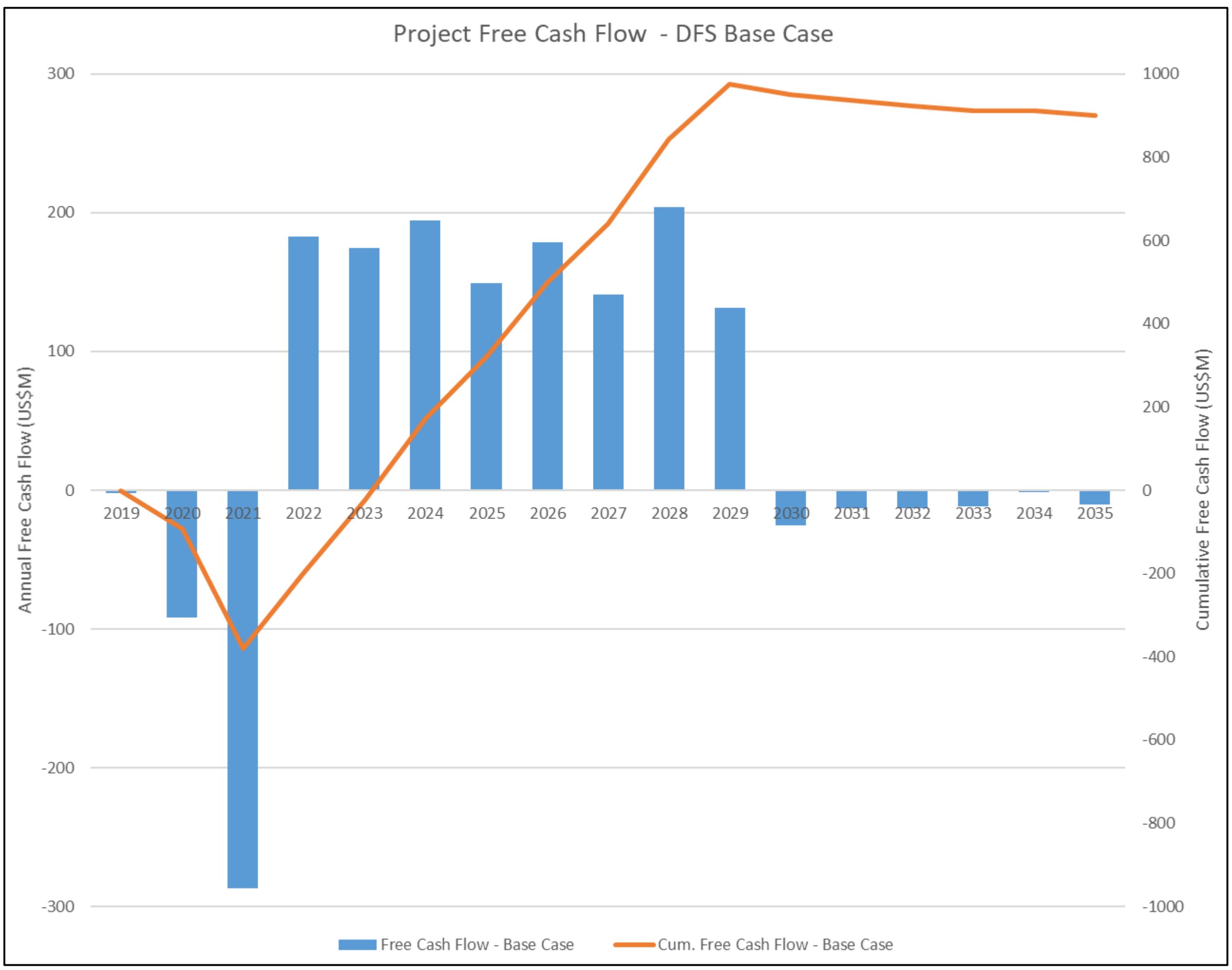


Figure 22-1: Annual Base Case Project Free Cash Flow

22.2 Taxation

The following discussion on taxation is based on the "Mining and Metals Tax Guide: Canada" produced by EY (EY, 2015) and updated here for recent changes.

22.2.1 Fiscal Regime

The fiscal regime that applies to the mining industry in Canada consists of a combination of income taxation at the federal level, and income taxation and mining taxes, duties or royalties at the provincial level:

- Income tax rate:
  - Federal corporate tax is 15%. Yukon corporate tax is 13%.
- Mining taxes, duties or royalties:
  - A progressive mining royalty applies in Yukon, based on annual mineral production and sales after deduction of eligible expenses and allowances.
- Investment incentives:
  - Research and development and mineral exploration tax credits.

Corporate Tax

For Canadian income tax purposes, a corporation's worldwide taxable income is computed in accordance with common principles of business (or accounting) practice, modified by certain statutory provisions in the *Canadian Income Tax Act* (the Act). In general, no special tax regime applies to mining enterprises.

Depreciation, depletion or amortization recorded for financial statement purposes is not deductible; rather, tax deductible capital cost allowances and deductions as specified in the Act are allowed. The annual tax deductions could vary from 6% to 100% of the capital expenditures depending upon the nature of a capital expenditure.

Mining corporations are taxed at the same rate as other corporations. Corporations are taxed by the Federal Government and by one or more provinces or territories. The basic rate of federal corporate tax is 38%[4], but it is further reduced to 28% by an abatement of 10% on a corporation's taxable income earned in a province or territory. The Yukon Territory tax rate is added to the federal tax.

No tax consolidation, group relief or profit transfer system applies in Canada. Each corporation computes and pays tax on a separate legal-entity basis. Business losses or non-capital losses may be carried back three years and forward 20 years.

Gains resulting from a disposal of a capital property are subject to income tax. Capital gains or losses are determined by deducting the adjusted cost base of an asset from proceeds of disposition (net of outlays incurred in connection with the disposition). For corporate taxpayers, one half of the capital gain (taxable capital gain) is taxed at normal income tax rates.

Capital losses are exclusively deductible against capital gains and not against other taxable income. However, non-capital losses are deductible against taxable capital gains, which are included in taxable income. Capital losses can be carried back three years and carried forward indefinitely for use in future years, provided an acquisition of control has not occurred. Mining rights and mineral resource properties are not capital properties for purposes of the Act.

[4] 38% is the general federal corporate tax rate (before abatement) for the 2019 calendar year and onward.

Mining Taxes, Duties or Royalties

Mining producers are required to pay a levy to the Crown (i.e. the Government) as the holder of the mineral rights on the extraction of minerals. In Canada, majority of the mineral rights are owned by the Crown on behalf of the people of Canada. Yukon mining royalties are a net profits royalty, based on annual mineral production and sales after deduction of eligible expenses and allowances. Further details are provided in Section 21.2.12.

The mining royalty is deductible in determining taxable income.

22.3 Sensitivity Analysis

The sensitivity of project value to key input parameters was assessed by varying each parameter over the range of -10% to +10%, and the results are detailed in Table 22-6 and Figure 22-2. All five metal inputs (zinc, copper, lead, gold and silver) were varied by the same amount for assessing sensitivities of metal price and head grade. Metal price and head grade inputs demonstrated the greatest sensitivity to project value. The exchange rate was the next most sensitive variable, followed by capital cost and finally operating cost.

Table 22-6: Sensitivity of after-tax NPV

	After-tax NPV @ 7% discount rate (US$M)								
Variation from Base Case	**-10%**	**-7.5%**	**-5%**	**-2.5%**	**0%**	**+2.5%**	**+5%**	**+7.5%**	**+10%**
Metal Price	$351	$395	$439	$483	$527	$571	$614	$658	$705
Head Grade	$381	$418	$454	$490	$527	$564	$601	$637	$674
Exchange Rate	$424	$452	$478	$503	$527	$549	$571	$591	$611
Capital Cost	$601	$583	$565	$546	$527	$507	$486	$465	$443
Operating Cost	$591	$575	$559	$543	$527	$511	$495	$479	$443

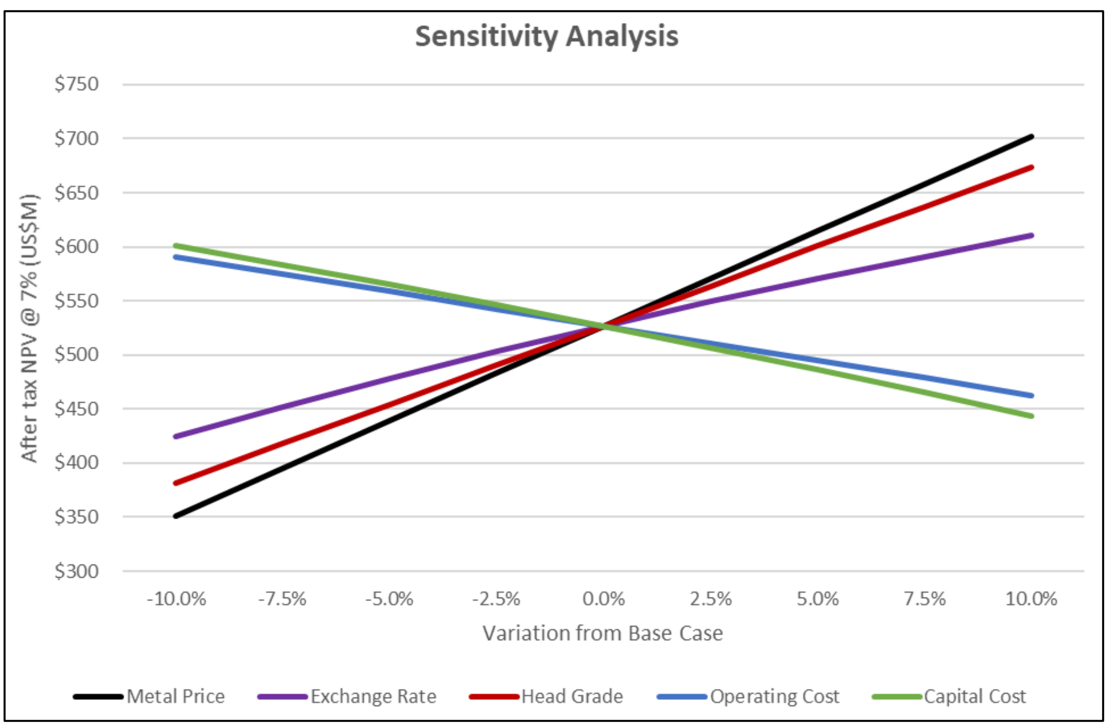


Figure 22-2: After-tax NPV @ 7% sensitivity analysis

A similar analysis was completed to assess impact on project fundamentals, again over a range of -10% to +10% for variations in metal price (Table 22-7) and exchange rate (Table 22-8).

Table 22-7: Sensitivity of economic parameters to metal price

	Change in metal price								
Variation from Base Case	**-10%**	**-7.5%**	**-5%**	**-2.5%**	**0%**	**+2.5%**	**+5%**	**+7.5%**	**+10%**
After-tax NPV @ 7% (US$M)	$351	$395	$439	$483	$527	$571	$614	$658	$705
IRR (%)	30%	33%	35%	37%	40%	42%	44%	46%	48%
Payback period (years)	2.7	2.5	2.3	2.3	1.9	1.9	1.8	1.8	1.8

Table 22-8: Sensitivity of economic parameters to exchange rate

	Change in exchange rate								
Variation from Base Case	**-10%**	**-7.5%**	**-5%**	**-2.5%**	**0%**	**+2.5%**	**+5%**	**+7.5%**	**+10%**
After Tax NPV @ 7% (US$M)	$424	$452	$478	$503	$527	$549	$571	$591	$611
IRR (%)	32%	34%	36%	38%	40%	42%	43%	45%	47%
Payback Period (years)	2.6	2.5	2.3	2.3	1.9	1.9	1.8	1.8	1.8

The impact on after-tax project value due to different discount rates is detailed in Table 22-9.

Table 22-9: Discount rate sensitivity on after-tax NPV

Discount rate (%)	After-tax NPV (US$M)
0%	$901
5%	$614
7%	$527
10%	$417

22.3.1 Tornado Analysis

The sensitivity of changes in key Project variables on Project After Tax NPV for Tornado Analysis has been determined by simple factoring of these elements. Most variables were assessed on a standard ±10% proportional change. The remaining variables are percentage variables (e.g. metal recovery) and were assessed based on flexing the variable by ±2 units to better represent the expected range of variability. The relative sensitivity to Project after-tax NPV, for the most sensitive variables, is shown in Figure 22-3 and results presented in Table 22-10.

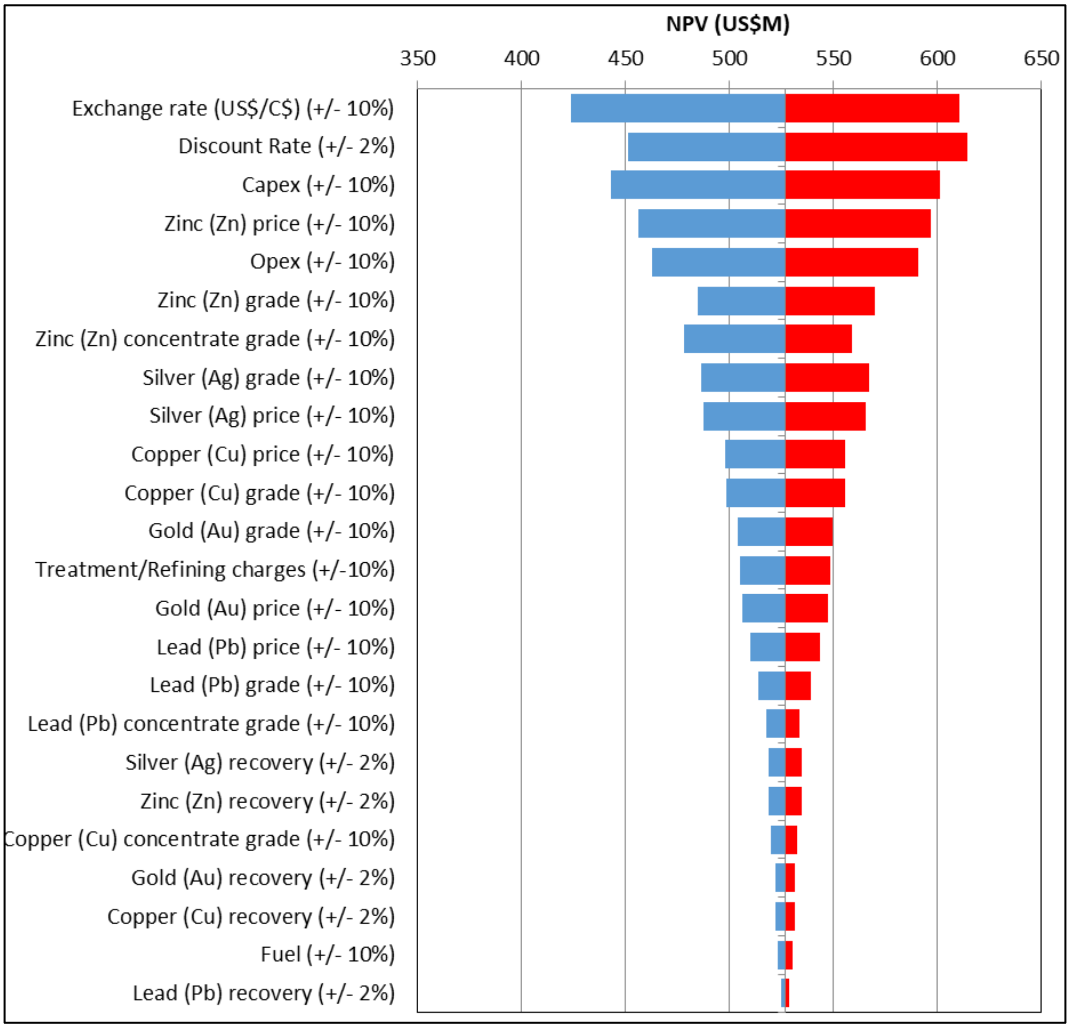


Figure 22-3: Relative sensitivities, Tornado analysis

Table 22-10: Sensitivity results, Tornado analysis

Variable sensitized	Low (US$M)	Base (US$M)	High (US$M)	Delta (US$M)
Exchange rate (US$/C$) (±10%)	$424	$527	$611	$186
Discount rate (±2%)	$451	$527	$614	$163
Capex (±10%)	$443	$527	$601	$158
Zinc (Zn) price (±10%)	$457	$527	$597	$140
Opex (±10%)	$463	$527	$591	$128
Zinc (Zn) grade (±10%)	$485	$527	$570	$85
Zinc (Zn) concentrate grade (±10%)	$478	$527	$559	$81
Silver (Ag) grade (±10%)	$487	$527	$567	$80
Silver (Ag) price (±10%)	$488	$527	$566	$78
Copper (Cu) price (±10%)	$498	$527	$556	$58
Copper (Cu) grade (±10%)	$499	$527	$556	$57
Gold (Au) grade (±10%)	$504	$527	$550	$46
'Treatment/Refining charges (±10%)	$505	$527	$548	$43
Gold (Au) price (±10%)	$506	$527	$547	$41
Lead (Pb) price (±10%)	$510	$527	$543	$33
Lead (Pb) grade (±10%)	$514	$527	$539	$25
Lead (Pb) concentrate grade (±10%)	$518	$527	$534	$16
Silver (Ag) recovery (±2%)	$519	$527	$535	$16
Zinc (Zn) recovery (±2%)	$519	$527	$535	$15
Copper (Cu) concentrate grade (±10%)	$520	$527	$533	$12
Gold (Au) recovery (±2%)	$522	$527	$531	$9
Copper (Cu) recovery (±2%)	$522	$527	$532	$9
Fuel (±10%)	$523	$527	$530	$7
Lead (Pb) recovery (±2%)	$525	$527	$529	$4
NPV range	$424	$519	$614	

23 Adjacent Properties

Significant VHMS deposits were discovered from 1994 to 1998 in the Finlayson Lake District. To date, at least 41 VHMS occurrences and five deposits have been discovered at different stratigraphic levels within the Finlayson Lake District (Ruijter et al., 2012). The five deposits; ABM, GP4F, Fyre Lake (Kona), Ice and Wolverine, collectively contain in excess of 40 Mt of base metal mineralization. With exception to the ABM and Fyre Lake deposits, the Qualified Person has been unable to verify the information and this information is not necessarily indicative of the mineralization on the property that is the subject of the technical report. Only the Wolverine deposit is considered to be an “adjacent property” for this report (Figure 23-1) as BMC has a beneficial interest in the Fyre Lake deposit.

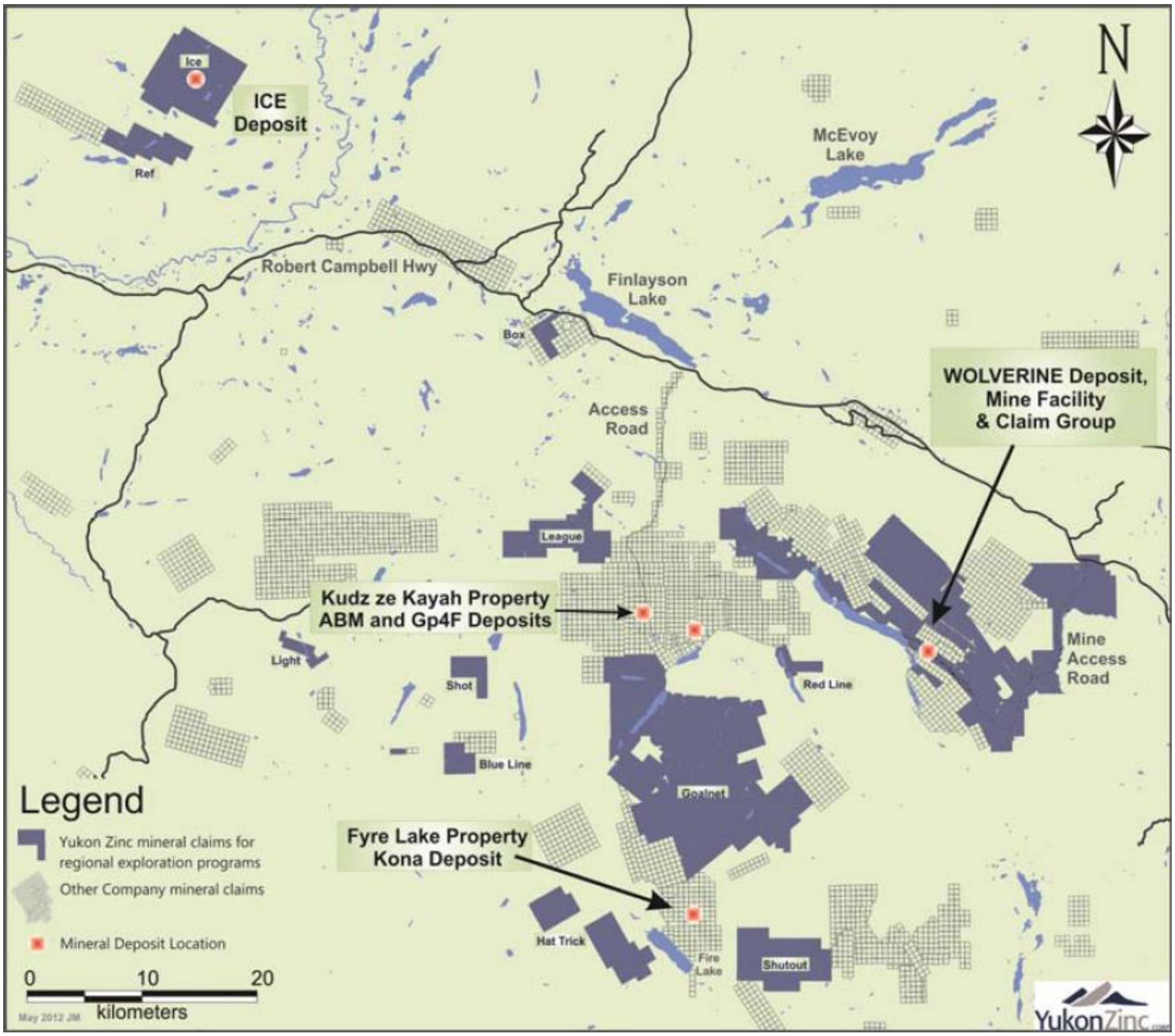


Figure 23-1: Adjacent property map
Source: Ruijter et al., 2012

23.1 Wolverine

The Qualified Person has not reviewed any technical data or technical reports for the Wolverine Property, and the following comments are based on data sourced from the public domain. The Qualified Person has been unable to verify the information and this information is not necessarily indicative of the mineralization on the property that is the subject of the technical report.

The Wolverine Mine is situated 30 km east of the ABM deposit. The mine, consisting of underground workings and a 750 kt/a processing facility (Figure 23-2), is wholly owned by Yukon Zinc Corporation and commenced full commercial production in 2013 with a Canadian NI 43-101 compliant Mineral Reserve (Proven and Probable) of 5.2 Mt @ 9.66% Zn, 0.91% Cu, 1.26% Pb, 281.8 g/t Ag and 1.36 g/t Au. The mine was place on care and maintenance in January 2015 (Yukon Zinc, 2016).

The Wolverine deposit was discovered in 1995 and is hosted by graphitic shales and felsic volcanic and volcaniclastic rocks. Sulphide mineralization occurs at the "Wolverine" and "Lynx" zones. They are laterally connected by stratabound, semi-massive replacement style Zn-Pb-Ag mineralization, called the "Saddle" zone. Strike lengths of both the Wolverine and Lynx zones are in the order of 150–250 m long with down-dip extents in excess of 450 m. True thicknesses of the Wolverine and Lynx zones are typically 3–5 m wide but can reach in excess of 16 m wide (Cowley and Song, 2014). Drilling by Expatriate in 2001 demonstrated that mineralization extends on to contiguous mineral claims held by BMC.

Remaining Mineral Resources or Mineral Reserves for the Wolverine deposit are unknown.

Figure 23-2: Overview of the Wolverine Mine and associated infrastructure
Source: Green, 2015b

24 Other Relevant Data and Information

24.1 Project Execution Plan

24.1.1 Introduction

This Project Execution Plan (PEP) was developed for the KZK Project DFS based on the latest information available. It describes the strategy for constructing and commissioning the KZK Project to bring it into the operational phase where marketable concentrates can be produced.

24.1.2 Health, Safety and Environment

The overarching objective of health, safety and environmental (HSE) management is to complete the implementation phase and transition into operations on a "zero-harm" basis. Fundamentally this will be achieved by continuously assessing and mitigating all unacceptable risks associated with each task.

BMC will develop overarching policies for the project relating to health, safety and environment, and are committed to developing a proactive safety and environmental aware culture with a "zero-harm" objective. BMC, the EPCM consultant and EPC contractors will establish HSE management plans that provide the details, management, and requirements to deliver the scope in a manner which aligns with BMC policies and complies with legislated requirements.

Some of the key elements of the HSE management plans will included the following:

- Project risk assessments and register
- Hazard and operability studies (HAZOP)
- Site Inductions
- Training, certification and registration
- Job Safety Analysis (JSA)
- Safe work procedures
- Personal safety checklists
- HSE Incident reporting and investigation
- HSE information and engagement meetings (toolbox meetings)
- Emergency response procedures.

Clinic first aid facilities and emergency response services will be provided by BMC for the KZK Project for the benefit of all stakeholders during construction and into operations. The EPCM consultant and other stakeholders will work with BMC to coordinate appropriate evacuation procedures, prior to commencement of the works.

During on-site construction and commissioning phases, the EPCM consultant and all stakeholders will work with BMC to ensure environmental compliance. Environmental risks will be assessed prior to authorizing the execution of any work. Regular audits and compliance reporting will be undertaken by BMC in conjunction with representatives of each stakeholder group. Corrective action registers will be maintained by BMC with compliance enforceable under all agreements.

Site safety and environmental performance and corrective action requirements will be communicated with all stakeholders associated with the KZK Project, including subcontractors, vendors and consultants to ensure consistent understanding and compliance.

24.1.3 Community Engagement

Although the KZK site is considered a remote location, there are several First Nation and other local communities within the region. BMC is committed to supporting these communities through providing employment opportunities during the various phases of the project.

During the implementation phase, BMC will engage with community leaders on a regular basis to assess the potential for members of the Frist Nations communities to participate in the project. In addition, it will be a commercial condition of all outsourced construction work that contractors actively seek to engage appropriately skilled First Nations labour where possible.

Cultural awareness programs will also be implemented during the construction phase to broaden the understanding and commitment of BMC, consultants and contractors alike.

24.1.4 Execution Strategy

The overarching strategy for managing execution of the KZK Project centres around clearly defined roles and accountabilities as well as best use of human resources based on experience and qualifications without the obstruction of corporate barriers.

Fundamentally, the strategy will be to use an Engineering Procurement and Contract Management (EPCM) contracting approach with functions shared between BMC and the EPCM consultant.

Management and Organization

BMC propose to operate a four-tier management structure during project implementation as illustrated in Figure 24-1.

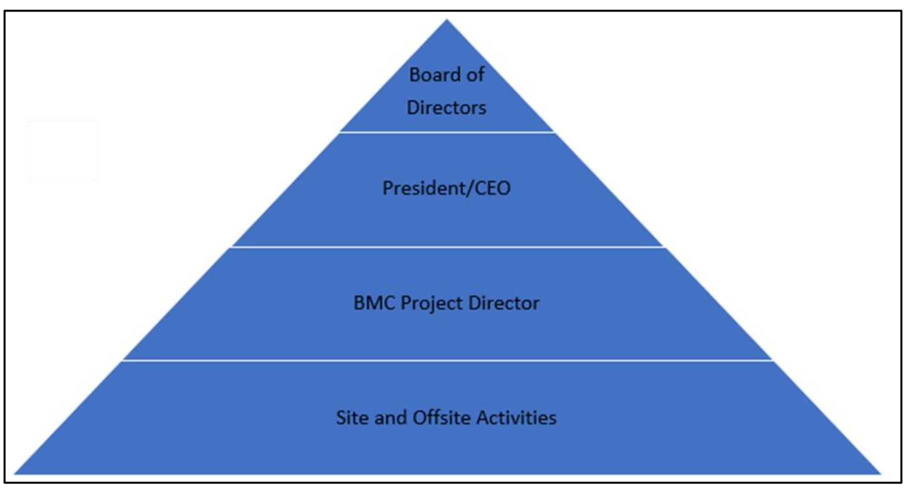


Figure 24-1: BMC management structure for KZK Project implementation

The BMC Project Director will oversee, monitor and manage the KZK Project from a strategic executive level. As the most senior authority over the project, this role is charged with managing project team members and ultimate decisions on the allocation of resources and directed effort. The role of the BMC Project Director will be critical to the coordination of activities and interoperability of the various departments, both on and off site.

There will be several subordinate departments reporting to the Project Director, as shown in Figure 24-2.

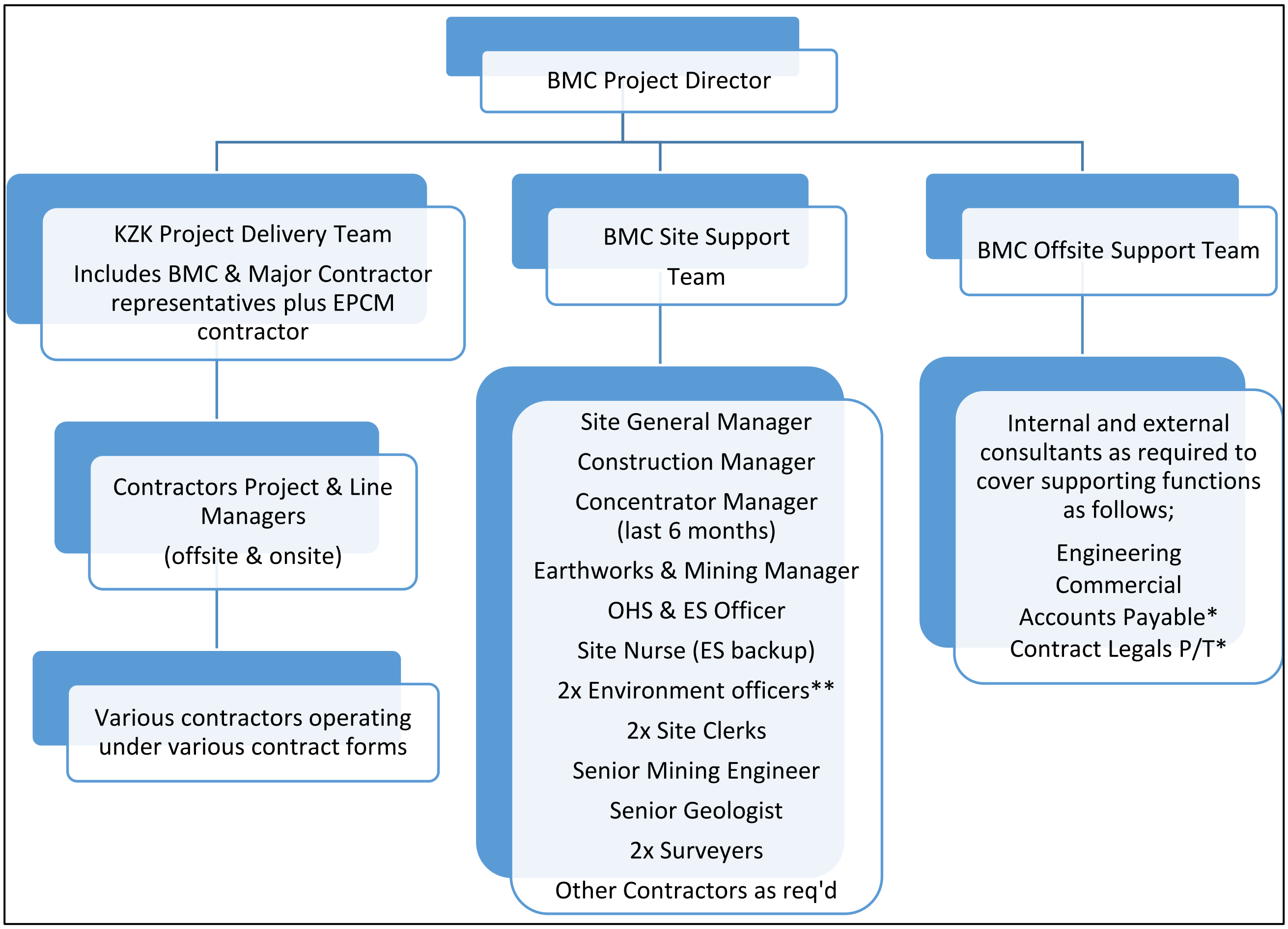


Figure 24-2: Project Director and subordinate departments

*Notes: *May report to CFO. **Quality control function through Environment Manager.*

The Project Delivery Team (PDT) will be responsible for the day-to-day management of activities during the execution phase. The team will operate as an integrated project team, with roles assigned based on competence, qualification and experience, regardless of their organization. The team will comprise primarily BMC and an EPCM consultant, as shown in Figure 24-3 and Figure 24-4. This may extend to other organizations depending on the talent pool available.

The site activities will also be supported by a number of offsite functions (e.g. Accounting) based in the BMC head office in Vancouver.

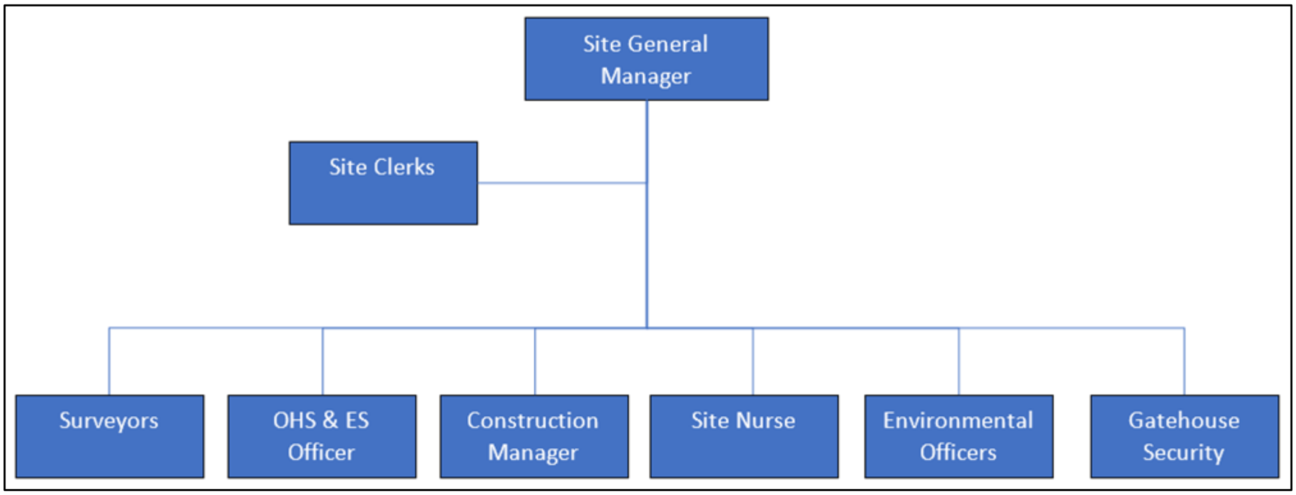


Figure 24-3 PDT – BMC personnel

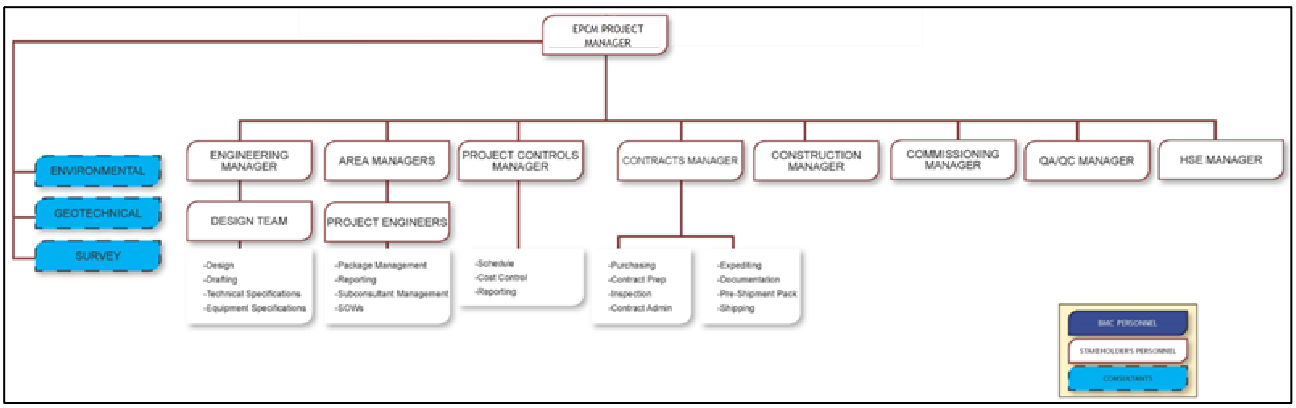


Figure 24-4: PDT – EPCM contractor personnel

Construction Management Plan

A detailed Construction Management Plan will be developed during the initial stages of Project Execution. The Construction Management Plan will be overseen and signed off by the Project Director. The plan will address the following areas:

- Project Scope
- Critical Success Factors
- Project Organization, Roles, Responsibilities and Authority
- Site Administration, Communication and Document Control
- Health, Safety and Environmental Management
- Human Resources/Industrial Relations
- Quality Control and Management
- Project Stakeholder Management
- Community Engagement
- Site Project Controls
- Contracts Management and Administration
- Construction Methodology and Sequence of Construction Packages

- Engineering
- Logistics
- Support Facilities, Services and Systems
- Construction Schedule
- Commissioning and Handover
- Construction Close Out and Handover.

Cost Control

A project Control Estimate will be set up following authority to proceed. The Control Estimate will include the original budget, keep track of the current project budget, current forecast, commitments, and expenditures. Budget and forecast costs will be input to the cost system by the project controls team, from estimates and deviations, in accordance with the PDT procedure.

The contingency provision will be managed by the BMC executive team and will be drawn down as required via variations or against each work package recommendation for award documentation. Contingency drawdown will be included in standard monthly reporting.

Progress Reporting

Progress reporting will occur on daily, weekly and monthly intervals with varying levels of detail and distribution, depending on their nature and content. All reports will be made available to the PDT and BMC Executive.

Early Works

Early works include all activities required to establish the project prior to the appointment of the EPCM consultant and mobilization to site of the main earthworks contractor. These activities are scheduled early to avoid impacting the critical path.

Early works activities include the following:

- Tendering for Temporary Facilities and Site Roads. These include phase 1 construction of the Tote Road, construction site offices, hard stand laydown areas and site roads necessary for the earthworks contractor.
- Construction and commissioning of the Phase 1 camp. This will provide 96 rooms and will be required to accommodate the earthworks contractor.
- Tendering for the Phase 2 camp to increase the camp capacity to 348 rooms, being the full design capacity for the facility. This will be required to accommodate peak labour force onsite during construction with bunking proposed to handle peak demand.
- Plant site geotechnical program involving site data collection and laboratory testing. This information is required to improve accuracy and understanding of the plant site conditions, develop effective dewatering and remedial action strategies as well as finalize foundation designs for the Process Plant and associated infrastructure.
- Water treatment plant treatability study and detailed engineering. This is required to confirm the performance and design of the WTP.

Mobilization

The BMC PDT will mobilize to site five months ahead of the EPCM consultant. Early works tasks will be completed by this stage. The BMC team will be tasked with establishing the site in preparation for the start of earthworks. This will include detailed planning and setting out the works so there is a seamless transition into executing the work.

A small temporary site office and associated facilities will be mobilized for BMC during this initial period. This facility will be later expanded upon arrival of the EPCM site personnel.

The earthworks contractor will commence mobilization at the same time the as the BMC PDT. In addition to mobile equipment, the earthworks contractors will mobilize temporary offices, workshops and refuelling facilities.

Various other specialist contractors associated with the installation of the Pit Rim Pond liner and discharge pipework will also be mobilized during this period. These will be allocated set areas within the initial hardstand.

The EPCM consultant will mobilize to site toward the end of the initial site earthworks (roads and diversion drains). Initially a small crew will be mobilized to site and will gradually increase in line with construction activities, peaking at an estimated 68 persons on site. Site facilities will be expanded accordingly to cater for this increase.

Earthworks

BMC will oversee all earthworks on site. The initial focus of the earthworks contractor will be to carry out the following activities in order of priority:

- Establish initial temporary diversions and coffer dams to capture and manage site runoff of the immediate construction area
- Construct Pit Rim Pond and associated access roads
- Construction hardstand laydown areas and Process Plant site access roads
- Plant site earthworks
- Development of Class A borrow pit
- Stage 1 A earthworks for Class A and B waste storage facility
- Fault Creek diversion channel
- Collection ponds for Class A, B and C and waste storage facilities
- Lower Water Management Pond.
- All other earthwork activities will be carried out by the mining contractor.

Planning and Scheduling

A WBS will be developed during the detailed engineering phase of the KZK Project. This will provide the basis for classification of facilities, asset management and elements for inclusion in specific work packages. The WBS will inform the structure of the project implementation schedule to three levels of detail.

The Project Master Schedule will ultimately be developed using Primavera P6 software or similar to four standard levels of detail:

- Level 1 – Management level schedule
- Level 2 – Project level schedule

- Level 3 – Control level schedule
- Level 4 – Vendor schedules.

The initial Project Master Schedule will be developed to Level 3 during detailed engineering phase over the course of several workshops. At the conclusion of the schedule planning workshops, the Project Director will review the Project Master Schedule for subsequent approval. Following approval of the Project Master Schedule, a baseline will be set to create a Project Baseline Schedule from which the progress of all activities will be monitored. Level 3 schedule was prepared for the KZK DFS with key milestones shown in Table 24-1.

Table 24-1: Key milestone dates

Milestone	Date
EPCM Contract Awarded	October 2019
Engineering Start	November 2019
Geotechnical Interpretation Report	November 2019
Decision Document Issued	January 2020
Quartz Mining Licence Issued	April 2020
Camp Complete	April 2020
Engineering Complete	September 2020
Process Plant Earthworks Completion	October 2020
Site Access for EPCM	October 2020
Commence Concrete	November 2020
Commence Open Pit Preproduction Works	March 2021
Type A Water Licence Issued	July 2021
Process Plant Building Complete	July 2021
Water Treatment Plant Complete	August 2021
Power Plant Commissioned	September 2021
Construction Complete (Commence Ore Commissioning)	December 2021
Port Facilities Complete	January 2022
Operations Handover	May 2022

Multiple critical paths were identified for the project. Following are key activities on the critical path:

- Mobilization of the EPCM contractor
- Finalization of process design
- Pre-engineered Process Plant building
- SAG and ball mills, flotation cells and regrind mills
- Mobilization of the concrete works
- Power station and associated LNG fuel storage
- Type A Water Licence.

24.1.5 Engineering

The detailed engineering phase of the KZK Project will begin with updating and finalizing the PDC ready for use by other disciplines. Subsequent to this, the plant and site general arrangements can be updated and issued. This will form the basis for earthworks design, roads and the reticulation of services between major project facilities, such as water, power and process pipelines. Equipment lists and datasheets will be updated in readiness for inclusion in the procurement packages. This will form an early focus of the engineering effort to ensure delivery of equipment does not negatively impact the project schedule and implementation sequence.

Development of the mechanical and structural design and drafting will enable the finalization of the civil design for the Process Plant. Electrical and instrumentation design can be finalized, once the process and instrumentation diagrams, and the mechanical equipment lists have been completed.

Other design work in this phase includes finalizing detailed mine design and scheduling for the ABM open pit up to and including the first year of production. This will also require review and update of the LOM mining strategy as appropriate.

The EPCM Project Manager will approve all designs (after sign-off by the Project Engineers and/or Area Managers) before presenting to the Project Director who will have the ultimate responsibility for the design.

The assigned Project Engineers will be responsible to ensure all presented design documentation, whether it be drawing or otherwise, be amended shortly after construction has been completed to reflect the "as-built" conditions on site.

24.1.6 Procurement and Contracts

The strategy for implementation of the processing plant and associated facilities will be driven by the opportunity to outsource the work to a suitably experienced contractor where the risk can be better managed and mitigated. The concept of using a consolidated management team is considered the most appropriate method of undertaking a major design and construct project of this nature. The PDT will include the services of an EPCM contractor who will be responsible for outsourcing and managing specific packages of work relating to the Process Plant, in line with implementation objectives.

The remaining works such as bulk earthworks for plant site, water storage infrastructure, waste storage facilities and all road works will be outsourced but managed by BMC personnel within the PDT. Mining will be also be outsourced to experienced open pit and underground contractors but overseen and technically supported by experienced BMC personnel.

Construction work will be outsourced in the form of individual work packages. Majority of the work will be carried out by Engineering, Procurement and Construction (EPC) contractors who will be utilized to complete works associated with mechanical installations, structural and platework steel installation and erection, electrical and instrumentation installation, and piping installations. Specialist subcontractors will be utilized to complete bulk earthworks and concrete batching and placement. A list of major work packages proposed for KZK implementation is given in Table 24-2.

Table 24-2: Major work packages for KZK Project implementation

Scope	Type of contract	Stakeholder	Managed by
Tote Road Upgrade	Schedule of Rates	Earthworks Contractor 1	BMC
Site Earthworks: Camp, Site Roads, Plant Site, Waste Storage Facilities, Water Management Structures, Borrow Pit	Schedule of Rates	Earthworks Contractor 2	BMC
Concrete Supply	Schedule of Rates	Specialist Contractor	EPCM contractor
Camp	Lump Sum	EPC Contractor No. 1	BMC
Non-processing Infrastructure inclusive of Process Plant Pre-Engineered Building, Administration Office, Plant Control Room, Plant Maintenance Workshop, Plant Warehousing Facilities, HVAC Systems, Assay Laboratory, Gatehouse, and Mine Dry Change Room	Lump Sum	EPC Contractor No. 2	EPCM contractor
Water Treatment Plant	Lump Sum	EPC Contractor No. 3	BMC
Bulk Diesel Fuel Storage and LV Fuel Storage and Dispensation System	Lump Sum	EPC Contractor No. 4	EPCM contractor
LNG Storage Facility	Lump Sum	EPC Contractor No. 5	EPCM contractor
Power Station	Lump Sum	EPC Contractor No. 7	EPCM contractor
Site wide power distribution	Lump Sum	EPC Contractor No. 8	EPCM contractor
Major Equipment (e.g. SAG, and Ball Mills, Concentrate, and Tailings Filters, Thickeners, etc.)	Lump Sum	Supplier Type 1	EPCM contractor
Site Communications	Lump Sum	Supplier Type 2	BMC
Port Facilities	Lump Sum	EPC Contractor No. 9	BMC
Mining Services	Schedule of Rates	Mining Contractor	BMC
Explosive Supply	Fixed and Variable	Explosive Services Contractor	BMC
Concentrate Transport Services	Schedule of Rates	Transport Contractor	BMC
BMC Mobile Equipment	Lump Sump	Equipment Vendor	BMC

For tendering of the construction contracts (including but not limited to civil earthworks, structural, mechanical, piping, electrical, and instrumentation), three primary methods have been considered, namely:

- Horizontal Package Contract, in which separate contracts are let by major construction discipline such as earthworks, concrete works, steelwork, piping or electrical installation
- Vertical Integration Contract, in which contracts are let for the construction of a complete building or facility
- Direct hire of equipment and labour and purchase of materials at agreed rates.

Other types of contract (e.g. Schedule of Rates, or Day Works hire) will only be used where the fixed lump sum contract is not practical. Due to the nature or timing of some of the work, a schedule of rates may be necessary for some construction works. This work will be supervised by BMC Project Management personnel familiar with the standards and work quality required. Rates will be established by competitive tender.

A Schedule of Rates style contract is proposed for the mining related works. This will provide BMC management the flexibility to optimize the mine plan within a range agreed with the contractor.

Standard documentation for all agreements will be drafted to streamline the tendering process. A Request for Approval (RFA) for each contract will be submitted to BMC, who will execute the contracts, which will then be managed by the appropriate PDT member. The duration of the procurement process will be scope dependent; however, it is expected that the use of standard form contracts and active participation by BMC executive during the construction phase in regard to review and acceptance of RFAs will expedite the process.

24.1.7 Construction Labour Requirement

The construction contracting strategy and DFS cost estimate were based on "open shop" construction works. This approach takes advantage of the vast pool of skills available from all types of union and non-union shops and allows for the use of local labour sources as well as contractors from anywhere in Canada. The Schedule is based on an 84-hour work week with rostering scheduled as three weeks on-site and one week off-site. During construction the site will generally operate one shift per day; however, a period of double shift will be required during peak construction activities.

An estimated 1,600,000 man-hours of direct and indirect on-site construction labour will be required for project construction, excluding mine pre-development and engineering. Construction manpower on site is estimated to peak at around 650 personnel on site, including construction workers and support staff.

24.1.8 Camp

The accommodation camp will be developed in two phases. The first phase will commission 96 rooms with associated messing. The second phase will complete the build out to a total of 348 rooms with expanded messing and recreation facilities.

The existing 40-bed exploration camp will be used during the early works period whilst the first phase of the permanent camp is commissioned. The exploration camp will be demobilized after this.

The camp will be operated on a motel-style basis with onsite storage provided for the workforce during rostered R&R. During peak construction, the workforce is expected to increase to around 650 persons and bunking will be used to accommodate the increase.

24.1.9 Mine Development

Given the proximity of the mineralization to surface, the ABM open pit requires limited pre-strip and does not need to be developed until nine months prior to ore commissioning. The pit will initially provide borrow material for the construction of the waste and water storage facilities. A stockpile of ore will also be mined for commissioning and further areas developed to ensure continuous feed for the Process Plant as operations begin to ramp up.

A Class A Water Licence will be required before draining Geona Creek. A number of dewatering trenches will be constructed across the valley floor by the mining contractor, at the same time as the ABM pit borrow areas are developed. This will allow the creek and overburden to be drained so the ABM pit can be expanded to the eastern embankment early in the operational phase without incurring trafficability issues.

24.1.10 Port Facilities

Given the location of the port, construction will be carried out under a separate EPCM arrangement. A small on-site team (one to two persons) will be based on site at the Stewart World Port facility to oversee the construction of the concentrate storage shed as well as interfacing of the conveyer system required for ship loading.

The construction of the facility will be outsourced under a single EPC work package including the truck tipple facility which is considered a specialist installation and may require a separate vendor work package. Typical

EPCM support services, such as survey set-out and quality control and testing will be outsourced and provided on an as required basis.

24.1.11 Housekeeping and Hazardous Waste Management

Specific procedures will be implemented for waste management and spill response during the construction period. These procedures will be defined in the Construction Management Plan and Environmental Management Plan. Procedures will be established regarding ongoing clean-up and rubbish removal as well as the safe handling, storage, and disposal of batteries, fuels, oil, and hazardous materials during the construction phase. Recycling programs will be implemented where feasible. Ongoing dust suppression and water management programs will also be established and observed for the duration of the construction phase. Specific procedures and storage areas will be designated for construction waste prior to recycling or removal from the site. Solid waste will be recycled or disposed off-site.

24.1.12 Construction Equipment

Construction equipment will be supplied by each contractor as part of the work package. All mobile or lifting equipment brought to site by a contractor will be inspected by the PDT representative for compliance prior to authorizing its use.

Large construction cranes will be provided by the PDT to improve safety and efficiency, avoid congestion and reduce costs.

The EPCM consultant will provide temporary office and ablution facilities during the construction phase. All other facilities will be provided by BMC and installed permanently where practical.

24.1.13 Communication

For the first 15 months of the construction period the site will operate using satellite links to global telephone and data networks. The permanent microwave link to the existing network will be commissioned by the end of this period when civil, structural, mechanic, piping and electrical works commence. Communications on site will be via UHF radio until fixed lines are installed.

24.1.14 Construction Power

The permanent power station will not be commissioned until near the end of the construction phase when it will be required for plant commissioning. Power for the construction work will therefore be reliant on temporary diesel generators. The camp will have a separate permanent generator which will be installed with the Phase 1 development of the camp facility.

24.1.15 Commissioning

Planning meetings will be held at least three months before commissioning to outline detailed requirements and coordination aspects with BMC personnel, the handover of responsibility for control of the plant and the new interfaces between the stakeholder and BMC operations.

Supplier assistance for the commissioning of new plant will also be reviewed and planned. It is anticipated there will be at least one supplier representative for each major package required for installation, pre-commissioning, and commissioning activities.

Personnel from the stakeholders' technical and construction crews will carry out the system commissioning under the control of the Commissioning Manager. BMC's operations and maintenance personnel will be incorporated into the commissioning team for training purposes, and to develop BMC's Operational Readiness Plan to enable a seamless handover to operations.

In summary, commissioning will consist of the following phases:

- C1 – Construction Verification
- C2 – Functional Testing
- C3 – No-Load Commissioning
- C4 – Load Commissioning.

The following records will be maintained by Document Control for the duration of the project and will form part of the handover documentation:

- Design input data
- Approved design calculations
- Approved data sheets
- As built design drawings
- Specifications
- Vendor data
- Spare parts listings
- Quality records
- Site records
- Other design output as required by the contract
- Technical queries
- Engineering change requests.

24.1.16 Production Ramp-Up

The commencement of ore processing follows no-load wet commissioning. This is considered to be the start of the operational phase of the project. It is scheduled to occur in December 2021 following 13 months of site construction for the Process Plant and non-process infrastructure.

Provision was made for a production ramp-up in the DFS. The ramp-up schedule was prepared by a commissioning consultant. The Process Plant will reach "nameplate" throughput capacity within nine months of commissioning, target recoveries within six to nine months of commissioning depending on flotation circuit, and target concentrate grades within four to seven months of commissioning depending on concentrate product.

24.2 Risk Management

A Risk Assessment Workshop was conducted during the DFS and a Risk Register developed (CSA Global, 2019b) which identified the following number of high-, medium- and low-level risks after mitigation measures:

- Five high-level risks
- 42 medium-level risks
- 115 low-level risks.

Most risks identified could be mitigated to manageable levels; however, of the five remaining high risks, realizing full value of the concentrates was predominantly the theme of the remaining elevated risks. This was either through changes in off-take terms, changes to import restrictions, increased sea freight charges or the ability to meet agreed specifications. Whilst the expected concentrate qualities are considered complex, assessments concluded that they should be able to be produced to the target specification and find a buyer, either as a direct sale to a smelter or through an established trading house. BMC propose to mitigate this risk during the project development phase through securing of sales agreements ahead of production.

With regard to sea freight charges, volatility is expected and provisions within the study are considered to be above market longterm median pricing.

The fifth remaining high risk was in relation to the Upper and Lower Water Management Pond embankments and reflects the importance of these structures to the operation. It should be noted that these structures are not considered to be an elevated risk; rather, the rating reflects the conservatism and limitations of the assessment criteria, insofar as the "Likelihood" was considered to be at the low end of the "rare" classification (significantly less than 10% chance of occuring threshold for a "rare" classification) and no more appropriate criteria was available to apply.

Environmental risks were not assessed independently under this process. An environmental risk assessment was undertaken by BMC as part of the permitting process.

25 Interpretation and Conclusions

The KZK Property is located in a region known to contain significant VHMS deposits. The KZK Project comprises a wide range of base metal exploration targets from near grass-roots geochemical anomalies, conceptual geological and geophysical targets to drill-ready targets (i.e. GP4F, Fault Creek Zone, Northwest ABM, Krakatoa extensions) and advanced-stage targets consisting of Inferred and Indicated Mineral Resources (ABM and Krakatoa zones). In addition to this, large sections of ground within the KZK Property remain under-explored.

The 2015 exploration program undertaken by BMC at the ABM deposit was an outstanding success, highlighted by the discovery of the Krakatoa VHMS Zone in the fault offset block southeast of the ABM Zone. This discovery, coupled with additional mineralization identified in extension drilling at the ABM Zone and confirmation of historical results, resulted in a significant (47%) increase in reported tonnage for the deposit. Additional drilling during the 2016 field season, particularly at the Krakatoa Zone, resulted in an improved understanding of the controls on mineralization despite the slight reduction in reported tonnage. The increased drilling information and improved confidence allowed 18.3 Mt or 96% of the ABM Zone Mineral Resource to be classified in the Indicated category.

The responsible QP considers that data collection techniques are consistent with industry good practice and suitable for use in the preparation of MREs to be reported in accordance with the CIM Definition Standards on Mineral Resources and Mineral Reserves. QC data supports the integrity of the analytical data which has been utilized.

High-quality diamond core samples from the ABM deposit were used to interpolate grades and bulk density into blocks using OK. The block models were validated visually and statistically.

The ABM deposit remains open (ie. Krakatoa down-dip extent) and additional drilling is required to fully define the extents of mineralization.

The KZK Project presents a viable development scenario of mining the ABM deposit primarily by open pit mining methods, with a smaller underground mine incorporated to mine the deeper section of the Krakatoa Zone. Mining contractors are expected to be engaged to undertake all mining works, under the direction of BMC.

The open pit design includes mining of approximately 110,000 tonnes of Inferred Resource, that presents an opportunity to add to reserves if resource confidence can be improved with future work. Similarly, inferred mineralization at depth in the Krakatoa Zone presents an opportunity to extend the life of the underground mine with additional work to improve confidence in the resource.

Metallurgical testwork has proved that the Kudz Ze Kayah ore can be processed using a conventional wet grinding circuit followed by sequential flotation to produce marketable concentrates of copper, lead and zinc. The concentrates will also contain significant precious metal credits.

Grade control drilling will be a key requirement for improving the confidence and understanding of the distribution or economic and deleterious element in the ore. Blending of metallurgical domains to feed into the Process Plant will be adopted to optimize Process Plant performance and concentrate quality.

Waste rock will be stored in separate facilities according to expected acid generation and metal leaching potential, with long term closure planning considered from the outset. Tailings from ore processing will be produced as a filtered tailing product that will be deposited in the Class A Waste Storage Facility together with Class A waste rock.

A composite basin liner system incorporating an HDPE geomembrane liner will be constructed at the base of all storage facilities that have the potential for acid generation and metal leaching characteristics. Progressive reclamation will be implemented to cover waste storage facilities as they are developed to minimize exposure to oxygen and water as well as promote active revegetation of the facilities.

All infrastructure has been designed to be situated within a single watershed to minimize impacts on the broader environment. Water that does not come into contact with the KZK Project footprint will be diverted around the site for discharge. Contact water not requiring chemical treatment for discharge will be kept separate from water that does in order to minimize chemical treatment requirements. Reuse of water within the mining and processing facilities will be prioritized to limit the amount of water that will require treatment prior to discharge from the site. A WTP will be constructed to treat water to meet site discharge quality limits.

Concentrates will be hauled to the Port of Stewart, BC for shipping to market. The Portland Waterway remains ice-free all year round, and enables Stewart's position as Canada's most northerly ice-free port. The DFS proposes that all concentrates will be sold into the East Asian region at generally standard commercial arrangements for sale of concentrates.

Through a process of collaboration, the Kudz Ze Kayah DFS study team identified 163 risks associated with the project. The vast majority of these risks could be mitigated to Moderate or Low risks, leaving only five risks remaining as Adjusted Risks in the High category, and no Adjusted Risk classified as Extreme. Four of the high-level risks relate to concentrate transport and sales and were considered typical levels for a project of this nature and which are normally mitigated during the project development phase through securing sales agreements. The fifth high-level risk, related to the Upper and Lower Water Management Ponds was not in reality, considered an elevated risk; rather, its rating was an artefact of the conservatism and limitations of the classification system.

The pre-production capital cost to develop the KZK Project is estimated to be CAD$496 million, including a contingency of CAD$47 million. The capital cost estimate has been prepared to within the normal limits expected for a Feasibility study as defined in Canadian Institute of Mining and Metallurgy and Petroleum ("CIM") Definition Standards for Mineral Resources & Mineral Reserves. The costs are considered current as of Q4 2018. Sustaining capital costs are estimated to be CAD$162 million and closure costs CAD$102 million.

The average operating cost over the life of the project is estimated to be CAD$184 per tonne[5] of ore processed.

The time required to develop the KZK Project was estimated to be 31 months from the commencement of initial engineering works. On-site construction of the KZK Project was estimated to be completed within approximately 13 months. This will be followed by a 9 month ramp up to full production.

The KZK Project, as described in this Technical Report is environmentally and technically feasible, delivering a positive case on which the project can move forward. The project presents a viable development scenario for open pit mining of the ABM Zone and upper Krakatoa Zone and underground mining the lower portion of the Krakatoa Zone. Mining will be completed within an 8.6-year period.

[5] Includes all site operating costs, concentrate transportation to smelters, concentrate treatment, refining and penalty charges, government royalties and First Nations payments.

26 Recommendations

Based on the findings of this Feasibility Study, it is recommended that BMC progress the KZK Project to detailed engineering and construction. These costs are detailed in Section 21.1 and the initial costs (inclusive of contingency) prior to commencing construction activities on site in April 2020 are summarized in Table 26-1.

Table 26-1: Initial costs of project development

Description	Total (CAD$M)
Process Plant	$1.8
Waste Storage Facilities	$0.4
Water Storage and Management Facilities	$0.4
Water Treatment Plant	$0.4
Accommodation Camp	$9.9
Owners Costs	$0.5
Engineering and Procurement	$6.9
Other Indirect Costs	$0.3
Total Initial Cost of Project Development (CAD$M)	**$20.7**

During the course of the study, a number of recommendations were made. The following key items were identified as having the potential to further improve the economics of the KZK Project and/or reduce risk to project development and should be pursued as part of the detailed engineering:

- Commence tender process for EPCM contractor to facilitate award of contract as soon as practicable following project sanction. Consideration should be given to the benefits of early contractor involvement.
- Commence negotiations with identified customers for the sale of Quality A concentrates with the view to establishing binding off take agreements as soon as practicable.
- In order for the KZK Project to progress to higher Mineral Resource classification levels (Measured and Indicated), further infill exploration drilling, or grade control drilling will be required. This drilling should also be planned to improve confidence in the prediction of Class A, B and C waste material classifications.
- Include in the mineral resource model modelling of absent potential revenue and penalty elements (Bi, Cd, Cr, F, Cl) to assist blending strategies to minimize penalizable elements in concentrate, where sufficient data exists to reasonably estimate these elements.
- Undertake detailed short term planning and scheduling consistent with the open pit mine designs to reduce haul distances for waste stripping of the western and eastern valley walls.
- Additional geotechnical drilling be undertaken into the west wall of the ABM Zone pit to investigate the possible locations of additional north-northeast trending features and confirm design parameters for the final west wall of the pit.
- A geotechnical drilling program should be completed to support detailed design for the underground mine, prior to commencement of underground mining operations. Given that the underground mine commences approximately 3.5 years after open pit mining operations commence, this work can be completed after the commencement of open pit mining.
- 3D elasto-plastic numerical modelling should be completed to support detailed underground stope design verification and stability assessments during implementation of the underground mine and continued into the production phase.

- Routinely update the hydrogeologcial model and conduct sensitivity testing to determine the impact of a range of results spanning the uncertainty associated with specific parameters and model configurations as new data becomes available.
- Complete additional paste backfill testwork. Given the lead time on the development of the underground mine, this work does not need to commence until the processing plant is operational and representative plant tailings are available for testwork purposes.
- Additional assays of existing variability flotation concentrates for minor elements (Bi, Cd, Hg, Se) are recommended to improve the dataset for predicting concentrate qualities.
- Investigate opportunities to increase gold content of copper and lead concentrates by reducing copper concentrate grades below 25%.
- Final process plant foundation designs to be confirmed against geotechnical data and water balance models to ensure they are fit for purpose prior to construction.
- Confirm location and design of construction material borrow sources, including locally sourced construction aggregates, to avoid importation costs.
- Undertake dynamic simulation of tailings filtration to confirm its flexibility to attend to the inherent variation in capacity, and availability between it and the upstream process units to ensure materials handling systems (i.e. feed tanks, dry storage, reject pond, pumping systems etc.) are designed with the required margin.
- Complete current laboratory treatability testwork for the WTP processes to improve confidence in meeting discharge quality targets.
- Assess the feasibility of self-performing the design, supply, fabrication, and erection of the Process Building in lieu of engaging a supplier to reduce capital costs and critical path dependencies.
- Re-assess the used equipment marketplace at the time of detailed engineering design to ascertain the availability of equipment that would be suitable for the duty performance in order to reduce pre-production costs.

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28 Certificates

Certificate of Qualified Person – Karl van Olden, FAusIMM (CSA Global)

CSA Global
Mining Industry Consultants

CERTIFICATE OF AUTHOR

I, Karl van Olden, BSc(Eng)., do hereby certify that:

1. I am currently employed as Principal Mining Engineer with CSA Global Pty Ltd with an office at Level 2, 3 Ord Street West Perth, WA 6005.
2. This certificate applies to the technical report titled "NI 43-101 Feasibility Study Technical Report for the Kudz Ze Kayah Project, Yukon Territory, Canada", with an Effective Date of June 30 2019, (the "Technical Report") prepared for BMC Minerals (No. 1) LTD. ("the Issuer");
3. I am a professional mining engineer registered as a Fellow member with the Australasian Institute of Mining and Metallurgy (AusIMM) (Membership No 226473).

   I am a graduate from the University of the Witwatersrand, Johannesburg (1993) and have a Graduate Diploma in Engineering (Mineral Economics) (1995) and a Masters in Business Administration from Latrobe University (2005). I have been involved or associated with the mining industry sincen 1994, in South Africa and Australia in production roles for 16 years and 9 years in consulting. I joined CSA Global in 2015, and am currently the Manager for the Mining Team.

   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.
4. I have visited the Kudz Ze Kayah Project site on 15 to 18 August 2016 for a period of four days.
5. I am responsible for Section numbers 1-3, 15, 16.1-16.3, 16.4.2-16.4.7, 16.5.1, 16.5.3-16.5.18, 18.7, 19, 20.1, 20.2.1, 20.2.2, 20.2.7-20.2.10,20.3, 21, 22, 23, 25-29 of the 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 no prior involvement with the property that is the subject of the Technical Report other than to contribute to the 2017 Pre-feasibility Study;
8. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1;
9. As of the effective date of the Technical Report and the date of this certificate, 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;

Effective Date: June 30, 2019

Signing Date: August 16, 2019

Original document signed by Karl van Olden, BSC(Eng), GDE, MBA, FAusIMM.

Certificate of Qualified Person – Aaron Green, MAIG (CSA Global)

CSA Global
Mining Industry Consultants

CERTIFICATE OF AUTHOR

I, Aaron Green, MAIG., do hereby certify that:

- I am currently employed as Director – Australasian Operations and Principal Resource Geologist with CSA Global Pty Ltd with an office at Level 2, 3 Ord St West Perth, WA 6005.
- This certificate applies to the technical report titled "NI 43-101 Feasibility Study Technical Report for the Kudz Ze Kayah Project, Yukon Territory, Canada", with an Effective Date of June 30 2019, (the "Technical Report") prepared for BMC Minerals (No. 1) LTD. ("the Issuer");
- I am a Professional Geologist (Member) registered with the Australian Institute of Geoscientists (Membership No. 1719).
- I am a graduate of La Trobe University, completing a BSc (Hons) in 1993. I have been involved or associated with the mining industry since 1994. I also have a Graduate Diploma in Applied Finance and Investment (Securities Institute of Australia, 2003). Having worked for almost 10 years in the Western Australian goldfields in both exploration and underground production roles, I moved into consulting in 2003 focussed on resource estimation and evaluation. In 2013, I joined CSA Global Pty Ltd as Australian Operations Manager and became a Director in 2014.
- 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 43101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
- I visited the Kudz Ze Kayah Project site on 11-13 October 2015 for a period of 2 days and subsequently on 26 July 2017 for 1 day.
- I am responsible for Section numbers 4-12 and 14 of the Technical Report;
- I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;
- I have had no prior involvement with the property that is the subject of the Technical Report other than to contribute to the 2017 Pre-feasibility Study in the same manner as the contribution to this update and in the preparation of several resource estimates since acquisition of the project by BMC in 2015.
- I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1;
- As of the effective date of the Technical Report and the date of this certificate, 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;

Effective Date: June 30, 2019
Signing Date: 16 August 2019
Original document signed by Aaron Green, MAIG

Certificate of Qualified Person – Geoff Davidson, FAusIMM (CSA Global)

CSA Global
Mining Industry Consultants

CERTIFICATE OF AUTHOR

I, Geoff Davidson, B.Eng(mining)., do hereby certify that:

- I am currently employed as Associate Principal Consultant with CSA Global Pty Ltd with an office at Level 2, 3 Ord Street West Perth, WA 6005.
- This certificate applies to the technical report titled "NI 43-101 Feasibility Study Technical Report for the Kudz Ze Kayah Project, Yukon Territory, Canada", with an Effective Date of June 30 2019, (the "Technical Report") prepared for BMC Minerals (No. 1) LTD. ("the Issuer");
- I am a Professional Mining Engineer registered as a Fellow member with the Australasian Institute of Mining and Metallurgy (AusIMM) (Membership No 112 127). I am a member of the Society for Mining, Metallurgy & Exploration (SME) (Membership No. 04133819).
- I am a graduate of Curtin University (WASM) . I have been involved or associated with the mining industry since 1989. I also received a Graduate Certificate in mineral economics from Curtin University. I have worked in both operational and technical roles in both open pit and underground mining. I was employed by a major mining contractor for approximately 9 years where I advanced to the position of Estimating Manager for open pit, underground and civil excavation contracts. I worked for a major consulting firm for approximately 5 years where I advanced to the position of Principal Consultant and lead feasibility studies for open pit and underground projects in Australia and overseas. I currently work as an Associate Principal Consultant for CSA Global providing technical advice and assistance on mine feasibility studies.
- 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 43101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
- I have visited the Kudz Ze Kayah Project site on 15 October 2017 and was present on site for a period of one day;
- I am responsible for Section number 24 of the Technical Report;
- I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;
- I have had no prior involvement with the property that is the subject of the Technical Report;
- I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1;
- As of the effective date of the Technical Report and the date of this certificate, 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;

Effective Date: June 30, 2019
Signing Date: August 16, 2019
Original document signed by Geoff Davidson, B.Eng(Mining), FAusIMM

Certificate of Qualified Person – John Fleay, FAusIMM (Minnovo)

CERTIFICATE OF AUTHOR

I, John Fleay B.Eng(Mineral Processing), do hereby certify that:

1. I am currently employed as Manager Metallurgy with Minnovo Pty Ltd. with an office at 256 Adelaide Terrace, Perth, 6000.
2. This certificate applies to the technical report titled "NI 43-101 Feasibility Study Technical Report for the Kudz Ze Kayah Project, Yukon Territory, Canada", with an Effective Date of June 30 2019, (the "Technical Report") prepared for BMC Minerals (No. 1) LTD. ("the Issuer").
3. My technical qualifications are Batchelor of Engineering (Mineral Processing) and I am a Fellow member of the Australian Institute of Metallurgy (AusIMM No:320872).

   I am a graduate of WA School Of Mines (WASM). I have appropriate experience in these matters, by way of my qualifications and 25 years of experience in the mining and resource sector.

   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.
4. I have not visited the Kudz Ze Kayah Project site.
5. I am responsible for Section numbers 13 and 17 of the 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 no prior involvement with the property that is the subject of the Technical Report other than to contribute to the 2017 Pre-feasibility Study in the same manner as the contribution to this update.
8. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1;
9. As of the effective date of the Technical Report and the date of this certificate, 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.

Effective Date: June 30, 2019
Signing Date: August 16, 2019

Original document signed by John Fleay B.Eng(Mineral Processing)

Certificate of Qualified Person – Les Galbraith, P.Eng (Knight Piésold)

Knight Piésold CONSULTING

CERTIFICATE OF AUTHOR

I, Les Galbraith, P.Eng., do hereby certify that:

1. I am currently employed as Specialist Engineer - Associate with Knight Piésold Ltd. with an office at Suite 1400 – 750 West Pender Street Vancouver, British Columbia, Canada, V6C 2T8.
2. This certificate applies to the technical report titled "NI 43-101 Feasibility Study Technical Report for the Kudz Ze Kayah Project, Yukon Territory, Canada", with an Effective Date of June 30 2019, (the "Technical Report") prepared for BMC Minerals (No. 1) LTD. ("the Issuer");
3. I am a Professional Engineer registered with the Association of Professional Engineers of Yukon. Membership NO 2368.
4. I am a graduate of the University of British Columbia. I have been involved or associated with the mining industry since 1996. My experience includes tailings and water dam design, geotechnical investigations, construction supervision, stability modelling, and foundation assessments.
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 Kudz Ze Kayah Project site (May 9-12, 2016);
7. I am responsible for Section numbers 18.1 and 18.9 of the Technical Report;
8. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;
9. I have had no prior involvement with the property that is the subject of the Technical Report other than to contribute to the 2017 Pre-feasibility Study in the same manner as the contribution to this update];
10. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1;
11. As of the effective date of the Technical Report and the date of this certificate, 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;

Effective Date: June 30, 2019
Signing Date: August 19, 2019

Original document signed and sealed by Les Galbraith, P.Eng.

Certificate of Qualified Person – Jaimie Cathcart, P.Eng (Knight Piésold)

Knight Piésold CONSULTING

CERTIFICATE OF AUTHOR

I, Jaime Cathcart, Ph.D., P.Eng., do hereby certify that:

1. I am currently employed as Specialist Hydrotechnical Engineer with Knight Piésold Ltd. with an office at 1400 – 750 West Pender Street, Vancouver, BC;
2. This certificate applies to the technical report titled "NI 43-101 Feasibility Study Technical Report for the Kudz Ze Kayah Project, Yukon Territory, Canada", with an Effective Date of June 30 2019, (the "Technical Report") prepared for BMC Minerals (No. 1) LTD. ("the Issuer");
3. I am a Professional Engineer registered with the Yukon Territory and British Columbia;
4. I am a graduate of the University of British Columbia. I have been involved or associated with the mining industry since 1993. I have a bachelor's degree in civil engineering (1987), a master's degree in water resource engineering (1993), and a Ph.D. in Hydrology (2001), all from UBC. I have practiced for over 25 years as a consulting engineer in the mining and hydroelectric sectors and am responsible for overseeing all hydrologic work in the KP Vancouver office. In addition, I am involved with water management and hydraulic design studies for mining, water supply and hydroelectric projects;
5. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
6. I have not visited the Kudz Ze Kayah Project site;
7. I am responsible for Section number 18.2 of the Technical Report;
8. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;
9. I have provided senior technical review related to hydrotechnical aspects of the project since 2016;
10. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1;
11. As of the effective date of the Technical Report and the date of this certificate, 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;

Effective Date: June 30, 2019
Signing Date: August 22, 2019

Original document signed and sealed by Jaime Cathcart, P.Eng.

Certificate of Qualified Person – Paul Hughes, P.Eng (Dempers and Seymour)

Dempers & Seymour Pty Ltd
Geotechnical and Mining Consultants

CERTIFICATE OF AUTHOR

I, Paul Hughes, P.Eng, Ph.D., do hereby certify that:

1. I am currently employed as Senior Geotechnical Engineer with Dempers and Seymour Pty Ltd. with an office at Suite 14 & 15 Aspire Office Park, 231 Balcatta WA 6021 Australia.
2. This certificate applies to the technical report titled "NI 43-101 Feasibility Study Technical Report for the Kudz Ze Kayah Project, Yukon Territory, Canada", with an Effective Date of June 30 2019, (the "Technical Report") prepared for BMC Minerals (No. 1) LTD. ("the Issuer");
3. I am a Registered Professional Engineer in good standing with the Association of Professional Engineers and Geoscientists Yukon Territory (Reg # 2657). I am a Registered Professional Engineer in good standing with Engineers and Geoscientists British Columbia (Reg # 36997).

   I am a graduate of the University of British Columbia in Vancouver, British Columbia having been conferred the degree Bachelor of Applied Science degree in Geological Engineering in 2004; Masters of Applied Science degree in Mining Engineering in 2008; and a Doctor of Philosophy in Mining Engineering in 2014. I have worked as an engineer continuously for 10 years in the areas of underground mining, rock mechanics, geotechnical engineering and consulting.
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.
5. I have visited the Kudz Ze Kayah Project site between July 28, 2017 and July 30, 2017 and between April 25, 2018 to April 29, 2018 and was present on site for a total period of 8 days.
6. I am responsible for Section numbers 16.4.1 and 16.5.2 of the 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 had no prior involvement with the property that is the subject of the Technical Report other than to contribute to the 2017 Pre-feasibility Study in the same manner as the contribution to this update;
9. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1;
10. As of the effective date of the Technical Report and the date of this certificate, 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;

Effective Date: June 30, 2019

Signing Date: August 15, 2019

Original document signed by Paul Hughes, P.Eng, Ph.D.

Certificate of Qualified Person – AJ MacDonald, P.Eng (Integrated Sustainability)

INTEGRATED SUSTAINABILITY

CERTIFICATE OF AUTHOR

I, AJ MacDonald, P.Eng., do hereby certify that:

1. I am currently employed as Vice President / Senior Technical Specialist with Integrated Sustainability Consultants Ltd with an office at 1050 West Pender Street, Vancouver BC, V6C 3S7
2. This certificate applies to the technical report titled "NI 43-101 Feasibility Study Technical Report for the Kudz Ze Kayah Project, Yukon Territory, Canada", with an Effective Date of June 30 2019, (the "Technical Report") prepared for BMC Minerals (No. 1) LTD. ("the Issuer");
3. I am a Professional Engineer registered with Association of Professional Engineers Yukon. I am also a professional engineer registered in the provinces of British Columbia, Alberta and Ontario

   I am a graduate of Queen's University, Kingston, Ontario and Carleton University, Ottawa, Ontario. I have been involved or associated with the mining industry since 2007. I have participated in dozens of mining and other resource sector projects, with a particular focus on water treatment, primarily in Western Canada. My experience spans all phases of project delivery including preliminary analysis, conceptual design, detailed design, construction, commissioning and optimization of infrastructure at industrial water treatment facilities in Canada and around the world.

   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.
4. I have not visited the Kudz Ze Kayah Project site;
5. I am responsible for Section numbers 18.3 of the 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 no prior involvement with the property that is the subject of the Technical Report;
8. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1;
9. As of the effective date of the Technical Report and the date of this certificate, 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;

Effective Date: June 30, 2019
Signing Date: August 15, 2019

Original document signed and sealed by AJ MacDonald, P.Eng.

Certificate of Qualified Person – Grant Morgan, P.Eng (Allnorth)

CERTIFICATE OF AUTHOR

I, Grant Morgan, P.Eng., do hereby certify that:

12. I am currently employed as Project Management Group Lead with Allnorth Consultants Ltd. with an office at 1200 - 1100 Melville Street, Vancouver BC V6E 4A6.
13. This certificate applies to the technical report titled "NI 43-101 Feasibility Study Technical Report for the Kudz Ze Kayah Project, Yukon Territory, Canada", with an Effective Date of June 30 2019, (the "Technical Report") prepared for BMC Minerals (No. 1) LTD. ("the Issuer");
14. I am a Professional Engineer registered with The Association of Professional Engineers and Geoscientists of the province of BC (Registration No. 17,382) and The Association of Professional Engineers and Geoscientists of Alberta (Registration No. 95606).

    I am a graduate of Queen's University at Kingston with a Bachelor of Science in Engineering, and received a Master of Applied Science degree from the University of BC. I have been involved or associated with the mining industry since1987. I have worked in research, consulting, and project management; involved with several different mining properties encompassing multiple different ore and concentrate materials.

    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.
15. I have not visited the Kudz Ze Kayah Project site.
16. I am responsible for Section numbers 18.4, 18.5, 18.6, 18.8, 18.11, 18.12, 18.14 and 18.15 of the Technical Report;
17. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;
18. I have had no prior involvement with the property that is the subject of the Technical Report;
19. I have read NI 43-101, and the DRAFT Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1;
20. As of the effective date of the Technical Report and the date of this certificate, 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;

Effective Date: June 30, 2019
Signing Date: August 15, 2019

Original document signed and sealed by Grant Morgan, P.Eng.

Certificate of Qualified Person – Bader Diab, P.E. (AqualisBraemar)

AqualisBraemar Technical Services Ltd.
5th floor, 6 Bevis Marks
London EC3A 7BA
United Kingdom

T +44(0) 203 142 4300

CERTIFICATE OF AUTHOR

I, Dr Badreddin (Bader) Diab, P.E., do hereby certify that:

1. I am currently employed as Regional Director, Americas with Aqualis Offshore, Inc. (Braemar Technical Services was acquired by Aqualis Offshore on 21 June 2019, forming AqualisBraemar). Our local office address is 2800 North Loop West, Suite 900, Houston, Texas 77092, USA..

2. This certificate applies to the technical report titled "NI 43-101 Feasibility Study Technical Report for the Kudz Ze Kayah Project, Yukon Territory, Canada", with an Effective Date of June 30 2019, (the "Technical Report") prepared for BMC Minerals (No. 1) LTD. ("the Issuer");

3. I am a Professional Engineer (P.E.) registered in the states of Texas (registration number 100416) and Alaska (registration number 132477).

   I am a qualified Structural Engineer with Bachelors, Masters and Doctorate qualifications, and a registered Professional Engineer in two US states. I have worked in the United Kingdom, the United Arab Emirates and the United States across numerous fields of structural and naval architectural disciplines. I have marine terminals and jetty structures experience as a project engineer, lead engineer and project manager for terminal integrity assessments, expansion feasibility studies and upgrade/life extension of existing facilities. Additionally, I have extensive offshore engineering experience in consulting, transportation and installation, jack-ups, semi-submersibles and floating production units including acting technical lead and project manager on numerous large-scale energy sector projects in Canada, the Gulf of Mexico, United Arab Emirates and Qatar. I have also authored and co-authored eight technical publications and authored the Installation Engineering chapter of the Handbook of Offshore Engineering".

   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.

4. I have not visited the Kudz Ze Kayah Project port facilities site at Stewart; however, I have been briefed on the site to my satisfaction, by Mr Brocque Preece C.Eng, who holds the position of Naval Architect with AqualisBraemar Technical Services Ltd and who attended the site on April 4th, 2018 and was present on site for a period of 2 days;

5. I am responsible for Section number 18.13 of the 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 no prior involvement with the property that is the subject of the Technical Report other than my contribution therein;

8. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1;

9. As of the effective date of the Technical Report and the date of this certificate, 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;

Effective Date: June 30, 2019
Signing Date: August 15, 2019

Original document signed and sealed by Dr Bader Diab, P.E.

Certificate of Qualified Person – Guy Roemer, P.E. (Tetra Tech)

TETRA TECH

CERTIFICATE OF AUTHOR

I, Guy Roemer, P.E. (Colorado, U.S.), do hereby certify that:

1. I am currently employed as an Associate Environmental Engineer with Tetra Tech, Inc. with an office at 1100 South McCaslin Blvd, Superior, Colorado
2. This certificate applies to the technical report titled "NI 43-101 Feasibility Study Technical Report for the Kudz Ze Kayah Project, Yukon Territory, Canada", with an Effective Date of June 30 2019, (the "Technical Report") prepared for BMC Minerals (No. 1) LTD. ("the Issuer");
3. I am a Professional Engineer registered with the State of Colorado (#36810) since July 2001.

   I am a graduate of Texas A&M University in College Station, Texas (1995) with a Bachelor's degree in Nuclear Engineering. I am also a graduate of the University of New Mexico (1997) with a Master's degree in Nuclear Engineering. I have been involved or associated with the mining industry since 1999. Besides this project, I have developed or reviewed groundwater flow and transport models for more than twenty mine sites in Australia, Canada, and U.S. I have also developed or reviewed eight water balance models for mine sites in Asia, Australia, Canada, and the U.S. I have also served as a qualified person on three other NI 43-101 reports.

   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.
4. I have not visited the Kudz Ze Kayah Project site;
5. I am responsible for Section numbers 20.2.4 of the 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 no prior involvement with the property that is the subject of the Technical Report other than to contribute to the 2017 Pre-feasibility Study in the same manner as the contribution to this update;
8. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1;
9. As of the effective date of the Technical Report and the date of this certificate, 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;

Effective Date: June 30, 2019
Signing Date: August 15, 2019

Original document signed by Guy Roemer, P.E.

Certificate of Qualified Person – Cheibany Elemine, P.Geo (Alexco Environmental Group)

 # 3 Calcite Business Centre, 151 Industrial Road
Whitehorse, Yukon Y1A 2V3
Phone (867) 668-6463 Fax (867) 633-4882
www.alexcoenv.com

CERTIFICATE OF AUTHOR

I, Cheibany Ould Elemine, P.Geo., PhD., do hereby certify that:

1. I am currently employed as Senior Geochemist with Alexco Environmental Group Inc. with an office at #3 Calcite Business Centre, 151 Industrial Rd., Whitehorse, YT Y1A 2V3
2. This certificate applies to the technical report titled "NI 43-101 Feasibility Study Technical Report for the Kudz Ze Kayah Project, Yukon Territory, Canada", with an Effective Date of June 30 2019, (the "Technical Report") prepared for BMC Minerals (No. 1) LTD. ("the Issuer");
3. I am a Professional Geoscientist registered with Engineers and Geoscientists of British Columbia, Membership No: 162640 and with the Association of Professional Engineers and Geoscientists of Alberta, Membership No: 237587.

   I am a graduate of University of Niigata and University of Nouakchott. I have been involved in the mining industry since 2011. I have been environmental scientist and geochemist on several mining projects worldwide. I have been involved in many aspects in those projects including but not limited to: environmental baseline data collection, detailed investigations, water quality studies and prediction, hydrological modeling, geochemical characterization studies, supervision of field investigation and construction, reclamation and closure design.

   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.
4. I have not visited the Kudz Ze Kayah Project site;
5. I am responsible for Sections 20.2.3, 20.2.5, 20.2.6 and 20.4 of the 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 no prior involvement with the property that is the subject of the Technical Report other than to contribute to the 2017 Pre-feasibility Study in the same manner as the contribution to this update];
8. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1;
9. As of the effective date of the Technical Report and the date of this certificate, 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;

Effective Date: June 30, 2019
Signing Date: August 17, 2019]

Original document signed by Cheibany Ould Elemine, P.Geo., PhD

Certificate of Qualified Person – Jeremy Araki, P.Eng (Onsite Engineering)

ONSITE
Engineering Ltd.

CERTIFICATE OF AUTHOR

I, Jeremy Araki, P.Eng., do hereby certify that:

1. I am currently employed as Senior Engineer with Onsite Engineering Ltd. with an office at 1040 Cedar Street, Campbell River, BC, V9W 2C8.
2. This certificate applies to the technical report titled "NI 43-101 Feasibility Study Technical Report for the Kudz Ze Kayah Project, Yukon Territory, Canada", with an Effective Date of June 30 2019, (the "Technical Report") prepared for BMC Minerals (No. 1) LTD. ("the Issuer");
3. I am a Professional Engineer registered with Engineers Yukon, Engineers and Geoscientists BC, and the Association of Professional Engineers and Geoscientists of Alberta.

   I am a graduate of the University of New Brunswick. I have been involved or associated with the mining industry since 2007. I have been involved in surface projects and project access for mining projects in BC, Yukon Territory, and Nunavut. My primary focus has been on mine access but I have worked on a wide variety of surface infrastructure projects including haul roads and bridges, retaining walls, ore chutes, structural assessments, and foundation geotechnical assessments. My remote linear access experience spans several other resource industries including forestry, hydro, and oils and gas.

   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.
4. I have visited the Kudz Ze Kayah Project site on May 26, 2015, and September 12, 2016, and was present on site for a period of 2 days and 5 days, respectively;
5. I am responsible for Section number 18.10 of the 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 no prior involvement with the property that is the subject of the Technical Report other than to contribute to the 2017 Pre-feasibility Study in the same manner as the contribution to this update;
8. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1;
9. As of the effective date of the Technical Report and the date of this certificate, 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;

Effective Date: June 30, 2019
Signing Date: August 19, 2019
Original document signed and sealed by Jeremy Araki, P.Eng.

29 Abbreviations and Glossary

29.1 Abbreviations and Units of Measurement

“	inch
°	degrees
°C	degrees Celsius
/	per (e.g. g/t = grams per tonne)
%w/w	mass fraction
2D	two-dimensional
3D	three-dimensional
AAS	atomic absorption spectroscopy
ABA	acid-base accounting
AEG	Alexco Environmental Group
AI	abrasion index
Allnorth	Allnorth Consultants Ltd
AP	acid potential
APS	azimuth positioning system
ARD	acid rock drainage
bbl	barrel (of oil)
BC	British Columbia
BCMoE	British Columbia Ministry of the Environment
BHEM	bore hole electromagnetic
BMC	BMC Minerals (No.1) Limited
Braemar	Braemar Technical Services LLC
BWI	bond ball mill work index
CAD$ or CAD	Canadian dollars
c.	circa
CCME	Canadian Council of Ministers of the Environment
CEP	Consultation and Engagement Plan
CERL	Cominco Exploration Research Laboratory
Challenger	Challenger Geomatics
CIBC	Canadian Imperial Bank of Commerce
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
cm	centimetre(s)
COPC	constituents of potential concern
COSEWIC	Committee on the Status of Endangered Wildlife in Canada
CRM	certified reference material
CSA Global	CSA Global Pty Ltd
CV	coefficient of variation

$D^{2,3,4}$	structural geology terms describing deformational events – D^1 oldest, $D^{4.}$ Youngest
D&S	Dempers & Seymour Pty Ltd
DD	diamond drilling
DFO	(Federal) Department of Fisheries and Oceans
DFS	definitive feasibility study
dmt	dry metric tonne
DWI	drop weight index
E	easting or east
Ecofor	Ecofor Consulting Ltd
EDTA	ethylene diamine tetra acetic acid
EGL	effective grinding length
EM	electromagnetic
EMIT	Electromagnetic Imaging Technology
EMR	Energy, Mines and Resources Department, Yukon
EPC	engineering, procurement and construction
EPCM	engineering, procurement, construction management
EPLT	equivalent point load testing
Equity Exploration	Equity Exploration Consultants
Expatriate	Expatriate Resources Ltd
EY	Ernst and Young
FCCOG	fully costed cut-off grade
FCH	Finlayson Caribou Herd
FEL	front-end loader
FLTEM	fixed-loop transient electromagnetic
FOB	free on board (INCO term)
FS	feasibility study
FW	footwall
g	gram(s)
G&A	general and administration
GJ	billion joule(s)
GPS	global positioning system
g/cm^3	grams per centimetre cubed
g/t	grams per tonne
h	hour(s)
ha	hectare(s)
Hatch	Hatch Pty Ltd
HDPE	high-density polyethylene
HDS	high-density sludge
HLEM	horizontal loop electromagnetic
HMS	heavy media separation

HSE	health, safety and environmental
HV	high voltage
HVAC	heating, ventilation, and air conditioning system
HW	hangingwall
Hz	hertz
ICOG	incremental cut-off grade
ICP	inductively coupled plasma
ICP-AES	inductively coupled plasma - atomic emission spectrometry
ICP-MS	inductively coupled plasma - mass spectrometry
ICP-OES	inductively coupled plasma - optical emission spectrometry
ID2	inverse distance squared
ID3	inverse distance cubed
IRR	internal rate of return
ISC	Integrated Sustainability Consultants Inc.
JDS	JDS Energy & Mining Inc.
KE	kriging efficiency
kg	kilogram
kg/t	kilograms per tonne
kJ	thousand Joules
KZK	Kudz Ze Kayah
KZK Project	Kudz Ze Kayah Project
KZK Property	Kudz Ze Kayah Property
km	kilometre(s)
km^2	square kilometre(s)
km/h	kilometres per hour
KNA	kriging neighbourhood analysis
koz	thousand ounces
KP	Knight Piésold Ltd
kt	thousand tonnes
kt/a	thousand tonnes per annum
kV	thousand volts
kW	thousand watts
kWh/t	kilowatt hour per tonne
kWh/m^3	kilowatt hour per cubic metre
L	litre(s)
lab	laboratory
L/s	litres per second
lb	pound(s)
LFN	Liard First Nation
LGO	low grade ore

LiDAR	light detection and ranging (survey)
LNG	liquefied natural gas
LOM	life of mine
LPG	liquefied petroleum gas
LSA	local study area
LV	low voltage
m	metre(s)
M	million(s)
m^3	cubic metre(s)
m^3/day	cubic metres/day
Ma	million year
MAIG	Member of the Australian Institute of Geoscientists
masl	metres above sea level
mg	milligram(s)
mg/l	milligrams per litre
mH	metres high
mil	millionth of an inch
Minnovo	Minnovo Pty Ltd
ML	metal leaching
Ml/t	million litres per tonne
mm	millimetre(s)
Mm^3	million cubic metres
Moz	million ounces
MRE	Mineral Resource estimate
mRL	reduced level in metres
m/s	metres per second
Mt	million tonne(s)
Mt/a	million tonnes per annum
mW	metres wide
MW	megawatt
N	northing or north
NI 43-101	National Instrument 43-101
NTS	National Topographic System of Canada
NPV	net present value
NRCan	Explosives Regulatory Division of Natural Resources Canada
NSR	net smelter return
OK	ordinary kriging
OMI	OMI Pty Ltd
OSA	online sample analyser
OSE	On Site Engineering Limited

oz	ounce
PAG	potential acid generating
PDC	process design criteria
PDT	project delivery team
PEP	project evaluation plan
PFS	preliminary feasibility study
PLC	programmable logic controller
ppm	parts per million
QA	quality assurance
QAQC	quality assurance and quality control
QC	quality control
QML	Quartz Mining Licence
Q-Q	quantile-quantile
RCP	reclamation and closure plan
RFA	request for approval
RFQ	request for tender
ROM	run-of-mine
RQD	rock quality designation
RRDC	Ross River Dena Council
RTK	real time kinematic
S	south
$S^{0,1,2,3}$	structural geology terms describing overprinting rock fabrics - S^0 oldest, S^3 youngest
SAG	semi-autogenous grinding (mill)
SCADA	Supervisory Control and Data Acquisition
SCSE	SAG circuit specific energy
SD	standard deviation
SEPA	Socio Economic Participation Agreement
SG	specify gravity
SK	Saskatchewan
SLL	safe lifting limit
SWP	Stewart World Port
t	tonne(s)
t_a	abrasion breakage parameter
t/h	tonnes per hour
t/m^3	tonnes per cubic metre
$t/m^2/h$	tonnes per square metre per hour
TEM	transient electromagnetic, or time-domain electromagnetic
Tetra Tech	Tetra Tech Inc.
TML	transportable moisture limit
US$ or USD	United States of America dollar

US$M	million US$
UTM	Universal Transverse Mercator
VHF	very high frequency
VHMS	volcanic hosted massive sulphide (deposit)
VSHMS	volcanic sediment hosted massive sulphide (deposit)
VTEM	versatile time-domain electromagnetic
W	west
WBS	work breakdown structure
wmt	wet metric tonne(s)
WTP	water treatment plant
XRF	x-ray fluorescence
YESAA	Yukon Environmental and Socio-economic Assessment Act
YESAB	Yukon Environmental and Socio-economic Assessment Board
YT	Yukon
µg	microgram

29.2 Chemical Symbols

Ag	silver
Al	aluminium
As	arsenic
Au	gold
Ba	barium
Bi	bismuth
Ca	calcium
Cd	cadmium
Co	cobalt
Cr	chromium
Cu	copper
Fe	iron
Hg	mercury
K	potassium
La	lanthanum
Mg	magnesium
Mn	manganese
Mo	molybdenum
Na	sodium
Ni	nickel
Pb	lead
S	sulphur
Sb	antimony

Se	selenium
Sn	tin
Sr	strontium
Ti	titanium
V	vanadium
W	tungsten
Y	yttrium
Zn	zinc

29.3 Geological Unit Abbreviations

For abbreviation of geological units used in this report, the reader is referred to Table 7-2 in Section 7.3.2.

29.4 Glossary of Terms

Access Road	Improved road from the Robert Campbell Highway to the KZK Project, suitable for freight haulage to and from an operating mine. Subject to a Lease Agreement under the *Territorial Lands Act* under which the road is classed as a "Haul Road".
All-in Sustaining Costs (AISC)	Includes C1 cash costs, plus exploration costs at the Project and sustaining capital expenditures (including progressive expansion of waste storage facilities, permitting and customary improvements to the operations over the life of the Project). AISC is divided by the number of payable pounds of zinc or ounces of silver, estimated to be produced for the period to arrive at AISC per zinc pound or silver ounce produced.
C1 Cash Cost	Net Direct Cash Cost (C1) represents the cash cost incurred at each processing stage, from mining through to recoverable metal delivered to market, less net by-product credits (if any), divided by the number of payable pounds of zinc or ounces of silver, estimated to be produced for the period to arrive at AISC per zinc pound or silver ounce produced
CIM Definition Standards	The CIM Definition Standards on Mineral Resources and Reserves establish definitions and guidance on the definitions for mineral resources, mineral reserves, and mining studies used in Canada. The Mineral Resource, Mineral Reserve, and Mining Study definitions are incorporated, by reference, into National Instrument 43-101.
Cut-off grade	The lowest grade, or quality, of mineralized material that qualifies as economically mineable and available in a given deposit. May be defined on the basis of economic evaluation, or on physical or chemical attributes that define an acceptable product specification.
Density (bulk density)	Measure of quantity of mass per unit volume.
Dilution	Waste which is unavoidably mined with ore.
Exploration	Prospecting, sampling, mapping, diamond drilling and other work involved in the search for mineralization.
Fully costed cut-off grade	Measured and Indicated material that covers all operating costs associated with mining.
Feasibility Study	A comprehensive technical and economic study of the selected development option for a mineral project that includes appropriately detailed assessments of applicable Modifying Factors together with any other relevant operational factors and detailed financial analysis that are necessary to demonstrate, at the time of reporting, that extraction is reasonably justified (economically mineable). The results of the study may reasonably serve as the basis for a final decision by a proponent or financial institution to proceed with, or finance, the development of the project. The confidence level of the

	study will be higher than that of a Preliminary Feasibility Study (PFS). The term proponent captures issuers who may finance a project without using traditional financial institutions. In these cases, the technical and economic confidence of the Feasibility Study is equivalent to that required by a financial institution. A Definitive Feasibility Study (DFS) has the same meaning as for a Feasibility Study.
Geotechnical investigation	A study which investigates the soil and rock structure of a particular site, as well as the passing of water above and beneath the surface.
Geotechnical IRA slope	Geotechnical inter-ramp slope angle.
Geotechnical OSA slope	Geotechnical overall slope angle.
Grade	Any physical or chemical measurement of the characteristics of the material of interest in samples or product. The units of measurement should be stated when figures are reported. Or the relative quality or percentage of ore mineral content.
Hydrological	Pertaining to water either above or below the surface.
Incremental cut-off grade	Measured and Indicated material that covers underground production mining, load and haul, direct processing, and general and administration costs.
Indicated Mineral Resource	Is that part of a Mineral Resource for which quantity, grade (or quality), densities, shape and physical characteristics are estimated with sufficient confidence to allow the application of Modifying Factors in sufficient detail to support mine planning and evaluation of the economic viability of the deposit. Geological evidence is derived from adequately detailed and reliable exploration, sampling and testing gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drillholes, and is sufficient to assume geological and grade (or quality) continuity between points of observation. An Indicated Mineral Resource has a lower level of confidence than that applying to a Measured Mineral Resource and may only be converted to a Probable Mineral Reserve.
Inferred Mineral Resource	Is that part of a Mineral Resource for which quantity and grade (or quality) are estimated on the basis of limited geological evidence and sampling. Geological evidence is sufficient to imply but not verify geological and grade (or quality) continuity. It is based on exploration, sampling and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drillholes. An Inferred Mineral Resource is based on limited information and sampling gathered through appropriate sampling techniques from locations such as outcrops, trenches, pits, workings and drillholes. Inferred Mineral Resources must not be included in the economic analysis, production schedules, or estimated mine life in publicly disclosed Pre-Feasibility or Feasibility Studies, or in the Life of Mine plans and cash flow models of developed mines. An Inferred Mineral Resource has a lower level of confidence than that applying to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.
Measured Mineral Resource	Is that part of a Mineral Resource for which quantity, grade (or quality), densities, shape, and physical characteristics are estimated with confidence sufficient to allow the application of Modifying Factors to support detailed mine planning and final evaluation of the economic viability of the deposit. Geological evidence is derived from detailed and reliable exploration, sampling and testing gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drillholes, and is sufficient to confirm geological and grade (or quality) continuity between points of observation. A Measured Mineral Resource has a higher level of confidence than that applying to either an Indicated Mineral Resource or an Inferred Mineral Resource. It may be converted to a Proved Mineral Reserve or under certain circumstances to a Probable Mineral Reserve.

Mineable Mineral Reserve	Is that portion of a resource for which extraction is technically and economically feasible. The economically mineable part of a Measured and/or Indicated Mineral Resource. It includes diluting materials and allowances for losses, which may occur when the material is mined or extracted and is defined by studies at Preliminary Feasibility or Feasibility level as appropriate that include application of Modifying Factors. Such studies demonstrate that, at the time of reporting, extraction could reasonably be justified. The reference point at which Mineral Reserves are defined, usually the point where the ore is delivered to the processing plant, must be stated. It is important that, in all situations where the reference point is different, such as for a saleable product, a clarifying statement is included to ensure that the reader is fully informed as to what is being reported. The waste-to-ore ration must be disclosed. The public disclosure of a Mineral Reserve must be demonstrated by a Preliminary Feasibility Study or Feasibility Study. Mineral Reserve estimate are determined and reported in accordance with NI 43-101 "Standards of Disclosure for Mineral Projects" (the "Instrument", June 2011) and the classifications adopted by the CIM Council in November 2014.
Mineral Resource	A concentration or occurrence of material of economic interest in or on the Earth's crust in such form, quality and quantity that there are reasonable and realistic prospects for eventual economic extraction. The location, quantity, grade or quality continuity and other geological characteristics of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge, including sampling. Mineral Resources are subdivided in order of increasing geological confidence, in respect of geoscientific evidence, into Inferred, Indicated and Measured categories. The Mineral Resource is stated inclusive of the Mineral Reserve.
Mineralization	Any single mineral or combination of minerals occurring in a mass, or deposit, of economic interest. The term is intended to cover all forms in which mineralization might occur, whether by class of deposit, mode of occurrence, genesis or composition.
Mining	All activities related to extraction of metals, minerals and gemstones from the earth whether surface or underground, and by any method (e.g. quarries, open cut, open cut, solution mining, dredging, etc.).
Modifying Factors	Modifying Factors are considerations used to convert Mineral Resources to Mineral Reserves. These include, but are not restricted to, mining, processing, metallurgical, infrastructure, economic, marketing, legal, environmental, social and governmental factors.
Net smelter return	Net Smelter Return (Free on Board, FOB) Mine Gate and Royalties by unit of metal in ROM feed.
Ore	A mixture of valuable and worthless minerals from which at least one of the minerals can be mined and processed at an economic profit.
Orebody	A continuous well-defined mass of material of sufficient ore content to make extraction economically feasible.
Preliminary Feasibility Study	A Preliminary Feasibility Study (PFS) is a comprehensive study of a range of options for the technical and economic viability of a mineral project that has advanced to a stage where a preferred mining method, in the case of underground mining, or the pit configuration, in the case of an open pit, is established and an effective method of mineral processing is determined. It includes a financial analysis based on reasonable assumptions on the Modifying Factors and the evaluation of any other relevant factors which are sufficient for a Qualified Person, acting reasonably, to determine if all or part of the Mineral Resource may be converted to a Mineral Reserve at the time of reporting. A PFS is at a lower confidence level than a Feasibility Study. The CIM Definition Standards requires the completion of a Prefeasibility Study as the minimum prerequisite for the conversion of Mineral Resources to Mineral Reserves.

Probable Mineral Reserve	A Probable Mineral Reserve is the economically mineable part of an Indicated, and in some circumstances, a Measured Mineral Resource. The confidence in the Modifying Factors applying to a Probable Mineral Reserve is lower than that applying to a Proven Mineral Reserve. The Qualified Person(s) may elect, to convert Measured Mineral Resources to Probable Mineral Reserves if the confidence in the Modifying Factors is lower than that applied to a Proven Mineral Reserve. Probable Mineral Reserve estimates must be demonstrated to be economic, at the time of reporting, by at least a Preliminary Feasibility Study.
Proven Mineral Reserve	A Proven Mineral Reserve is the economically mineable part of a Measured Mineral Resource. A Proven Mineral Reserve implies a high degree of confidence in the Modifying Factors. 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 the potential economic viability of the deposit. Proven Mineral Reserve estimates must be demonstrated to be economic, at the time of reporting, by at least a Pre-Feasibility Study. Within the CIM Definition standards the term Proved Mineral Reserve is an equivalent term to a Proven Mineral Reserve.
Run of mine	The Process Plant primary crushers maximum run-of-mine (ROM) throughput capacity.
Specific gravity	A ratio which for all practical purposes is equivalent to density expressed in t/m^3.
Tote Road	Existing road from the Robert Campbell Highway to the KZK Project, authorized as an “Access Road (Kudz Ze Kayah Access Road) and Gatehouse” under a Lease Agreement with Yukon Government issued under the *Territorial Lands Act*.
Waste rock	Rock with an insufficient metal content to justify processing.
YESAA	Yukon Environmental and Socio-economic Assessment which mandates a public process for assessing the Project’s potential socio-economic and environmental impacts.
YESAB	The Yukon Environmental and Socio-economic Assessment Board.

Appendix 1: BMC KZK Project Tenements

District = Watson Lake, Claim Owner = BMC Minerals (No.1) Ltd - 100%, Status = Active

Grant no.	Claim name and no.	Operation recording date	Staking date	Claim expiry date	NTS map no.	Non-standard size	Ops no.
YB46227	TAG 1	20/08/1993	18/08/1993	2/04/2038	105G07		1000130232
YB46228	TAG 2	20/08/1993	18/08/1993	2/04/2038	105G07		1000130233
YB46229	TAG 3	20/08/1993	18/08/1993	2/04/2038	105G07		1000130234
YB46230	TAG 4	20/08/1993	18/08/1993	2/04/2038	105G07		1000130235
YB46231	TAG 5	20/08/1993	18/08/1993	2/04/2038	105G07		1000130236
YB46232	TAG 6	20/08/1993	18/08/1993	2/04/2038	105G07		1000130237
YB46233	TAG 7	20/08/1993	18/08/1993	2/04/2038	105G07		1000130238
YB46234	TAG 8	20/08/1993	18/08/1993	2/04/2038	105G07		1000130239
YB46235	TAG 9	20/08/1993	18/08/1993	2/04/2038	105G07		1000130240
YB46236	TAG 10	20/08/1993	18/08/1993	2/04/2038	105G07		1000130241
YB46237	TAG 11	20/08/1993	18/08/1993	2/04/2038	105G07		1000130242
YB46238	TAG 12	20/08/1993	18/08/1993	2/04/2038	105G07		1000130243
YB46239	TAG 13	20/08/1993	18/08/1993	2/04/2038	105G07		1000130244
YB46240	TAG 14	20/08/1993	18/08/1993	2/04/2038	105G07		1000130245
YB46241	TAG 15	20/08/1993	18/08/1993	2/04/2038	105G07		1000130246
YB46242	TAG 16	20/08/1993	18/08/1993	2/04/2038	105G07		1000130247
YB46243	TAG 17	20/08/1993	18/08/1993	2/04/2038	105G07		1000130248
YB46244	TAG 18	20/08/1993	18/08/1993	2/04/2038	105G07		1000130249
YB46245	TAG 19	20/08/1993	18/08/1993	2/04/2038	105G07		1000130250
YB46246	TAG 20	20/08/1993	18/08/1993	2/04/2038	105G07		1000130251
YB46247	TAG 21	20/08/1993	18/08/1993	2/04/2038	105G07		1000130252
YB46248	TAG 22	20/08/1993	18/08/1993	2/04/2038	105G07		1000130253
YB46249	TAG 23	20/08/1993	18/08/1993	2/04/2038	105G07		1000130254
YB46250	TAG 24	20/08/1993	18/08/1993	2/04/2038	105G07		1000130255
YB46251	TAG 25	20/08/1993	18/08/1993	2/04/2038	105G07		1000130256
YB46252	TAG 26	20/08/1993	18/08/1993	2/04/2038	105G07		1000130257
YB46253	TAG 27	20/08/1993	18/08/1993	2/04/2038	105G07		1000130258
YB46254	TAG 28	20/08/1993	18/08/1993	2/04/2038	105G07		1000130259
YB46255	TAG 29	20/08/1993	18/08/1993	2/04/2038	105G07		1000130260
YB46256	TAG 30	20/08/1993	18/08/1993	2/04/2038	105G07		1000130261
YB46325	PLATE 1	29/09/1993	15/09/1993	2/04/2038	105G07		1000130330
YB46326	PLATE 2	29/09/1993	15/09/1993	2/04/2038	105G07		1000130331
YB46327	PLATE 3	29/09/1993	15/09/1993	2/04/2038	105G07		1000130332
YB46328	PLATE 4	29/09/1993	15/09/1993	2/04/2038	105G07		1000130333
YB46329	PLATE 5	29/09/1993	15/09/1993	2/04/2038	105G07		1000130334
YB46330	PLATE 6	29/09/1993	15/09/1993	2/04/2038	105G07		1000130335
YB46331	PLATE 7	29/09/1993	15/09/1993	2/04/2038	105G07		1000130336
YB46332	PLATE 8	29/09/1993	15/09/1993	2/04/2038	105G07		1000130337
YB46333	PLATE 9	29/09/1993	15/09/1993	2/04/2038	105G07		1000130338
YB46334	PLATE 10	29/09/1993	15/09/1993	2/04/2038	105G07		1000130339
YB46335	PLATE 11	29/09/1993	15/09/1993	2/04/2038	105G07		1000130340

Grant no.	Claim name and no.	Operation recording date	Staking date	Claim expiry date	NTS map no.	Non-standard size	Ops no.
YB46336	PLATE 12	29/09/1993	15/09/1993	2/04/2038	105G07		1000130341
YB46337	PLATE 13	29/09/1993	15/09/1993	2/04/2038	105G07		1000130342
YB46338	PLATE 14	29/09/1993	15/09/1993	2/04/2038	105G07		1000130343
YB46339	PLATE 15	29/09/1993	15/09/1993	2/04/2038	105G07		1000130344
YB46340	PLATE 16	29/09/1993	15/09/1993	2/04/2038	105G07		1000130345
YB46341	PLATE 17	29/09/1993	15/09/1993	2/04/2038	105G07		1000130346
YB46342	PLATE 18	29/09/1993	15/09/1993	2/04/2038	105G07		1000130347
YB46343	PLATE 19	29/09/1993	15/09/1993	2/04/2038	105G07		1000130348
YB46344	PLATE 20	29/09/1993	15/09/1993	2/04/2038	105G07		1000130349
YB46345	PLATE 21	29/09/1993	15/09/1993	2/04/2038	105G07		1000130350
YB46346	PLATE 22	29/09/1993	15/09/1993	2/04/2038	105G07		1000130351
YB46347	PLATE 23	29/09/1993	15/09/1993	2/04/2038	105G07		1000130352
YB46348	PLATE 24	29/09/1993	15/09/1993	2/04/2038	105G07		1000130353
YB46349	PLATE 25	29/09/1993	15/09/1993	2/04/2038	105G07		1000130354
YB46350	HOME 1	29/09/1993	15/09/1993	2/04/2038	105G07		1000130355
YB46351	HOME 2	29/09/1993	15/09/1993	2/04/2038	105G07		1000130356
YB46352	HOME 3	29/09/1993	15/09/1993	2/04/2038	105G07		1000130357
YB46353	HOME 4	29/09/1993	15/09/1993	2/04/2038	105G07		1000130358
YB46354	HOME 5	29/09/1993	15/09/1993	2/04/2038	105G07		1000130359
YB46355	HOME 6	29/09/1993	15/09/1993	2/04/2038	105G07		1000130360
YB46356	HOME 7	29/09/1993	15/09/1993	2/04/2038	105G07		1000130361
YB46357	HOME 8	29/09/1993	15/09/1993	2/04/2038	105G07		1000130362
YB46358	HOME 9	29/09/1993	15/09/1993	2/04/2038	105G07		1000130363
YB46359	HOME 10	29/09/1993	15/09/1993	2/04/2038	105G07		1000130364
YB46360	HOME 11	29/09/1993	15/09/1993	2/04/2038	105G07		1000130365
YB46361	HOME 12	29/09/1993	15/09/1993	2/04/2038	105G07		1000130366
YB46362	HOME 13	29/09/1993	15/09/1993	2/04/2038	105G07		1000130367
YB46363	HOME 14	29/09/1993	15/09/1993	2/04/2038	105G07		1000130368
YB46364	HOME 15	29/09/1993	15/09/1993	2/04/2038	105G07		1000130369
YB46365	HOME 16	29/09/1993	15/09/1993	2/04/2038	105G07		1000130370
YB46366	HOME 17	29/09/1993	15/09/1993	2/04/2038	105G07		1000130371
YB47461	TAG 31	15/04/1994	9/04/1994	2/04/2036	105G07		1000130566
YB47462	TAG 32	15/04/1994	9/04/1994	2/04/2036	105G07		1000130567
YB47463	TAG 33	15/04/1994	9/04/1994	2/04/2036	105G07		1000130568
YB47464	TAG 34	15/04/1994	9/04/1994	2/04/2036	105G07		1000130569
YB47465	TAG 35	15/04/1994	9/04/1994	2/04/2036	105G07		1000130570
YB47466	TAG 36	15/04/1994	9/04/1994	2/04/2036	105G07		1000130571
YB47467	TAG 37	15/04/1994	9/04/1994	2/04/2036	105G07		1000130572
YB47468	TAG 38	15/04/1994	9/04/1994	2/04/2036	105G07		1000130573
YB47469	TAG 39	15/04/1994	9/04/1994	2/04/2036	105G07		1000130574
YB47470	TAG 40	15/04/1994	9/04/1994	2/04/2036	105G07		1000130575
YB47471	TAG 41	15/04/1994	9/04/1994	2/04/2036	105G07		1000130576
YB47472	TAG 42	15/04/1994	9/04/1994	2/04/2036	105G07		1000130577
YB47473	TAG 43	15/04/1994	9/04/1994	2/04/2036	105G07		1000130578
YB47474	TAG 44	15/04/1994	9/04/1994	2/04/2036	105G07		1000130579
YB47475	TAG 45	15/04/1994	9/04/1994	2/04/2036	105G07		1000130580

Grant no.	Claim name and no.	Operation recording date	Staking date	Claim expiry date	NTS map no.	Non-standard size	Ops no.
YB47476	TAG 46	15/04/1994	9/04/1994	2/04/2036	105G07		1000130581
YB47477	TAG 47	15/04/1994	9/04/1994	2/04/2036	105G07		1000130582
YB47478	TAG 48	15/04/1994	9/04/1994	2/04/2036	105G07		1000130583
YB47479	TAG 49	15/04/1994	9/04/1994	2/04/2036	105G07		1000130584
YB47480	TAG 50	15/04/1994	9/04/1994	2/04/2036	105G07		1000130585
YB47481	TAG 51	15/04/1994	9/04/1994	2/04/2036	105G07		1000130586
YB47482	TAG 52	15/04/1994	9/04/1994	2/04/2036	105G07		1000130587
YB47483	TAG 53	15/04/1994	9/04/1994	2/04/2036	105G07		1000130588
YB47484	TAG 54	15/04/1994	9/04/1994	2/04/2036	105G07		1000130589
YB47485	TAG 55	15/04/1994	9/04/1994	2/04/2036	105G07		1000130590
YB47486	TAG 56	15/04/1994	9/04/1994	2/04/2036	105G07		1000130591
YB47487	TAG 57	15/04/1994	9/04/1994	2/04/2036	105G07		1000130592
YB47488	TAG 58	15/04/1994	9/04/1994	2/04/2036	105G07		1000130593
YB47489	TAG 59	15/04/1994	9/04/1994	2/04/2036	105G07		1000130594
YB47490	TAG 60	15/04/1994	9/04/1994	2/04/2036	105G07		1000130595
YB47491	TAG 61	15/04/1994	9/04/1994	2/04/2036	105G07		1000130596
YB47492	TAG 62	15/04/1994	9/04/1994	2/04/2036	105G07		1000130597
YB47493	TAG 63	15/04/1994	9/04/1994	2/04/2036	105G07		1000130598
YB47494	TAG 64	15/04/1994	9/04/1994	2/04/2036	105G07		1000130599
YB47495	TAG 65	15/04/1994	9/04/1994	2/04/2036	105G07		1000130600
YB47496	TAG 66	15/04/1994	9/04/1994	2/04/2036	105G07		1000130601
YB47497	TAG 67	15/04/1994	9/04/1994	2/04/2036	105G07		1000130602
YB47498	TAG 68	15/04/1994	9/04/1994	2/04/2036	105G07		1000130603
YB47499	TAG 69	15/04/1994	9/04/1994	2/04/2036	105G07		1000130604
YB47500	TAG 70	15/04/1994	9/04/1994	2/04/2036	105G07		1000130605
YB47501	TAG 71	15/04/1994	9/04/1994	2/04/2036	105G07		1000130606
YB47502	TAG 72	15/04/1994	9/04/1994	2/04/2036	105G07		1000130607
YB47503	TAG 73	15/04/1994	9/04/1994	2/04/2036	105G07		1000130608
YB47504	TAG 74	15/04/1994	9/04/1994	2/04/2036	105G07		1000130609
YB47505	TAG 75	15/04/1994	9/04/1994	2/04/2036	105G07		1000130610
YB47506	TAG 76	15/04/1994	9/04/1994	2/04/2036	105G07		1000130611
YB47507	TAG 77	15/04/1994	9/04/1994	2/04/2036	105G07		1000130612
YB47508	TAG 78	15/04/1994	9/04/1994	2/04/2036	105G07		1000130613
YB47509	TAG 79	15/04/1994	9/04/1994	2/04/2036	105G07		1000130614
YB47510	TAG 80	15/04/1994	9/04/1994	2/04/2036	105G07		1000130615
YB47511	TAG 81	15/04/1994	9/04/1994	2/04/2036	105G07		1000130616
YB47512	TAG 82	15/04/1994	9/04/1994	2/04/2036	105G07		1000130617
YB47513	TAG 83	15/04/1994	9/04/1994	2/04/2036	105G07		1000130618
YB47514	TAG 84	15/04/1994	9/04/1994	2/04/2036	105G07		1000130619
YB47515	TAG 85	15/04/1994	9/04/1994	2/04/2036	105G07		1000130620
YB47516	TAG 86	15/04/1994	9/04/1994	2/04/2036	105G07		1000130621
YB47517	TAG 87	15/04/1994	9/04/1994	2/04/2036	105G07		1000130622
YB47518	TAG 88	15/04/1994	9/04/1994	2/04/2036	105G07		1000130623
YB47519	TAG 89	15/04/1994	9/04/1994	2/04/2036	105G07		1000130624
YB47520	TAG 90	15/04/1994	9/04/1994	2/04/2036	105G07		1000130625
YB47521	TAG 91	15/04/1994	9/04/1994	2/04/2036	105G07		1000130626

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YB47522	TAG 92	15/04/1994	9/04/1994	2/04/2036	105G07		1000130627
YB47523	TAG 93	15/04/1994	9/04/1994	2/04/2036	105G07		1000130628
YB47524	TAG 94	15/04/1994	9/04/1994	2/04/2036	105G07		1000130629
YB47525	TAG 95	15/04/1994	9/04/1994	2/04/2036	105G07		1000130630
YB47526	TAG 96	15/04/1994	9/04/1994	2/04/2036	105G07		1000130631
YB47527	TAG 97	15/04/1994	9/04/1994	2/04/2036	105G07		1000130632
YB47528	TAG 98	15/04/1994	9/04/1994	2/04/2036	105G07		1000130633
YB47529	TAG 99	15/04/1994	9/04/1994	2/04/2036	105G07		1000130634
YB47530	TAG 100	15/04/1994	9/04/1994	2/04/2036	105G07		1000130635
YB47531	TAG 101	15/04/1994	9/04/1994	2/04/2036	105G07		1000130636
YB47532	TAG 102	15/04/1994	9/04/1994	2/04/2036	105G07		1000130637
YB47533	TAG 103	15/04/1994	9/04/1994	2/04/2036	105G07		1000130638
YB47534	TAG 104	15/04/1994	9/04/1994	2/04/2036	105G07		1000130639
YB47535	TAG 105	15/04/1994	9/04/1994	2/04/2036	105G07		1000130640
YB47536	TAG 106	15/04/1994	9/04/1994	2/04/2036	105G07		1000130641
YB47537	TAG 107	15/04/1994	9/04/1994	2/04/2036	105G07		1000130642
YB47538	TAG 108	15/04/1994	9/04/1994	2/04/2036	105G07		1000130643
YB47539	TAG 109	15/04/1994	9/04/1994	2/04/2036	105G07		1000130644
YB47540	TAG 110	15/04/1994	9/04/1994	2/04/2036	105G07		1000130645
YB47541	TAG 111	15/04/1994	9/04/1994	2/04/2036	105G07		1000130646
YB47542	TAG 112	15/04/1994	9/04/1994	2/04/2036	105G07		1000130647
YB47543	TAG 113	15/04/1994	9/04/1994	2/04/2036	105G07		1000130648
YB47544	TAG 158	15/04/1994	10/04/1994	2/04/2036	105G07		1000130649
YB47545	TAG 159	15/04/1994	10/04/1994	2/04/2036	105G07		1000130650
YB47546	TAG 160	15/04/1994	10/04/1994	2/04/2036	105G07		1000130651
YB47547	TAG 161	15/04/1994	10/04/1994	2/04/2036	105G07		1000130652
YB47548	TAG 162	15/04/1994	10/04/1994	2/04/2036	105G07		1000130653
YB47549	TAG 163	15/04/1994	10/04/1994	2/04/2036	105G07		1000130654
YB47550	TAG 164	15/04/1994	10/04/1994	2/04/2036	105G07		1000130655
YB47551	TAG 165	15/04/1994	10/04/1994	2/04/2036	105G07		1000130656
YB47552	TAG 166	15/04/1994	10/04/1994	2/04/2036	105G07		1000130657
YB47553	TAG 167	15/04/1994	10/04/1994	2/04/2036	105G07		1000130658
YB47554	TAG 168	15/04/1994	10/04/1994	2/04/2036	105G07		1000130659
YB47555	TAG 169	15/04/1994	10/04/1994	2/04/2036	105G07		1000130660
YB47556	TAG 170	15/04/1994	10/04/1994	2/04/2036	105G07		1000130661
YB47557	TAG 171	15/04/1994	10/04/1994	2/04/2036	105G07		1000130662
YB47558	TAG 172	15/04/1994	10/04/1994	2/04/2036	105G07		1000130663
YB47559	TAG 173	15/04/1994	10/04/1994	2/04/2036	105G07		1000130664
YB47560	TAG 174	15/04/1994	10/04/1994	2/04/2036	105G07		1000130665
YB47561	TAG 175	15/04/1994	10/04/1994	2/04/2036	105G07		1000130666
YB47562	TAG 176	15/04/1994	10/04/1994	2/04/2036	105G07		1000130667
YB47563	TAG 177	15/04/1994	10/04/1994	2/04/2036	105G07		1000130668
YB47564	TAG 178	15/04/1994	10/04/1994	2/04/2036	105G07		1000130669
YB47565	TAG 179	15/04/1994	10/04/1994	2/04/2036	105G07		1000130670
YB47566	TAG 180	15/04/1994	10/04/1994	2/04/2036	105G07		1000130671
YB47567	TAG 181	15/04/1994	10/04/1994	2/04/2036	105G07		1000130672

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YB47568	TAG 182	15/04/1994	10/04/1994	2/04/2036	105G07		1000130673
YB47569	TAG 183	15/04/1994	10/04/1994	2/04/2036	105G07		1000130674
YB47570	TAG 184	15/04/1994	10/04/1994	2/04/2036	105G07		1000130675
YB47571	TAG 185	15/04/1994	10/04/1994	2/04/2036	105G07		1000130676
YB47572	TAG 186	15/04/1994	10/04/1994	2/04/2036	105G07		1000130677
YB47573	TAG 187	15/04/1994	10/04/1994	2/04/2036	105G07		1000130678
YB47574	TAG 188	15/04/1994	10/04/1994	2/04/2036	105G07		1000130679
YB47575	TAG 189	15/04/1994	10/04/1994	2/04/2036	105G07		1000130680
YB47576	TAG 190	15/04/1994	10/04/1994	2/04/2036	105G07		1000130681
YB47577	TAG 191	15/04/1994	10/04/1994	2/04/2036	105G07		1000130682
YB47578	TAG 192	15/04/1994	10/04/1994	2/04/2036	105G07		1000130683
YB47579	TAG 193	15/04/1994	10/04/1994	2/04/2036	105G07		1000130684
YB47580	TAG 194	15/04/1994	10/04/1994	2/04/2036	105G07		1000130685
YB47581	TAG 195	15/04/1994	10/04/1994	2/04/2036	105G07		1000130686
YB47582	TAG 196	15/04/1994	10/04/1994	2/04/2036	105G07		1000130687
YB47583	TAG 197	15/04/1994	10/04/1994	2/04/2036	105G07		1000130688
YB47584	TAG 198	15/04/1994	10/04/1994	2/04/2036	105G07		1000130689
YB47585	TAG 199	15/04/1994	10/04/1994	2/04/2036	105G07		1000130690
YB47586	TAG 200	15/04/1994	10/04/1994	2/04/2036	105G07		1000130691
YB47587	TAG 201	15/04/1994	10/04/1994	2/04/2036	105G07		1000130692
YB47588	TAG 202	15/04/1994	10/04/1994	2/04/2036	105G07		1000130693
YB47590	TAG 204	15/04/1994	10/04/1994	2/04/2036	105G07		1000130695
YB47592	TAG 206	15/04/1994	10/04/1994	2/04/2036	105G07		1000130697
YB47593	TAG 207	15/04/1994	10/04/1994	2/04/2036	105G07		1000130698
YB47594	TAG 208	15/04/1994	10/04/1994	2/04/2036	105G07		1000130699
YB47595	TAG 209	15/04/1994	10/04/1994	2/04/2036	105G07		1000130700
YB47596	TAG 210	15/04/1994	10/04/1994	2/04/2036	105G07		1000130701
YB47597	TAG 211	15/04/1994	10/04/1994	2/04/2036	105G07		1000130702
YB47598	TAG 212	15/04/1994	10/04/1994	2/04/2036	105G07		1000130703
YB47599	TAG 213	15/04/1994	10/04/1994	2/04/2036	105G07		1000130704
YB47600	TAG 214	15/04/1994	10/04/1994	2/04/2036	105G07		1000130705
YB47601	TAG 215	15/04/1994	10/04/1994	2/04/2036	105G08		1000130706
YB47602	TAG 216	15/04/1994	10/04/1994	2/04/2036	105G08		1000130707
YB47603	TAG 217	15/04/1994	10/04/1994	2/04/2036	105G08		1000130708
YB47604	TAG 218	15/04/1994	10/04/1994	2/04/2036	105G08		1000130709
YB47605	TAG 219	15/04/1994	10/04/1994	2/04/2036	105G08		1000130710
YB47606	TAG 220	15/04/1994	10/04/1994	2/04/2036	105G08		1000130711
YB47607	TAG 221	15/04/1994	10/04/1994	2/04/2036	105G08		1000130712
YB47608	TAG 222	15/04/1994	10/04/1994	2/04/2036	105G08		1000130713
YB47609	TAG 223	15/04/1994	10/04/1994	2/04/2036	105G08		1000130714
YB47610	TAG 224	15/04/1994	10/04/1994	2/04/2036	105G08		1000130715
YB47611	TAG 225	15/04/1994	10/04/1994	2/04/2036	105G08		1000130716
YB47612	TAG 226	15/04/1994	10/04/1994	2/04/2036	105G08		1000130717
YB47613	TAG 227	15/04/1994	10/04/1994	2/04/2036	105G08		1000130718
YB47614	TAG 228	15/04/1994	10/04/1994	2/04/2036	105G08		1000130719
YB47615	TAG 229	15/04/1994	10/04/1994	2/04/2036	105G08		1000130720

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YB47616	TAG 230	15/04/1994	10/04/1994	2/04/2036	105G08		1000130721
YB47617	TAG 231	15/04/1994	10/04/1994	2/04/2036	105G08		1000130722
YB47618	TAG 232	15/04/1994	10/04/1994	2/04/2036	105G08		1000130723
YB47619	TAG 233	15/04/1994	11/04/1994	2/04/2036	105G07		1000130724
YB47620	TAG 234	15/04/1994	11/04/1994	2/04/2036	105G07		1000130725
YB47621	TAG 235	15/04/1994	11/04/1994	2/04/2036	105G07		1000130726
YB47622	TAG 236	15/04/1994	11/04/1994	2/04/2036	105G07		1000130727
YB47623	TAG 237	15/04/1994	11/04/1994	2/04/2036	105G07		1000130728
YB47624	TAG 238	15/04/1994	11/04/1994	2/04/2036	105G07		1000130729
YB47625	TAG 239	15/04/1994	11/04/1994	2/04/2036	105G07		1000130730
YB47626	TAG 240	15/04/1994	11/04/1994	2/04/2036	105G07		1000130731
YB47627	TAG 241	15/04/1994	11/04/1994	2/04/2036	105G07		1000130732
YB47628	TAG 242	15/04/1994	11/04/1994	2/04/2036	105G07		1000130733
YB47629	TAG 243	15/04/1994	11/04/1994	2/04/2036	105G07		1000130734
YB47630	TAG 244	15/04/1994	11/04/1994	2/04/2036	105G07		1000130735
YB47631	TAG 245	15/04/1994	11/04/1994	2/04/2036	105G07		1000130736
YB47632	TAG 246	15/04/1994	11/04/1994	2/04/2036	105G07		1000130737
YB47633	TAG 247	15/04/1994	11/04/1994	2/04/2036	105G07		1000130738
YB47634	TAG 248	15/04/1994	11/04/1994	2/04/2036	105G07		1000130739
YB47635	TAG 249	15/04/1994	11/04/1994	2/04/2036	105G07		1000130740
YB47636	TAG 250	15/04/1994	11/04/1994	2/04/2036	105G07		1000130741
YB47637	TAG 251	15/04/1994	11/04/1994	2/04/2036	105G07		1000130742
YB47638	TAG 252	15/04/1994	11/04/1994	2/04/2036	105G07		1000130743
YB47639	TAG 253	15/04/1994	11/04/1994	2/04/2036	105G07		1000130744
YB47640	TAG 254	15/04/1994	14/04/1994	2/04/2036	105G07		1000130745
YB47641	TAG 255	15/04/1994	14/04/1994	2/04/2036	105G07		1000130746
YB47642	TAG 256	15/04/1994	14/04/1994	2/04/2036	105G07		1000130747
YB47643	TAG 257	15/04/1994	14/04/1994	2/04/2036	105G07		1000130748
YB47644	TAG 258	15/04/1994	14/04/1994	2/04/2036	105G07		1000130749
YB47645	TAG 259	15/04/1994	14/04/1994	2/04/2036	105G07		1000130750
YB47646	TAG 260	15/04/1994	14/04/1994	2/04/2036	105G07		1000130751
YB47647	TAG 261	15/04/1994	14/04/1994	2/04/2036	105G07		1000130752
YB47648	TAG 262	15/04/1994	14/04/1994	2/04/2036	105G07		1000130753
YB47649	TAG 263	15/04/1994	14/04/1994	2/04/2036	105G07		1000130754
YB47668	TAG 114	26/04/1994	9/04/1994	2/04/2036	105G07		1000130773
YB47669	TAG 115	26/04/1994	9/04/1994	2/04/2036	105G07		1000130774
YB47670	TAG 116	26/04/1994	9/04/1994	2/04/2036	105G07		1000130775
YB47671	TAG 117	26/04/1994	9/04/1994	2/04/2036	105G07		1000130776
YB47672	TAG 118	26/04/1994	9/04/1994	2/04/2036	105G07		1000130777
YB47673	TAG 119	26/04/1994	9/04/1994	2/04/2036	105G07		1000130778
YB47674	TAG 120	26/04/1994	9/04/1994	2/04/2036	105G07		1000130779
YB47675	TAG 121	26/04/1994	9/04/1994	2/04/2036	105G07		1000130780
YB47676	TAG 122	26/04/1994	9/04/1994	2/04/2036	105G07		1000130781
YB47677	TAG 123	26/04/1994	9/04/1994	2/04/2036	105G07		1000130782
YB47678	TAG 124	26/04/1994	9/04/1994	2/04/2036	105G07		1000130783
YB47679	TAG 125	26/04/1994	9/04/1994	2/04/2036	105G07		1000130784

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YB47680	TAG 126	26/04/1994	9/04/1994	2/04/2036	105G07		1000130785
YB47681	TAG 127	26/04/1994	9/04/1994	2/04/2036	105G07		1000130786
YB47682	TAG 128	26/04/1994	9/04/1994	2/04/2036	105G07		1000130787
YB47683	TAG 129	26/04/1994	9/04/1994	2/04/2036	105G07		1000130788
YB47684	TAG 130	26/04/1994	9/04/1994	2/04/2036	105G07		1000130789
YB47685	TAG 131	26/04/1994	9/04/1994	2/04/2036	105G07		1000130790
YB47686	TAG 132	26/04/1994	9/04/1994	2/04/2036	105G07		1000130791
YB47687	TAG 133	26/04/1994	9/04/1994	2/04/2036	105G07		1000130792
YB47688	TAG 134	26/04/1994	10/04/1994	2/04/2036	105G07		1000130793
YB47689	TAG 135	26/04/1994	10/04/1994	2/04/2036	105G07		1000130794
YB47690	TAG 136	26/04/1994	10/04/1994	2/04/2036	105G07		1000130795
YB47691	TAG 137	26/04/1994	10/04/1994	2/04/2036	105G07		1000130796
YB47692	TAG 138	26/04/1994	10/04/1994	2/04/2036	105G07		1000130797
YB47693	TAG 139	26/04/1994	10/04/1994	2/04/2036	105G07		1000130798
YB47694	TAG 140	26/04/1994	10/04/1994	2/04/2036	105G07		1000130799
YB47695	TAG 141	26/04/1994	10/04/1994	2/04/2036	105G07		1000130800
YB47696	TAG 142	26/04/1994	10/04/1994	2/04/2036	105G07		1000130801
YB47697	TAG 143	26/04/1994	10/04/1994	2/04/2036	105G07		1000130802
YB47698	TAG 144	26/04/1994	10/04/1994	2/04/2036	105G07		1000130803
YB47699	TAG 145	26/04/1994	10/04/1994	2/04/2036	105G07		1000130804
YB47700	TAG 146	26/04/1994	10/04/1994	2/04/2036	105G07		1000130805
YB47701	TAG 147	26/04/1994	10/04/1994	2/04/2036	105G07		1000130806
YB47702	TAG 148	26/04/1994	10/04/1994	2/04/2036	105G07		1000130807
YB47703	TAG 149	26/04/1994	10/04/1994	2/04/2036	105G07		1000130808
YB47704	TAG 150	26/04/1994	10/04/1994	2/04/2036	105G07		1000130809
YB47705	TAG 151	26/04/1994	10/04/1994	2/04/2036	105G07		1000130810
YB47706	TAG 152	26/04/1994	10/04/1994	2/04/2036	105G07		1000130811
YB47707	TAG 153	26/04/1994	10/04/1994	2/04/2036	105G07		1000130812
YB47708	TAG 154	26/04/1994	10/04/1994	2/04/2036	105G07		1000130813
YB47709	TAG 155	26/04/1994	10/04/1994	2/04/2036	105G07		1000130814
YB47710	TAG 156	26/04/1994	10/04/1994	2/04/2036	105G07		1000130815
YB47711	TAG 157	26/04/1994	10/04/1994	2/04/2036	105G07		1000130816
YB48413	TAG 264	2/05/1994	26/04/1994	2/04/2036	105G07		1000130917
YB48414	TAG 265	2/05/1994	26/04/1994	2/04/2036	105G07		1000130918
YB48415	TAG 266	2/05/1994	26/04/1994	2/04/2038	105G07		1000130919
YB48416	TAG 267	2/05/1994	26/04/1994	2/04/2038	105G07		1000130920
YB48417	TAG 268	2/05/1994	26/04/1994	2/04/2038	105G07		1000130921
YB48418	TAG 269	2/05/1994	26/04/1994	2/04/2038	105G07		1000130922
YB48419	TAG 270	2/05/1994	26/04/1994	2/04/2036	105G07		1000130923
YB48420	TAG 271	2/05/1994	26/04/1994	2/04/2036	105G07		1000130924
YB48421	TAG 272	2/05/1994	26/04/1994	2/04/2036	105G07		1000130925
YB48422	TAG 273	2/05/1994	26/04/1994	2/04/2036	105G07		1000130926
YB48423	TAG 274	2/05/1994	26/04/1994	2/04/2036	105G07		1000130927
YB48424	TAG 275	2/05/1994	26/04/1994	2/04/2036	105G07		1000130928
YB48425	TAG 276	2/05/1994	26/04/1994	2/04/2036	105G07		1000130929
YB48426	TAG 277	2/05/1994	26/04/1994	2/04/2036	105G07		1000130930

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YB48427	TAG 278	2/05/1994	26/04/1994	2/04/2036	105G07		1000130931
YB48428	TAG 279	2/05/1994	26/04/1994	2/04/2036	105G07		1000130932
YB48429	TAG 280	2/05/1994	26/04/1994	2/04/2036	105G07		1000130933
YB48430	TAG 281	2/05/1994	26/04/1994	2/04/2036	105G07		1000130934
YB48431	TAG 282	2/05/1994	26/04/1994	2/04/2036	105G07		1000130935
YB48432	TAG 283	2/05/1994	26/04/1994	2/04/2036	105G07		1000130936
YB48433	TAG 284	2/05/1994	26/04/1994	2/04/2036	105G10		1000130937
YB48434	TAG 285	2/05/1994	26/04/1994	2/04/2036	105G10		1000130938
YB48435	TAG 286	2/05/1994	26/04/1994	2/04/2036	105G10		1000130939
YB48436	TAG 287	2/05/1994	26/04/1994	2/04/2036	105G10		1000130940
YB48437	TAG 288	2/05/1994	26/04/1994	2/04/2036	105G10		1000130941
YB48438	TAG 289	2/05/1994	26/04/1994	2/04/2036	105G10		1000130942
YB48439	TAG 290	2/05/1994	26/04/1994	2/04/2036	105G10		1000130943
YB48440	TAG 291	2/05/1994	26/04/1994	2/04/2036	105G10		1000130944
YB48441	TAG 292	2/05/1994	26/04/1994	2/04/2036	105G10		1000130945
YB48442	TAG 293	2/05/1994	26/04/1994	2/04/2036	105G10		1000130946
YB48443	TAG 294	2/05/1994	26/04/1994	2/04/2036	105G10		1000130947
YB48444	TAG 295	2/05/1994	26/04/1994	2/04/2036	105G10		1000130948
YB48445	TAG 296	2/05/1994	26/04/1994	2/04/2036	105G10		1000130949
YB48446	TAG 297	2/05/1994	26/04/1994	2/04/2036	105G10		1000130950
YB48447	TAG 298	2/05/1994	26/04/1994	2/04/2036	105G10		1000130951
YB48448	TAG 299	2/05/1994	26/04/1994	2/04/2036	105G10		1000130952
YB48449	TAG 300	2/05/1994	26/04/1994	2/04/2036	105G10		1000130953
YB48450	TAG 301	2/05/1994	26/04/1994	2/04/2036	105G10		1000130954
YB48451	TAG 302	2/05/1994	26/04/1994	2/04/2036	105G10		1000130955
YB48452	TAG 303	2/05/1994	26/04/1994	2/04/2036	105G10		1000130956
YB48455	TAG 306	2/05/1994	26/04/1994	2/04/2036	105G10		1000130959
YB48456	TAG 307	2/05/1994	26/04/1994	2/04/2036	105G10		1000130960
YB48457	TAG 308	2/05/1994	26/04/1994	2/04/2036	105G10		1000130961
YB48458	TAG 309	2/05/1994	26/04/1994	2/04/2036	105G10		1000130962
YB48459	TAG 310	2/05/1994	26/04/1994	2/04/2036	105G10		1000130963
YB48460	TAG 311	2/05/1994	26/04/1994	2/04/2036	105G10		1000130964
YB48461	TAG 312	2/05/1994	27/04/1994	2/04/2036	105G10		1000130965
YB48464	TAG 315	2/05/1994	27/04/1994	2/04/2036	105G10		1000130968
YB48465	TAG 316	2/05/1994	27/04/1994	2/04/2036	105G10		1000130969
YB48466	TAG 317	2/05/1994	27/04/1994	2/04/2036	105G10		1000130970
YB48467	TAG 318	2/05/1994	27/04/1994	2/04/2036	105G09		1000130971
YB48468	TAG 319	2/05/1994	27/04/1994	2/04/2036	105G09		1000130972
YB48477	TAG 328	2/05/1994	26/04/1994	2/04/2036	105G07		1000130981
YB48478	TAG 329	2/05/1994	26/04/1994	2/04/2036	105G07		1000130982
YB48479	TAG 330	2/05/1994	26/04/1994	2/04/2036	105G07		1000130983
YB48480	TAG 331	2/05/1994	26/04/1994	2/04/2036	105G07		1000130984
YB48481	TAG 332	2/05/1994	26/04/1994	2/04/2036	105G08		1000130985
YB48482	TAG 333	2/05/1994	26/04/1994	2/04/2036	105G08		1000130986
YB48483	TAG 334	2/05/1994	26/04/1994	2/04/2036	105G08		1000130987
YB48484	TAG 335	2/05/1994	26/04/1994	2/04/2036	105G08		1000130988

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YB48485	TAG 336	2/05/1994	26/04/1994	2/04/2036	105G08		1000130989
YB48486	TAG 337	2/05/1994	26/04/1994	2/04/2036	105G08		1000130990
YB48507	TAG 358	2/05/1994	26/04/1994	2/04/2036	105G10		1000131011
YB48508	TAG 359	2/05/1994	26/04/1994	2/04/2036	105G10		1000131012
YB48509	TAG 360	2/05/1994	26/04/1994	2/04/2036	105G10		1000131013
YB48510	TAG 361	2/05/1994	26/04/1994	2/04/2036	105G10		1000131014
YB48511	TAG 362	2/05/1994	26/04/1994	2/04/2036	105G10		1000131015
YB48512	TAG 363	2/05/1994	26/04/1994	2/04/2036	105G10		1000131016
YB48513	TAG 364	2/05/1994	26/04/1994	2/04/2036	105G10		1000131017
YB48514	TAG 365	2/05/1994	26/04/1994	2/04/2036	105G10		1000131018
YB48515	TAG 366	2/05/1994	26/04/1994	2/04/2036	105G10		1000131019
YB48516	TAG 367	2/05/1994	26/04/1994	2/04/2036	105G10		1000131020
YB48517	TAG 368	2/05/1994	27/04/1994	2/04/2036	105G10		1000131021
YB48518	TAG 369	2/05/1994	27/04/1994	2/04/2036	105G10		1000131022
YB48519	TAG 370	2/05/1994	27/04/1994	2/04/2036	105G10		1000131023
YB48520	TAG 371	2/05/1994	27/04/1994	2/04/2036	105G10		1000131024
YB48521	TAG 372	2/05/1994	27/04/1994	2/04/2036	105G10		1000131025
YB48522	TAG 373	2/05/1994	27/04/1994	2/04/2036	105G10		1000131026
YB48523	TAG 374	2/05/1994	27/04/1994	2/04/2036	105G10		1000131027
YB48524	TAG 375	2/05/1994	27/04/1994	2/04/2036	105G10		1000131028
YB48525	TAG 376	2/05/1994	27/04/1994	2/04/2036	105G10		1000131029
YB48526	TAG 377	2/05/1994	27/04/1994	2/04/2036	105G10		1000131030
YB48532	TAG 383	2/05/1994	27/04/1994	2/04/2036	105G10		1000131036
YB48534	TAG 385	2/05/1994	27/04/1994	2/04/2036	105G10		1000131038
YB48535	TAG 386	2/05/1994	27/04/1994	2/04/2036	105G10		1000131039
YB48536	TAG 387	2/05/1994	27/04/1994	2/04/2036	105G10		1000131040
YB48917	EL 1	27/05/1994	17/05/1994	2/04/2028	105G10		1000131167
YB48918	EL 2	27/05/1994	17/05/1994	2/04/2028	105G10		1000131168
YB48919	EL 3	27/05/1994	17/05/1994	2/04/2028	105G10		1000131169
YB48920	EL 4	27/05/1994	17/05/1994	2/04/2028	105G10		1000131170
YB48921	EL 5	27/05/1994	17/05/1994	2/04/2028	105G10		1000131171
YB48922	EL 6	27/05/1994	17/05/1994	2/04/2028	105G10		1000131172
YB48923	EL 7	27/05/1994	17/05/1994	2/04/2028	105G10		1000131173
YB48924	EL 8	27/05/1994	17/05/1994	2/04/2028	105G10		1000131174
YB48925	LY 1	27/05/1994	17/05/1994	2/04/2036	105G10		1000131175
YB48926	LY 2	27/05/1994	17/05/1994	2/04/2036	105G10		1000131176
YB48927	LY 3	27/05/1994	17/05/1994	2/04/2036	105G10		1000131177
YB48928	LY 4	27/05/1994	17/05/1994	2/04/2036	105G10		1000131178
YB48929	LY 5	27/05/1994	17/05/1994	2/04/2036	105G10		1000131179
YB48930	LY 6	27/05/1994	17/05/1994	2/04/2036	105G10		1000131180
YB48931	LY 7	27/05/1994	17/05/1994	2/04/2036	105G10		1000131181
YB48932	LY 8	27/05/1994	17/05/1994	2/04/2036	105G10		1000131182
YB48933	LY 9	27/05/1994	17/05/1994	2/04/2036	105G10		1000131183
YB48934	LY 10	27/05/1994	17/05/1994	2/04/2036	105G10		1000131184
YB48935	LY 11	27/05/1994	17/05/1994	2/04/2036	105G10		1000131185
YB48936	LY 12	27/05/1994	17/05/1994	2/04/2036	105G10		1000131186

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YB48937	LY 13	27/05/1994	17/05/1994	2/04/2036	105G10		1000131187
YB48938	LY 14	27/05/1994	17/05/1994	2/04/2036	105G10		1000131188
YB48939	LY 15	27/05/1994	17/05/1994	2/04/2036	105G10		1000131189
YB48940	TAG 398	27/05/1994	16/05/1994	2/04/2036	105G07		1000131190
YB48941	TAG 399	27/05/1994	16/05/1994	2/04/2036	105G07		1000131191
YB48942	TAG 400	27/05/1994	16/05/1994	2/04/2036	105G07		1000131192
YB48943	TAG 401	27/05/1994	16/05/1994	2/04/2036	105G07		1000131193
YB48944	TAG 402	27/05/1994	16/05/1994	2/04/2036	105G07		1000131194
YB48945	TAG 403	27/05/1994	16/05/1994	2/04/2036	105G07		1000131195
YB48946	TAG 404	27/05/1994	16/05/1994	2/04/2036	105G07		1000131196
YB48947	TAG 405	27/05/1994	16/05/1994	2/04/2036	105G07		1000131197
YB48948	TAG 406	27/05/1994	16/05/1994	2/04/2036	105G07		1000131198
YB48949	TAG 407	27/05/1994	16/05/1994	2/04/2036	105G07		1000131199
YB48950	TAG 408	27/05/1994	16/05/1994	2/04/2036	105G07		1000131200
YB48951	TAG 409	27/05/1994	16/05/1994	2/04/2036	105G07		1000131201
YB48952	TAG 410	27/05/1994	16/05/1994	2/04/2036	105G07		1000131202
YB48953	TAG 411	27/05/1994	16/05/1994	2/04/2036	105G07		1000131203
YB48954	TAG 412	27/05/1994	16/05/1994	2/04/2036	105G07		1000131204
YB48955	TAG 413	27/05/1994	16/05/1994	2/04/2036	105G07		1000131205
YB48956	TAG 414	27/05/1994	16/05/1994	2/04/2036	105G07		1000131206
YB48957	TAG 415	27/05/1994	16/05/1994	2/04/2036	105G07		1000131207
YB48958	TAG 416	27/05/1994	16/05/1994	2/04/2036	105G07		1000131208
YB48959	TAG 417	27/05/1994	16/05/1994	2/04/2036	105G07		1000131209
YB48960	TAG 418	27/05/1994	16/05/1994	2/04/2036	105G07		1000131210
YB48961	TAG 419	27/05/1994	16/05/1994	2/04/2036	105G07		1000131211
YB48962	TAG 420	27/05/1994	16/05/1994	2/04/2036	105G07		1000131212
YB48963	TAG 421	27/05/1994	16/05/1994	2/04/2036	105G07		1000131213
YB48964	TAG 422	27/05/1994	16/05/1994	2/04/2036	105G07		1000131214
YB48965	TAG 423	27/05/1994	16/05/1994	2/04/2036	105G07		1000131215
YB48966	TAG 424	27/05/1994	16/05/1994	2/04/2036	105G07		1000131216
YB48967	TAG 425	27/05/1994	16/05/1994	2/04/2036	105G07		1000131217
YB48968	TAG 426	27/05/1994	16/05/1994	2/04/2036	105G07		1000131218
YB48969	TAG 427	27/05/1994	16/05/1994	2/04/2036	105G07		1000131219
YB48970	TAG 428	27/05/1994	16/05/1994	2/04/2036	105G07		1000131220
YB48971	TAG 429	27/05/1994	16/05/1994	2/04/2036	105G07		1000131221
YB48972	TAG 430	27/05/1994	16/05/1994	2/04/2036	105G07		1000131222
YB48973	TAG 431	27/05/1994	16/05/1994	2/04/2036	105G07		1000131223
YB48974	TAG 432	27/05/1994	16/05/1994	2/04/2036	105G07		1000131224
YB48975	TAG 433	27/05/1994	16/05/1994	2/04/2036	105G07		1000131225
YB48976	TAG 434	27/05/1994	16/05/1994	2/04/2036	105G07		1000131226
YB48977	TAG 435	27/05/1994	16/05/1994	2/04/2036	105G07		1000131227
YB48978	TAG 436	27/05/1994	16/05/1994	2/04/2036	105G07		1000131228
YB48979	TAG 437	27/05/1994	16/05/1994	2/04/2036	105G07		1000131229
YB48980	TAG 438	27/05/1994	16/05/1994	2/04/2036	105G07		1000131230
YB48981	TAG 439	27/05/1994	16/05/1994	2/04/2036	105G07		1000131231
YB48982	TAG 440	27/05/1994	16/05/1994	2/04/2036	105G07		1000131232

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YB48983	TAG 441	27/05/1994	16/05/1994	2/04/2036	105G07		1000131233
YB48984	TAG 442	27/05/1994	16/05/1994	2/04/2036	105G07		1000131234
YB48985	TAG 443	27/05/1994	16/05/1994	2/04/2036	105G07		1000131235
YB48986	TAG 444	27/05/1994	16/05/1994	2/04/2036	105G07		1000131236
YB48987	TAG 445	27/05/1994	16/05/1994	2/04/2036	105G07		1000131237
YB48988	TAG 446	27/05/1994	16/05/1994	2/04/2036	105G07		1000131238
YB48989	TAG 447	27/05/1994	16/05/1994	2/04/2036	105G07		1000131239
YB48990	TAG 448	27/05/1994	16/05/1994	2/04/2036	105G07		1000131240
YB48991	TAG 449	27/05/1994	16/05/1994	2/04/2036	105G07		1000131241
YB48992	TAG 450	27/05/1994	16/05/1994	2/04/2036	105G07		1000131242
YB48993	TAG 451	27/05/1994	16/05/1994	2/04/2036	105G07		1000131243
YB48994	TAG 452	27/05/1994	16/05/1994	2/04/2036	105G07		1000131244
YB48995	TAG 453	27/05/1994	16/05/1994	2/04/2036	105G07		1000131245
YB48996	TAG 454	27/05/1994	16/05/1994	2/04/2036	105G07		1000131246
YB48997	TAG 455	27/05/1994	16/05/1994	2/04/2036	105G07		1000131247
YB48998	TAG 456	27/05/1994	16/05/1994	2/04/2036	105G07		1000131248
YB48999	TAG 457	27/05/1994	16/05/1994	2/04/2036	105G07		1000131249
YB49000	TAG 458	27/05/1994	16/05/1994	2/04/2036	105G07		1000131250
YB49001	TAG 459	27/05/1994	16/05/1994	2/04/2036	105G07		1000131251
YB49002	TAG 460	27/05/1994	16/05/1994	2/04/2036	105G07		1000131252
YB49003	TAG 461	27/05/1994	16/05/1994	2/04/2036	105G07		1000131253
YB49565	TAG 489	22/06/1994	1/06/1994	2/04/2036	105G07		1000131815
YB49566	TAG 490	22/06/1994	1/06/1994	2/04/2036	105G07		1000131816
YB49567	TAG 491	22/06/1994	1/06/1994	2/04/2036	105G07		1000131817
YB49568	TAG 492	22/06/1994	1/06/1994	2/04/2036	105G07		1000131818
YB49569	TAG 493	22/06/1994	1/06/1994	2/04/2036	105G07		1000131819
YB49570	TAG 494	22/06/1994	1/06/1994	2/04/2036	105G07		1000131820
YB49571	TAG 495	22/06/1994	1/06/1994	2/04/2036	105G07		1000131821
YB49572	TAG 496	22/06/1994	1/06/1994	2/04/2036	105G07		1000131822
YB49573	TAG 497	22/06/1994	1/06/1994	2/04/2036	105G07		1000131823
YB49574	TAG 498	22/06/1994	1/06/1994	2/04/2036	105G07		1000131824
YB49575	TAG 499	22/06/1994	1/06/1994	2/04/2036	105G07		1000131825
YB49576	TAG 500	22/06/1994	1/06/1994	2/04/2036	105G07		1000131826
YB49577	TAG 501	22/06/1994	1/06/1994	2/04/2036	105G07		1000131827
YB49578	TAG 502	22/06/1994	1/06/1994	2/04/2036	105G07		1000131828
YB49579	TAG 503	22/06/1994	1/06/1994	2/04/2036	105G07		1000131829
YB49580	TAG 504	22/06/1994	1/06/1994	2/04/2036	105G07		1000131830
YB49581	TAG 505	22/06/1994	1/06/1994	2/04/2036	105G07		1000131831
YB49582	TAG 506	22/06/1994	1/06/1994	2/04/2036	105G07		1000131832
YB49583	TAG 507	22/06/1994	1/06/1994	2/04/2036	105G07		1000131833
YB49584	TAG 508	22/06/1994	1/06/1994	2/04/2036	105G07		1000131834
YB49585	TAG 509	22/06/1994	1/06/1994	2/04/2036	105G07		1000131835
YB49586	TAG 510	22/06/1994	1/06/1994	2/04/2036	105G07		1000131836
YB49587	TAG 511	22/06/1994	1/06/1994	2/04/2036	105G07		1000131837
YB49588	TAG 512	22/06/1994	1/06/1994	2/04/2036	105G07		1000131838
YB49589	TAG 513	22/06/1994	1/06/1994	2/04/2036	105G07		1000131839

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YB49590	TAG 514	22/06/1994	1/06/1994	2/04/2036	105G07		1000131840
YB49591	TAG 515	22/06/1994	1/06/1994	2/04/2036	105G07		1000131841
YB49592	TAG 516	22/06/1994	1/06/1994	2/04/2036	105G07		1000131842
YB49654	LIMY 1	22/06/1994	10/06/1994	2/04/2032	105G10		1000131904
YB49655	LIMY 2	22/06/1994	10/06/1994	2/04/2032	105G10		1000131905
YB49656	LIMY 3	22/06/1994	10/06/1994	2/04/2032	105G10		1000131906
YB49657	LIMY 4	22/06/1994	10/06/1994	2/04/2032	105G10		1000131907
YB49658	LIMY 5	22/06/1994	10/06/1994	2/04/2032	105G10		1000131908
YB49659	LIMY 6	22/06/1994	10/06/1994	2/04/2032	105G10		1000131909
YB49660	LIMY 7	22/06/1994	10/06/1994	2/04/2032	105G10		1000131910
YB49661	LIMY 8	22/06/1994	10/06/1994	2/04/2032	105G10		1000131911
YB49662	LIMY 9	22/06/1994	10/06/1994	2/04/2032	105G10		1000131912
YB50436	TAG 559	7/07/1994	23/06/1994	2/04/2036	105G10		1000132686
YB50437	TAG 560	7/07/1994	23/06/1994	2/04/2036	105G10		1000132687
YB50438	TAG 561	7/07/1994	23/06/1994	2/04/2036	105G10		1000132688
YB50439	TAG 562	7/07/1994	23/06/1994	2/04/2036	105G10		1000132689
YB50510	TAG 633	7/07/1994	23/06/1994	2/04/2036	105G10		1000132760
YB50511	TAG 634	7/07/1994	23/06/1994	2/04/2036	105G10		1000132761
YB50512	TAG 635	7/07/1994	23/06/1994	2/04/2036	105G10		1000132762
YB50513	TAG 636	7/07/1994	23/06/1994	2/04/2036	105G10		1000132763
YB50514	TAG 637	7/07/1994	23/06/1994	2/04/2036	105G10		1000132764
YB50515	TAG 638	7/07/1994	23/06/1994	2/04/2036	105G10		1000132765
YB50516	TAG 639	7/07/1994	23/06/1994	2/04/2036	105G10		1000132766
YB50517	TAG 640	7/07/1994	23/06/1994	2/04/2036	105G10		1000132767
YB50518	TAG 641	7/07/1994	23/06/1994	2/04/2036	105G10		1000132768
YB50519	TAG 642	7/07/1994	23/06/1994	2/04/2036	105G10		1000132769
YB50521	TAG 644	7/07/1994	23/06/1994	2/04/2036	105G10		1000132771
YB50589	TAG 712	7/07/1994	23/06/1994	2/04/2036	105G10		1000132839
YB50590	TAG 713	7/07/1994	23/06/1994	2/04/2036	105G10		1000132840
YB50591	TAG 714	7/07/1994	23/06/1994	2/04/2036	105G10		1000132841
YB50592	TAG 715	7/07/1994	23/06/1994	2/04/2036	105G10		1000132842
YB50593	TAG 716	7/07/1994	23/06/1994	2/04/2036	105G10		1000132843
YB50594	TAG 717	7/07/1994	23/06/1994	2/04/2036	105G10		1000132844
YB50595	TAG 718	7/07/1994	23/06/1994	2/04/2036	105G10		1000132845
YB50596	TAG 719	7/07/1994	23/06/1994	2/04/2036	105G10		1000132846
YB50600	TAG 723	7/07/1994	23/06/1994	2/04/2036	105G10		1000132850
YB50602	TAG 725	7/07/1994	23/06/1994	2/04/2036	105G10		1000132852
YB50604	TAG 727	7/07/1994	23/06/1994	2/04/2036	105G10		1000132854
YB50606	TAG 729	7/07/1994	23/06/1994	2/04/2036	105G10		1000132856
YB50607	TAG 730	7/07/1994	23/06/1994	2/04/2036	105G10		1000132857
YB50608	TAG 731	7/07/1994	23/06/1994	2/04/2036	105G10		1000132858
YB50609	TAG 732	7/07/1994	23/06/1994	2/04/2036	105G10		1000132859
YB50611	TAG 734	7/07/1994	23/06/1994	2/04/2036	105G10		1000132861
YB50613	TAG 736	7/07/1994	24/06/1994	2/04/2036	105G10		1000132863
YB50615	TAG 738	7/07/1994	24/06/1994	2/04/2036	105G10		1000132865

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YB50617	TAG 740	7/07/1994	24/06/1994	2/04/2036	105G10	Full Quartz fraction (25+ acres)	1000132867
YB50623	TAG 746	7/07/1994	24/06/1994	2/04/2032	105G10		1000132873
YB50625	TAG 748	7/07/1994	24/06/1994	2/04/2032	105G10		1000132875
YB50627	TAG 750	7/07/1994	24/06/1994	2/04/2032	105G10		1000132877
YB50629	TAG 752	7/07/1994	24/06/1994	2/04/2032	105G10		1000132879
YB50631	TAG 754	7/07/1994	25/06/1994	2/04/2032	105G10		1000132881
YB50633	TAG 756	7/07/1994	25/06/1994	2/04/2032	105G10		1000132883
YB50635	TAG 758	7/07/1994	25/06/1994	2/04/2032	105G10		1000132885
YB50637	TAG 760	7/07/1994	26/06/1994	2/04/2032	105G10		1000132887
YB50639	TAG 762	7/07/1994	26/06/1994	2/04/2032	105G10		1000132889
YB50641	TAG 764	7/07/1994	26/06/1994	2/04/2032	105G10		1000132891
YB50643	TAG 766	7/07/1994	26/06/1994	2/04/2032	105G10		1000132893
YB50664	TAG 787	7/07/1994	23/06/1994	2/04/2036	105G10		1000132914
YB50665	TAG 788	7/07/1994	23/06/1994	2/04/2036	105G10		1000132915
YB50666	TAG 789	7/07/1994	23/06/1994	2/04/2036	105G10		1000132916
YB50667	TAG 790	7/07/1994	23/06/1994	2/04/2036	105G10		1000132917
YB50670	TAG 793	7/07/1994	23/06/1994	2/04/2036	105G10		1000132920
YB50671	TAG 794	7/07/1994	23/06/1994	2/04/2036	105G10		1000132921
YB50692	TAG 815	7/07/1994	24/06/1994	2/04/2036	105G10	Full Quartz fraction (25+ acres)	1000132942
YB50716	TAG 839	7/07/1994	26/06/1994	2/04/2032	105G10		1000132966
YB50718	TAG 841	7/07/1994	26/06/1994	2/04/2032	105G10		1000132968
YB50720	TAG 843	7/07/1994	26/06/1994	2/04/2032	105G10		1000132970
YB50722	TAG 845	7/07/1994	26/06/1994	2/04/2032	105G10		1000132972
YB50723	TAG 846	7/07/1994	26/06/1994	2/04/2032	105G10		1000132973
YB50725	TAG 848	7/07/1994	26/06/1994	2/04/2032	105G10		1000132975
YB50734	TAG 857	7/07/1994	23/06/1994	2/04/2036	105G10		1000132984
YB50735	TAG 858	7/07/1994	23/06/1994	2/04/2036	105G10		1000132985
YB50736	TAG 859	7/07/1994	23/06/1994	2/04/2036	105G10		1000132986
YB50737	TAG 860	7/07/1994	23/06/1994	2/04/2036	105G10		1000132987
YB50740	TAG 863	7/07/1994	23/06/1994	2/04/2036	105G10		1000132990
YB50741	TAG 864	7/07/1994	23/06/1994	2/04/2036	105G10		1000132991
YB50798	TAG 921	7/07/1994	26/06/1994	2/04/2032	105G10		1000133048
YB50800	TAG 923	7/07/1994	26/06/1994	2/04/2032	105G10		1000133050
YB50801	TAG 924	7/07/1994	26/06/1994	2/04/2032	105G10		1000133051
YB50802	TAG 925	7/07/1994	26/06/1994	2/04/2032	105G10		1000133052
YB50805	TAG 928	7/07/1994	23/06/1994	2/04/2036	105G10		1000133055
YB50806	TAG 929	7/07/1994	23/06/1994	2/04/2036	105G10		1000133056
YB50807	TAG 930	7/07/1994	23/06/1994	2/04/2036	105G10		1000133057
YB50808	TAG 931	7/07/1994	23/06/1994	2/04/2036	105G10		1000133058
YB51214	TAG 1057	19/07/1994	27/06/1994	2/04/2036	105G07		1000133464
YB51215	TAG 1058	19/07/1994	27/06/1994	2/04/2036	105G07		1000133465
YB51216	TAG 1059	19/07/1994	27/06/1994	2/04/2036	105G07		1000133466
YB51217	TAG 1060	19/07/1994	27/06/1994	2/04/2036	105G07		1000133467
YB51218	TAG 1061	19/07/1994	27/06/1994	2/04/2036	105G07		1000133468

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YB51219	TAG 1062	19/07/1994	27/06/1994	2/04/2036	105G07		1000133469
YB51220	TAG 1063	19/07/1994	27/06/1994	2/04/2036	105G07		1000133470
YB51221	TAG 1064	19/07/1994	27/06/1994	2/04/2036	105G07		1000133471
YB51222	TAG 1065	19/07/1994	27/06/1994	2/04/2036	105G07		1000133472
YB51223	TAG 1066	19/07/1994	27/06/1994	2/04/2036	105G07		1000133473
YB51224	TAG 1067	19/07/1994	27/06/1994	2/04/2036	105G07		1000133474
YB51225	TAG 1068	19/07/1994	27/06/1994	2/04/2036	105G07		1000133475
YB51226	TAG 1069	19/07/1994	27/06/1994	2/04/2036	105G07		1000133476
YB51227	TAG 1070	19/07/1994	27/06/1994	2/04/2036	105G07		1000133477
YB51228	TAG 1071	19/07/1994	27/06/1994	2/04/2036	105G07		1000133478
YB51229	TAG 1072	19/07/1994	27/06/1994	2/04/2036	105G07		1000133479
YB51230	TAG 1073	19/07/1994	27/06/1994	2/04/2036	105G07		1000133480
YB51231	TAG 1074	19/07/1994	27/06/1994	2/04/2036	105G07	Full Quartz fraction (25+ acres)	1000133481
YB51232	TAG 1075	19/07/1994	27/06/1994	2/04/2036	105G07		1000133482
YB51233	TAG 1076	19/07/1994	27/06/1994	2/04/2036	105G07	Full Quartz fraction (25+ acres)	1000133483
YB51234	TAG 1077	19/07/1994	27/06/1994	2/04/2036	105G07	Full Quartz fraction (25+ acres)	1000133484
YB51235	TAG 1078	19/07/1994	27/06/1994	2/04/2036	105G07		1000133485
YB51236	TAG 1079	19/07/1994	27/06/1994	2/04/2036	105G07	Full Quartz fraction (25+ acres)	1000133486
YB51237	TAG 1080	19/07/1994	27/06/1994	2/04/2036	105G07		1000133487
YB51238	TAG 1081	19/07/1994	27/06/1994	2/04/2036	105G07		1000133488
YB51239	TAG 1082	19/07/1994	27/06/1994	2/04/2036	105G07		1000133489
YB51240	TAG 1083	19/07/1994	27/06/1994	2/04/2036	105G07		1000133490
YB51241	TAG 1084	19/07/1994	27/06/1994	2/04/2036	105G07		1000133491
YB51242	TAG 1085	19/07/1994	27/06/1994	2/04/2036	105G07		1000133492
YB51243	TAG 1086	19/07/1994	27/06/1994	2/04/2036	105G07		1000133493
YB51244	TAG 1087	19/07/1994	27/06/1994	2/04/2036	105G07		1000133494
YB51245	TAG 1088	19/07/1994	27/06/1994	2/04/2036	105G07		1000133495
YB51246	TAG 1089	19/07/1994	27/06/1994	2/04/2036	105G07		1000133496
YB51247	TAG 1090	19/07/1994	27/06/1994	2/04/2036	105G07		1000133497
YB51248	TAG 1091	19/07/1994	27/06/1994	2/04/2036	105G07		1000133498
YB51249	TAG 1092	19/07/1994	27/06/1994	2/04/2036	105G07		1000133499
YB51250	TAG 1093	19/07/1994	27/06/1994	2/04/2036	105G07		1000133500
YB51251	TAG 1094	19/07/1994	27/06/1994	2/04/2036	105G07		1000133501
YB51252	TAG 1095	19/07/1994	27/06/1994	2/04/2036	105G07		1000133502
YB51253	TAG 1096	19/07/1994	27/06/1994	2/04/2038	105G07		1000133503
YB51254	TAG 1097	19/07/1994	27/06/1994	2/04/2036	105G07		1000133504
YB51255	TAG 1098	19/07/1994	27/06/1994	2/04/2038	105G07		1000133505
YB51256	TAG 1099	19/07/1994	27/06/1994	2/04/2036	105G07		1000133506
YB51257	TAG 1100	19/07/1994	27/06/1994	2/04/2038	105G07		1000133507
YB51258	TAG 1101	19/07/1994	27/06/1994	2/04/2036	105G07		1000133508
YB51259	TAG 1102	19/07/1994	27/06/1994	2/04/2038	105G07		1000133509
YB51260	TAG 1103	19/07/1994	27/06/1994	2/04/2036	105G07		1000133510

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YB51261	TAG 1104	19/07/1994	27/06/1994	2/04/2036	105G07		1000133511
YB51262	TAG 1105	19/07/1994	27/06/1994	2/04/2038	105G07	Full Quartz fraction (25+ acres)	1000133512
YB51263	TAG 1106	19/07/1994	27/06/1994	2/04/2038	105G07		1000133513
YB51264	TAG 1107	19/07/1994	27/06/1994	2/04/2038	105G07		1000133514
YB51265	TAG 1108	19/07/1994	27/06/1994	2/04/2038	105G07		1000133515
YB51266	TAG 1109	19/07/1994	27/06/1994	2/04/2038	105G07		1000133516
YB51267	TAG 1110	19/07/1994	27/06/1994	2/04/2038	105G07		1000133517
YB51268	TAG 1111	19/07/1994	27/06/1994	2/04/2038	105G07		1000133518
YB51269	TAG 1112	19/07/1994	27/06/1994	2/04/2038	105G07		1000133519
YB51270	TAG 1113	19/07/1994	27/06/1994	2/04/2036	105G07		1000133520
YB51271	TAG 1114	19/07/1994	27/06/1994	2/04/2036	105G07		1000133521
YB55325	TAG 1449	31/08/1994	18/08/1994	2/04/2036	105G07		1000134575
YB55326	TAG 1450	31/08/1994	18/08/1994	2/04/2036	105G07		1000134576
YB55327	TAG 1451	31/08/1994	18/08/1994	2/04/2036	105G07		1000134577
YB55328	TAG 1452	31/08/1994	18/08/1994	2/04/2036	105G07		1000134578
YB55329	TAG 1453	31/08/1994	18/08/1994	2/04/2036	105G07		1000134579
YB55330	TAG 1454	31/08/1994	18/08/1994	2/04/2036	105G07		1000134580
YB55331	TAG 1455	31/08/1994	18/08/1994	2/04/2036	105G07		1000134581
YB55332	TAG 1456	31/08/1994	18/08/1994	2/04/2036	105G07		1000134582
YB55333	TAG 1457	31/08/1994	18/08/1994	2/04/2036	105G07		1000134583
YB55334	TAG 1458	31/08/1994	18/08/1994	2/04/2036	105G07		1000134584
YB55335	TAG 1459	31/08/1994	18/08/1994	2/04/2036	105G07		1000134585
YB55336	TAG 1460	31/08/1994	18/08/1994	2/04/2036	105G07		1000134586
YB55337	TAG 1461	31/08/1994	18/08/1994	2/04/2036	105G07		1000134587
YB55338	TAG 1462	31/08/1994	18/08/1994	2/04/2036	105G07		1000134588
YB55339	TAG 1463	31/08/1994	18/08/1994	2/04/2036	105G07		1000134589
YB55340	TAG 1464	31/08/1994	18/08/1994	2/04/2036	105G07		1000134590
YB55341	TAG 1465	31/08/1994	18/08/1994	2/04/2036	105G07		1000134591
YB55342	TAG 1466	31/08/1994	18/08/1994	2/04/2036	105G07		1000134592
YB55343	TAG 1467	31/08/1994	18/08/1994	2/04/2036	105G07		1000134593
YB55344	TAG 1468	31/08/1994	18/08/1994	2/04/2036	105G07		1000134594
YB55346	TAG 1505	31/08/1994	23/08/1994	2/04/2036	105G08		1000134596
YB55348	TAG 1507	31/08/1994	23/08/1994	2/04/2036	105G08		1000134598
YB55350	TAG 1509	31/08/1994	23/08/1994	2/04/2036	105G08		1000134600
YB55377	TAG 1538	31/08/1994	23/08/1994	2/04/2036	105G08		1000134627
YB55899	TAG 1469	6/09/1994	18/08/1994	2/04/2036	105G08		1000135149
YB55900	TAG 1470	6/09/1994	18/08/1994	2/04/2036	105G08		1000135150
YB55901	TAG 1471	6/09/1994	18/08/1994	2/04/2036	105G08		1000135151
YB55902	TAG 1472	6/09/1994	18/08/1994	2/04/2036	105G08		1000135152
YB55903	TAG 1473	6/09/1994	18/08/1994	2/04/2036	105G08		1000135153
YB55904	TAG 1474	6/09/1994	18/08/1994	2/04/2036	105G08		1000135154
YB55905	TAG 1475	6/09/1994	18/08/1994	2/04/2036	105G08		1000135155
YB55906	TAG 1476	6/09/1994	18/08/1994	2/04/2036	105G08		1000135156
YB55907	TAG 1477	6/09/1994	18/08/1994	2/04/2036	105G08		1000135157
YB55908	TAG 1478	6/09/1994	18/08/1994	2/04/2036	105G08		1000135158

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YB55909	TAG 1479	6/09/1994	18/08/1994	2/04/2036	105G08		1000135159
YB55910	TAG 1480	6/09/1994	18/08/1994	2/04/2036	105G08		1000135160
YB55911	TAG 1481	6/09/1994	18/08/1994	2/04/2036	105G08		1000135161
YB55912	TAG 1482	6/09/1994	18/08/1994	2/04/2036	105G08		1000135162
YB55913	TAG 1483	6/09/1994	18/08/1994	2/04/2036	105G08		1000135163
YB55914	TAG 1484	6/09/1994	18/08/1994	2/04/2036	105G08		1000135164
YB55915	TAG 1485	6/09/1994	18/08/1994	2/04/2036	105G08		1000135165
YB55916	TAG 1486	6/09/1994	18/08/1994	2/04/2036	105G08		1000135166
YB55917	TAG 1487	6/09/1994	18/08/1994	2/04/2036	105G08		1000135167
YB55918	TAG 1488	6/09/1994	18/08/1994	2/04/2036	105G08		1000135168
YB55919	TAG 1489	6/09/1994	18/08/1994	2/04/2036	105G08		1000135169
YB55920	TAG 1490	6/09/1994	18/08/1994	2/04/2036	105G08		1000135170
YB55921	TAG 1491	6/09/1994	18/08/1994	2/04/2036	105G08		1000135171
YB55922	TAG 1492	6/09/1994	18/08/1994	2/04/2036	105G08		1000135172
YB55923	TAG 1493	6/09/1994	18/08/1994	2/04/2036	105G08		1000135173
YB55924	TAG 1494	6/09/1994	18/08/1994	2/04/2036	105G08		1000135174
YB55925	TAG 1495	6/09/1994	18/08/1994	2/04/2036	105G08		1000135175
YB55926	TAG 1496	6/09/1994	22/08/1994	2/04/2036	105G08		1000135176
YB55927	TAG 1497	6/09/1994	22/08/1994	2/04/2036	105G08		1000135177
YB55928	TAG 1498	6/09/1994	22/08/1994	2/04/2036	105G08		1000135178
YB55929	TAG 1499	6/09/1994	22/08/1994	2/04/2036	105G08		1000135179
YB55930	TAG 1500	6/09/1994	22/08/1994	2/04/2036	105G08		1000135180
YB55931	TAG 1501	6/09/1994	22/08/1994	2/04/2036	105G08		1000135181
YB55934	TAG 1536	6/09/1994	22/08/1994	2/04/2036	105G07		1000135184
YB55935	TAG 1537	6/09/1994	22/08/1994	2/04/2036	105G08		1000135185
YB55936	TAG 1539	6/09/1994	22/08/1994	2/04/2036	105G07		1000135186
YB55937	TAG 1540	6/09/1994	22/08/1994	2/04/2036	105G07		1000135187
YB55938	TAG 1541	6/09/1994	22/08/1994	2/04/2036	105G07		1000135188
YB56713	TAG 1544	30/11/1994	16/11/1994	2/04/2036	105G07		1000135963
YB56714	TAG 1545	30/11/1994	16/11/1994	2/04/2036	105G07		1000135964
YB56715	TAG 1546	30/11/1994	16/11/1994	2/04/2036	105G07		1000135965
YB56716	TAG 1547	30/11/1994	16/11/1994	2/04/2036	105G07		1000135966
YB56717	TAG 1548	30/11/1994	16/11/1994	2/04/2036	105G07		1000135967
YB56718	TAG 1549	30/11/1994	16/11/1994	2/04/2036	105G07		1000135968
YB56719	TAG 1550	30/11/1994	16/11/1994	2/04/2036	105G07		1000135969
YB56720	TAG 1551	30/11/1994	16/11/1994	2/04/2036	105G07		1000135970
YB56721	TAG 1552	30/11/1994	16/11/1994	2/04/2036	105G07		1000135971
YB56722	TAG 1553	30/11/1994	16/11/1994	2/04/2036	105G07		1000135972
YB56729	TAG 1560	30/11/1994	16/11/1994	2/04/2036	105G07		1000135979
YB56730	TAG 1561	30/11/1994	16/11/1994	2/04/2036	105G07		1000135980
YB56731	TAG 1562	30/11/1994	16/11/1994	2/04/2036	105G07		1000135981
YB56732	TAG 1563	30/11/1994	16/11/1994	2/04/2036	105G07		1000135982
YB56733	TAG 1564	30/11/1994	16/11/1994	2/04/2036	105G07		1000135983
YB56734	TAG 1565	30/11/1994	16/11/1994	2/04/2036	105G07		1000135984
YB56735	TAG 1566	30/11/1994	16/11/1994	2/04/2036	105G07		1000135985
YB56736	TAG 1567	30/11/1994	16/11/1994	2/04/2036	105G07		1000135986

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YB56737	TAG 1568	30/11/1994	16/11/1994	2/04/2036	105G07		1000135987
YB56738	TAG 1569	30/11/1994	16/11/1994	2/04/2036	105G07		1000135988
YB62677	ON 21	2/10/1995	14/09/1995	2/04/2036	105G08		1000140927
YB62678	ON 22	2/10/1995	14/09/1995	2/04/2036	105G08		1000140928
YB62679	ON 23	2/10/1995	14/09/1995	2/04/2036	105G08		1000140929
YB62680	ON 24	2/10/1995	14/09/1995	2/04/2036	105G08		1000140930
YB62681	ON 25	2/10/1995	14/09/1995	2/04/2036	105G08		1000140931
YB62682	ON 26	2/10/1995	14/09/1995	2/04/2036	105G08		1000140932
YB62683	ON 27	2/10/1995	14/09/1995	2/04/2036	105G08		1000140933
YB62684	ON 28	2/10/1995	14/09/1995	2/04/2036	105G08		1000140934
YB62685	ON 29	2/10/1995	14/09/1995	2/04/2036	105G08		1000140935
YB62686	ON 30	2/10/1995	14/09/1995	2/04/2036	105G08		1000140936
YB62687	ON 31	2/10/1995	14/09/1995	2/04/2036	105G08		1000140937
YB62688	ON 32	2/10/1995	14/09/1995	2/04/2036	105G08		1000140938
YB62689	ON 33	2/10/1995	14/09/1995	2/04/2036	105G08		1000140939
YB62690	ON 34	2/10/1995	14/09/1995	2/04/2036	105G08		1000140940
YB62691	ON 35	2/10/1995	14/09/1995	2/04/2036	105G08		1000140941
YB62692	ON 36	2/10/1995	14/09/1995	2/04/2036	105G08		1000140942
YB62693	ON 37	2/10/1995	14/09/1995	2/04/2036	105G08		1000140943
YB62694	ON 38	2/10/1995	14/09/1995	2/04/2036	105G08		1000140944
YB62695	ON 39	2/10/1995	17/09/1995	2/04/2036	105G08		1000140945
YB62696	ON 40	2/10/1995	17/09/1995	2/04/2036	105G08		1000140946
YB62697	ON 41	2/10/1995	17/09/1995	2/04/2036	105G08	Full Quartz fraction (25+ acres)	1000140947
YB62698	ON 42	2/10/1995	14/09/1995	2/04/2036	105G08		1000140948
YB62699	ON 43	2/10/1995	14/09/1995	2/04/2036	105G08		1000140949
YB62700	ON 44	2/10/1995	14/09/1995	2/04/2036	105G08		1000140950
YB62701	ON 45	2/10/1995	14/09/1995	2/04/2036	105G08		1000140951
YB62702	ON 46	2/10/1995	14/09/1995	2/04/2036	105G08		1000140952
YB62703	ON 47	2/10/1995	14/09/1995	2/04/2036	105G08		1000140953
YB62704	ON 48	2/10/1995	14/09/1995	2/04/2036	105G08		1000140954
YB62705	ON 49	2/10/1995	14/09/1995	2/04/2036	105G08		1000140955
YB62706	ON 50	2/10/1995	14/09/1995	2/04/2036	105G08		1000140956
YB62707	ON 51	2/10/1995	14/09/1995	2/04/2036	105G08		1000140957
YB62708	ON 52	2/10/1995	14/09/1995	2/04/2036	105G08		1000140958
YB62709	ON 53	2/10/1995	14/09/1995	2/04/2036	105G08		1000140959
YB62710	ON 54	2/10/1995	14/09/1995	2/04/2036	105G08		1000140960
YB62711	ON 55	2/10/1995	14/09/1995	2/04/2036	105G08		1000140961
YB62712	ON 56	2/10/1995	14/09/1995	2/04/2036	105G08		1000140962
YB62713	ON 57	2/10/1995	14/09/1995	2/04/2036	105G08		1000140963
YB62714	ON 58	2/10/1995	14/09/1995	2/04/2036	105G08		1000140964
YB62715	ON 59	2/10/1995	14/09/1995	2/04/2036	105G08		1000140965
YB62716	ON 60	2/10/1995	14/09/1995	2/04/2036	105G08		1000140966
YB62717	ON 61	2/10/1995	14/09/1995	2/04/2036	105G08		1000140967
YB62718	ON 62	2/10/1995	14/09/1995	2/04/2036	105G08		1000140968
YB62719	ON 63	2/10/1995	14/09/1995	2/04/2036	105G08		1000140969

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YB62720	ON 64	2/10/1995	14/09/1995	2/04/2036	105G08		1000140970
YB62721	ON 65	2/10/1995	14/09/1995	2/04/2036	105G08		1000140971
YB62722	ON 66	2/10/1995	14/09/1995	2/04/2036	105G08		1000140972
YB62723	ON 67	2/10/1995	14/09/1995	2/04/2036	105G08		1000140973
YB62724	ON 68	2/10/1995	14/09/1995	2/04/2036	105G08		1000140974
YB62725	ON 69	2/10/1995	14/09/1995	2/04/2036	105G08		1000140975
YB62726	ON 70	2/10/1995	14/09/1995	2/04/2036	105G08		1000140976
YB62727	ON 71	2/10/1995	14/09/1995	2/04/2036	105G08		1000140977
YB62728	ON 72	2/10/1995	14/09/1995	2/04/2036	105G08		1000140978
YB62729	ON 73	2/10/1995	14/09/1995	2/04/2036	105G08		1000140979
YB62730	ON 74	2/10/1995	14/09/1995	2/04/2036	105G08		1000140980
YB62731	ON 75	2/10/1995	14/09/1995	2/04/2036	105G08		1000140981
YB62732	ON 76	2/10/1995	14/09/1995	2/04/2036	105G08		1000140982
YB62733	ON 77	2/10/1995	14/09/1995	2/04/2036	105G08		1000140983
YB62734	ON 78	2/10/1995	14/09/1995	2/04/2036	105G08		1000140984
YB62735	ON 79	2/10/1995	14/09/1995	2/04/2036	105G08		1000140985
YB62736	ON 80	2/10/1995	14/09/1995	2/04/2036	105G08		1000140986
YB62737	ON 81	2/10/1995	14/09/1995	2/04/2036	105G08		1000140987
YB62738	ON 82	2/10/1995	14/09/1995	2/04/2036	105G08		1000140988
YB62739	ON 83	2/10/1995	14/09/1995	2/04/2036	105G08		1000140989
YB62740	ON 84	2/10/1995	14/09/1995	2/04/2036	105G08		1000140990
YB62741	ON 85	2/10/1995	14/09/1995	2/04/2036	105G08		1000140991
YB62742	ON 86	2/10/1995	14/09/1995	2/04/2036	105G08		1000140992
YB62743	ON 87	2/10/1995	14/09/1995	2/04/2036	105G08		1000140993
YB62744	ON 88	2/10/1995	14/09/1995	2/04/2036	105G08		1000140994
YB62745	ON 89	2/10/1995	14/09/1995	2/04/2036	105G08		1000140995
YB62746	ON 90	2/10/1995	14/09/1995	2/04/2036	105G08		1000140996
YB62747	ON 91	2/10/1995	14/09/1995	2/04/2036	105G08		1000140997
YB62748	ON 92	2/10/1995	14/09/1995	2/04/2036	105G08		1000140998
YB62749	ON 93	2/10/1995	14/09/1995	2/04/2036	105G08		1000140999
YB62750	ON 94	2/10/1995	14/09/1995	2/04/2036	105G08		1000141000
YB62751	ON 95	2/10/1995	14/09/1995	2/04/2036	105G08		1000141001
YB62752	ON 96	2/10/1995	14/09/1995	2/04/2036	105G08		1000141002
YB62753	ON 97	2/10/1995	14/09/1995	2/04/2036	105G08		1000141003
YB62754	ON 98	2/10/1995	14/09/1995	2/04/2036	105G08		1000141004
YB62755	ON 99	2/10/1995	14/09/1995	2/04/2036	105G08		1000141005
YB62756	ON 100	2/10/1995	14/09/1995	2/04/2036	105G08		1000141006
YB62757	ON 101	2/10/1995	14/09/1995	2/04/2036	105G08		1000141007
YB62760	ON 104	2/10/1995	14/09/1995	2/04/2036	105G08		1000141010
YB62761	ON 105	2/10/1995	14/09/1995	2/04/2036	105G08		1000141011
YB62762	ON 106	2/10/1995	14/09/1995	2/04/2036	105G08		1000141012
YB62763	ON 107	2/10/1995	14/09/1995	2/04/2036	105G08		1000141013
YB62764	ON 108	2/10/1995	14/09/1995	2/04/2036	105G08		1000141014
YB62765	ON 109	2/10/1995	14/09/1995	2/04/2036	105G08		1000141015
YB62766	ON 110	2/10/1995	14/09/1995	2/04/2036	105G08		1000141016
YB62767	ON 111	2/10/1995	14/09/1995	2/04/2036	105G08		1000141017

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YB62768	ON 112	2/10/1995	14/09/1995	2/04/2036	105G08		1000141018
YB62769	ON 113	2/10/1995	14/09/1995	2/04/2036	105G08		1000141019
YB62772	ON 116	2/10/1995	14/09/1995	2/04/2036	105G08		1000141022
YB62773	ON 117	2/10/1995	14/09/1995	2/04/2036	105G08		1000141023
YB62774	ON 118	2/10/1995	14/09/1995	2/04/2036	105G08		1000141024
YB62775	ON 119	2/10/1995	14/09/1995	2/04/2036	105G08		1000141025
YB62776	ON 120	2/10/1995	14/09/1995	2/04/2036	105G08		1000141026
YB62777	ON 121	2/10/1995	14/09/1995	2/04/2036	105G08		1000141027
YB62778	ON 122	2/10/1995	14/09/1995	2/04/2036	105G08		1000141028
YB62779	ON 123	2/10/1995	14/09/1995	2/04/2036	105G08		1000141029
YB62780	ON 124	2/10/1995	14/09/1995	2/04/2036	105G08		1000141030
YB62781	ON 125	2/10/1995	14/09/1995	2/04/2036	105G08		1000141031
YB62816	ON 162	2/10/1995	16/09/1995	2/04/2036	105G08		1000141066
YB62817	ON 163	2/10/1995	16/09/1995	2/04/2036	105G08		1000141067
YB62818	ON 164	2/10/1995	16/09/1995	2/04/2036	105G08		1000141068
YB62819	ON 165	2/10/1995	16/09/1995	2/04/2036	105G08		1000141069
YB62820	ON 166	2/10/1995	16/09/1995	2/04/2036	105G08		1000141070
YB62821	ON 167	2/10/1995	16/09/1995	2/04/2036	105G08		1000141071
YB62822	ON 168	2/10/1995	16/09/1995	2/04/2036	105G08		1000141072
YB62823	ON 169	2/10/1995	16/09/1995	2/04/2036	105G08		1000141073
YB62824	ON 170	2/10/1995	16/09/1995	2/04/2036	105G08		1000141074
YB62825	ON 171	2/10/1995	16/09/1995	2/04/2036	105G08		1000141075
YB62826	ON 172	2/10/1995	16/09/1995	2/04/2036	105G08		1000141076
YB62827	ON 173	2/10/1995	16/09/1995	2/04/2036	105G08		1000141077
YB62828	ON 174	2/10/1995	16/09/1995	2/04/2036	105G08		1000141078
YB62830	ON 176	2/10/1995	16/09/1995	2/04/2036	105G08		1000141080
YB62832	ON 178	2/10/1995	16/09/1995	2/04/2036	105G08		1000141082
YB62834	ON 180	2/10/1995	16/09/1995	2/04/2036	105G08		1000141084
YB62851	ON 197	2/10/1995	16/09/1995	2/04/2036	105G08		1000141101
YB62852	ON 198	2/10/1995	16/09/1995	2/04/2036	105G08		1000141102
YB62853	ON 199	2/10/1995	16/09/1995	2/04/2036	105G08		1000141103
YB62854	ON 200	2/10/1995	16/09/1995	2/04/2036	105G08		1000141104
YB62855	ON 201	2/10/1995	16/09/1995	2/04/2036	105G08		1000141105
YB62856	ON 202	2/10/1995	16/09/1995	2/04/2036	105G08		1000141106
YB62857	ON 203	2/10/1995	16/09/1995	2/04/2036	105G08		1000141107
YB62859	ON 205	2/10/1995	17/09/1995	2/04/2036	105G08		1000141109
YB85276	KZK 1	12/07/1996	1/07/1996	2/04/2036	105G07	Partial Quartz fraction (<25 acres)	1000156726
YB85277	KZK 2	12/07/1996	1/07/1996	2/04/2036	105G07	Partial Quartz fraction (<25 acres)	1000156727
YB85278	KZK 3	12/07/1996	1/07/1996	2/04/2036	105G07	Partial Quartz fraction (<25 acres)	1000156728
YB85279	KZK 4	12/07/1996	1/07/1996	2/04/2036	105G07	Partial Quartz fraction (<25 acres)	1000156729
YB85280	KZK 5	12/07/1996	1/07/1996	2/04/2036	105G07	Partial Quartz fraction (<25 acres)	1000156730

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YB85281	KZK 6	12/07/1996	1/07/1996	2/04/2036	105G07	Partial Quartz fraction (<25 acres)	1000156731
YB85282	KZK 7	12/07/1996	1/07/1996	2/04/2036	105G07	Partial Quartz fraction (<25 acres)	1000156732
YB85283	KZK 8	12/07/1996	1/07/1996	2/04/2036	105G07	Partial Quartz fraction (<25 acres)	1000156733
YB85284	KZK 9	12/07/1996	1/07/1996	2/04/2036	105G07	Partial Quartz fraction (<25 acres)	1000156734
YB85285	KZK 10	12/07/1996	1/07/1996	2/04/2036	105G07	Partial Quartz fraction (<25 acres)	1000156735
YB85286	KZK 11	12/07/1996	1/07/1996	2/04/2036	105G07	Partial Quartz fraction (<25 acres)	1000156736
YB85287	KZK 12	12/07/1996	1/07/1996	2/04/2036	105G07	Partial Quartz fraction (<25 acres)	1000156737
YB85288	KZK 13	12/07/1996	1/07/1996	2/04/2036	105G07	Partial Quartz fraction (<25 acres)	1000156738
YB85289	KZK 14	12/07/1996	1/07/1996	2/04/2036	105G07	Partial Quartz fraction (<25 acres)	1000156739
YB85290	KZK 15	12/07/1996	1/07/1996	2/04/2036	105G07	Partial Quartz fraction (<25 acres)	1000156740
YB85291	KZK 16	12/07/1996	1/07/1996	2/04/2036	105G07	Partial Quartz fraction (<25 acres)	1000156741
YB85292	KZK 17	12/07/1996	1/07/1996	2/04/2036	105G07	Partial Quartz fraction (<25 acres)	1000156742
YB85293	KZK 18	12/07/1996	1/07/1996	2/04/2036	105G07	Partial Quartz fraction (<25 acres)	1000156743
YB85294	KZK 19	12/07/1996	1/07/1996	2/04/2036	105G07	Partial Quartz fraction (<25 acres)	1000156744
YB85295	KZK 20	12/07/1996	1/07/1996	2/04/2036	105G07	Partial Quartz fraction (<25 acres)	1000156745
YB85296	KZK 21	12/07/1996	1/07/1996	2/04/2036	105G07	Partial Quartz fraction (<25 acres)	1000156746
YB85297	KZK 22	12/07/1996	1/07/1996	2/04/2036	105G07	Partial Quartz fraction (<25 acres)	1000156747
YB85298	KZK 23	12/07/1996	1/07/1996	2/04/2036	105G07	Partial Quartz fraction (<25 acres)	1000156748
YB85299	KZK 24	12/07/1996	1/07/1996	2/04/2036	105G07	Full Quartz fraction (25+ acres)	1000156749
YB85300	KZK 25	12/07/1996	1/07/1996	2/04/2036	105G07	Full Quartz fraction (25+ acres)	1000156750
YB85301	KZK 26	12/07/1996	1/07/1996	2/04/2036	105G07	Full Quartz fraction (25+ acres)	1000156751
YB85302	KZK 27	12/07/1996	1/07/1996	2/04/2036	105G07	Full Quartz fraction (25+ acres)	1000156752
YB85303	KZK 28	12/07/1996	1/07/1996	2/04/2036	105G07	Partial Quartz fraction (<25 acres)	1000156753
YB85304	KZK 29	12/07/1996	1/07/1996	2/04/2036	105G07	Partial Quartz fraction (<25 acres)	1000156754

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