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Technical Report

Technical Report and Preliminary Economic Assessment Update for the Klaza Property, Yukon, Canada

Rockhaven Resources Ltd.

Yukon, Canada

In accordance with the requirements of National Instrument 43-101 "Standards of Disclosure for Mineral Projects" of the Canadian Securities Administrators

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1 Summary

AMC Mining Consultants (Canada) Ltd. (AMC) was commissioned by Rockhaven Resources Ltd. (Rockhaven) to prepare an updated Preliminary Economic Assessment (PEA) and National Instrument 43-101 (NI 43-101) Technical Report for the Klaza Property (Property) in Yukon, Canada. This report discloses the results of an updated PEA that is based on the Mineral Resource estimate publicly reported in August 2018.

The Property hosts gold-silver-lead-zinc mineralization associated with an extensive system of subparallel vein and breccia zones. It is situated in the Mount Nansen Gold Camp (MNGC), which is located in the south-eastern part of the more regionally extensive Dawson Range Gold Belt, in south-western Yukon.

The Property comprises 1,478 mineral claims that are 100% owned by Rockhaven. A total of 207 claims are subject to a 1.5% Net Smelter Return (NSR) royalty, and a further six claims are subject to a 1% NSR on precious metals and a 0.5% NSR on non-precious metals. The remaining 1,265 claims, including the claims covering the areas of the current Mineral Resources, are not subject to any underlying royalties.

The Property encompasses an area of 28,620 hectares and is located approximately 50 kilometres (km) due west of the village of Carmacks in south-western Yukon. Access is via the Mount Nansen Road, which extends from the Klondike Highway at the village of Carmacks to the former Mount Nansen Mine site and from there along 13 km of unnamed placer access roads to the Property.

1.1 Geology and mineralization

Most of the Property is underlain by Mid-Cretaceous granodiorite. A moderately-sized, Late Cretaceous quartz-rich, granite to quartz monzonite stock intrudes the granodiorite in the south-east corner of the Property and is thought to be the main heat source for hydrothermal cells that deposited mineralization along a series of north-westerly trending, structural conduits.

A swarm of north-westerly trending, Late Cretaceous feldspar porphyry dykes emanate from the stock in the south-eastern part of the Property and cut the granodiorite in the main areas of interest. These porphyry dykes are up to 30 metres (m) wide and commonly occupy the same structural zones as the mineralization. The dykes are coeval with, or slightly older than, the mineralization.

Mineralization on the Property is hosted in a number of parallel zones, which individually range from 1 m to 100 m wide and collectively form a 2 km wide structural corridor in the granodiorite. Mineralization within the structural corridor has been intermittently traced for a length of 4.5 km, but most exploration has concentrated on 2.4 km of lengths along two of the main trends. The mineralization occurs within steeply dipping veins, sheeted veinlets and tabular breccia bodies.

The two areas that have received focused exploration by Rockhaven since 2010 are the BRX and Klaza zones, which have each been traced by trenching and diamond drilling along strike for 2,400 m and from surface to depths of 520 m and 325 m down-dip, respectively. The current Mineral Resource estimation includes the Western and Central BRX zones and the Western and Central Klaza zones.

All of the mineralization comprising the Mineral Resources, except in the Western Klaza zone, lies alongside or cross-cuts feldspar porphyry dykes. A major, post-mineralization cross-fault divides the central portions of the zones from their respective western portions. A second, post-mineralization cross-cutting fault, parallel to the one separating the western and central zones, is believed to be the boundary between the central and eastern zones.

The Western BRX zone is the highest grade area of mineralization discovered to date on the Property. It features discrete veins containing abundant pyrite, arsenopyrite, galena, sphalerite, chalcopyrite, and sulphosalts. Manganiferous carbonate (rhodochrosite) and quartz are the main gangue minerals in these veins.

The Central BRX zone hosts veins that are dominated by quartz, pyrite, and iron-rich carbonates (siderite and ankerite). Pyrite, sphalerite, and galena are the main sulphide minerals in these veins.

The Western Klaza zone is defined by two veins, both of which are laterally continuous. The mineral assemblages in this zone contain higher proportions of arsenopyrite and sulphosalts than are common further east in the Central Klaza zone, and silver to gold ratios are higher. The dominant gangue minerals are quartz and ankerite.

The Central Klaza zone comprises a complex of veins, breccias and sheeted veinlets that are associated with several narrow feldspar porphyry dykes. The strongest veins are typically found along the margins of the dykes. Pyrite and arsenopyrite are the main sulphide minerals in this zone. Quartz and ankerite are the most abundant gangue minerals.

Mineralization within the eastern portions of the BRX and Klaza zones comprises a series of closely spaced, narrow, sub-parallel veins and vein zones dominated by quartz, pyrite, and lesser chalcopyrite. Unlike the Central and Western zones, sulphide mineralization in the Eastern zones contains little arsenopyrite, galena, and sphalerite.

1.2 History

While no hard rock commercial mining is documented on any of the claims comprising the Property, placer mining has been done on some creeks draining the Property. Independent placer mines are still active on some placer claims that partially overlap mineral claims comprising the Property.

A modest amount of historical exploration was conducted on various parts of the Property by previous owners between 1937 and 2014. This work was intermittent and mostly focused on small, isolated portions of the main gold-silver bearing structures and a poorly developed copper-gold-molybdenum porphyry centre. Rockhaven purchased claims in the core of the Property in 2009, and since then has greatly expanded its claim holdings.

Much of the historical work was completed in the areas hosting the current Mineral Resource. This work included soil geochemical surveys, mechanical trenching, geophysical surveys, and limited diamond drilling.

1.3 Exploration and drilling

Between 2010 and 2019, Rockhaven conducted systematic exploration that better defined north-westerly trending structures comprising the BRX, Klaza and other gold-silver enriched zones on the Property. Exploration work by Rockhaven has included grid soil geochemical surveys, ground and airborne geophysical surveys, 24,231 m of mechanized trenching, and 100,200.85 m of diamond drilling.

The most extensively explored zones, the BRX and Klaza zones have each been traced along strike for 2,400 m and to depths of 520 m and 325 m down-dip, respectively. Neither zone outcrops, but mineralization has been exposed in excavator trenches, beneath a thin, 1 - 2 m thick, veneer of overburden. Trenches and drillholes referenced below only include those completed by Rockhaven between 2010 and 2019.

To further divide the Property for Mineral Resource work, the BRX and Klaza zones have been subdivided into the Western, Central, and Eastern BRX zones as well as the Western and Central Klaza zones.

The Western BRX zone is 500 m long and has been tested by both diamond drillholes and trenches. It was tested with the deepest hole completed to date on the Property, and it intersected mineralization at a depth of 520 m down-dip of surface. The Central and Eastern BRX zones have been tested by both diamond drillholes and excavator trenches. Mineralization within these zones has been traced cumulatively for 1,900 m along strike and from surface to a maximum depth of 400 m down-dip.

The Western and Central Klaza zones are located approximately 800 m north-east of the corresponding BRX subzones. They have been tested by both diamond drillholes and excavator trenches. Mineralization within these subzones extends along a 1,200 m strike length and from surface to a maximum depth of 325 m down-dip.

Between 2010 and 2019, a total of 171 holes have been drilled at the BRX zone, while 209 holes have been drilled at the Klaza zones. A further 88 holes were drilled on nearby exploration targets.

The 2010 to 2019 programs on the Property were all managed by Archer, Cathro & Associates (1981) Limited (Archer Cathro) on behalf of Rockhaven.

Samples were collected and processed onsite or at Archer Cathro's Whitehorse facility. After logging and processing, samples were dispatched to ALS Mineral's Whitehorse facility for sample preparation. Prepared samples were subsequently sent to ALS Mineral's North Vancouver Laboratory for analysis. Rockhaven has implemented procedures to ensure a secure chain of custody of samples. Rockhaven routinely inserted Certified Reference Materials (CRMs), blank material, and coarse duplicates (crush duplicates and quarter core). Umpire check assays have also been sent to a separate laboratory to assess the accuracy of the primary laboratory. Rockhaven monitors quality assurance / quality control (QA/QC) on a batch by batch basis immediately upon receipt of the assay certificate.

In the opinion of the Qualified Person (QP), the sampling, sample preparation, security, and analytical procedures adopted by Rockhaven for its exploration programs meet accepted Industry standards. The QA/QC results confirm that the assay results may be relied upon for Mineral Resource Estimation purposes.

The QP undertook random cross checks of assay results in the Rockhaven drillhole database against assay certificates received directly from ALS Minerals. A total of 494 samples of the 9,798 diamond drillhole samples collected in 2016 and 2017 were reviewed to ensure that Au, Ag, Pb, and Zn values were consistent. No errors were detected. Additional QA/QC was collected in 2019 but does not pertain to the current Mineral Resource estimate.

The QP considers the database fit-for-purpose and suitable for use in the estimation of Mineral Resources.

1.4 Mineral processing and metallurgical testing

Metallurgical testing has consisted of a basic flotation and leaching scoping program on four composites at SGS in 2014, and a more in depth program which has been conducted at Blue Coast Research from 2015 to 2018 investigating a number of zones from within the Property. The majority of testing has focused on samples from the Eastern and Western BRX and the Central and Western Klaza zones.

Mineralogical examination of several samples showed that quartz, feldspar, and muscovite dominate the samples representing 75% to 85% of the mineral mass. Sulphides present include pyrite, arsenopyrite, sphalerite and galena, with minor chalcopyrite in the Eastern BRX zone. Carbonates are not abundant and there was no evidence of the presence of preg-robbing carbonaceous material. Gold occurs both as discrete grains and solid solution in both arsenopyrite and pyrite. Outside of the Eastern BRX zone, the distribution of gold project-wide is approximately 55% refractory (solid solution) gold contained in arsenopyrite, 8% within pyrite, and 37% discrete gold. Gold in the Eastern BRX zone occurs more as discrete, leachable grains.

Comminution testwork showed the material to have moderate resistance to grinding either by semi-autogenous grinding (SAG) or ball milling. The Bond Ball Work Index is 16.4 kWh/t and Bond Rod Work Index is 15.3 kWh/t.

Flotation testwork focused on developing a flowsheet consisting of sequential lead, zinc, and arsenopyrite flotation with the aim of creating saleable lead and zinc concentrates and a gold bearing arsenopyrite concentrate that could either be economically processed on site or sold. The final flowsheet features a primary grind of 80% passing 70 microns and standard flotation reagents. Flotation performance from stable locked-cycle testing on a project-wide composite is shown in the following table.

Table 1.1 Metallurgical performance from locked cycle testing of project-wide composite

Product	Weight		Grade						
	g	%	Pb (%)	Zn (%)	Fe (%)	Ag (g/t)	Au (g/t)	As (%)	S (%)
Lead cleaner 3 Conc.	46	1.1	59.8	3.1	9.3	5,957	129.9	3.6	19.4
Zinc cleaner 2 Conc.	89	2.2	2.0	48.0	9.0	1,318	13.5	1.0	30.7
AsPy Conc.	485	12.1	0.3	1.0	35.0	73	30.7	6.7	33.4
Rougher tail	3,389	84.5	0.04	0.04	2.4	4	0.27	0.05	0.9
Feed	**4,009**	**100**	**0.8**	**1.3**	**6.5**	**110**	**5.73**	**0.9**	**5.7**
Product	**Weight**		**% distribution**						
	g	**%**	**Pb**	**Zn**	**Fe**	**Ag**	**Au**	**As**	**S**
Lead cleaner 3 Conc.	46	1.1	85	3	2	62	26	4	4
Zinc cleaner 2 Conc.	89	2.2	6	85	3	27	5	2	12
AsPy Conc.	485	12.1	5	10	65	8	65	88	71
Rougher tail	3,389	84.5	4	3	31	3	4	5	13
Feed	**4,009**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**

Source: Blue Coast Metallurgy Ltd.

The lead concentrate is relatively high-grade and should be attractive to many smelters, however arsenic, antimony, and mercury levels may incur penalties, although this should not affect marketability. Without further treatment, high gold grades may also limit the number of smelters which can accept the lead concentrate. Zinc concentrates are relatively low-grade but are saleable at 48% zinc, though payment on the precious metals will, at best, be poor.

The base case assumption for the 2020 PEA Update is that arsenopyrite concentrates will be treated on-site to produce doré, the marketability of which is not a factor. The base case also includes intensive leaching of the lead concentrate to extract much of the gold into doré form.

To date pressure oxidation (POX) has been assumed to be the pre-oxidation process of choice for the refractory gold bearing arsenopyrite concentrate. The extraction of gold by carbon-in-leach after POX was 98%.

Intensive leach testing has been conducted on several samples of lead concentrate. The leach achieved 80% – 85% gold extraction within four hours. Reagent consumption was low indicating that the process would likely be economic. Therefore, intensive cyanidation of this concentrate has been built into the processing flowsheet.

The overall response of the project-wide composite, in terms of key metal recoveries is shown below. Both lead and zinc recoveries, to their respective concentrates, were 85%. Total silver recovery was 91% and gold recovery was 96%. Recovery to doré for gold and silver was 85% and 11% respectively, accordingly the sale of doré will account for the vast majority of the revenue from the operation.

A summary of the metal recovery to doré and concentrates from testwork on the project wide composite is provided in Table 1.2.

Table 1.2 Summary of metal recovery to doré and concentrates

	Lead	Zinc	Silver	Gold
Pb to lead concentrate	85%			
Zn to zinc concentrate		85%		
Au to doré				87%
Au to lead concentrate				4%
Au to zinc concentrate				5%
Ag to doré			11%	
Ag to lead concentrate			53%	
Ag to zinc concentrate			27%	
Total metal recoveries to payables products	**85%**	**85%**	**91%**	**96%**

Source: Blue Coast Metallurgy Ltd.

Prior to the 2020 PEA study, the application of gravity pre-concentration was tested more extensively. Mean gold recoveries to the pre-concentrate were 95% or higher to 50% of the mass for all deposits except from the Western Klaza zone. Recoveries were lower when treating lower grade samples.

In the preparation of the current PEA, pre-concentration has been removed from the flowsheet as it has so far failed to substantially increase the size of the overall resource or offer much in terms of capital or operating cost savings. However, the option remains available and should further drilling in the future open considerable marginal grade resources pre-concentration could add significant value to the project.

1.5 Mineral Resource estimates

The Mineral Resource estimate was completed using 322 drillholes on the Property comprising 68,948 m of diamond core and 27,266 assays. Grade interpolation was completed predominantly using ordinary kriging (OK), except for the Central Klaza zone where grade was interpolated using a combination of OK, multiple indicator kriging (MIK), local multiple indicator kriging (LMIK), and restricted ordinary kriging (ROK). Gold, silver, lead, zinc, copper, iron, and arsenic were estimated. Copper and iron were estimated to provide data for a base metal to rock density regression analysis to facilitate density modelling. Arsenic was estimated to aid metallurgical assessment.

Mineralized domains were constructed by Archer Cathro to constrain the continuous higher-grade main structures and veins and accepted by the QP. Dyke intersections were used as a marker to help constrain the orientation and position of mineralization. In the previous estimate, high-grade solids were built to capture only vein mineralization and often consisted of only one or two samples in a drillhole. In this estimate, the three-dimensional solids were wider and incorporated waste.

For the OK estimation, 25 of Archer Cathro's mineralization wireframes were used to constrain the interpolation.

A combination of LMIK and ROK estimation was used to model complex, splay hosted mineralization delineated by relatively limited drilling in the Central Klaza area. AMC created a new mineralization envelope for the LMIK model based on a nominal 0.3 g/t AuEq boundary and broad geological control. An intermediate MIK model incorporating change of support (COS) was used to produce a selective mining unit (SMU) model for gold and silver. ROK was used to estimate lead, zinc, copper, iron, and arsenic. The MIK models were converted to simplified, localized versions of the MIK models (LMIK) to replicate recoverable panel models on the SMU block dimensions, and then incorporating the ROK modelled elements as required.

AMC's review of Rockhaven's 2,644 density data measurements demonstrates strong linear relationship between rock density and the sum of Cu, Pb, Zn, and Fe. Densities were assigned to the block model based on a linear regression formula incorporating the sum of these elements.

The pit constrained Mineral Resources are reported for blocks occuring 3 m below the topographic surface to account for weathering, and above a conceptual pit shell based on a US$1,400/ounce gold price. The cut-off applied for reporting the open pit Mineral Resources is 1.0 g/t gold equivalency (AuEq, defined in Table 1.3).

The underground Mineral Resources are reported outside of the conceptual pit shells. No allowances have been made for crown pillars. The cut-off applied to the underground Mineral Resources is 2.30 g/t AuEq. Assumptions made to derive a cut-off grade included mining costs, processing costs and recoveries and were obtained from this report and comparable industry situations.

Table 1.3 Gold equivalent (AuEq) formula

Zone	Au	Recovery %				AuEq (g/t)
		Au	Ag	Pb	Zn	
Central Klaza	>1.0 g/t	92	86	79	81	AUEQ = 1 x AU + AG/94.40 + PB/3.38 + ZN/3.21
	<1.0 g/t	78	76	73	71	AUEQ = 1 x AU + AG/90.56 + PB/3.10 + ZN/3.11
Western Klaza	All	96	91	85	85	AUEQ = 1 x AU + AG/93.09 + PB/3.28 + ZN/3.20
Western BRX & Central BRX	>1.0 g/t	94	87	80	79	AUEQ = 1 x AU + AG/95.34 + PB/3.41 + ZN/3.37
	<1.0 g/t	64	57	49	42	AUEQ = 1 x AU + AG/99.08 + PB/3.79 + ZN/4.31

Note: AuEq values assume metal prices of US$1,400/oz Au, US$19/oz Ag, US$1.10/lb Pb, and US$1.25/lb Zn, and payable values of 97% Au, 81% Ag, 62% Pb, and 52% Zn.
Source: AMC Mining Consultants (Canada) Ltd.

The Mineral Resource for the Klaza deposit has been estimated by Dr A. Ross. P.Geo. Principal Geologist of AMC, who takes responsibility for the portions of the model estimated by OK and Mr Ingvar Kirchner FAusIMM, M.A.I.G., Principal Geologist of AMC Consultants Pty Ltd (Perth, Australia) who takes responsibility for the portions of the model estimated by MIK, LMIK, and ROK.

AMC is not aware of any known environmental, permitting, legal, title, taxation, socioeconomic, marketing, political, or other similar factors that could materially affect the stated Mineral Resource estimate.

Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

The summary results of the estimate are shown in Table 1.4.

Table 1.4 Summary of Mineral Resources as of 5 June 2018

Resource classification	Tonnes (kt)	Grade					Contained metal				
		Au (g/t)	Ag (g/t)	Pb (%)	Zn (%)	AuEq (g/t)	Au (Koz)	Ag (Koz)	Pb (Klbs)	Zn Klbs)	AuEq (Koz)
Indicated											
Pit constrained	2,447	5.3	90	0.7	1.0	6.7	414	7,096	39,143	52,935	529
Underground	2,010	4.2	108	0.8	0.9	5.8	272	6,974	34,125	39,172	378
Indicated total	**4,457**	**4.8**	**98**	**0.7**	**0.9**	**6.3**	**686**	**14,071**	**73,268**	**92,107**	**907**
Inferred											
Pit constrained	1,754	2.6	43	0.4	0.5	3.3	147	2,429	14,897	18,599	187
Underground	3,960	2.8	90	0.7	0.8	4.2	359	11,472	62,647	70,578	538
Inferred total	**5,714**	**2.8**	**76**	**0.6**	**0.7**	**3.9**	**507**	**13,901**	**77,544**	**89,176**	**725**

Notes:
- CIM Definition Standards (2014) were used for the Mineral Resource.
- Estimate includes drill results to 31 December 2017.
- Near surface Mineral Resources are constrained by an optimized pit shell at metal prices of $1,400/oz Au, $19/oz Ag, $1.10/lb Pb, and $1.25/lb Zn.
- Cut-off grades applied to the pit constrained and underground resources are 1.0 g/t AuEq and 2.3 g/t AuEq respectively.
- Gold equivalent values were calculated using parameters outlined in Table 14.4.
- Numbers may not add due to rounding.
- All metal prices are quoted in US$ at an exchange rate of US$0.80 to C$1.00.
- Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

Source: AMC Mining Consultants (Canada) Ltd.

1.6 Mineral Reserve estimates

There are no Mineral Reserve estimates to report for the Property.

1.7 Mining methods

Mineral Resources occur as a selection of vein systems contained in two distinct zones – Klaza and BRX. Each zone can be further broken down by relative location: Western, Central and Eastern. At Central Klaza and BRX potential exists in the eastern extremity for underground mining. The eastern extent is naturally separated from the Central zone by low grade mineralization. The eastern extent of the Central Klaza zone will be accessed from the Klaza pit and at Central BRX zone via underground level development.

The Klaza and BRX zones lend themselves to open pit mining as the mineralized veins are located close to surface. The surrounding topography is moderately steep with sufficient flat areas suitable for the placement of waste dumps and stockpiles. Open pit mining plans to use conventional drill-and-blast equipment and truck and excavator mining equipment supplied by a contractor. Recommendations for the pit slope design are based on probabilistic kinematic analyses. The bench height was limited to 10 m. The berm width of 6.0 m is considered to be sufficient to ensure retention of bench-scale size failures.

Mineralization has been identified through exploration drilling below the potential open pits to a depth of approximately 450 m below surface. Both zones remain open at depth and along strike. Both the Klaza and BRX zones are amenable to mining by underground methods. The proposed underground mining method is Longhole stoping with waste rock fill. Underground mining is planned to be owner-operator with the equipment owned, and personnel employed by Rockhaven.

In order to determine an appropriate production rate that can be supported by the deposit, AMC has used a combination of Taylor`s rule of thumb and vertical tonnes per metre to determine expected production ranges. AMC recommends that the BRX deposit and Klaza deposit be mined as two virtually independent operations at a combined average peak production rate of 690 kilotonnes per annum (ktpa). This production rate is well supported by the detailed production scheduling.

The Klaza and BRX zones are approximately parallel and 800 m apart, and as such separate declines were designed for Klaza and BRX. Access to the Klaza zone underground mine would be via two independent 5 m by 5 m declines for the Western Klaza and Central Klaza zones with crosscuts on each level. The Central Klaza zone also has an in-pit portal to access one level of stopes. Levels are spaced at a vertical distance of 25 m floor to floor. Ore development (4 m by 4 m) was designed to follow the vein along strike from a central access crosscut. Main access to the BRX zone has a similar design with two independent (5 m by 5 m) access decline for the Western and Central BRX zones. Decline access is designed with a 1:7 gradient.

In general, the rock quality values typically range from Poor to Good. Based on a floor to back height of 25 m, at a 65° dip, stope lengths of 18 m for unsupported walls and 49 m for supported walls will be stable. Dilution for Longhole stopes has been estimated using the equivalent linear overbreak slough (ELOS) technique and was estimated by AMC to be less than 0.5 m from the hangingwall (HW).

For both zones, the mine will be ventilated by a “Pull” or exhausting type ventilation system. That is, the primary mine ventilation fans will be located at the primary exhaust airways of the mine. Fresh air will enter each mine via the decline portals with exhaust to the surface via a dedicated return airway. During the winter, air will be heated by direct propane gas fired heaters at the portal, with the heat ducted to the intake airflow. Based upon the equipment required, a peak of approximately 159 m^3/s is planned for ventilation of each of the Central Klaza and Western Klaza zones. Central BRX zone has a peak demand of approximately 189 m^3/s and Western BRX zone 217 m^3/s.

Tetra Tech EBA Inc. (Tetra Tech EBA) was retained by Rockhaven to install a groundwater monitoring well network in the area of the identified resource and to conduct a preliminary hydrogeological assessment for the area of the mineralized zones at Klaza. A preliminary monitoring well network consisting of five nested monitoring wells was installed; four additional observations wells were fitted with vibrating wire piezometers (VWPs) to monitor pore pressures at different depths in each of the observation wells. Permafrost appears to act as a confining layer for the deeper bedrock aquifer. The groundwater flow regime at the site is controlled by the steep terrain with groundwater flow from areas at higher elevations on the mountain slopes toward the valley bottoms.

Based on Tetra Tech EBA's observations, AMC has assumed a low ground-water inflow that can be sufficiently pumped from the mine workings using submersible pumps and a four inch (100 mm) discharge pipeline. The majority of the discharge water will be service water for operating equipment.

During development the decline will be equipped with power for distribution underground as well as a six inch (150 mm) compressed air line, a four inch (100 mm) pipeline for mine service water and a four inch (100 mm) pipeline for dewatering. Telecommunications will be provided by a conventional leaky feeder system.

On all levels, the planned main escape route is either the main decline or to the return air raise (RAR). RARs will be equipped with ladder ways for personnel egress. Refuge stations will be placed strategically in the underground mine, they will be portable for flexibility of location.

Typical equipment for a mechanized Longhole stoping underground mine were selected by AMC. Development will be undertaken using 2 boom Jumbo development drills, stope drilling will be undertaken using production drill rigs. Bolters and cablebolters will undertake all ground support installation. Forty tonne articulated trucks will be used to haul mineralized material from the underground mines to surface. Suitable quantities of auxiliary equipment such as a water truck, grader, scissor lifts, personnel transporters, and utility vehicles were also estimated by AMC.

In order to optimize the overall value of the project and the sequence of mining each zone, AMC has determined potential revenue for each pit and each underground zone. The zones were then ranked in order of value. The potential revenue from each source provides a basis for the order in which the pits and underground zones are scheduled. In addition, there is a focus to have the open pits mined early in the mine life in order to allow development of the underground mines simultaneously.

The proposed combined life-of-mine (LOM) production schedule for the open pit and underground is summarized in Table 1.5.

Table 1.5 Combined open pit (OP) and underground (UG) LOM production schedule

Production	**YR0**	**YR1**	**YR2**	**YR3**	**YR4**	**YR5**	**YR6**	**YR7**
Waste (kt)		5,749	3,149	475	240	245	143	75
Mineralization (kt)		607	688	688	686	688	688	688
NSR ($/t)		200	218	350	362	351	314	315
UG production	**YR8**	**YR9**	**YR10**	**YR11**	**YR12**			**Total**
Waste (kt)	28	26						10,130
Mineralization (kt)	672	675	675	456	255			7,464
NSR ($/t)	249	206	198	187	174			268

Source: AMC Mining Consultants (Canada) Ltd.

Proposed production from the open pit and underground is stockpiled in YR0, during the construction of the process plant. It was assumed that the process plant will be capable of reaching 90% capacity (610 ktpa) during YR1 and 100% capacity in YR2. The process plant has been designed for a maximum throughput of 1,900 tpd (690 ktpa). The proposed processing schedule is summarized in Table 1.6.

Table 1.6 Processing schedule

Total production	**YR0**	**YR1**	**YR2**	**YR3**	**YR4**	**YR5**	**YR6**	**YR7**
Mill feed (kt)		607	688	688	686	688	688	688
NSR ($/t)		200	218	350	362	351	314	315
Au (g/t)		2.70	2.87	4.57	4.83	4.60	4.04	3.93
Ag (g/t)		41.33	47.45	102.11	93.30	95.79	88.75	96.79
Pb (%)		0.36	0.52	0.55	0.62	0.65	0.67	0.77
Zn (%)		0.63	0.74	0.66	0.68	0.71	0.77	0.85
As (%)		3,233	4,015	6,582	6,881	6,820	6,100	6,320
Total production	**YR8**	**YR9**	**YR10**	**YR11**	**YR12**			**Total**
Mill feed (kt)	672	675	675	456	255			7,464
NSR ($/t)	249	206	198	187	174			268
Au (g/t)	3.14	2.50	2.27	1.96	1.67			3.40
Ag (g/t)	72.75	65.24	71.68	84.14	93.18			78.88
Pb (%)	0.60	0.60	0.67	0.72	0.69			0.61
Zn (%)	0.74	0.68	0.76	0.81	0.80			0.73
As (%)	4,987	3,669	4,033	3,936	4,757			5,192

Source: AMC Mining Consultants (Canada) Ltd.

1.8 Recovery methods

Run-of-mine (ROM) ore will be crushed and fed to a grinding circuit comprising a primary SAG mill and a secondary ball mill.

After primary grinding, sequential rougher and cleaner flotation would produce a gold-rich lead concentrate and a zinc product. The lead concentrate would undergo intensive leaching to extract most of the gold prior to being shipped to market. The gold extracted through this leach would be upgraded to doré. The zinc product is poorer in gold and would be shipped directly.

The zinc circuit tails would be subjected to flotation of the gold-bearing sulphides. These would be a blend of pyrite and arsenopyrite, the precise mix of which would depend on whether the concentrate was to be processed on-site using pressure hydrometallurgy or sold to a third party. Mill tailings, a mix of flotation tails and hydrometallurgical residues, would be stored in the tailings facility.

1.9 Tailing Storage Facility

The tailings management strategy focusses on the protection of the regional groundwater and surface waters both during operations and in the long-term (after closure), and to achieve effective reclamation at mine closure. The tailings management strategy has been developed to manage the different tailings streams in separate facilities based on the tailings geochemical properties. Two tailings streams will be produced:

- A hydromet residue.
- A flotation tailings.

Hydromet residue tailings and flotation tailings will be managed in a separate Tailings Storage Facilities (TSF) sized to contain the production schedule volumes. The Flotation Tailings Facility (FTF) will be located downstream of the Hydromet Residue Tailings Facility (HRTF).

A preliminary site water balance has indicated a net water input requirement to provide sufficient process water. It is proposed that additional water to support operations will come from the Klaza river. The aim of the water management plan will be to utilize water within the project area to the maximum practical extent. The plan involves collecting and managing site runoff from disturbed areas and maximizing the recycle of process water. Excess water will be stored in the supernatant pond within the FTF and recycled to the mill for use in the processing plant. The TSF designs minimize the project footprint, prevent surface effluent discharge during operations, and allow for simple and effective water management at closure.

1.10 Project infrastructure

A trade-off study was performed to investigate providing electric power using diesel generators or using grid power from the territorial utility. The results indicate that grid electrical power provides more value to the project over the life of the mine. The grid power option would require a transmission power line to be constructed from Carmacks to the mine-site along the existing Mount Nansen and site access roads.

The water treatment plant will be located at the processing plant operation. Underground mine water from operations, surface water from the open pits, and grey water from the office and mine dry will be routed via dual wall heat traced high-density polyethylene (HDPE) piping systems, partially or completely buried, to the plant for processing as part of the tailings system.

Potable water will also be provided by a treatment plant and will be capable of providing 60 gallons per day per man to the offices and mine dry.

A network of light vehicle roads will be provided to keep personnel vehicles separate from the open pit and underground mine haul traffic. These roads will be constructed in accordance with applicable permafrost design requirements.

The mine offices will be an assembly of standard construction industry grade, portable trailers. The trailer complex will provide for a perimeter of offices, a common area in the centre, meeting rooms, lunch rooms, and training rooms. The mine dry will also be an assembly of construction industry grade, portable trailers. Space for 240 lockers and baskets will be provided along with showers and laundry facilities. The underground mines will be supported by a centrally located maintenance facility near the offices, a heated warehouse, and a cold storage warehouse.

Labour will be sourced from the local area around Klaza, as well as fly-in fly-out (FIFO) workers. The FIFO workforce will be housed in the onsite camp facilities. Daily bus service will also be provided to employees that live in Carmacks, which is 73 km to the mine and back. The total underground mining labour and supervision requirements were estimated to be 180 employees; this excludes contractor labour for the open pit, however an additional 80 personnel are estimated for the open pit. A further 124 employees are required for processing, administration and support, camp operation and spare workers. The total workforce is 304 employees.

Mine water will be supplied via a four-inch (100 mm) steel line down the declines that will be installed as the decline progresses. At required levels, pressure reducing and isolation valves will be installed to maintain the system at operating pressures. A two-inch (50 mm) distribution system of steel pipe and hoses will be laid out on the operating levels and relocated as required during the mine life.

The mine dewatering system will consist of staged 50 horsepower (hp) submersible dirty water pumps at 60 m levels. Each sump will have two pumps to provide continual redundancy. A four-inch (100 mm) steel line will remove the water from the mines and direct it to the process water treatment facility.

Underground telecommunications will be provided by a conventional leaky feeder system strung down the decline and feeding the operating levels.

Compressed air will be supplied by two separate compressor plants, one for each of the two underground mining areas. Compressed air will be carried underground via DN150 HDPE pipe down the main ramps. Reticulation on the levels will be via DN100 HDPE pipe.

The main part of the explosives will be stored on surface as part of the open pit mine operations. The underground mines will continue to use the open pit facilities as the project completes the open pit phase of the work.

Fuel storage will consist of two tanks that will have the capacity to support two months consumption at peak production. The tank system will be enclosed by a lined berm of sufficient capacity to contain 110% of the contents of a full tank in the event of a major leak or spillage. Fuel will be trucked to site on a year round basis.

Portable refuge stations in the operating levels as well as lunchrooms near the maintenance area will be provided. Facilities for self-rescue storage in the lunchrooms as well as first aid kits at the refuge stations will also be in place.

Main egress is provided by the declines, and a second means of egress via the ladders in the ventilation raises.

1.11 Market studies and contracts

Initial metallurgical tests showed elevated levels of penalty elements in the forecast lead and zinc concentrates. In addition, the low levels of zinc in zinc concentrates and the high levels of gold in lead concentrates were seen to be potentially difficult in the marketing of both concentrates.

Although this is a PEA based on Inferred Mineral Resources, Rockhaven decided to undertake an additional degree of research regarding the marketability of the concentrates. The high percentage of precious metals reporting to the lead concentrate was determined to potentially impact marketability and lead to inclusion of the Acacia leach process. In addition, a concerted effort was made to reduce the potential deleterious elements present in the concentrates and, thereby, improve the marketability.

While H. M. Hamilton & Associates Inc. (HMH) has investigated possible markets and potential terms, no detailed market study has been undertaken at this stage of the project.

1.12 Environmental studies and permitting

A preliminary environmental assessment (EA) was undertaken for the Klaza Property that addressed the following considerations:

- Reviewed and summarized existing relevant social and environmental data, including government data sets.
- Developed a list of key regulatory approvals required and an outline of the Yukon regulatory approval process.
- Identified principal areas of concern / key issues related to permitting and the environment.
- Described site specific environmental and land-use issues (parks, land use conflicts, specific flora, fauna issues, Species at Risk, other potentially significant constraints).
- Provided a general outline of the sequence, timeline and cost estimates to advance the project through the permitting / approvals stage (up to approval to construct).

- Developed recommendations on how to proceed and types of studies that may be required to support the regulatory review and approvals process.

Based on the information reviewed as part of this preliminary EA, there were no known significant environmental issues or sensitive receptors / features identified that could materially influence project viability, nor affect the major design components for future mine development.

The level of information contained in the existing environmental and social data reviewed as part of the PEA is considered sufficient to facilitate the scoping of a comprehensive environmental baseline study in order to meet future assessment and approval requirements. Any future baseline studies would require targeted biophysical and socio-economic considerations be identified and assessed. Once the initial stages of mine planning have been completed and conceptual level detail is determined, it will be possible to identify and define a baseline assessment program.

1.13 Capital and operating costs

The operating cost estimate allows for all labour, equipment, consumables, supervision, and technical services. Operating costs for underground mining are based on AMC's database of underground mine costs. AMC has then validated the benchmark costs against a number of Canadian mining operations. Actual costs for an operating mine near Klaza and contractor quotes were also used to validate the assumptions.

The process operating cost estimate was provided by Blue Coast Metallurgy Ltd. (BCM). The process operating costs are separated out in two components. Firstly, the mineral processing sections of the mill: grinding, milling, flotation concentrate filtration and load-out. Secondly, for the hydromet sections, which include POX, hot cure, CCD's, neutralization, cyanide leach – carbon-in-pulp (CIP), and gold recovery to doré.

Process operating costs were determined from both first principles make-up and vendor quotes from other recent projects. These include all labour and supervision, consumables, reagents and power. Vendor quoted delivered costs were obtained for the major consumables: power, flotation reagents, lime, limestone, cyanide, and oxygen. Maintenance costs were determined from similar projects and industry standards.

General and administration (G&A) costs generally cover site administration and corporate costs. AMC benchmarked its estimate of \$15/t against knowledge of the G&A cost (\$13/t) for a similar operation in northern Canada.

The total operating cost is summarized in Table 1.7.

Table 1.7 Total operating cost

Description	Cost (C$/t)	Cost (C$M)
Mining cost	55.14	412
Processing and tailings storage cost	41.64	311
G&A cost	15.00	112
Total operating cost	**111.78**	**834**

Note: Totals may not add up exactly due to rounding.
Source: AMC Mining Consultants (Canada) Ltd.

The capital cost estimate is split into project capital (first three years) and sustaining capital (remainder of the mine life). No royalties apply to the project. Project capital includes the cost of the process plant, underground equipment and infrastructure, underground development, and surface infrastructure.

AMC has assumed that, due to the short life of the pits (three years), a contractor will be used to mine the open pits. Mark-ups on the operating costs have been assumed to cover for the capital costs and no capital has been allowed for the open pits.

The process plant capital cost estimate was provided by BCM. Process plant civils and building costs were estimated by AMC. AMC estimated the capital cost for the underground mines and surface infrastructure.

The major components of the surface infrastructure cost estimate are refurbishment of existing access roads and new site roads, mine office, mine dry, and maintenance workshop.

The underground capital cost comprises primarily of underground development (lateral and vertical), underground mobile equipment and underground infrastructure. Capital costs for equipment are based on supplier quotes for other recent projects. Equipment numbers were estimated to meet the production target of 688 ktpa. Underground infrastructure costs are based on estimated quantities, some supplier quotes from other recent projects or from benchmark construction costs and assumptions. The underground infrastructure largely consists of electrical distribution, ventilation, and dewatering costs.

The total capital cost is estimated to be C$358M and is summarized in Table 1.8.

Table 1.8 Total capital cost

Description	Total cost (C$M)
Underground lateral development	102
Underground vertical development	13
Floatation tailings storage and residue tailings storage	17
Underground mine infrastructure	21
Mobile equipment	32
Processing plant	103
Surface infrastructure	16
Capital indirect costs	10
Contingency	32
Additional 5% sustaining for equipment rebuilds	12
Total capital cost	**358**
Project capital	**244**
Sustaining capital	**114**

Note: Totals may not add up exactly due to rounding.
Source: AMC Mining Consultants (Canada) Ltd.

1.14 Economic analysis

All currency is in Canadian dollars (C$) unless otherwise stated. Foreign exchange rates were applied as required. Pricing submitted in US dollars (US$) were converted to C$ using the exchange rate C$1 : US$0.72. The cost estimate was prepared with a base date of Year 0 and does not include any escalation beyond this date. For economic analysis (Net Present Value (NPV)) all costs and revenues are discounted at 5% from the base date. Metal prices were selected in discussion with

Rockhaven, and in keeping with three year forecasts. A corporate tax rate of 27% is applied as the mining income will be earned in the Yukon. It is assumed there are no royalties to be paid.

AMC conducted a high level economic assessment of the combined Klaza open pit and underground mine. The combined open pit and underground mine generates C$529M pre-tax NPV and C$378M post-tax NPV at a 5% discount rate, pre-tax IRR of 45% and post-tax IRR of 37%. Project capital is estimated at C$358M with a payback period of 4 years (discounted pre-tax cash flow from base date of Year 0). The Klaza combined open pit and underground mine economics are provided in Table 1.9.

Pre-tax and post-tax NPV is most sensitive to changes in the gold price (as well as grade or recovery). It is also significantly sensitive to changes in total capital costs and exchange rate.

Table 1.9 Klaza combined open pit and underground mine economics

Klaza	Unit	Value
Total mineralized rock	kt	7,464
Total waste production	kt	10,130
Gold grade	g/t	3.4
Silver grade	g/t	79
Lead grade	%	0.6
Zinc grade	%	0.7
Gold price	US$/oz	1,450
Silver price	US$/oz	17.00
Lead price	US$/lb	0.95
Zinc price	US$/lb	1.00
Payable gold metal	kg	23,373
Payable silver metal	kg	429,222
Payable lead metal	t	22,691
Payable zinc metal	t	22,706
Revenue by commodity (gold)	%	77
Revenue by commodity (silver)	%	16
Revenue by commodity (lead)	%	3
Revenue by commodity (zinc)	%	4
Total net revenue	C$M	1,975
Capital costs	C$M	358
Operating costs (total)	C$M	834
Mine operating costs	C$/t mineralized rock	55.1
Process and tails storage operating costs	C$/t mineralized rock	41.6
Operating costs (total)	C$/t mineralized rock	111.8
Operating cash cost (AuEq)	US$/oz AuEq	612.6
Total all in sustaining cost (AuEq)	US$/oz AuEq	875.3
Payback period	Yrs	4
Cumulative cash flows (pre-tax)	C$M	783
Pre-tax NPV [1]	C$M	529
Pre-tax IRR	%	45
Post-tax NPV [1]	C$M	378
Post-tax IRR	%	37

Note: [1] Discount rate of 5%.
Source: AMC Mining Consultants (Canada) Ltd.

1.15 Interpretations and conclusions

The results of the PEA suggest that the Project has good economic potential and warrants further study.

Standard industry practices, equipment and processes were used in this study. The authors of this report are not aware of any unusual or significant risks, or uncertainties that could materially affect the reliability or confidence in the project based on the data and information made available.

The typical risks associated with open pit and underground mining related to geotechnical conditions, equipment availability and productivity, and personnel productivity are generally similar to those expected at other remote operations.

The PEA is preliminary in nature. It includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves. There is no certainty that the PEA will be realized.

1.16 Recommendations

1.16.1 Geology and Mineral Resources

Work at the Klaza Property has defined significant, high-grade gold-silver-lead-zinc Mineral Resources. AMC recommends the following:

- Conduct further drilling of the BRX East and other Eastern Zones in order to sufficiently increase the drillhole density to enable completion of a Mineral Resource estimate.
- Infill drilling in the BRX and Klaza zones to upgrade Inferred Mineral Resources currently considered in the PEA to the Indicated category.
- Collect samples from previously unsampled drill core intervals within the Eastern Zones in order to complete the sample record.
- Conduct exploration diamond drilling beneath and along strike of prospective targets identified by trenching and diamond drilling.
- Update the Mineral Resource estimate on completion of the drill program and additional sampling within the Eastern Zones.
- Going forward, take duplicate samples only from mineralized material.

1.16.2 Hydrology

The following recommendations are made for further hydrogeological assessment at Klaza:

- Continue seasonal groundwater monitoring for the existing monitoring and observation wells.
- Survey all monitoring wells for their location and elevation of the top of the PVC casing with an accuracy of about ±1 cm or better.
- Collect additional ground temperature and hydrogeological data from the existing observation and monitoring wells, and drill additional wells as required. This data will allow updates to the preliminary conceptual hydrogeological model with an emphasis on permafrost-groundwater interaction.
- As mine planning progresses, install additional monitoring wells in the areas up and down gradient of proposed mine infrastructure.
- Intergrate groundwater and surface water baseline data collection and interpret both datasets to assess groundwater-surface water interaction.

1.16.3 Geotechnical

Obtain a better understanding of the factors affecting open pit and stope stability and the proposed mining method from additional data collection, interpretation, and analysis, including the following:

- Develop a series of 3D models that includes lithology, alteration and major structure.
- Using data from these models develop a 3D geotechnical model.
- Continue collecting geotechnical information during infill and exploration drilling. Preferably using oriented core whenever possible to increase confidence and understanding of structures.
- Implement a laboratory testing program on the various lithologies to assist in understanding rock properties. The following suite of rock property tests is recommended: Uniaxial compressive strength (UCS) with Young's modulus (E) and Poisson's ratio (v), Confined compressive strength (triaxial), Indirect tensile strength (Brazilian test).
- As the mine is likely to be developed to depths > 300 m below ground level, in-situ stress testing will likely be needed. This could be carried out once mining has commenced.

1.16.4 Mining and infrastructure

AMC recommends the following work to be undertaken during the next phase of study:

- Re-evaluate open pit and underground mining opportunities for any updates to the Mineral Resource estimate.
- Reassess open pit-underground interface and specifics of crown pillar requirements.
- Further investigate the open pit mining method and bench height to evaluate means of reducing dilution.
- Prepare detailed development and production schedules.
- Determine groundwater inflow to the proposed pits and underground mines from updated hydrogeological modelling.
- Should the hydrogeological modelling and study of the ground water regime indicate potential for large quantities of inflow into the mine, investigate a non-contact dewatering system. Water captured prior to entering the mining floor can reduce the cost of water treatment later.
- Undertake detailed cost estimation and obtain contractor quotes for operating costs.
- Increase the level of detail for infrastructure engineering to better define capital costs.

1.16.5 Processing and metallurgical testwork

BCM recommends the following for the Klaza project, ahead of preparation of a more detailed study:

- Pre-concentration: Conduct sufficient testing to establish metallurgical projections and process design for the use of pre-concentration on all zones except Western Klaza.
- Refractory gold concentrate development: Continue testing aimed at maximizing the gold grade / recovery relationship for arsenopyrite-hosted refractory gold.
- Conduct a marketing study on the refractory gold concentrate.

1.16.6 Tailings Storage Facility

- Complete rheology testing and geotechnical testing of the tailings streams.
- Complete geotechnical investigations to evaluate foundation conditions and construction material sources.

1.16.7 Environmental

Continue ongoing collection and evaluation of baseline data.

1.16.8 Proposed budget for recommendations

An approximate budget for the recommended work described above is presented in Table 1.10.

Table 1.10 Estimated cost to complete recommended work

Parameter	Cost (C$000's)
BRX and Klaza zones infill drilling (23,000 m @ $220/m)	5,060
Hydrological monitoring	50
Geotechnical testwork, modelling, and interpretation	50
Metallurgical studies	60
Marketing study	20
Updated Resource and next level of study	500
Environmental baseline studies to stage ready for EA	600
Contingency @ 15%	950
Total (excluding taxes)	**7,290**

Source: Archer, Cathro & Associates (1981) Limited.

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2 Introduction

AMC Mining Consultants (Canada) Ltd. (AMC) was commissioned by Rockhaven Resources Ltd. (Rockhaven) to prepare an updated Preliminary Economic Assessment (PEA) and National Instrument 43-101 (NI 43-101) Technical Report (Technical Report) for the Klaza Property (Property) in Yukon, Canada. This report discloses the results of an updated PEA that is based on the Mineral Resource estimate publicly reported in August 2018.

The Property is located in the south-western Yukon Territory of Canada approximately 50 kilometres (km) due west of the village of Carmacks. The Property is 100% owned by Rockhaven. Rockhaven acquired the original Property from ATAC Resources Ltd. (ATAC) in 2009 and subsequently added more claims through both acquisition and additional staking. Of a total of 1,478 claims, 207 claims are subject to a 1.5% Net Smelter Return (NSR) royalty. Six claims are subject to a 1% NSR royalty on precious metals and a 0.5% royalty on non-precious metals. The other 1,273 claims, including the claims over the existing Mineral Resources, are not subject to any underlying royalties.

The Technical Report has focused on the Mineral Resources potentially mineable by open pit and underground methods for the Central and Estern BRX zones and the Central and Western Klaza zones.

AMC was responsible for managing and preparing the Technical Report with inputs from Archer, Cathro & Associates (1981) Limited (Archer Cathro), Blue Coast Metallurgy Ltd. (BCM) and Knight Piésold Ltd (KP).

Table 2.1 Persons who prepared or contributed to this technical report

Qualified Persons responsible for the preparation of this Technical Report						
Qualified Person	**Position**	**Employer**	**Independent of Rockhaven?**	**Date of last site visit**	**Professional designation**	**Sections of report**
Dr A Ross	Principal Geologist	AMC Mining Consultants (Canada) Ltd.	Yes	18 – 19 Aug 2015	P.Geo. (BC), P.Geol. (AB)	1 (part), 11 ,12, 14 (part) 25 (part), 26 (part), 27 (part)
Mr I Kirchner	Principal Geologist	AMC Consultants Pty Ltd	Yes	No visit	FAusIMM, M.A.I.G.	1 (part) and 14 (part)
Mr C Martin	Principal Metallurgist	Blue Coast Metallurgy Ltd.	Yes	No visit	C.Eng.	1 (part), 13, 17 (part), 19, 25 (part), 26 (part), 27 (part)
Mr M Dumala	Senior Engineer and Partner	Archer, Cathro & Associates (1981) Limited.	No	14 Oct 2017	P.Eng. (BC)	1 (part), 3 (part), 4-10, 23, 24, 25 (part), 26 (part), 27 (part)
Mr G Methven	Principal Mining Engineer	AMC Mining Consultants (Canada) Ltd.	Yes	18 – 19 Aug 2015	P.Eng. (BC)	1 (part), 2, 3 (part), 15, 16 (part), 20, 21 (part), 22, 25 (part), 26 (part), 27 (part)
Mr M Molavi	Principal Mining Engineer	AMC Mining Consultants (Canada) Ltd.	Yes	No visit	P.Eng. (BC)	1 (part), 18, 25 (part), 26 (part)
Mr D Warren	Principal Mining Engineer	AMC Mining Consultants (Canada) Ltd.	Yes	No visit	P.Eng. (BC)	1 (part), 16 (part), 21 (part), 25 (part) 26 (part), 27 (part)
Mr B Borntraeger	Specialist Geotechnical Engineer / Associate	Knight Piésold Ltd.	Yes	No visit	P.Eng. (BC)	1 (part), 17 (part), 26 (part)

Other Experts who assisted the Qualified Persons					
Expert	**Position**	**Employer**	**Independent of Rockhaven?**	**Visited site**	**Sections of Report**
Mr G R Yeadon	Secretary & Director	Tupper, Jonsson & Yeadon	No	No	4
Mr Al Strang	Senior Environmental Planner	Morrison Hershfield Limited	Yes	No	20
Mr L Donaldson	Chief Financial Officer and Principal	Rockhaven / Donaldson Brohman Martin	No	No	22

Source: AMC Mining Consultants (Canada) Ltd.

The key information used in this report is listed in Section 27, References.

All currency amounts and commodity prices are in Canadian dollars unless stated otherwise. Quantities are stated in metric (SI) units. Commodity weights of measure are in grams (g) or percent (%) unless stated otherwise.

This Technical Report includes the tabulation of numerical data which involves a degree of rounding for the purpose of Mineral Resource estimation. AMC does not consider any rounding of the numerical data to be material to the Property.

This Technical Report has been produced in accordance with the Standards of Disclosure for Mineral Projects as contained in NI 43-101 and accompanying policies and documents. NI 43-101 utilizes the definitions and categories of Mineral Resources and Mineral Reserves as set out in the May 2014 edition of the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards on Mineral Resources and Mineral Reserves (CIM Definition Standards).

A draft of the Technical Report was provided to Rockhaven to check for factual accuracy. The Technical Report is effective as at 10 July 2020.

The last site visit was carried out on 14 October 2017, by Mr M. Dumala. There was no exploration activity carried out in 2018 and drilling in 2019 was conducted peripheral to the deposit area and is not included in the Mineral Resource estimation. As a result no additional information is available on site for the current Mineral Resource estimate to warrant a more recent visit.

3 Reliance on other experts

The Qualified Persons (QPs) have relied, in respect of legal aspects, upon the work of the Expert listed below. To the extent permitted under NI 43-101, the QPs disclaim responsibility for the relevant section of the Technical Report.

The following disclosure is made in respect of this Expert:

Glenn R. Yeadon, Attorney, Tupper, Jonsson & Yeadon, Vancouver, BC, Canada.

Report, opinion, or statement relied upon: information on mineral tenure and status, title issues, royalty obligations, etc. Mr Yeadon is also a director of Rockhaven.

Extent of reliance: full reliance following a review by the QP(s).

Portion of Technical Report to which disclaimer applies: Section 4.

The QPs have relied, in respect of environmental aspects, upon the work of the Expert listed below. To the extent permitted under NI 43-101, the QPs disclaim responsibility for the relevant section of the Technical Report.

The following disclosure is made in respect of this Expert:

Al Strang, Senior Environmental Planner, Morrison Hershfield Limited, Burnaby, BC, Canada.

Report, opinion, or statement relied upon: information on permitting, environmental, social, and community factors.

Extent of reliance: full reliance following a review by the QP(s).

Portion of Technical Report to which disclaimer applies: Section 20.

The following disclosure is made in respect of this Expert:

Larry Donaldson, Principal, Donaldson Brohman Martin, Port Moody, BC, Canada and Chief Financial Officer of Rockhaven.

Report, opinion, or statement relied upon: information on taxation regarding the Property.

Extent of reliance: full reliance following a review by the QP(s).

Portion of Technical Report to which disclaimer applies: Section 22.

4 Property description and location

The Property is located in south-western Yukon at latitude 62°15ˈ11" north and longitude 137°7ˈ23" west on NTS 115I/3 (Figure 4.1). It comprises 1,478 contiguous mineral claims (totalling approximately 28,620 hectares) registered with the Whitehorse Mining Recorder in the names of Rockhaven or Archer Cathro, which holds them in trust for Rockhaven.

A total of 207 claims (Dic, Eagle, Etzel, VG, VIC, Jon-Wedge, Rat, Wedge, Ox, Bull, and parts of J. Bill#) are subject to a 1.5% NSR royalty payable to Janet Dickson of Whitehorse. The six Desk claims have a 1.0% NSR royalty on precious metals and a 0.5% NSR royalty on non-precious metals payable to R. Hulstein and R. Stroshein Estate. The other 1,265 claims are not subject to any underlying royalties. Specifics concerning claim registration are tabulated in Table 4.1, while the locations of individual claims are shown on Figure 4.2 and Figure 4.3.

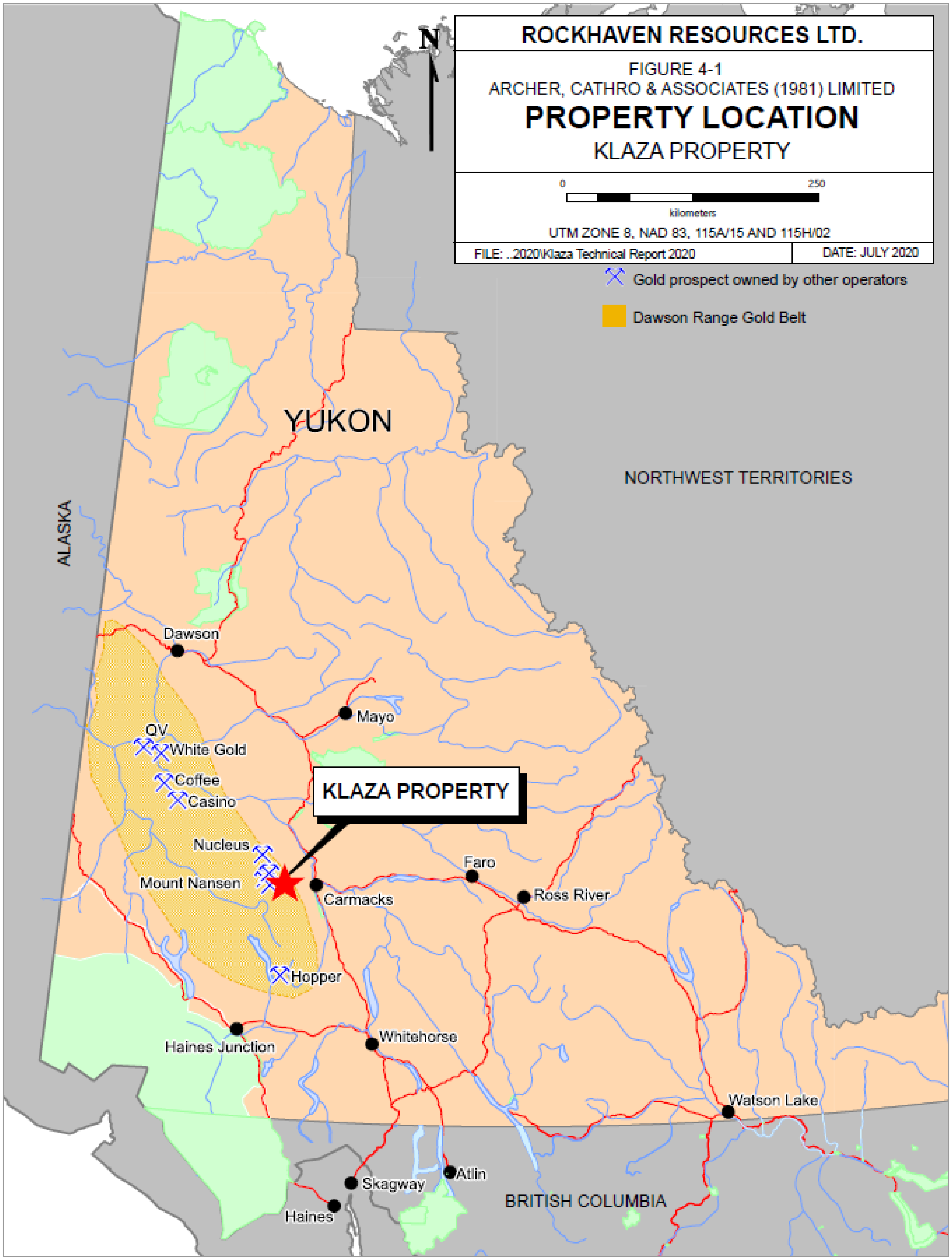
Table 4.1 Claim data

Claim name		**Grant number**	**Expiry date****
BBB	1-96	YD56331-YD56426	15 April 2035
	97-152	YD58527-YD58582	15 April 2035
	153-172	YD62853-YD62872	15 April 2035
	173-255	YD113413-YD113495	15 April 2035
	256-384	YE60326-YE60454	15 April 2035
Dic	1-7[1]	YA93470-YA93476	11 January 2045
	101-106[1]	YB35470-YB35475	11 January 2046
Eagle	1-12[1]	YB35415-YB35426	11 January 2046
Etzel	1-12[1]	YA86336-YA86347	1 December 2052
	13-17[1]	YA86348-YA86352	1 December 2051
	18-20[1]	YA86353-YA86355	1 December 2052
	21-28[1]	YA86356-YA86363	1 December 2051
	29-32[1]	YA86364-YA86367	1 December 2052
	33[1]	YS86368	1 December 2048
	34[1]	YA86369	1 December 2052
	35-44[1]	YA86370-YA86379	1 December 2049
	45-50[1]	YA86380-YA86385	1 December 2051
JCS	1-3	YC25916-YC25918	1 December 2036
Klaza	1-17*	YC37984-YC38000	11 January 2048
	18-24*	YC39051-YC39057	11 January 2048
	25-40	YD09205-YD09220	7 January 2048
	43-64	YD09223-YD09244	7 January 2044
	65F-66F	YC99541-YC99542	11 January 2048
	68-129	YD07149-YD07210	11 January 2048
	133-166	YD07214-YD07247	11 January 2048
	167-308	YD119737-YD119878	11 January 2044
	309	YD110502	11 January 2044
	310-311	YC97706-YC97707	11 January 2045
	314-316	YC97722-YC97724	11 January 2045
	317-319	YC99801-YC99803	11 January 2039
	320-357	YE66241-YE66278	11 January 2036
VG	1-4[1]	YA86406-YA86409	1 December 2049
	5-8[1]	YA86410-YA86413	1 December 2041
VIC	2[1]	YA86309	1 December 2051
	75[1]	YC19429	1 December 2049
	76-78[1]	YC19430-YC19432	1 December 2052

Claim name		**Grant number**	**Expiry date****
Wedge	11-14[1]	YA82177-YA82180	1 December 2041
Dade	1-16	YD07685-YD07700	23 March 2039
	17-54	YD108507-YD108544	23 March 2036
	77-90	YD108567-YD108580	23 March 2032
	91-96	YC97716-YC97721	23 March 2037
	97-106	YD07248-YD07257	23 March 2033
Krast	1-32	YD74101-YD74070	11 January 2032
Queen	1-121	YE60731-YE60851	24 April 2032
Val	1-9	YC25903-YC25911	24 February 2038
	10-15*	YE85801-YE85806	24 February 2038
Nor	1-74	YE60651-YE60724	24 April 2032
Bull	1-2[1]	YA81420-YA81421	1 December 2048
	12[1]	YA86291	29 February 2040
	14[1]	YA86293	29 February 2040
	16-20[1]	YA86295-YA86299	28 February 2039
	21-28[1]	YA86300-YA86307	28 February 2034
D	1-2	YB57373-YB57374	20 January 2034
	3-4	YB57375-YB57376	20 January 2042
Desk	1-6[2]	YC47461-YC47466	23 March 2032
J. Bill	1-2[1]	YA78049-YA78050	28 February 2034
	3-4[1]	YA78051-YA78052	28 February 2035
	5-8[1]	YA78053-YA78056	28 February 2034
	9-12[1]	YA78057- YA78060	2 February 2034
	13[1]	YA78061	2 February 2042
	14[1]	YA78062	2 February 2046
	15-16[1]	YA78063-YA78064	2 February 2042
	17-24[1]	YA78065-YA78072	02 February 2034
	25-28[1]	YA78073-YA78076	28 February 2034
	29-30[1]	YA78077-YA78078	28 February 2042
	31-32[1]	YA78079-YA78080	28 February 2046
JBF	6	YB36958	1 December 2040
	10	YB54543	5 December 2041
Jon-Wedge	1[1]	YB35895	1 December 2042
	2[1]	YB35896	1 December 2040
	3[1]	YB35897	1 December 2032
	4[1]	YB35898	1 December 2033
	5-6[1]	YB35899-YB35900	1 December 2032
Ox	1-20[1]	YA86386-YA86405	20 December 2032
Rat	1-8[1]	YA81428-YA81435	28 February 2034
	9-24[1]	YA81436-YA81451	28 February 2035
	25-40[1]	YA81452-YA81467	28 February 2034
Sked	1-30	YD07655-YA07684	23 March 2034
	31-36	YC99722-YC99727	23 March 2032
Lone	1-161	YF59161-YF59321	11 January 2027

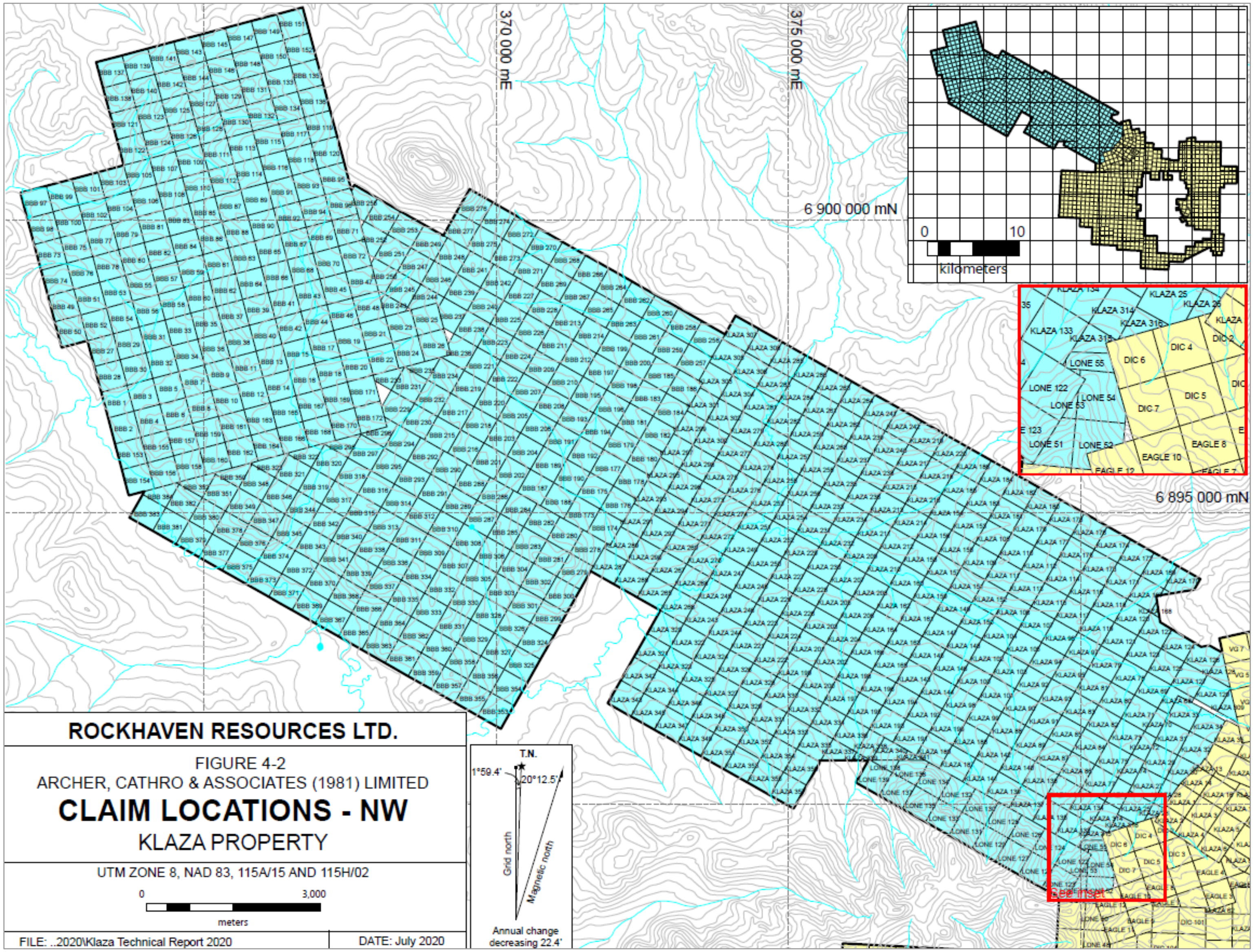
Notes:
[1] A total of 207 mineral claims (the Dic, the Eagle, the Etzel, the VG, the VIC, the Wedge, the Bull, the Jon-Wedge, the Ox, the Rat and parts of the J. Bill) are subject to a 1.5% NSR royalty payable to Janet Dickson of Whitehorse, Yukon Territory.
[2] The six Desk mineral claims are subject to a 1.0% NSR royalty related to precious metals and a 0.5% NSR royalty related to non-precious metals payable to each of Roger Hulstein of Whitehorse, Yukon Territory and the Estate of Robert W. Stroshein.
* Includes fractional claims.
**Expiry dates include 2019 work which has been filed for assessment credit but not yet accepted.
Source: Archer, Cathro & Associates (1981) Limited.

Figure 4.1 Property location – Klaza Property



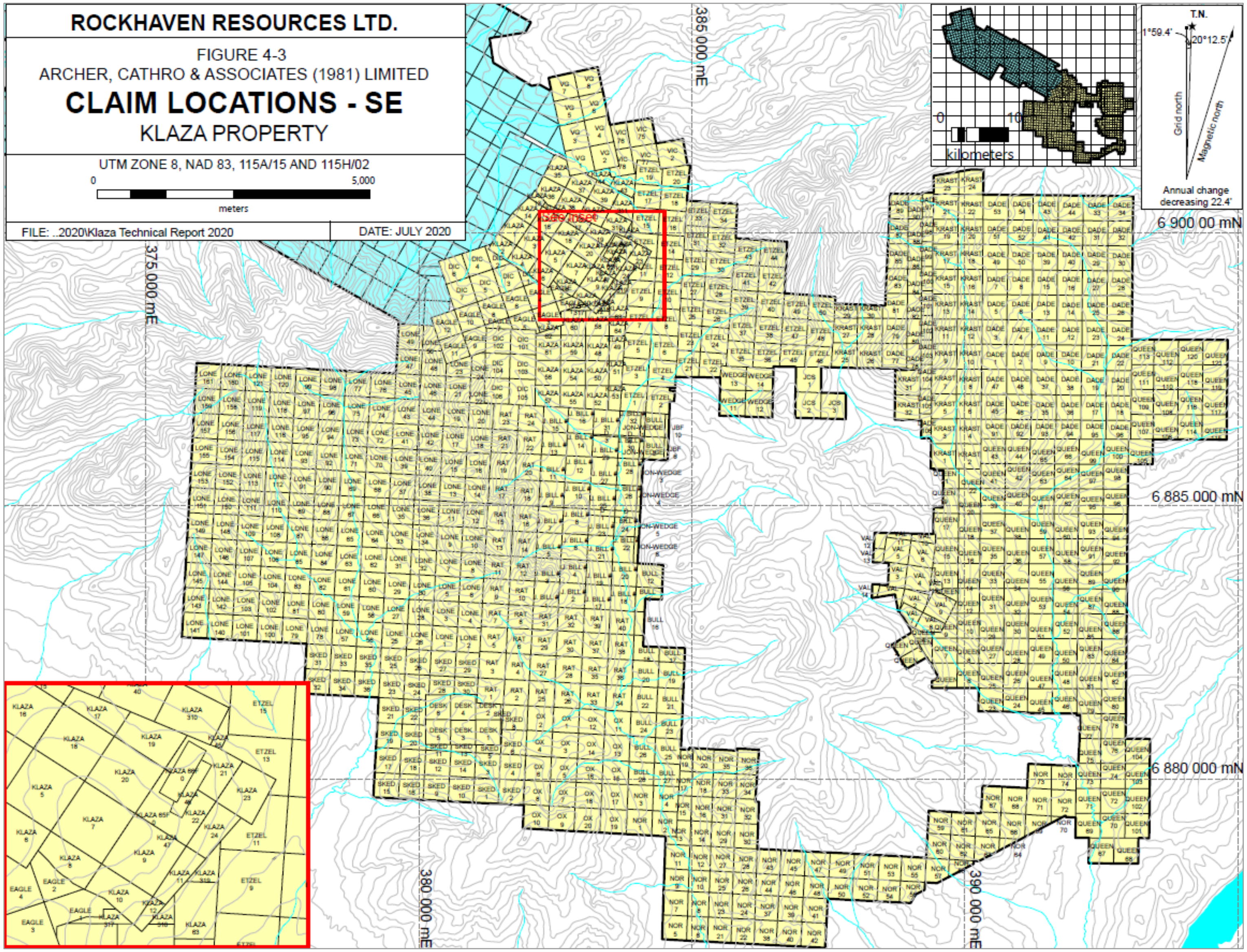
Source: Archer, Cathro & Associates (1981) Limited.

Figure 4.2 Claims location – north-western part of Property



Source: Archer, Cathro & Associates (1981) Limited.

Figure 4.3 Claims location – south-eastern part of Property

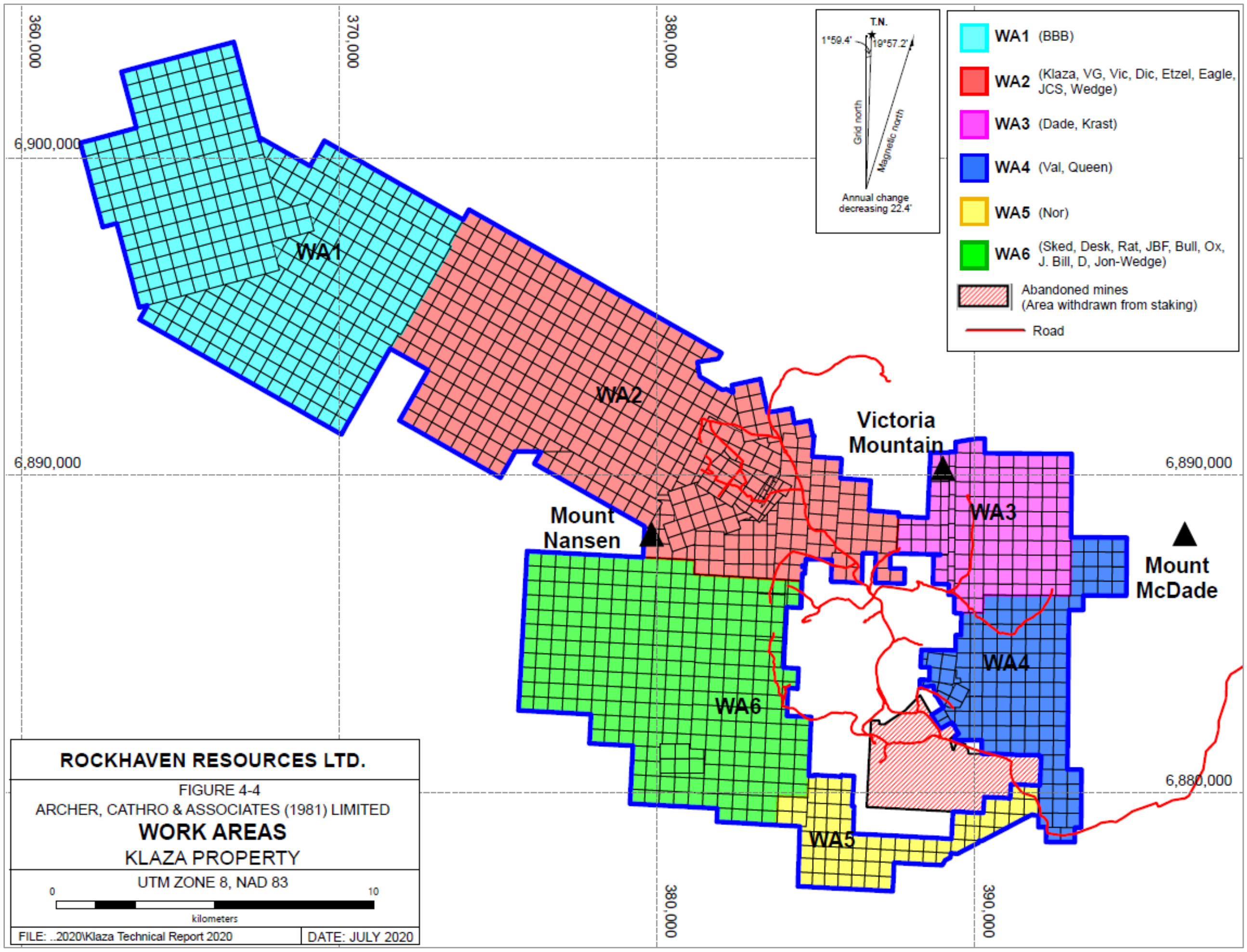


Source: Archer, Cathro & Associates (1981) Limited.

For the purpose of this report the Property has been subdivided into Work Areas 1 – 6. Historical exploration programs can cross multiple Work Areas, typically where several claim groups are closely spaced, or early claims were restaked.

Figure 4.4 highlights the boundaries of these Work Areas, and Table 4.2 describes the claim groups within each Work Area. An area of about 40 km^2 is centred between Work Areas 4 and 6, encompassing the former Mount Nansen mining area, since declared an abandoned site by the Canadian Federal Government.

Figure 4.4 Work Area locations



Source: Archer, Cathro & Associates (1981) Limited.

Table 4.2 Work Area claims

Work Area	Claim groups	Area (km^2)
1	BBB	78.5
2	Dic, Eagle, Etzel, JCS, Klaza, Lone, VG, Vic, and Wedge	89.8
3	Dade, Krast	21.0
4	Queen, Val	25.2
5	Nor	13.2
6	Bull, D, Desk, J. Bill, JBF, Jon-Wedge, Lone, Ox, Rat, and Sked	57.8

Source: Archer, Cathro & Associates (1981) Limited.

The Klaza Property is the primary focus of this report. No Mineral Resources or Mineral Reserves have been defined within any of the other Work Areas.

The mineral claims comprising the Property can be maintained in good standing by performing approved exploration work to a dollar value of $100 per claim per year and an additional $5 fee per claim for an Application for a Certificate of Work. The QP is not aware of any unusual encumbrances associated with lands underlain by the Property, except that some of the mineral claims overlap with placer claims owned by independent miners. Placer claims give the owner the right to extract metals and minerals from near-surface unconsolidated gravels, while mineral claims apply to metals and minerals in bedrock. There are no agreements relating to the overlapping placer claims.

Exploration is subject to Mining Land Use Regulations of the Yukon Mining Quartz Act and the Yukon Environmental and Socio-economic Assessment Act (YESAA). Yukon Environmental and Socio-economic Assessment Board (YESAB) approval must be obtained and a Land Use Approval must be issued, before large-scale exploration is conducted. Approval for this scale of exploration has been obtained, for the deposit area, by Rockhaven under Class III Mining Land Use Approval LQ00434, which covers the majority of Work Area 2, and expires 6 December 2020. Rockhaven has also obtained three additional Class III Mining Land Use Approvals for other areas of the Property. These are LQ00344 (Work Area 3), LQ00357 (parts of Work Area 2), and LQ00493 (Work Area 6) which expire on 3 April 2022, 3 May 2022, and 25 July 2028, respectively.

Potential mine development on the Property will require YESAB approval, a Yukon Mining License and Lease issued by the Yukon Government and a permit issued by the Yukon Water Board.

The claim posts on the Property have been located by Rockhaven using hand-held GPS devices.

The Property lies within the traditional territory of the Little Salmon Carmacks First Nation (LSCFN), and the north-western corner overlaps with the traditional territory of the Selkirk First Nation (SFN). In 2012, the White River First Nation made a unilateral claim that its traditional territory covers an area that includes the Property. The validity of this claim is uncertain. To the best of the QP's knowledge there are no encumbrances to the Property relating to First Nation Settlement Lands.

On 5 August 2015, Rockhaven and LSCFN signed an exploration benefits agreement (EBA) related to Rockhaven's exploration activities at its Klaza project, which is located within the LSCFN traditional territory. The EBA provides a framework under which Rockhaven and LSCFN will advance the Klaza Project through a mutually beneficial working relationship.

Outstanding environmental liabilities relating to the Property are currently limited to progressive reclamation during seasonal exploration activities and final decommissioning required prior to expiration of the Land Use Approval. Progressive reclamation generally entails backfilling or recontouring disturbed sites and leaving them in a manner conducive to re-vegetation of native plant species. Back-hauling scrap materials, excess fuel and other seasonal supplies is also done. Final decommissioning requires that: all vegetated areas disturbed by Rockhaven's exploration be left in a manner conducive to re-vegetation by native plant species; all petroleum products and hazardous substances be removed from the site; all scrap metal, debris and general waste be completely disposed of; structures be removed; and, the site be restored to its previous level of utility.

The QP does not know of any other significant factors that may affect access, title, surface rights, or ability of Rockhaven to perform work on the Property.

5 Accessibility, climate, local resources, infrastructure, and physiography

The Property lies 50 km due west of the village of Carmacks, which is the nearest supply centre. Carmacks can be reached from Whitehorse by driving 180 km north on Highway #2, the Klondike Highway. In addition to being road accessible from Whitehorse, the Yukon’s territorial capital and main transportation hub, Carmacks is also located 351 km from the year-round tidewater port at Skagway, Alaska (Figure 5.1). From Carmacks, the Property is accessible by a 69 km road.

Carmacks formerly serviced the mine and mill operations of the Mount Nansen Mine. The Yukon Territorial Government maintains a haulage road that extends 60 km from Carmacks to the Mount Nansen Mine site, which is located 9 km by road south of the Property through moderately hilly terrain.

The proposed facility will be powered by territorial grid power delivered via a power line from Carmacks to be constructed along the existing roadway.

The proposed mineral processing plant will be located near the open pits to minimize haul distances. A site has been identified down slope of the pits on a gently sloping area. Mineralized rock stockpiles will be avoided during operations by careful scheduling of production. Tailings will be stored in nearby containment areas.

The existing road from Carmacks will be extended to the mine-site to support operations. Many services are also available in Carmacks including hotel accommodations, restaurants, fuel sales, a nurse’s station, a 5,000 foot gravel east-west air-strip, various types of aircraft, and a Royal Canadian Mounted Poilice (RCMP) detachment.

The water demands of the process plant and potable water will be served by the water treatment plant, which is located at the processing plant operation.

The proposed project infrastructure details are covered in Section 18.

The existing site uses portable electrical generators to provide sufficient power for exploration programs currently planned on the Property. Local creeks provide sufficient water for camp and diamond drilling requirements.

All work programs to date have been conducted from a tent frame camp on the Property. Drilling and excavator trenching sites have been accessed using All Terrain Vehicles, four-by-four trucks or heavy equipment. In 2013, EBA Engineering Consultants Ltd. of Whitehorse was retained to prepare a Terrain and Geohazard Assessment and Access Route Evaluation.

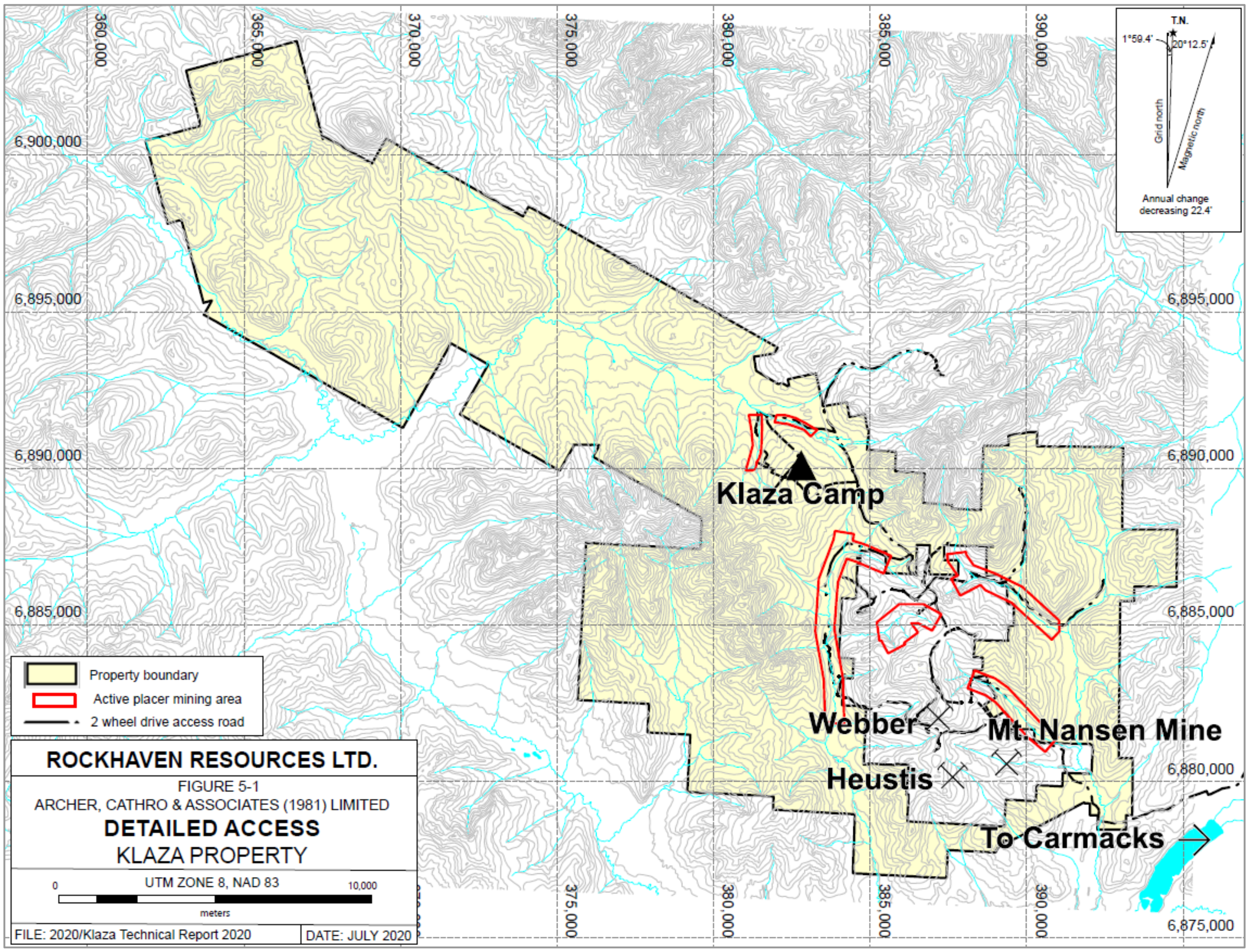
Since acquiring the Property in 2009, Rockhaven has consulted with LSCFN in recognition and respect of its traditional territory and has discussed the project with the local community. Meetings have been held, or written descriptions of work programs have been submitted, at least twice annually to provide updates of exploration conducted and work proposed.

Matrix Research Ltd. (Matrix) of Whitehorse conducted a Heritage Resources Overview Assessment of the Klaza area in 2011. This work classified zones of high, moderate and low potential for heritage resources within the Property and immediately peripheral lands. In 2012, Matrix conducted ground studies and did not locate any heritage sites within the main areas of interest.

In 2019 a Heritage Resource Overview Assessment was conducted by Ecofor Consulting of Whitehorse in exploration areas on the Property, east of the Study Area. While this assessment identified multiple areas with elevated potential for surface / subsurface heritage resource sites, none are within the Study Area.

Surface rights will be secured during the conversion of mineral claims to Quartz Mining Leases at a later stage in the project. The Company does not anticipate any concerns from stakeholders.

Figure 5.1 Detailed access – Klaza Property



Source: Archer, Cathro & Associates (1981) Limited.

The Property is situated in the southern part of the Dawson Range, a belt of low mountains, hills, and relatively mature river systems. The Property is drained by tributaries of the Klaza River and Nansen Creek, both of which flow into the Nisling River, part of the Yukon River watershed.

The eastern part of the Property covers several broad ridges and valleys. The rest of the Property is characterized by low hills and valley bottoms, flanking the Klaza River. The main areas of interest lie along a north-westerly elongated ridge. Elevations on the Property range between 1,200 m and 1,500 m above sea level (asl). Tree line is at 1,200 m asl on north-facing slopes and about 1,400 m asl on south facing slopes. Areas above treeline are vegetated with low-lying grass, moss, and sparse brush. The density and size of vegetation gradually increases toward lower slopes and valley bottoms, where stunted spruce are surrounded by an understory of dwarf birch and a thick layer of sphagnum moss.

The Klaza area escaped Pleistocene continental glaciation but experienced some local Pleistocene to Holocene valley and alpine glaciation. Outcrop is non-existent across most of the Property and overburden typically consists of a few centimetres of organics, 0 to 5 centimetres (cm) of volcanic ash and up to 200 cm of loess and immature soil mixed with locally derived rock fragments, over weathered bedrock. At lower elevations, thick layers of fluvial material, glacio-fluvial outwash and till blanket the valley floors. Permafrost is extensive, particularly on north- and west-facing slopes.

The area has a continental climate with low levels of precipitation and a wide temperature range. Summers are normally pleasant with extended daylight hours whereas winters are long and cold. Although summers are relatively warm, snowfall can occur in any month at higher elevations. The Property is mostly snow free from late May to late September. According to Environment Canada, summer temperatures in the nearest community of Carmacks average 18°C during the day and 5°C at night. Winter temperatures average -12°C during the daytime. Total annual precipitation over the 1961 to 1990 period averaged 277 mm, with about 92 cm of snow (Environment Canada 2015). Typical exploration programs in the Yukon can extend from May to October with mining occurring year round.

6 History

Exploration history was mostly compiled from the Yukon Minfile Database (Deklerk 2005) and assessment reports submitted to the Whitehorse Mining Recorder. The assessment reports were written prior to the implementation of NI 43-101. Nonetheless, they were consistent with professional standards at the time and were accepted by the mining recorder.

Between 1899 and 2014, several operators worked on various claim groups that now lie within the boundaries of the Property. Although strong geochemical and geophysical anomalies were identified by this work, follow-up trenching and drilling produced sporadic results, in part because of physical and technological limitations. Early technical limitations included early bulldozer trenches that rarely reached bedrock, often because of permafrost, and small diameter drillholes that typically gave poor core recoveries, especially in the more fractured, mineralized intervals.

As discussed in Section 4, for the purpose of this report, the Property has been subdivided into Work Areas 1 – 6. Work area 2 closely approximates the boundary of the Property prior to acquisitions in July 2015.

The following tables, organized by Work Area, summarize historical exploration and list the year of work, owner / operator, claim group name, work performed, and highlight results for each exploration program (Tarswell and Turner 2014).

Table 6.1 Work Area 1 exploration history

Year of work (report)	Owner / operator	Claim group / target	Work performed	Results
1969 (Cathro and Culbert, 1969)	Dawson Range Joint Venture	BBB area	Regional exploration including geochemical sampling	A stream sediment sample returned 10 parts per million (ppm) Cu and 51 ppm Pb.
1974 (Cathro, 1974)	Klotassin Joint Venture	BBB area	Prospecting and mapping	N/A
1975 (Cathro and Culbert, 1976)	Klotassin Joint Venture	BBB area	Soil and stream sampling	Highest samples ran at 114 ppm Cu, 48 ppm Pb, and 155 ppm Zn.
1980 (Archer and Onasick, 1980)	NAT Joint Venture	BBB area	Reanalysis of over 5,000 geochemical samples.	Most anomalous samples graded 110 - 500 parts per billion (ppb) Au.
1985	Geological Survey of Canada	BBB area	Stream and water sampling	N/A
1986 (Main, 1987)	Chevron Minerals Ltd.	Toast	Toast claims staked - some overlap with current BBB property.	N/A
1987 (Main, 1987)	Big Creek Joint Venture	Toast	Mapping, prospecting, geochemical sampling.	Highest soil sample value was 55 ppb Au.
1987 (Curley, 1987)	E. Curley	Jam	Staked Jam claims	Failed to locate the source of previous stream anomaly.
2010 (Chung, 2011)	Strategic Metals Ltd.; Wolverine Metals Ltd.	BBB 1-16	Staked BBB 1-16, geochemical sampling. Optioned claims to Wolverine Minerals Ltd., which then staked BBB 17-255.	The best sample returned 43 ppb Au, 84 ppm As, 158 ppm Cu, 192 ppm Zn, and 16 ppm Pb.
2011	Wolverine Minerals Ltd.	BBB 1-255	Geochemical sampling (1,846 soil samples), prospecting, geophysical surveys.	Maximum soil sample values graded 836 ppm Au, 82 ppm As, 32.9 ppm Ag, and 186 ppm Cu.
2012 (Mac Gearailt, 2012)	Goldstrike Resources Ltd.	BBB 1-255	Prospecting, mapping immediately SE and E of the BBB property.	High-grade samples returned 116.5 ppb Au, 1.7 Ag, and 479.9 ppm As.
2013 (Burrell, 2013b)	StrategicMetals Ltd.	BBB 1-255	Prospecting, mapping and completed an airborne magnetic survey.	Mapped linear lows that correlated with geophysical lows.
2014 (Burrell, 2014)	Strategic Metals Ltd.	BBB 1-384	Staked BBB 256-384 claims and ran a 987 sample soil grid.	Correlation between geophysical lows and soil anomalies was poor, possibly due to thick overburden and difficulties in locating geophysical trends.

Source: Archer, Cathro & Associates (1981) Limited.

In 2010, Strategic Metals Ltd. (Strategic) staked the BBB 1-16 claims and optioned the Property to Wolverine Minerals Ltd., which expanded the claim block to adjoin the Property (Chung 2011). Wolverine subsequently dropped its option. In early August 2014, Strategic staked the BBB 256-384 claims and sold all of the BBB claims to Rockhaven in 2015.

Table 6.2 Work Area 2 exploration history

Year of work (report)	Owner / operator	Claim group / target	Work performed	Results
1937 (none)	K. Paulson	N/A	Prospecting	Rumour of high-grade silver-lead float (Eaton, 1986).
1948	G. Dickson	N/A	Bulldozer trenching	N/A (Eaton, 1986)
1967 (none)	J. Smith	May	Soil sampling and bulldozer trenching	Anomalous silver and lead soil geochemistry, but no vein was intersected by trenching (Campbell and Guardia, 1969).
1968 (Parker, 1968)	Esansee Explorations Limited	May	Geochemical and geophysical surveys and bulldozer trenching	Peak soil values were 8,200 ppm lead, 125 ppm silver and 800 ppm arsenic. Specimen samples of "fissure" vein cut in a bulldozer trench returned peak values of 34.3 g/t gold, 2,057.1 g/t silver, 44% lead, and less than 1% zinc.
1969 (Campbell and Guardia, 1969)	Esansee Explorations Limited	May	Bulldozer trenching and road building	A chip sample returned 15.09 g/t gold and 483 g/t silver over 1.83 m. A 14 km road (considered an extension of the Mount Nansen Mine road) was built from the Mount Nansen Mine campsite to the May claims.
1971 (McClintock, 1986)	Cyprus Mines Corporation	Wedge	Mapping, geochemical sampling, geophysical survey, trenching, 6 diamond drillholes, and 1 percussion hole totalling 1,115 m drilled.	Drill logs were submitted to the Yukon Government, but were not discussed. No other results available.
1973 (Dickinson and Lewis, 1973)	Area Exploration Company	Betty, Bun, and Crow	Percussion (283.5 m) and diamond (776.1 m) drilling.	Two percussion drillholes (283.5 m) and three diamond drillholes (776.1 m) were completed to test a 700 by 900 m copper-in-soil geochemical anomaly.
1975 (Aho et al., 1975)	Kerr Addison Mines Limited	Dic	Geological, geochemical, and geophysical surveys.	A total of 216 soil and 45 rock samples were collected for analysis and 4.0 line kilometres of magnetic and 2.8 line kilometres of very low frequency electromagnetic, magnetic (VLF-EM) surveys were conducted.
1980 (Sauders, 1980a and 1980b)	BRX Mining & Petroleum Corp.	Tawa	Geochemical sampling, bulldozer trenching, and diamond drilling (447.3 m in 7 holes).	Soil sampling identified north-westerly trending linear anomalies. The best interval from diamond drilling returned 8.64 g/t gold and 25.68 g/t silver over 6.0 m including 24.5 g/t gold and 50.1 g/t silver over 1.5 m (80-6).
1981 (Brownlee, 1981)	BRX Mining & Petroleum Corp.	Tawa	Electromagnetic and proton magnetometer surveys	Both surveys highlighted coincident, north-westerly trending anomalies.
1984 (McClintock, 1986)	G. Dickson	Wedge	Staked the Wedge claims	N/A
1985 (McClintock, 1986)	G. Dickson	Wedge	Limited trenching	N/A

Year of work (report)	Owner / operator	Claim group / target	Work performed	Results
1986 (McClintock, 1986)	Pearl Resources Ltd.	Wedge	222 soil geochemical samples	Identified several NW trending soil anomalies (Au, Ag, Pb, and Zn).
1986 (Heberlein and Lyons, 1986)	Kerr Addison Mines Ltd.	Vic, VG	Mapping, sampling, geophysical surveys, trenching, and diamond drilling.	Magnetometer survey matched known alteration zones and highlighted possible extensions of these zones. Drilling revealed that quartz veining is discontinuous.
1986 (Eaton, 1986)	Chevron Minerals Limited	Tawa	Mapping, prospecting, bulldozer trenching and an electromagnetic survey.	Deepening historical trenches returned 5.28 g/t gold and 132.0 g/t silver over 2 m. Geophysical and geochemical anomalies extended to 1,900 and 2,000 m, respectively.
1986 (McClintock, 1986)	Pearl Resources Ltd.	Etzel	Geological mapping and geochemical sampling.	Geochemical sampling returning gold-in-soil values up to 310 ppb and silver-in-soil values up to 54.5 ppm. The best chip sample returned 0.99 g/t gold over 1 m.
1987 (Eaton and Walls, 1987)	Chevron Minerals Limited	Tawa	Road building, bulldozer trenching and claim staking.	Trench T-11 at the Klaza zone returned 4.22 g/t gold and 47.3 g/t silver over 8.0 m including 4.27 g/t gold and 86.7 g/t silver over 1.0 m. Trenching at the BRX zone intersected 3.12 g/t gold and 46.3 g/t silver over 7.0 m including 6.99 g/t gold and 41.1 g/t silver over 1.5 m (T-14) and 6.86 g/t gold and 160.1 g/t silver over 2.5 m (T-16).
1988 (Eaton and Walls, 1988)	Chevron Minerals Limited	Tawa	Excavator trenching (1,924 m) and six diamond drillholes (377 m).	Trenching exposed a vein in T-22 that returned 16.3 g/t gold and 1,289.1 g/t silver over 1.7 m. A drillhole testing the down-dip continuity of this interval returned 6.03 g/t gold and 129.9 g/t silver over 1.36 m (Hole 88-6).
1988 (Sutherland, 1988)	Chesbar Resources Inc.	Dic	Stream sediment sampling, prospecting and trenching.	Silt sampling returned values up to 1,050 ppb gold, while 75% of values were less than 10 ppb gold. Prospecting yielded a peak value of 220 ppb gold. Historical trenches (7.8 km) were deepened, but hindered by frozen ground.
1989 (Eaton, 1989)	BYG Natural Resources Inc. and Chevron Minerals Ltd.	Tawa	Road construction and excavator trenching (580 m).	N/A
1992 (Langdon, 1992)	Aurchem Exploration	Wedge	RC drilling	RC drilling confirmed anomalies identified by geophysics, trenching, and soil geochemistry. Some samples returned Cu-Mo anomalies, with minor Au and Ag.
1996 (Dujakovic et al., 1996)	BYG Natural Resources Inc.	Tawa, KR and Dic	VLF-EM and geochemical surveys.	North-westerly trending VLF-EM and magnetic anomalies and soil geochemical values up to 1,825 ppb gold and 1,049 ppm copper.

Year of work (report)	Owner / operator	Claim group / target	Work performed	Results
1999 (Stroshein, 1999)	BYG Natural Resources Inc.	Gerald and Tawa	Overburden stripping and diamond drilling (307.8 m in 3 holes).	Klaza zone drilling returned 3.82 g/t gold and 84.7 g/t silver over 5.05 m (TA-98-8). BRX zone drilling returned 0.24 g/t gold and 1.3 g/t silver over 55.75 m (TA-98-9).
2001 (Stroshein, 2001)	Aurchem Exploration	Vic	Prospecting, soil sampling, 3 trenches exposing Au-bearing quartz veins.	Trenching exposed faults post-dating mineralization.
2001 (Stroshein, 2001)	Aurchem Exploration	Wedge	Mapping, soil sampling, and trenching collecting 42 chip samples.	Silicification was revealed in trenches. Chip samples have anomalous Au and Ag.
2002 (Stroshein, 2003)	Aurchem Exploration	Vic	Prospecting, mapping, geochemical sampling, trenching.	Best trench assays were 2.55 g/t Au over 5.4 m.
2003 (Stroshein, 2004)	Aurchem Exploration	Vic, JCS	Prospecting, 816 soil samples on Vic claims, 173 samples on JCS claims, 5 trenches, and 122 chip / grab / float samples.	Trenching revealed quartz breccia vein zone and a quartz stringer zone, both bearing Au.
2003 (Stroshein, 2004)	Aurchem Exploration Ltd.	Etzel	Excavator trenching	The best trench result was from a clay-rich zone that graded 6.05 g/t gold and 15.3 g/t silver over 6.0 m.
2004-2006 (Ellemers and Stroshein, 2005; Stroshein, 2008)	Aurchem Exploration	Vic	Trenching, RC drilling, and diamond drilling.	2004 drilling found grades of 12.68 g/t Au over 1.22 m and 2.56 g/t Au over 1.07 m in the 28 Main zone. The 28 Extension veining appears to be discontinuous.
2005 (Wengzynowski, 2006)	ATAC Resources Ltd.	Klaza	Staked Klaza 1-24 claims before optioning them to Bannockburn Resources Limited.	N/A
2007 (Stroshein, 2008)	Aurchem Exploration	Vic	Soil geochemistry sampling, trenching, and 63 chip samples.	Veins are associated with porphyry dykes and fault zones trending 80°-115°, steeply dipping N and S. Average Maverick vein samples graded at 49.89 g/t Au. Average 2,650 vein samples graded 16.01 g/t Au.
2009 (Turner and Tarswell, 2011)	ATAC Resources Ltd. – Rockhaven Resources Ltd.	Klaza	ATAC sold the Klaza 1-24 claims to Rockhaven Resources Ltd.	N/A
Includes work carried out by Issuer				
2010 (Turner and Tarswell, 2011)	Rockhaven Resources Ltd.	Klaza	Claim staking, geophysical surveying, diamond drilling, excavator trenching, and soil geochemical sampling.	Best drill intercept returned 3.23 g/t gold and 117.7 g/t silver over 36.50 m. Several additional coincident geochemical and geophysical anomalies were identified. Peak soil geochemical values were 856 ppb gold, 5.8 ppm silver, 494 ppm lead and 349 ppm arsenic.

Year of work (report)	Owner / operator	Claim group / target	Work performed	Results
2011 (Great Bear Resource, 2012a)	Great Bear Resources Ltd.	Etzel	Diamond drilling, excavator trenching, and soil geochemical sampling.	Best drill intercept returned 0.58 g/t gold and 2.4 g/t silver over 40.65 m.
2012 (Great Bear Resource, 2012b)	Great Bear Resources Ltd.	Etzel	Diamond drilling and soil geochemical sampling.	Best drill intercept returned 2.30 g/t gold and 7.0 g/t silver over 1.16 m.
2012	Rockhaven Resources Ltd.	Dic and Eagle	Purchased Dic and Eagle claims from J. Dickson	N/A
2012	Rockhaven Resources Ltd.	Etzel	Purchased Etzel claims from Ansell Capital Corp.	N/A
2012	Rockhaven Resources Ltd.	VIC, VG, J. Bill#, D, Bull, JBF and Jon-Wedge	Purchased claims from Aurchem Exploration Ltd.	N/A

Source: Archer, Cathro & Associates (1981) Limited.

The main exploration programs and results for Work Area 2 are described in more detail in the technical report entitled "Geology, Mineralization, Geochemical Surveys, Geophysical Surveys, Diamond and Percussion Drilling, Metallurgical Testing and Mineral Resources on the Klaza Property, Yukon, Canada" dated 11 March 2015 and amended 19 June 2015 (Wengzynowski et al. 2015).

In 2005, ATAC staked the Klaza 1 – 24 claims, which form the core of Work Area 2. Rockhaven purchased the Klaza claims from ATAC in 2009.

In June 2011, Ansell Capital Corp. (Ansell) purchased the Etzel claims from Aurchem Exploration Ltd. (Aurchem). Rockhaven purchased the Etzel claims from Ansell in 2012 along with the VG, VIC, J. Bill#, D, Bull, JBF, and Jon-Wedge claims from Aurchem. These claims now form the eastern edge of Work Area 2.

In fall 2011, Rockhaven purchased the Dic and Eagle claims from Aurchem. These claims adjoin the Klaza claims and are the southernmost claims in Work Area 2.

Table 6.3 Work Area 3 exploration history

Year of work (report)	Owner / operator	Claim group / target	Work performed	Results
1987 (Hulstein, 1988)	G. Dickson	Nulee, JS, Moon, and Robert	Mapping, trenching, and geochemical sampling.	The Bear zone returned anomalous soil geochemistry. The source of Montgomery Creek zone anomalous float was untraceable.
1989 (Brent, 1991)	E. Curley	Grizzly	4 bulldozer trenches, hand trenching, and rock samples.	Trench intervals of 7.2 g/t Au over 3.5 m and 15.4 g/t Au over 1.5 m.
1990 (Brent, 1991)	E. Curley	Grizzly	8 trenches, chip sampling, and mapping.	Located felsic dykes associated with the Grizzly Vein (now the V1 vein). A grab sample graded 42.5 g/t Au, 57.9 g/t Ag, >3% As, 185 ppm Cu, 28 ppm Mo, 979 ppm Pb, 91 ppm Sb, 34 ppm W, and 410 ppb Hg.
1994 (Pautler, 1994)	E. Curley, Teck Corporation	Grizzly	Trench mapping and rock sampling.	Trench chip samples graded at 3.52 g/t Au and 8.8 g/t Ag over 1.5 m.
2003 (Hulstein, 2003)	J. Dickson	JRW	Prospecting, chip samples, and soil sampling to explore V1.	Vein samples ran 1.24 g/t Au, 6,756 ppm As, 68.3 ppm Bi, and 51.2 ppm W over 2.2 m. Soil geochemistry revealed an anomaly containing 31.3 ppb Au, 53.1 ppm As, and 1.5 ppm Bi.
2009 (Smith, 2010)	Strategic	Dade	Staked the Dade 1-16 claims and ran a small soil geochemistry grid. Dade claims 17-96 were staked after assay results were returned.	Soil anomalies graded up to 113 ppb Au, while samples from trench floors ran 4,280 ppb Au.
2011 (Smith, 2011)	Wolverine	Dade	CanDig and excavator trenching, soil geochemistry, and geophysical surveys.	Trenching verified V1 and located V2 veining and stockwork.
2012 (Burrell, 2013a)	Strategic	Dade	23 diamond drillholes totalling 2,043.39 m and 24 RC drillholes totalling 1,426.47 m.	Diamond holes did not intersect quartz veining, but found some results in alteration zones such as 2.45 g/t Au over 1.37 m. RC drilling had few significant intervals, including 5.32 g/t Au over 1.53 m.

Source: Archer, Cathro & Associates (1981) Limited.

In December 2009, Strategic staked the Dade 1 – 16 claims and sold them to Rockhaven in 2015.

Table 6.4 Work Area 4 exploration history

Year of work (report)	Owner / operator	Claim group / target	Work performed	Results
1934 (none)	G. Dickson	Val	Staked the Billy claims (now called Val).	N/A
1958 (Robinson, 1959)	Asbestos Corporation Ltd.	Val	Optioned the Billy claims, mapping, trenching, packsack drilling.	Exposed a quartz-feldspar porphyry dyke; alteration with galena and pyrite.
1979	Rex Silver Mines Ltd. (formerly Peso Silver Mines Ltd.)	Val	Transferred the Val property to Schweizerische Geselleschaft.	N/A
1981	Mount Nansen Corporation	Val	Acquired the Val property.	N/A
1981	BYG Natural Resources	Val	Re-staked some of the Val area as DD claims.	N/A
1983	Mount Nansen Corporation	Val	Conducted a feasibility study.	N/A
1984	BYG Natural Resources	Val	Purchased the Val property.	N/A
1985	BYG Natural Resources	Val	Re-staked some Val claims as ONT claims; Chevron Minerals Ltd optioned some claims.	N/A
1986-1988	BYG Natural Resources	Val	Ran an exploration program including mapping, soil geochemistry, geophysical surveys, trenching, and diamond drilling.	Identified a multi-element anomaly (Au, Ag, Zn, Sb, As, Cd, Bi, Cu, Mo) trending N-NW.
1988	Chevron Minerals Ltd.	Val	Dropped its options.	N/A
1995 (Carlyle, 1997)	E. Curley	Queen	Prospecting and surface mapping.	Located quartz-feldspar porphyry dykes.
1997 (Carlyle, 1997)	E. Curley	Queen	Program included geochemical soil sampling followed by trenching.	Soil geochemistry and trench samples gave low Au values (138 ppb Au and 47 ppb Au, respectively).
2003	Mr Trerice	Val	Staked the Val claims.	N/A
Includes work carried out by Issuer				
2011	Rockhaven Resources Ltd.	Val	Signed an option agreement for the Val property.	N/A
2012	Rockhaven Resources Ltd.	Val	Carried out geochemical soil sampling.	N/A

Source: Archer, Cathro & Associates (1981) Limited.

The Val claims were staked by Mr Trerice in 2003 in conjunction with placer activities along Back Creek. 38857 Yukon Inc. currently owns the placer claims on Back Creek. Rockhaven signed an option agreement with Mr Trerice on 21 September 2011, acquiring the right to earn a 100% interest in the Val property. In 2015, Rockhaven purchased a 100% interest in the Val property.

Table 6.5 Work Area 5 exploration history

Year of work (report)	Owner / operator	Claim group / target	Work performed	Results
1996	Conquest Yellowknife Resources Ltd.	Cow (Nor)	Geophysics VLF-EM, magnetometer. (Restaked as Nor claims by Strategic in 2015).	Delineated multiple VLF-EM and magnetic anomalies.

Source: Archer, Cathro & Associates (1981) Limited.

Table 6.6 Work Area 6 exploration history

Year of work (report)	Owner / operator	Claim group / target	Work performed	Results
1965	Mount Nansen Mines Ltd	Bit	Staked Bit claims 1-6	N/A
1966	Mount Nansen Mines Ltd	Bit	Mapping and geochemical sampling	N/A
1971	Area Exploration Company Ltd	Bit, Rusk	Optioned the Bit claims and staked Rusk claim 1-39	N/A
1972	Area Exploration Company Ltd	Rusk	Grid soil sampling	N/A
1973	Area Exploration Company Ltd	Rusk	Drilled one diamond drillhole	N/A
1974	J. Dickson	Lone	Staking around the Lone porphyry	N/A
1976	G. Dickson	LD, Swiss	Restaked as LD cl 1-14	N/A
1979	G. Dickson	LD, Swiss	Restaked as Swiss cl 1-62	N/A
1980	G. Dickson	LD, Swiss	Trenching	N/A
1981	G. Dickson	LD, Swiss	Trenching	N/A
1983	G. Dickson	J. Bill	Restaked as J. Bill cl 1-32	N/A
1984	G. Dickson	Rat, Bull	Trenching, added Rat c1 1-24 and Bull cl 1-28	N/A
1984	Kerr Addison Mines Ltd.	Lone	Prospecting	Rock samples yeilding peak values of 1.65 g/t gold and 14.8 g/t silver.
1985	Kerr Addison Mines Ltd.	Lone	Staking the Only claims	N/A
1986	Kerr Addison Mines Ltd.	Lone	Mapping, soil sampling, and VLF surveys	Identified VLF-EM north-west trending linear anomalies. Peak gold-in-soil of 150 ppb.
1987	E. Curley	Dows	Dows claims were staked. Hand trenching.	N/A
1988	Noranda Exploration Company Ltd	Dows	Optioned and expanded the Dows claims. Mapping, grid soil sampling, mechanized trenching, and geophysical surveys.	Two best soil samples returned: 1,100 ppb gold, 2.0 ppm silver, 460 ppm arsenic and 1,100 ppb mercury; and, 490 ppb gold, 4.4 ppm silver, 1,100 ppm arsenic, and 13,200 ppb mercury, respectively.
1988	Kerr Addison Mines Ltd.	Lone	Soil sampling and hand trenching	The best rock sample retruned 115 ppb gold, 555 ppm copper, and 115 ppb silver.
1989	Noranda Exploration Company Ltd	Dows	One diamond drillhole	Intersected a quartz breccia, which averaged 2.43 g/t gold over 7.5 m including 10.2 g/t gold over 1.5 m.
1995	Atna Resources Ltd.	Dows	Optioned Dows claims from E. Curley. Mapping, trenching, soil sampling, chip sampling.	N/A
1995	Conquest Yellowknife Resources Ltd.	Dows	Optioned the Property from Atna. Staked the Dows 119-124 claims.	N/A
1995	Aurchem Exploration Ltd	J. Bill, Rat, Bull	Aurchem acquired Dickson's claims J Bill, Rat, and Bull.	N/A

Year of work (report)	Owner / operator	Claim group / target	Work performed	Results
1996	Conquest Yellowknife Resources Ltd.	Dows	Diamond drilling	Anomalous drill intercepts, 0.51 g/t gold and 13.13 g/t silver over 2.61 m (DDH-96-2); 6.64 g/t with low silver over 5.90 m (DDH-96-6); and 0.34 g/t gold and 5.09 g/t silver over 11.10 m (DDH-96-8).
2003	Aurchem Exploration Ltd	J. Bill	Limited soil sample program. Pre-stripping for future trenching.	N/A
2006	R. Hulstein	Desk	Staked expired, Dows claims as Desk.	N/A
2007	Aurchem Exploration Ltd	J. Bill	Trenching and sampling stripped areas from 2003.	N/A
2009	Strategic Metals Ltd.	Sked	Stakes Sked claims	N/A
2010	Strategic Metals Ltd.	Sked	Grid soil sampling	The best results from soil sampling were strongly anomalous arsenic (up to 181 ppm) and copper (up to 140 ppm) and background to weakly anomalous gold (up to 23 ppb), lead (up to 14 ppm) and zinc (up to 85 ppm).
2010	Wolverine Minerals Corp.	Desk, Sked	Wolverine purchased Desk and Sked claims.	N/A

Source: Archer, Cathro & Associates (1981) Limited.

In 1983 G. Dickson staked the J. Bill 1-32 and expanded by adding the Rat claims 1 – 24 to the south in 1983 and the Bull claims 1-8 to the east in 1984. In 1994 the J. Bill, Rat, and Bull claims were transferred to Dickson's widow, J. Dickson (YGS Minfile 115I 096).

The Desk claims were staked by R. Hulstein in 2006.

In winter 2009, Strategic staked the Sked claims to cover the potential along-strike extension of the mineralized zone on the Desk claims. In summer 2010, Strategic expanded the Property to the north-west to cover a historical gold anomaly mapped on the west side of an unnamed tributary of Lonely Creek (Chung 2011).

In summer 2015, Rockhaven purchased the J. Bill, Rat, Bull, Desk, and Sked claims.

The Lone claims were staked by Rockhaven in summer of 2017.

7 Geological setting and mineralization

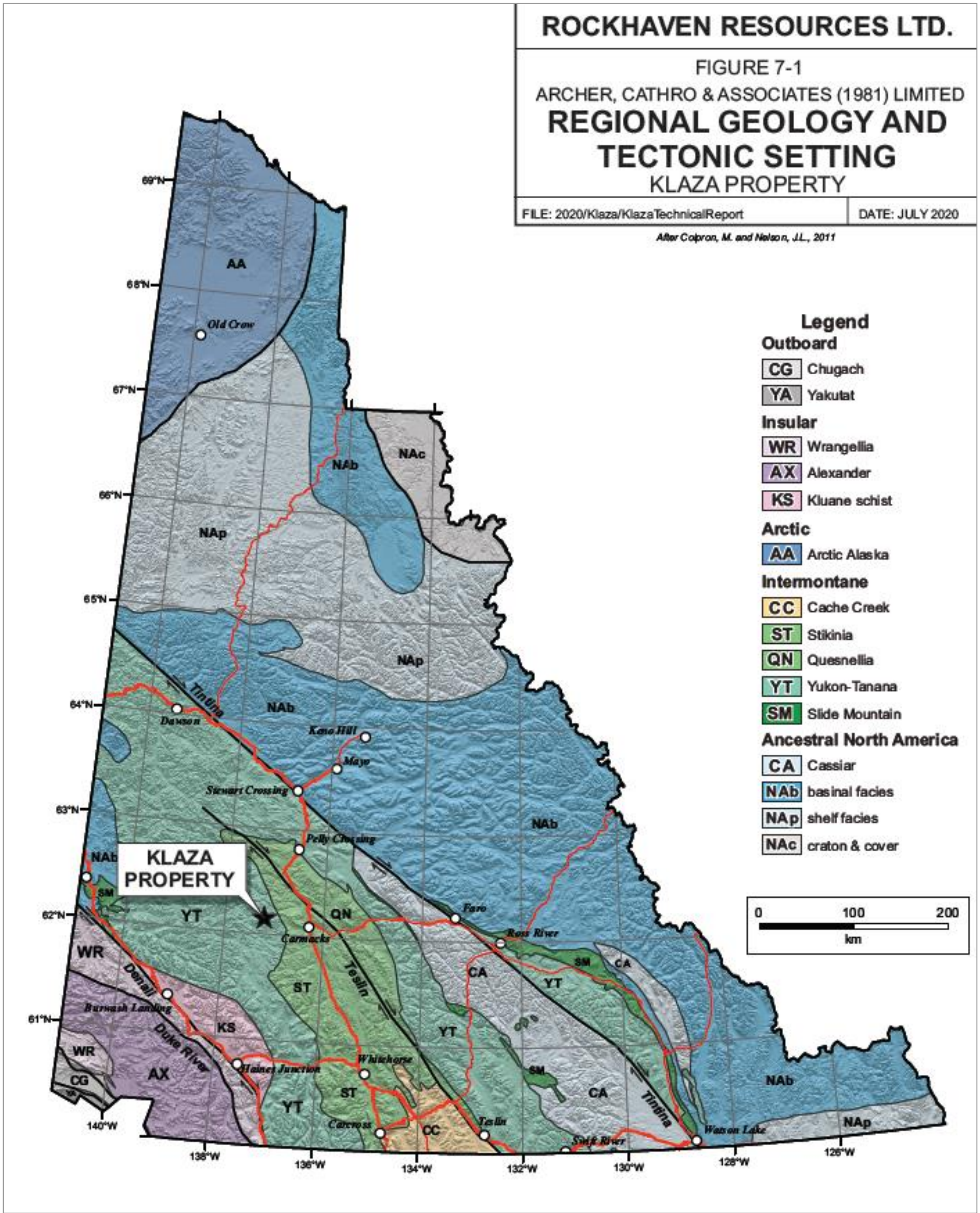
7.1 Regional geology

The area underlain by the Property was visited by J.B. Tyrrell and D.D. Cairnes for the Geological Survey of Canada (GSC) in 1898 and 1914, respectively, and has been mapped by H.S. Bostock (1936), D.J. Tempelman-Kluit (1974 and 1984) and G.G. Carlson (1987). The geology was revised in a compilation by Gordey and Makepeace (2000). Most recently, the GSC remapped the Mount Nansen area in 2016 (Ryan, et al. 2016) The following discussion is primarily based on maps prepared by Gordey and Makepeace, the Yukon Geological Survey (YGS) website and the most recent mapping by the GSC.

The Property lies within the Yukon-Tanana Terrane (YTT) approximately 100 km south-west of the Tintina Fault and 100 km north-east of the Denali Fault (Figure 7.1). YTT comprises a variety of Proterozoic and Paleozoic metavolcanic, metasedimentary and metaplutonic rocks, and represents both arc and back-arc environments (Colpron et al. 2006; Piercey et al. 2006). The Tintina Fault is a transcurrent structure that experienced about 450 km of dextral strike-slip movement during the Eocene. This movement offset an outlier of YTT in the Finlayson Lake District of south-eastern Yukon from the main body of YTT, which lies south-west of the fault. The Denali Fault is another major transcurrent structure that has seen hundreds of kilometres of dextral strike-slip movement.

Regional lithologies in the area of the Property are summarized in Table 7.1. The basement rocks are mainly schists and gneisses, which include metaplutonic, metasedimentary and metavolcanic rocks (Simpson Range Suite and Snowcap and Finlayson Assemblages) (Ryan et al. 2016). Basement rocks are cut by weakly foliated plutonic rocks (Long Lake Suite) that were metamorphosed and uplifted in the Jurassic, along with the schists and gneisses. The youngest rocks are unfoliated and are represented by five plutonic / volcanic events that occurred in the Cretaceous and Tertiary (Whitehorse Suite, Mount Nansen volcanics, Prospector Mountain Suite, Carmacks volcanics and the newly identified, Casino Suite) (Sanchez et al. 2014). The Casino Suite is of particular significance, because it is associated with most of the epithermal vein and porphyry deposits in the Dawson Range Gold Belt. Intrusions related to the Casino Suite were emplaced from 72 to 79 million years (Ma).

Figure 7.1 Regional geology and tectonic setting Klaza Property



Source: Archer, Cathro & Associates (1981) Limited.

Table 7.1 Regional lithologies

Upper Cretaceous	
uKC	**uKC: Carmacks Group** A volcanic succession dominated by basic volcanic strata (1), but including felsic volcanic rocks dominantly (?) at the base of the succession (2) and locally, basal clastic strata (3) (70 ma approx): 1 Augite olivine basalt and breccia; hornblende feldspar porphyry andesite and dacite flows; vesicular, augite phyric andesite and trachyte; minor sandy tuff, granite boulder conglomerate, agglomerate and associated epiclastic rocks (**Carmacks Gp., Little Ridge Volcanics, Casino Volcanics**). 2 Acid vitric crystal tuff, lapilli tuff and welded tuff including feeder plugs and necks; felsic volcanic flow rocks and quartz feldspar porphyries; green and purple massive tuff-breccia with feldspar phyric fragments (**Carmacks Gp., Donjek Volcanics, some rocks formerly mapped as Mount Nansen Gp.; the felsic part of the Carmacks Gp. is difficult to distinguish from similar Tertiary and Mid- Cretaceous (Mount Nansen) felsic volcanic strata**). 3 Medium bedded, poorly sorted, coarse to fine-grained sandstone, pebble conglomerate, shale, tuff, and coal; massive to thick bedded locally derived granite or quartzite pebble to boulder conglomerate (**Carmacks Gp**.).
Late Cretaceous to Tertiary	
LKgP	**LKgP: Prospector Mountain Suite** Grey, fine to coarse-grained, massive, granitic rocks of felsic (q), intermediate (g) and rarely mafic (d) composition plus related felsic dykes (f): q. Quartz monzonite, biotite quartz rich granite; porphyritic alaskite and granite with plagioclase and quartz-eye phenocrysts; biotite and hornblende quartz monzodiorite, granite, and leucocratic granodiorite with local alkali feldspar phenocrysts (**Prospector Mountain Suite, Carcross Pluton**). g. Hornblende-biotite granodiorite, hornblende diorite, quartz diorite (**Wheaton Valley Granodiorite**). d. Coarsely crystalline gabbro and diorite. f. Quartz-feldspar porphyry.
LKgC	**LKgC: Casino Suite** Grey fine to coarse-grained, massive granitic rocks of intermediate (g) composition and related felsick dykes (f): g. hornblende-biotite granodiorite, hornblende diorite, quartz diorite (**Wheaton Valley Granodiorite**) f. quartz-feldspar porphyry
Mid-Cretaceous	
mKN	**mKN: Mount Nansen Group** Massive aphyric or feldspar-phyric andesite to dacite flows, breccia and tuff; massive, heterolithic, quartz- and feldspar-phyric, felsic lapilli tuff; flow-banded quartz-phyric rhyolite and quartz-feldspar porphyry plugs, dykes, sills and breccia (**Mount Nansen Gp., Byng Creek Volcanics, Hutshi Gp**.).
mKW	**mKW: Whitehorse Suite** Grey, medium to coarse-grained, generally equigranular granitic rocks of felsic (q), intermediate (g), locally mafic (d) and rarely syenitic (y) composition: q. Biotite quartz-monzonite, biotite granite and leucogranite, pink granophyric quartz monzonite, porphyritic biotite leucogranite, locally porphyritic (K-feldspar) hornblende monzonite to syenite, and locally porphyritic leucocratic quartz monzonite (**Mount McIntyre Suite, Whitehorse Suite, Casino Intrusions, Mount Ward Granite, Coffee Creek Granite**). g. Biotite-hornblende granodiorite, hornblende quartz diorite and hornblende diorite; leucocratic, biotite hornblende granodiorite locally with sparse grey and pink potassium feldspar phenocrysts (**Whitehorse Suite, Casino Granodiorite, McClintock Granodiorite, Nisling Range Granodiorite**). d. Hornblende diorite, biotite-hornblende quartz diorite and mesocratic, often strongly magnetic, hypersthene-hornblende diorite, quartz diorite and gabbro (**Whitehorse Suite, Coast Intrusions**). y. Hornblende syenite, grading to granite or granodiorite (**Whitehorse Suite**).

Early Jurassic	
EJL	**EJL: Long Lake Suite** Mostly felsic granitic rocks (q) but locally grading to syenitic (y): q. Massive to weakly foliated, fine to coarse grained biotite, biotite- muscovite and biotite-hornblende quartz monzonite to granite, including abundant pegmatite and aplite phases; commonly K-feldspar megacrystic (**Long Lake Suite**). y. Resistant, dark weathering, massive, coarse to very coarse-grained and porphyritic, mesocratic hornblende syenite; locally sheared, commonly fractured and saussuritized; locally has well developed layering of aligned pink K-feldspar tablets (**Big Creek Syenite**).
Late Triassic	
LTrS:	**LTrS: Stikinie Suite** Coarse-grained, foliated, gabbroic hornblende orthogneiss; coarse-grained hornblende-biotite granite and granodiorite with K-feldspar megacrysts; foliated, fine to medium-grained hornblende quartz diorite to diorite with minor biotite (**Tally Ho leucogabbro, Friday Creek diorite, King Lake granite**).
Upper Devonian to Lower Mississippian	
DMF	**DMF: Finlayson Assemblage** assemblage of mafic (v) to felsic (f) metavolcanic rocks of arc and back-arc affinities; carbonaceous pelite, metachert (bp); minor quartzite, metavolcaniclastic rocks (s); marble (c); ultramafic rocks and metagabbro (um): v. Medium to dark green intermediate to mafic volcanic and volcaniclastic rocks; fine-grained amphibolite and greenstone (**Little Kalzas, Ram Creek, Cleaver Lake, Tutchitua, Fire Lake fms; Big Salmon Complex**). f. Felsic metavolcanic rocks, white quartz-muscovite schist, metaporphyry (**Kudz Ze Kayah, Wolverine, Waters Creeek fms**). bp. Dark grey to black carbonaceous metasedimentary rocks, metachert (**Nisana, Swift River, Grass Lakes, Wolverine fms**). s. Light green to grey, fine-grained siliciclastic and metavolcaniclastic rocks; arkosic grit and sandstone; chert and minor limestone (**Drury, Pelmac, Little Kalzas, Tutchitua fms; Big Salmon**). c. Light grey to white marble, locally crinoidal (**Little Kalzas fm**). Um. Ultramafic rocks, serpentinite; metagabbro (**Fire Lake fm**).
Early Mississippian	
MSR	**MSR: Simpson Range Suite** Foliated granitoid of mainly granodiorite to tonalite compsition (g), locally granite and augen granite (q), rare gabbro (gb): g. Foliated to strongly foliated, fine to medium-grained, hornblende-bearing metagranodiorite, metadiorite and metatonalite. q. Foliated metagranite, quartz monzonite and granodiorite; augen granite. gb. Locally metagabro
Neoproterozoic and Paleozoic	
PDS	**PDS: Snowcap Assemblage** Assemblage of dominantly metasiliciclastic rocks (s), minor marble (c), mafic metavolcanic rocks (v), and ultramafic rocks (um); intruded by Devonian-Mississippian calc-alkaline plutons of the Grass Lakes and Simpson suites; locally metamorphosed to blueschist and eclogite facies (e): s. Polydeformed and metamorphosed quartzite, psammite, pelite and marble; minor greenstone and amphibolite (**Snowcap, Dorsey, part of Big Salmon complexes; North River fm**). c. Light grey to buff weathering marble, generally lenticular and discontinueous. v. Medium to coarse-grained amphibolite, commonly garnet-bearing; greenstone; minor marble (**Snowcap, Dorsey complexes**). Um. Ultramafic rocks, serpentinite; metagrabbro; metapyroxenite (Dorsey complex). e. Metasiliciclastic and mafic meta-ignewous rocks, locally metamorphosed to eclogite blueshist (**Quiet Lake, Faro-Ross River, Simpson Lake**).

Source: Archer, Cathro & Associates (1981) Limited. Modified after Gordey and Makepeace (2000).

7.2 Property geology

Detailed mapping on the Property has been limited by sparse outcrop exposure and extensive vegetation cover. Cursory mapping has been done on the flank of Mount Nansen and from frost boils in the Klaza River valley (Aho et al. 1975). The geology map shown in Figure 7.2 has been interpreted from regional mapping, trenching, drilling and geophysical surveys, conducted on various parts of the Property. This corresponds to Work Area 2 as discussed in Section 6.

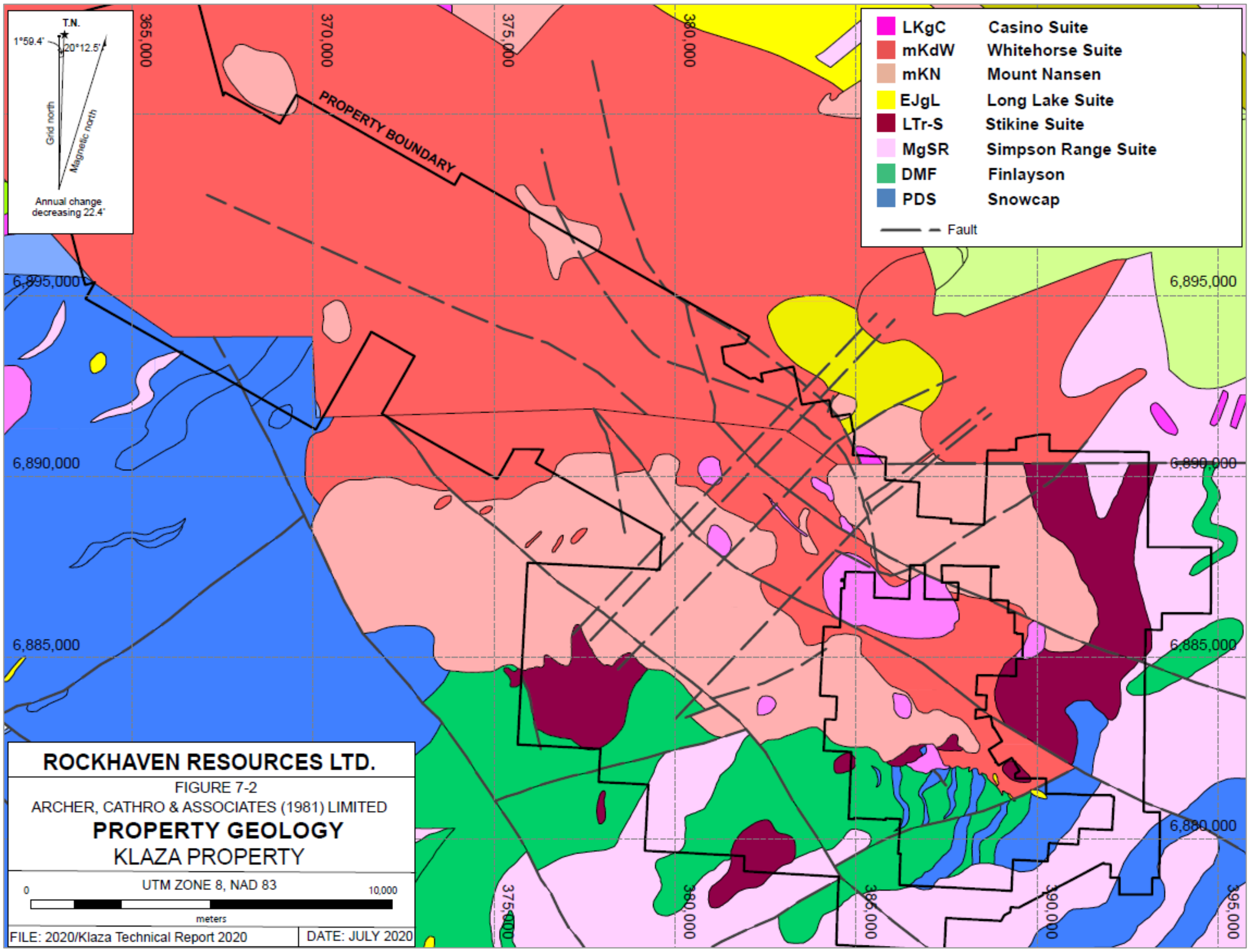
Limited exploration or property mapping has been done by Rockhaven on the newly acquired claims. The mineral occurrences are briefly discussed below in Section 7.3 but the information presented has not been verified by either the QP or Rockhaven. The Property geology discussed below pertains to Work Area 2.

The oldest exposed unit, comprised of Snowcap Assemblage schists, limestones and amphibolites, underlies a large region to the west of the Property. To the south and east of the Property, large bodies of orthogneiss belonging to the Simpson Range Suite surround younger Finlayson Assemblage metavolcanics. Paleozoic units are sporadically intruded by irregular bodies of Stikine Suite orthogneiss and granite.

The northern and western parts of the Property are mostly underlain by Mid-Cretaceous Whitehorse Suite granodiorite. This granodiorite contains 30% hornblende and biotite. It is coarse-grained and non-foliated.

Sub-aerial volcanic and volcaniclastic rocks of the Mount Nansen volcanics are common throughout the central part of the Property. They include medium green to grey andesite flows and pyroclastic rocks with occasional buff to tan rhyolitic tuff. These rocks are believed to be extrusive equivalents of Middle Cretaceous intrusions.

Figure 7.2 Property geology – Klaza Property



Source: Archer, Cathro & Associates (1981) Limited.

A moderate size quartz-rich granite to quartz monzonite stock intrudes granodiorite in the south-east corner of the map area and is thought to be the main heat source for hydrothermal cells responsible for mineralization. This pluton, along with feldspar porphyry dykes scattered throughout the Property, belong to the Casino Suite, which has been dated between Mount Nansen volcanics and Prospector Mountain Suite formation (Isreal, pers. comm. 2014). Geochronological work indicates that porphyry dykes, which are spatially and genetically related to porphyry and vein mineralization on the Property, are Late Cretaceous (78.2 – 76.3 Ma) in age (Mortensen et al. 2016). The Casino Suite is associated with most porphyry-style mineralization within the Dawson Range.

A series of north-westerly trending feldspar porphyry dykes, emanating from the stock in the south-eastern part of the map area, cut the Whitehorse Suite granodiorite in the main areas of interest. These porphyry dykes are up to 30 m wide and consist of buff aphanitic groundmass containing up to 15% orthoclase phenocrysts (1 to 2 millimetre (mm)) with minor biotite and rare quartz phenocrysts. Commonly the dykes occupy the same structural zones as the mineralized veins, and they are often strongly fractured. Some veins cross-cut dykes.

Two main fault trends (NW and NE) are present in the map area. The first set strikes north-westerly and dips between 60 and 80° to the south-west. Although these faults lack strong topographic expression, they are very important because they host mineralized veins and breccia zones and appear to control the distribution of porphyry dykes. The second set of faults strike north-easterly, almost perpendicular to the primary set, and dip sub-vertically. They form prominent topographic linears and offset the mineralized zones in a number of places, creating apparent left lateral displacements of up to 80 m in magnitude. The exact relationship between these structures and the mineralized north-westerly trending structures is still uncertain, but they appear to have been in part coeval and may have played an important role in ground preparation.

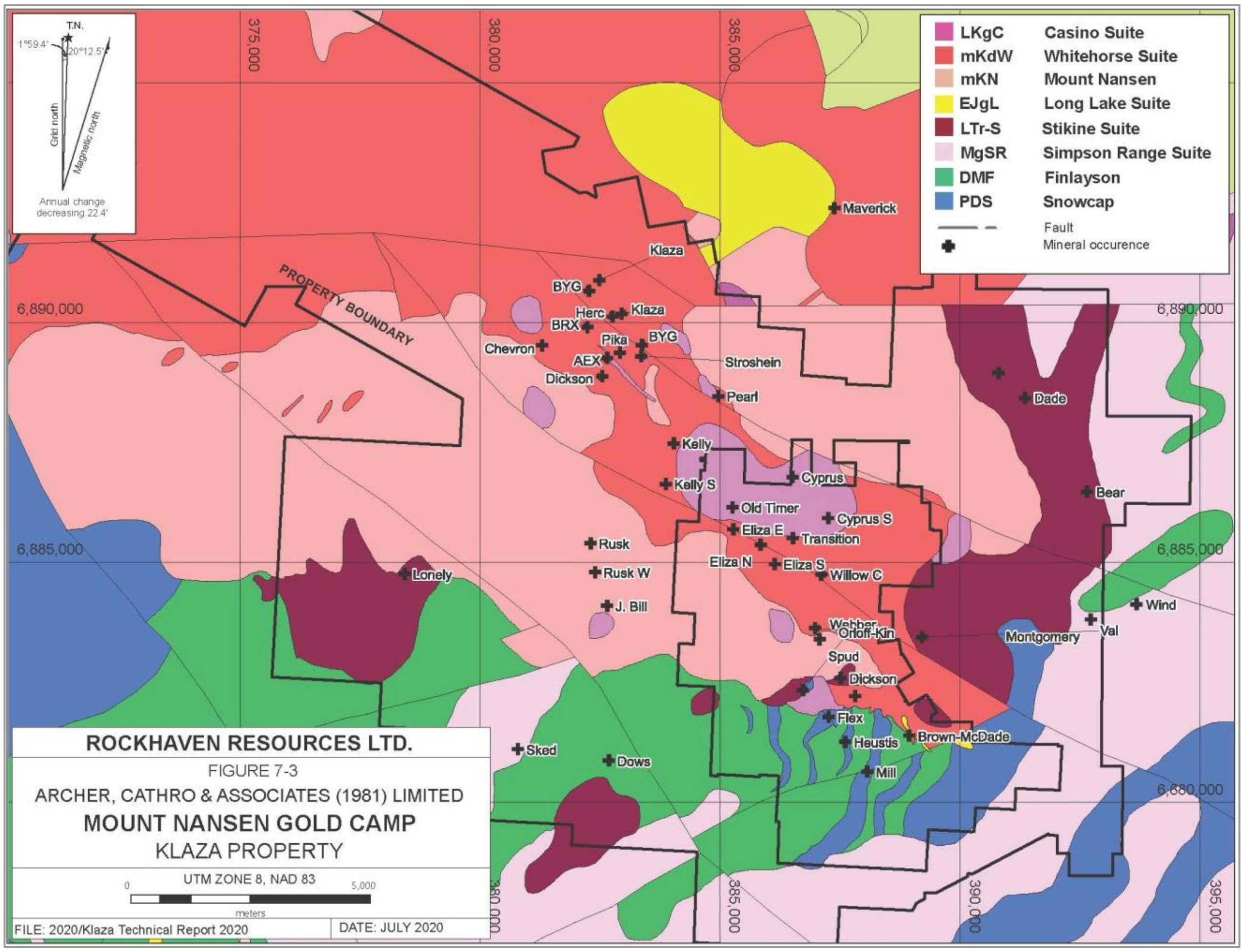
A third set of structures is slightly oblique to the main mineralized faults, striking more westerly. They are less continuous and are considered to be Riedel shears. High-grade mineralization is sometimes localized at junctions between these shears and the north-westerly trending structures.

7.3 Mineralization

The Property lies within the northern part of the Mount Nansen Gold Camp (MNGC), a north-west trending structural belt that hosts more than 30 known mineral occurrences (Figure 7.3), several of which are categorized as deposits and have produced historically and as recently as 1999 (Hart and Langdon 1997).

Mineralization within the MNGC is dominated by gold-silver rich structures associated with a zonation model ranging from weak porphyry copper-molybdenum centres, outward to transitional anastomosing sheeted veins, and lastly to more cohesive and continuous base and precious metal veins. The age of the mineralizing events within the MNGC is now considered to be Late Cretaceous.

Figure 7.3 Mount Nansen Gold Camp – Klaza Property



Source: Archer, Cathro & Associates (1981) Limited.

7.3.1 Mineralization – Work Area 2

The hydrothermal system interpreted to have deposited mineralization in Work Area 2 is centred on two porphyry centres (Cyprus and Kelly zones) related to Late Cretaceous plutonism. Mineralized zones identified on and adjacent to the Work Area 2, and the generalized metal zonation model are shown on Figure 7.4. The larger and better defined porphyry centre (Cyprus zone) is located in the south-east corner of the map area. It was explored in the late 1960s and early 1970s with approximately 4,500 m of drilling in 26 holes. Average hypogene grades of 0.12% copper and 0.01% molybdenum were reported at depths exceeding 60 to 90 m below surface. Hypogene copper grades are approximately double those in the overlying leached cap. There is no significant supergene enrichment zone. Higher grade zones (0.6% copper and 0.06% molybdenum) and elevated precious metal values are associated with local areas of intensive fracturing (Sawyer and Dickinson 1976). These metal enriched zones are found in weakly potassic altered areas within the dominantly phyllic altered porphyry system. The potassic altered areas often feature tourmaline breccias, abundant quartz veining and / or secondary biotite.

The western porphyry centre (Kelly zone) is located on the Property and was explored as early as 1973. The Kelly zone is defined by coincident geochemical and geophysical anomalies, including: 1) strongly elevated gold, copper and molybdenum soil geochemical response; 2) high chargeabilities with moderate resistivities; and, 3) a large area of low magnetic susceptibility observed in both ground and airborne surveys. The coincident anomalies cover a semicircular area approximately 2,500 m across. Trenching and diamond drilling done in 2012 by Rockhaven on the western edge of the Kelly zone discovered minor chalcopyrite, chalcocite and molybdenum, with rare bornite. The mineralization is hosted in several, 25 to 100 m wide bands of strongly phyllic altered and heavily quartz veined granodiorite, which are separated by barren porphyry dykes.

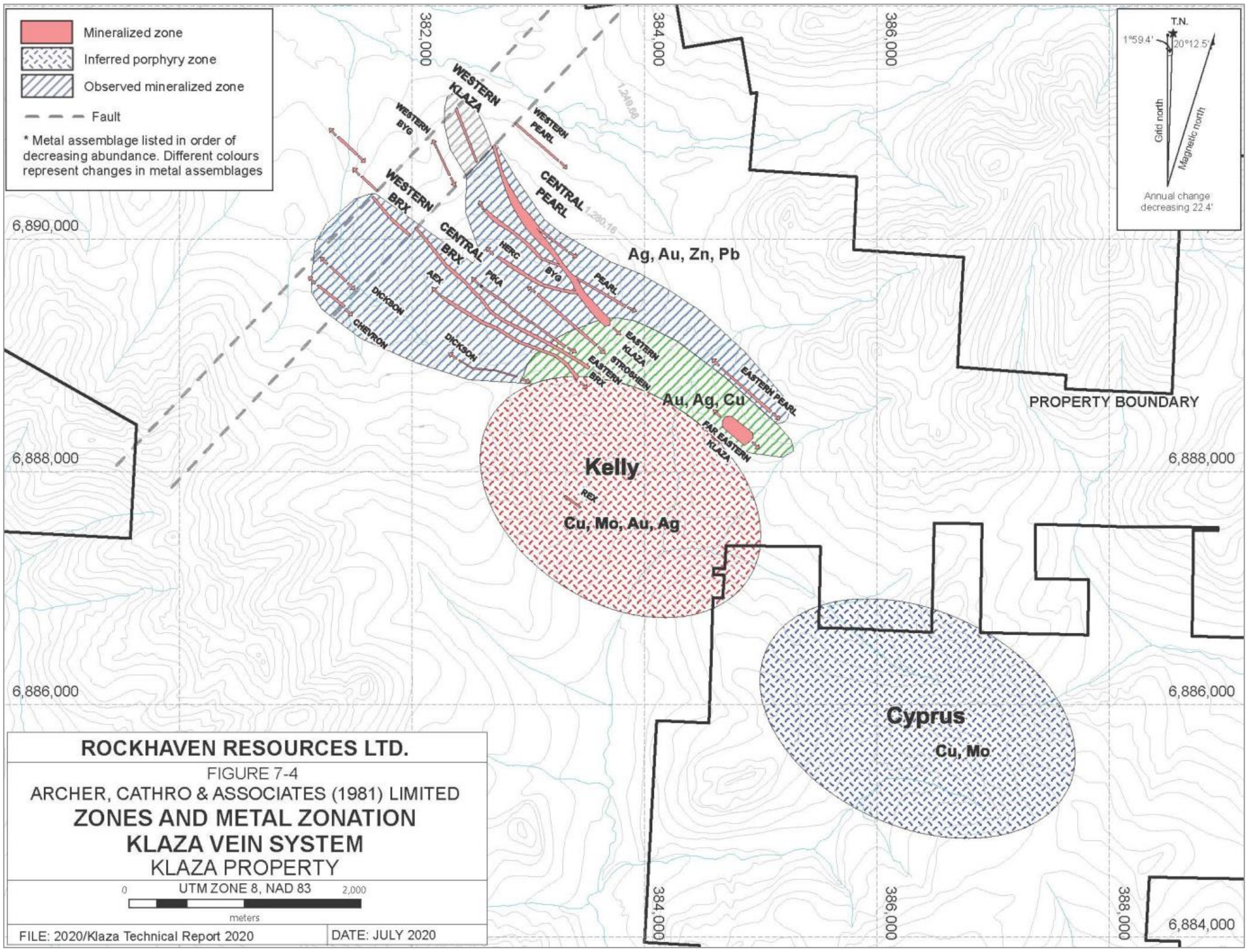
Structurally controlled gold-silver mineralization in the core of the Property is interpreted to be related to the hydrothermal system that is cored by the Cyprus and Kelly zones. Re-Os dating of the Cyprus zone has established Late Cretaceous age for the pluton and the associated mineralization (Mortensen et al. 2003).

The majority of Rockhaven's exploration activities have been conducted in the distal part of the local hydrothermal system where copper-deficient precious metal-rich veins predominate. This work has identified twelve main mineralized structural zones in Work Area 2 that are developed north-west of the porphyry targets. The structural zones collectively form a 4 km long by 3 km wide corridor that cuts north-westerly through Mid- Cretaceous granodiorite country rocks. Individual zones exhibit exceptional lateral and down-dip continuity, and all of them remain open for extension along strike and to depth. From south to north, the zones are named Rex, Chevron, Dickson, AEX, BRX, Pika, Stroshein, Herc, BYG, Klaza, Pearl, and Victoria. Rockhaven's exploration has focused mainly on the Klaza and BRX zones, which have been subdivided into the Western BRX, Central BRX, Eastern BRX, Western Klaza, Central Klaza, and Eastern Klaza zones. The current Mineral Resource estimate contains mineralization from the western and central zones.

The eastern portion of the mineralized structural zones comprises multiple, sub-parallel veins, that include the Eastern Klaza, Eastern BRX, Pika, Stroshein, and part of the AEX zones, all of which are open to the east, towards the mostly untested Kelly Porphyry.

The western portion of the main mineralized structural zones range from 1 to 100 m wide and are usually associated with feldspar porphyry dykes. Mineralization occurs within veins, sheeted veinlets and some tabular breccia bodies. The host granodiorite exhibits pervasive weak argillic alteration immediately adjacent to, and up to 30 m peripherally from, them. Sericitization and potassic alteration are developed directly adjacent to hydrothermal channel ways. The granodiorite is magnetite-bearing except where the magnetite has been replaced by sulphide minerals around and within mineralized structures.

Figure 7.4 Zones and zonation – Klaza Property



Source: Archer, Cathro & Associates (1981) Limited.

Depth of surface oxidation ranges from 5 m to 100 m below surface, depending on fracture intensity, the type of mineralization and local geomorphology. The deepest weathering occurs in wide, pyritic veins located along ridge tops or on south facing slopes.

Detailed evaluation of oriented drill core and measurements taken from trench exposures has identified two main structural orientations that control mineralization. The primary structural set strikes between 135° and 155° and dips 60° to 80° to the south-west. The secondary mineralized trend strikes between 110° and 130° and dips 60° to 70° to the south. The secondary structures may represent either Riedel shears of the primary structural set or a separate structural event altogether. The best gold mineralization is sometimes localized in areas where the two structural trends converge. The plunge of these structural intersections is towards the south-east.

Petrographic work demonstrated veins, veinlets and breccia material hosting disseminated to semi-massive pyrite, arsenopyrite, galena, sphalerite, stibnite, and jamesonite in quartz, carbonate, and barite gangue (Payne 2012). The sulphide minerals typically comprise 1% to 10% of the sample, often increasing to between 20% and 80% over 25 cm to 200 cm intervals. The petrographic work also identified native gold / electrum (Tarswell and Turner 2012).

Quartz is the dominant gangue mineral in veins in Work Area 2. It occurs in a variety of textures including chalcedonic, comb, banded, speckled and vuggy. Smoky quartz is the most common colour variation, but milky and clear quartz are locally abundant. Carbonate occurs mainly as ankerite and rhodochrosite and typically ranges between 5% and 20% of the veins by volume.

Breccias form tabular bodies consisting of heterolithic wallrock clasts, which include granodiorite and various volcanic or sub-volcanic lithologies. Matrices are enriched with fine-grained, disseminated to blebby pyrite, arsenopyrite, sphalerite and galena. Breccias are mostly observed within drill core from the Klaza zone where they range up to 2 m in width.

Mineralization within most structures is interpreted to be spatially and genetically related to porphyry dykes, which strike north-westerly and dip steeply to moderately toward the south. The dykes pinch and swell in three dimensions and are usually unmineralized. Some faults identified to date likely post-date emplacement of the dykes as they are occasionally cut by mineralized veins.

Two parallel, north-east trending faults have been observed to cut across the north-western portion of the Klaza and BRX zones. The easterly cross-fault appears to offset the western sections of the mineralized zones about 80 m to the south; however, the exact sense of motion is uncertain. Detailed exploration has not been conducted yet on the western side of the westerly cross-fault, so displacement on it has not been determined. The westerly cross-fault appears to be a stronger structure. The relative timing of movement on these faults has not yet been determined, but they are thought to be coeval with, or slightly younger than, the vein structures. Some of the better mineralized sections of the vein structures occur in what appear to be dilatant zones immediately east of the cross-faults. Drillholes and trenches are aligned subparallel to the orientation of the cross-faults – therefore only a few holes have intersected them. The extent to which the north-east trending faults are mineralized is not yet known. In the Klaza zone, the easterly cross-fault marks a sharp change in mineralogy with increasing arsenopyrite and sulphosalt contents coupled with higher silver to gold ratios in the Western Klaza zone relative to the Central Klaza zone. At the BRX zone, the same cross-fault separates bonanza-grade rhodochrosite-facies mineralization in the Western BRX zone from lower-grade, iron-carbonate facies mineralization in the Central BRX zone.

For a more detailed description of mineralization, mineral paragenesis, alteration facies and gangue facies, please refer to the technical report entitled "Geology, Mineralization, Geochemical Surveys, Geophysical Surveys, Diamond and Percussion Drilling, Metallurgical Testing and Mineral Resources on the Klaza Property, Yukon, Canada" dated 11 March 2015 and amended 19 June 2015 (Wengzynowski et al. 2015).

7.3.2 Mineralization – other work areas

Historical work on Rockhaven's recently acquired claims identified various mineralized trends, which warrant further investigation. The most noteworthy of these veins are found in Work Areas 3, 4, and 6. Rockhaven has conducted limited mapping, prospecting, geochemical sampling, and geophysical surveys on portions of the Property outside of Work Area 2.

The known surface mineralization on the Dade claims (Work Area 3) is hosted in two, sinusoidal zones of quartz veining and stockwork (V1 and V2) striking about 040° and dipping 60° to 75° north, hosted in coarse-grained hornblende-quartz granodiorite to diorite gneiss (Burrell 2013). V1 (formerly, the Grizzly Vein) and V2 are epithermal quartz vein and stockwork zones that exhibit pervasive silicification and moderate to strong clay alteration. In 2011, trenching exposed V1 over widths of 9 m to 20 m along a 175 m strike length and V2 over widths of 2 m to 12 m along a 125 m strike length (Burrell 2013). The veins comprise white to grey quartz with boxwork limonite and locally 1% to 3% disseminated arsenopyrite and pyrite.

On the Val claims (Work Area 4), mineralization consists of fault-controlled, gold- and silver-bearing veins, and breccias hosted within one structural zone (Turner 2014). This zone ranges from 10 m to 20 m wide and mineralization occurs within sheeted veins and veinlets. Sulphide minerals on the Val claims consist of arsenopyrite, pyrite, galena and sphalerite and occur as disseminations and stringers within quartz and carbonate gangue.

The J Bill, Rat, and Bull claims (Work Area 6) host two types of mineralization located about 500 m west of the vein proximal to the contact of a silicified porphyry stock. One type of mineralization consists of pyrite-arsenopyrite-galena-sphalerite-quartz, while the other is composed of finely disseminated molybdenite and chalcopyrite with minor pyrite and pyrrhotite (YGS Minfile 115I 096). The mineralized trend was traced for 1,150 m (YGS Minfile 115I 096).

8 Deposit types

8.1 Mineralization style on the Property

The metals of primary interest at the Property are gold and silver. These metals are intimately associated with lead, zinc and copper in various forms and concentrations throughout the mineralizing system. Gold and silver enriched mineralization is developed within a north-west trending structural corridor, which is interpreted to have focused fluid flow away from weak porphyry centres related to a Late Cretaceous stock in the south-eastern corner of the Property. Several of the mineralized structural zones are continuously mineralized for strike lengths of up to 2,400 m, and at least one of the structures is mineralized to a depth of 520 m down-dip from the current geographic surface. The mineralized structures remain open to extension along strike and down-dip.

Fluid inclusion work reveals that the veins formed at shallow depths (<1 km) and have low to intermediate sulphidation epithermal fluid characteristics (Main 2015). Textures and mineralogy observed at the Property share a number of similarities with carbonate base metal (CBM) deposits (Tarswell and Turner 2013).

CBM deposits are a recently recognized sub-class of epithermal deposits that encompass a family of similar deposits located around the world. CBM deposits have mainly been discovered around the Pacific Rim and include multi-million ounce gold deposits such as Porgera (Papua New Guinea), Buritica (Colombia) and Kelian (Indonesia).

The CBM class of deposits has yet to be positively identified in the Yukon, but some researchers have recognized that mineralization on the Property has some of the characteristics of mineralization now categorized as CBM deposits (ex. Smuk 1999). Given the limited academic research on the Property and the absence of significant syn-mineralization carbonate, more studies need to be undertaken.

Similarities and differences between CBM deposits and the Klaza mineralization are discussed below.

8.2 Characteristics of carbonate base metal gold deposits

CBM deposits are formed by the mixing of rising mineralized fluids with bicarbonate waters (Corbett and Leach 1998). Mineralization styles are highly zoned, depending on the crustal level of the system, with silver-rich CBMs formed at higher levels. Characteristic zonation of carbonate compositions develops when upwelling mineralizing fluids are progressively cooled as they mix with descending bicarbonate groundwater. These carbonate compositions vary from proximal (hot) calcium (Ca) through magnesium (Mg) and manganese (Mn) to distal (cool) iron (Fe) facies. Gold mineralization is believed to be preferentially distributed within veins containing Mn / Mg carbonate facies, notably rhodochrosite.

Key diagnostic features of CBM deposits are compared to features observed at the Property in Table 8.1.

Table 8.1 CBM comparison with BRX and Klaza zones

Diagnostic features of CBM deposits	Diagnostic features of Klaza mineralization
Mineralization hosted in veins and breccias	Mineralization hosted in veins and breccias
Large vertical extent of mineralization (> 1,000 m)	Large vertical extent of mineralization (520 m and open to depth)
Gold and silver generally well liberated (native or in electrum)	Gold and silver generally well liberated (native gold, electrum, and silver in tetrahedrite)
Veins and breccias emplaced adjacent to mineralizing intrusive	Veins and breccias emplaced adjacent to mineralizing intrusive
Carbonate (dominant), quartz, pyrite, sphalerite, and galena gangue	Quartz (dominant), carbonate, pyrite, sphalerite, and galena gangue
Multiple mineralized structures with long strike lengths (> 700 m)	Multiple (eleven) mineralized structures with long strike lengths (> 2,400 m)
Bonanza grade gold mineralization	Some bonanza grade intercepts

Source: Archer, Cathro & Associates (1981) Limited.

The CBM model is the geological concept on which exploration is planned. Although further studies may place the Klaza mineralization into a more general epithermal category, this difference does not materially affect the exploration strategy.

9 Exploration

Exploration programs performed by Rockhaven between 2010 and 2019 within the main area of interest, Work Area 2, are described below, except for drilling which is discussed in detail in Section 10. Work Areas 1 and 3 – 6 are on ground acquired by Rockhaven in the summer of 2015 and summer of 2017. Limited exploration work, comprising soil-geochemical sampling and ground-based VLF-EM and magnetic surveys,has been conducted by Rockhaven on parts of Work Areas 4 and 6. No exploration work has been completed on Work Areas 1, 3, and 5 by the issuer.

9.1 Geological mapping

Conventional geological mapping over much of the Property is hampered by the presence of pervasive overburden and vegetation cover. Data obtained from sparse outcrops, excavator trenching and drilling have been used in conjunction with information inferred from geophysical surveys to create geological maps of Work Area 2.

9.2 Soil geochemical surveys

From 1967 to present, various operators collected soil geochemical samples from the eastern part of Work Area 2. Historical samples were taken on baseline-controlled grids established using hip-chain and compass. Baselines were marked with one metre high wooden lath and sample sites were marked with 0.5 m wooden lath; however, very few of these markers are currently standing and legible. Early soil sampling identified linear gold ± silver ± lead anomalies, which correspond to some of the known mineralized structural zones, and a large (2,000 m by 3,000 m) area of moderately to strongly anomalous copper-in-soil response, which partially defines the Kelly porphyry target in the south-eastern corner of Work Area 2.

Grid soil sampling was performed within Work Area 2 by Rockhaven from 2010 to 2012, and 2015. Contour - controlled lines were completed in the north-western part of Work Area 2 and north-eastern part of Work Area 6. Soil sampling methods, spacing and analytical techniques are described in Section 11.

Effectiveness of soil sampling is often limited by thick layers of organic material and overburden, and in many areas, by permafrost. Despite these limitations, soil sampling has been one of the most effective surface exploration techniques for identifying trenching or drilling targets on the Property.

Results for gold, silver, lead, arsenic, and copper from historical surveys and Rockhaven's sampling are illustrated together on Figure 9.1 to Figure 9.5, respectively using gradient contour techniques. Table 9.1 lists the anomalous thresholds and peak values obtained by Rockhaven's surveys for these elements.

Table 9.1 Geochemical data for soil samples from Work Area 2

	Anomalous thresholds			Peak values
Element	**Weak**	**Moderate**	**Strong**	
Gold (ppb)	≥ 5 < 10	≥ 10 < 20	≥ 20	920
Silver (ppm)	≥ 0.5 < 1	≥ 1 < 2	≥ 2	61.3
Lead (ppm)	≥ 10 < 20	≥ 20 < 50	≥ 50	722
Copper (ppm)	≥ 20 < 50	≥ 50 < 100	≥ 100	1,870
Arsenic (ppm)	≥ 10 < 20	≥ 20 < 50	≥ 50	1,750

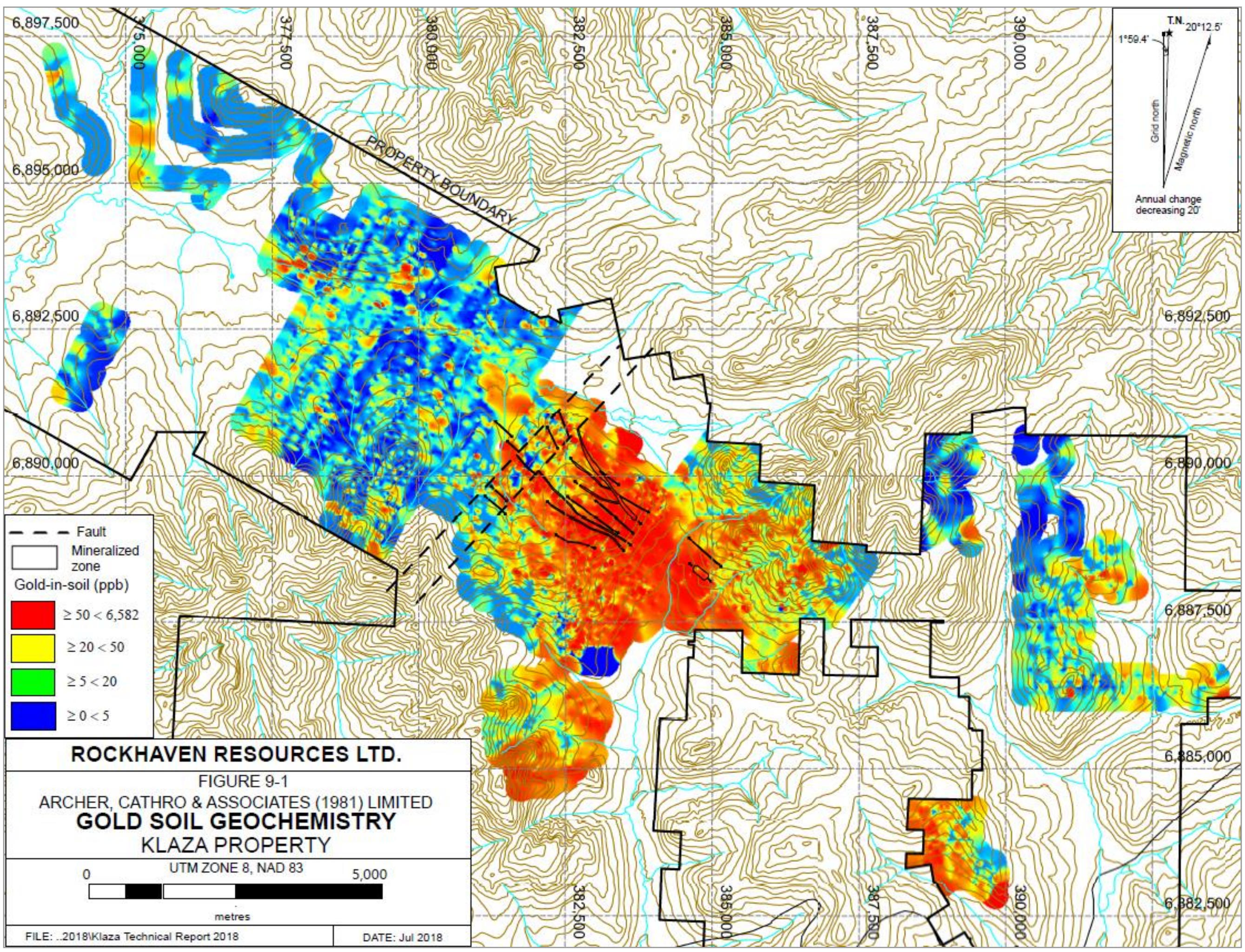
Source: Archer, Cathro & Associates (1981) Limited.

The structural corridor hosting the twelve known mineralized zones is defined by linear trends of moderately to strongly anomalous values for gold, silver, lead, and arsenic. Similar but more discontinuous anomalies have been identified south-west and north-east of the structural corridor, where no mineralized zones have been discovered to date. North-west along strike of the known mineralized zones, elevated soil values occur as isolated samples or in small clusters. The lack of continuity in these outlying anomalies may be due in part to more difficult sampling conditions resulting from lower elevations and increased overburden depths.

Work Area 2 exhibits distinct copper zonation from east to west. Copper is strongest in the south-eastern part of the area, in proximity to the intrusive centre at the Kelly zone. Response across the remainder of the gridded area is more subdued. The more southerly BRX, AEX, Dickson, and Chevron zones have weakly elevated copper-in-soil signatures, while the other zones, further to the north, show only background copper response.

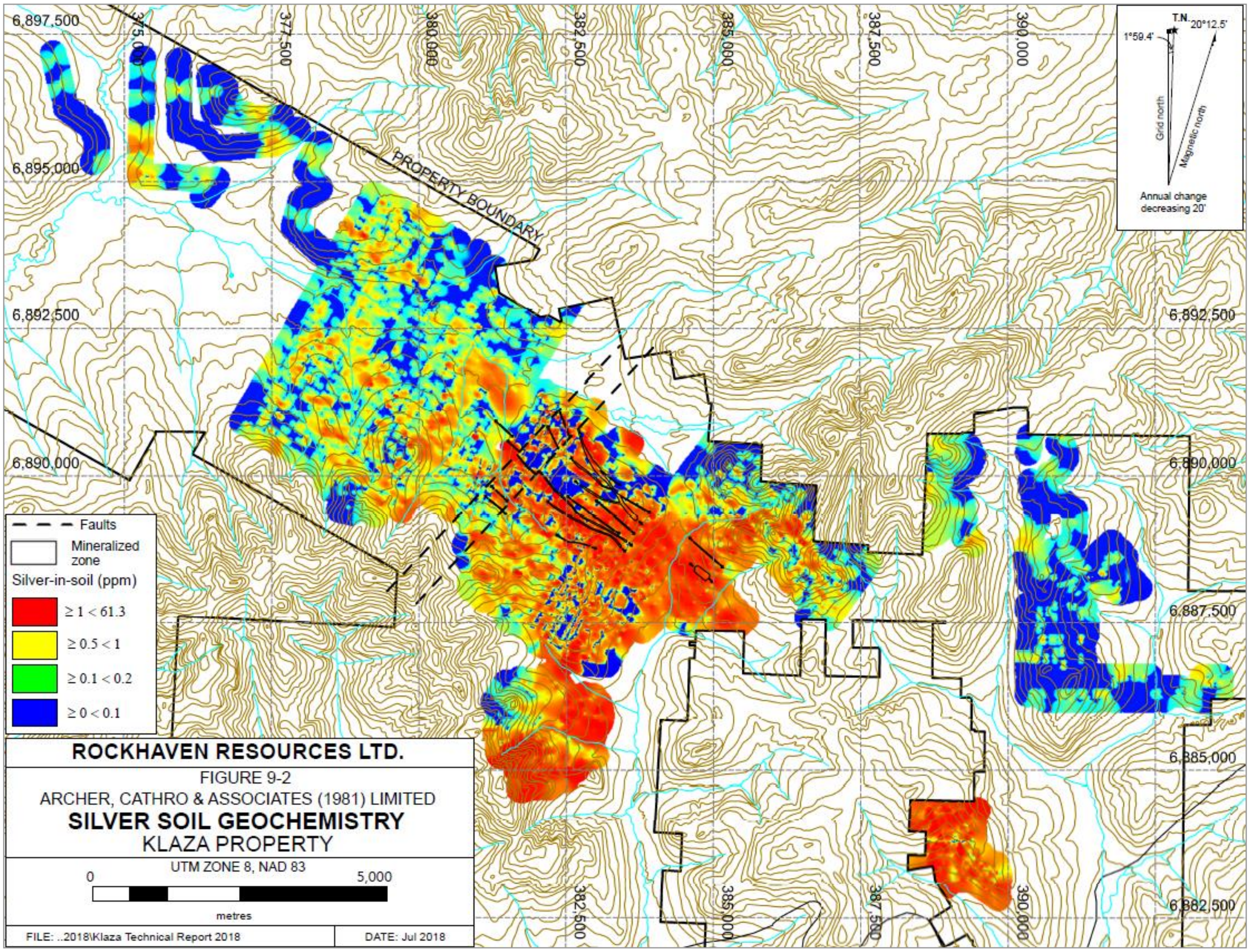
Contour sampling at the Rusk showing, located in the southern corner of Work Area 2 and Northern edge of Work Area 6, is characterized by coincident, strongly anomalous arsenic, silver, gold, and lead. This area has not been followed up by diamond drilling.

Figure 9.1 Gold soil geochemical values – Work Area 2 Klaza Property



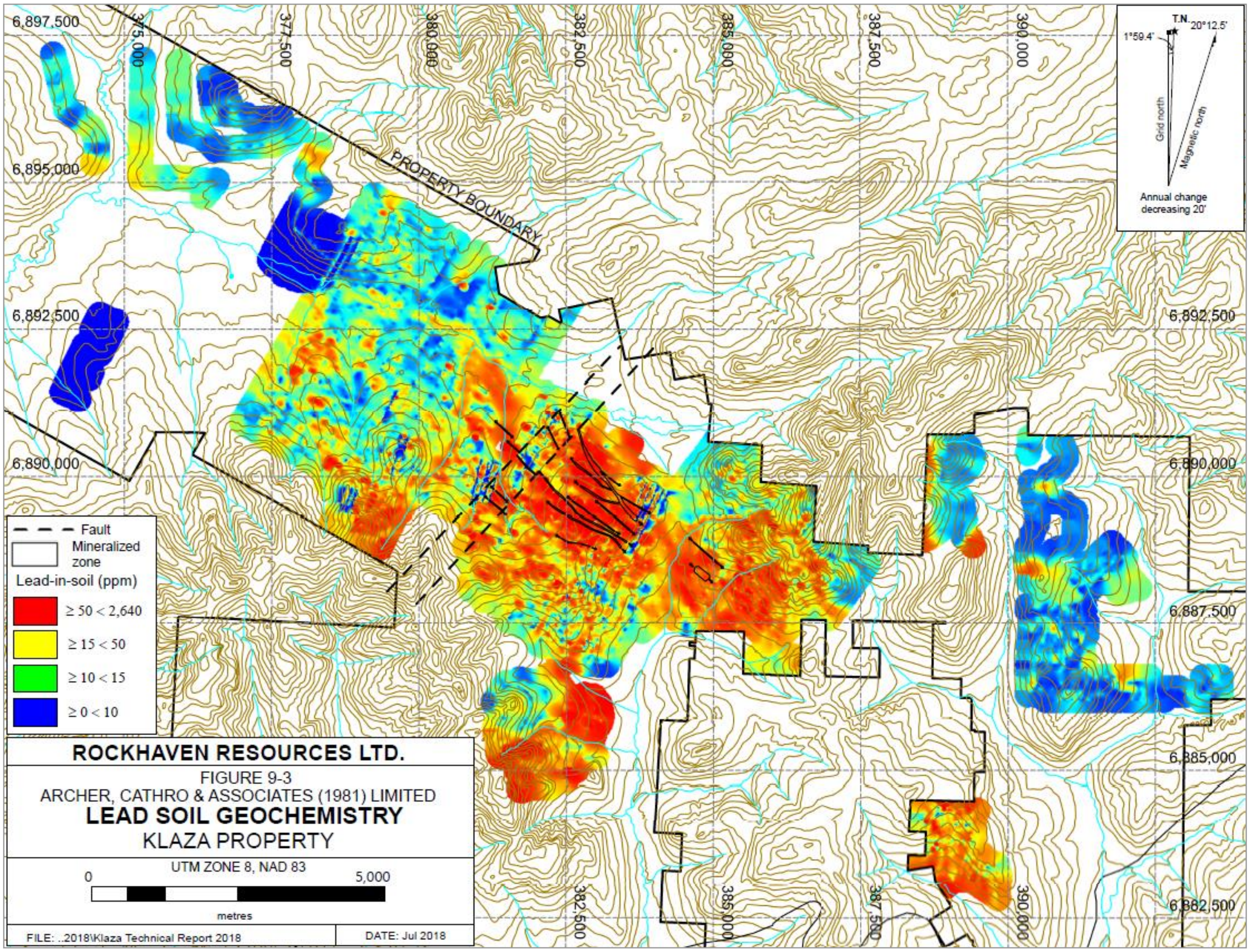
Source: Archer, Cathro & Associates (1981) Limited.

Figure 9.2 Silver soil geochemical values – Work Area 2 Klaza Property



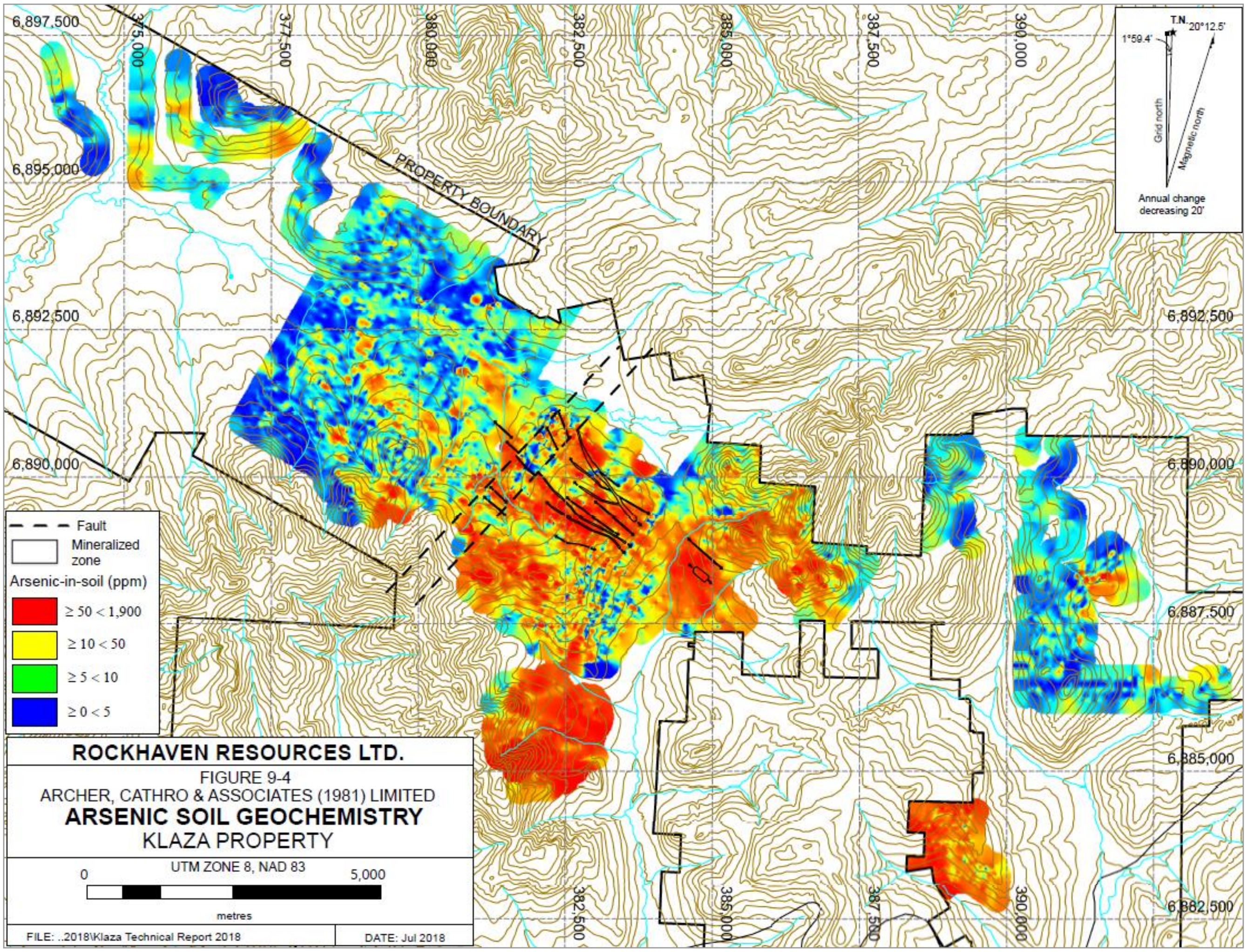
Source: Archer, Cathro & Associates (1981) Limited.

Figure 9.3 Lead soil geochemical values – Work Area 2 Klaza Property



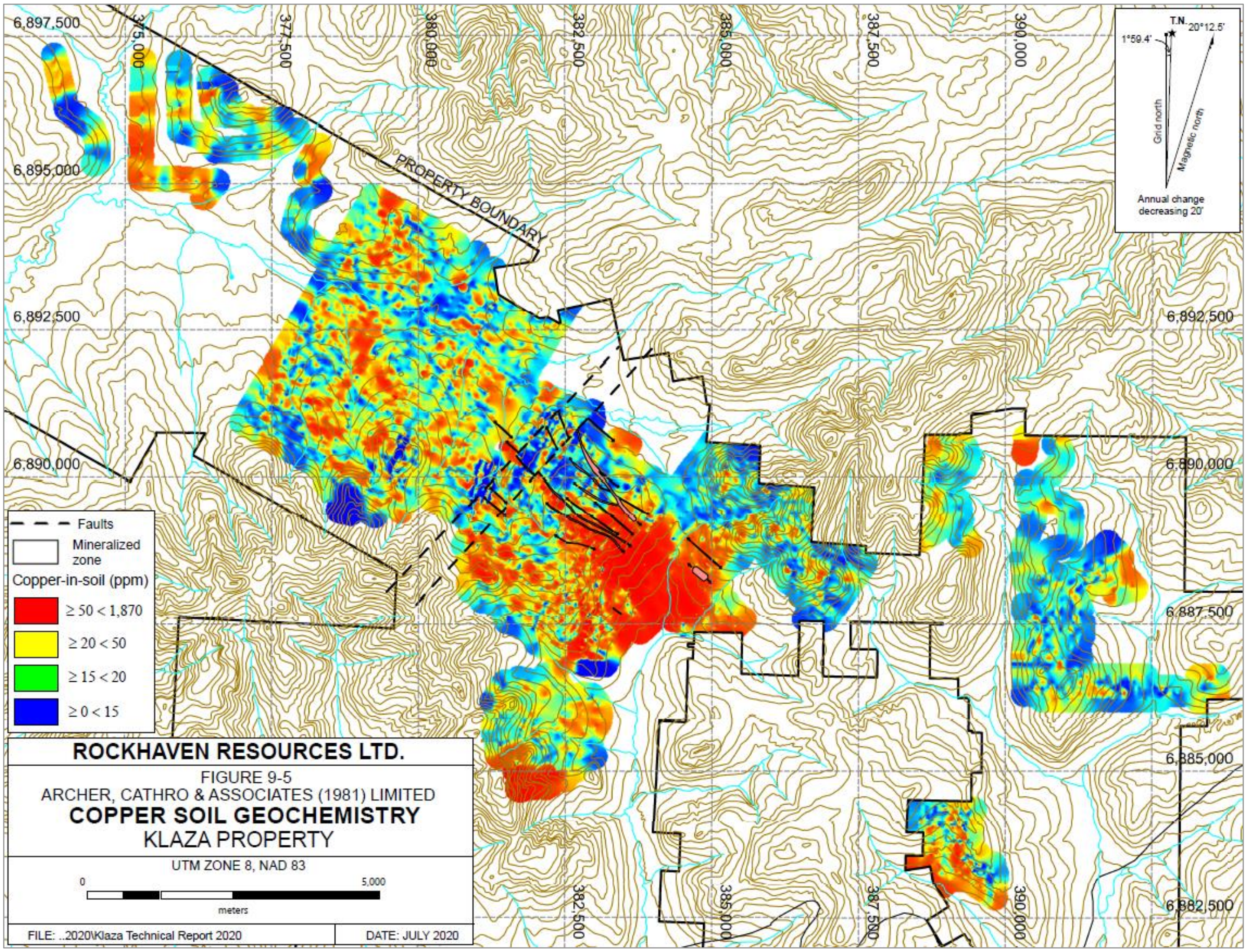
Source: Archer, Cathro & Associates (1981) Limited.

Figure 9.4 Arsenic soil geochemical values – Work Area 2 Klaza Property



Source: Archer, Cathro & Associates (1981) Limited.

Figure 9.5 Copper soil geochemical values – Work Area 2 Klaza Property



Source: Archer, Cathro & Associates (1981) Limited.

9.3 Excavator trenching

Excavator trenching in geochemically anomalous areas has been the most effective tool for identifying near surface but non-outcropping, mineralized zones. Within the main areas of exploration, overburden generally consists of 5 to 20 cm of vegetation and soil organics covering a discontinuous layer of white volcanic ash and 50 to 125 cm of loess and / or residual soil, which cap decomposed bedrock.

Typical trench exposures within the mineralized vein zones exhibit strong limonite and clay alteration that is often water saturated and more deeply weathered than the surrounding wallrocks. These zones are more intensely fractured and have higher porosity as a result of near surface oxidation. Residual sulphide minerals are rarely present in trenches and, where seen, they are usually encapsulated in silica. The locations and orientations of lithological contacts in trenches correspond very well with those predicted from nearby drillholes, indicating little solifluction has occurred. Outside of the mineralized zones, trench exposures are dominated by blocky, weakly oxidized granodiorite.

Rockhaven performed 24,231 m of trenching in 101 trenches between 2010 and 2017. Table 9.2 lists the total number and combined lengths of trenches completed by Rockhaven each year during that period. No trenching was conducted in 2018 or 2019.

Table 9.2 2010 to 2017 Excavator trenching summary

Year	Number of trenches	Total length (m)
2010	21	8,000
2011	12	4,050
2012	11	4,000
2013	38	5,000
2014	5	880
2015	2	436
2016	8	1,270
2017	4	595
Total	**101**	**24,231**

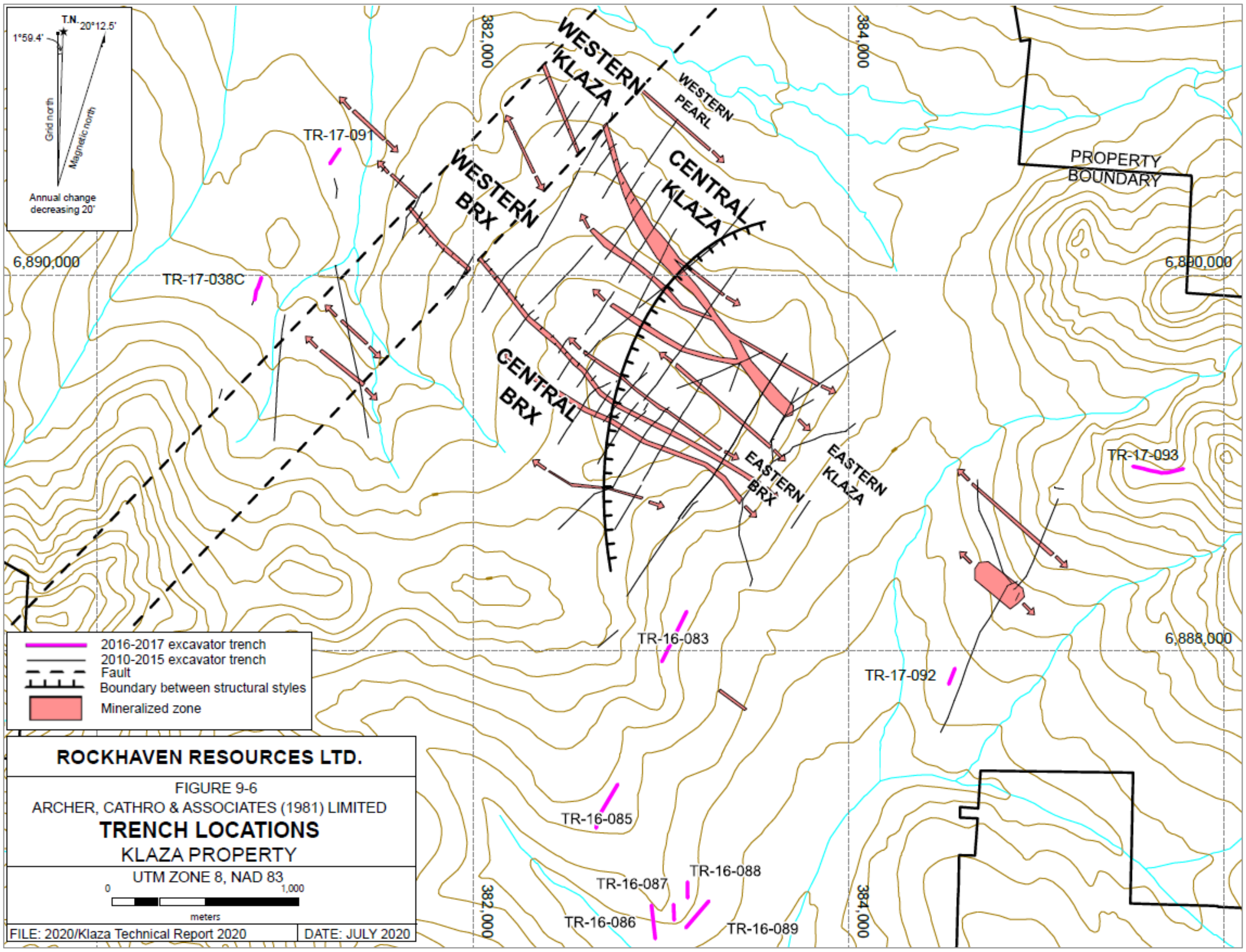
Source: Archer, Cathro & Associates (1981) Limited.

The majority of Rockhaven's trench locations were selected based on results from historical programs. Where possible, trenches were excavated in areas that had previously been stripped of soil and vegetation. The trenches were aligned at about 030°, which is perpendicular to the anomalous trends of the main soil geochemical anomalies. Figure 9.6 is a plan view map showing Rockhaven's trench locations and approximate surface traces of the nine main mineralized structural zones. Excavator trenching methods and analytical techniques are described in Section 11.

Individual zones and key trench results obtained prior to 2016 are discussed in the technical report entitled "Technical Report and PEA for the Klaza Au-Ag Deposit, Yukon, Canada for Rockhaven Resources Ltd." dated 26 February 2016 (Ross et al. 2016).

In 2016 and 2017, twelve trenches were dug. Locations of Klaza area trenches are shown in Figure 9.6.

Figure 9.6 Trench locations – Klaza Property



Note: 2016 and 2017 trenchs shown in pink.
Source: Archer, Cathro & Associates (1981) Limited.

9.4 Geophysical surveys

To date, a number of geophysical surveys have been completed on Work Area 2 by four different companies:

1. SJ Geophysics Ltd. of Delta, British Columbia conducted ground-based VLF-EM and magnetic surveys on behalf of BYG Natural Resources Inc. in 1996 (Dujokovic et al. 1996) and Rockhaven in 2014 (Dumala et al. 2015); and, a Volterra-3D IP survey in 2016 (Turner and Cruz 2018).
2. Aurora Geosciences Ltd. of Whitehorse, Yukon conducted a gradient array induced polarization survey on behalf of Bannockburn Resources Limited in 2006 (Wengzynowski 2006).
3. New-Sense Geophysics Ltd. (NSG) of Markham, Ontario conducted high sensitivity helicopter-borne magnetic and gamma-ray spectrometric surveys for Rockhaven during the 2010 (Tarswell and Turner 2011) and 2011 (Tarswell and Turner 2012) field seasons.
4. Ground Truth Exploration of Dawson City, Yukon conducted high resolution induced polarization surveys along two experimental lines in the Central Klaza and Central BRX zones for Rockhaven during the 2013 field season (Tarswell and Turner 2014).

The NSG surveys resulted in 326 line kilometres being flown on a grid that covered most of Work Area 2. Condor Consulting Inc. of Lakewood, Colorado was retained to ensure quality control and produced a 3D model of the total field magnetics as well as various vertical derivatives.

The magnetic surveys identified a number of prominent, linear magnetic lows in Work Area 2. Subsequent trenching and drilling have shown that many of the north-westerly trending lows coincide with mineralized structural zones, while north-easterly trending breaks in the magnetic patterns correspond to cross-faults. These relationships are consistent with the low magnetic susceptibility results returned from core samples within the altered structural zones compared to higher values from surrounding unaltered wallrocks. Several of the magnetic lows extend outside the main areas of exploration and have not yet been tested by drilling or trenching. Figure 9.7 shows the first vertical derivative of the magnetic data overlain with the interpreted surface traces of the structural zones.

Elevated potassic radioactivity is evident in the general area of the main zones in the eastern part of Work Area 2 but does not specifically coincide with individual mineralized zones. Numerous porphyry dykes and frost boils containing porphyry fragments lie within this area, and they are the probable source of the elevated radioactivity. The Klaza River valley generally has a subdued radiometric response, which is likely due to thick vegetation and water saturation in the flats adjacent to the river. However, a band of elevated radioactivity that directly correlates with the bed of the Klaza River may be caused by exposed gravels, which include abundant potassium feldspar bearing, intrusive material.

The SJ Geophysics VLF-EM and ground-based magnetic surveys covered 330 line kilometres on a 4.5 km by 8 km grid in the eastern and central part of Work Area 2. SJ Geophysics interpreted the data and produced images relating to it. These surveys delineated numerous linear magnetic lows and VLF-EM conductors that coincide with known mineralized zones. Northerly trending breaks in the VLF-EM conductors correspond to known or suspected cross-faults. Figure 9.8 shows the results of the VLF-EM survey overlain with the interpreted surface traces of the mineralized structural zones and their possible extensions along strike.

A Volterra-3D IP survey was completed by SJ Geophysics in 2016 to the east of the deposit area. A total of 18.4 line kilometres were completed on a grid with seven survey lines spaced at 400 m. Survey stations along these lines were set up at 100 m intervals. The work was designed to better define the Kelly Zone, where porphyry-style mineralization had been inferred from a few widely-spaced drillholes, strong soil geochemistry and a broad magnetic low. This survey identified a 2,400 m by 2,200 m sub-circular chargeability anomaly containing greater than 40 millisecond (ms) values, as shown in Figure 9.9. Within this anomaly there is a 1,700 m by 800 m core with greater than 50 ms values. North-trending resistivity lows (less than 100 ohm-metre) coincide with the core of the chargeability anomaly, as shown in Figure 9.9.

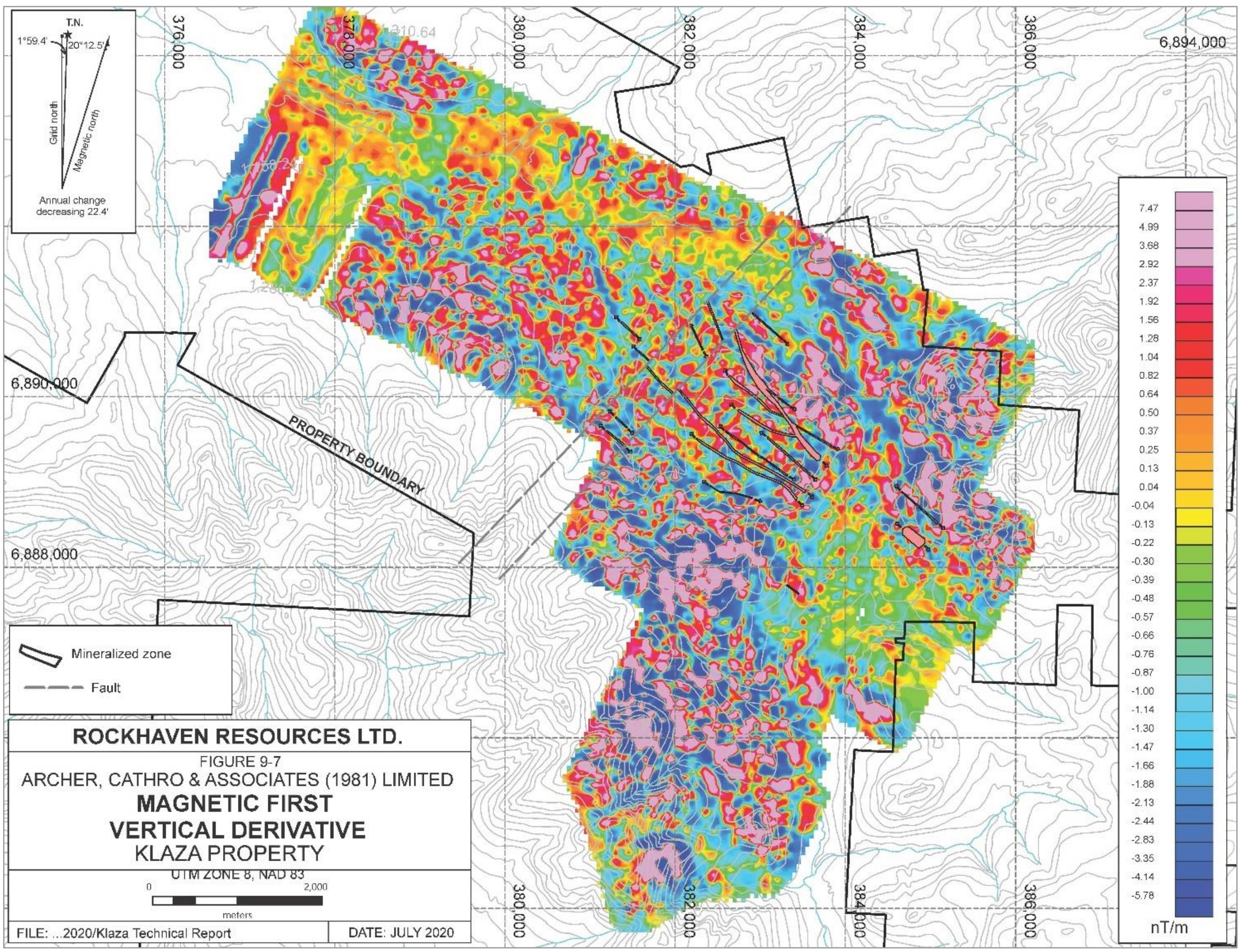
The gradient array and pole-dipole IP survey conducted by Aurora Geosciences covered an 1,800 m by 1,450 m area in the east-central part of Work Area 2. Readings were collected at 25 m intervals along lines spaced 100 m apart. This survey identified two main anomalies, both of which feature elevated chargeability with coincident resistivity lows.

The most prominent anomaly is located in the south-eastern corner of the grid. It is only partially defined and currently comprises a 1,000 m diameter, semicircular area characterized by moderate chargeability and low resistivity, which corresponds to the chargeability anomaly identified by the 2016 Volterra-3D IP survey. This anomaly coincides with an area of weak to strong gold-in-soil geochemistry (25 to 100 ppb) and strong copper geochemistry (>200 ppm) as well as porphyry style mineralization that is part of the Kelly zone.

The second IP anomaly includes three westerly trending chargeability features of weak to moderate intensity. These chargeability features are 710 m to 1,200 m long and are offset 30 m to 190 m to the south from parallel resistivity lows. The IP anomaly also includes three other, smaller chargeability highs that directly coincide with resistivity lows. These latter features correspond with parts of the BRX, AEX, and BYG zones.

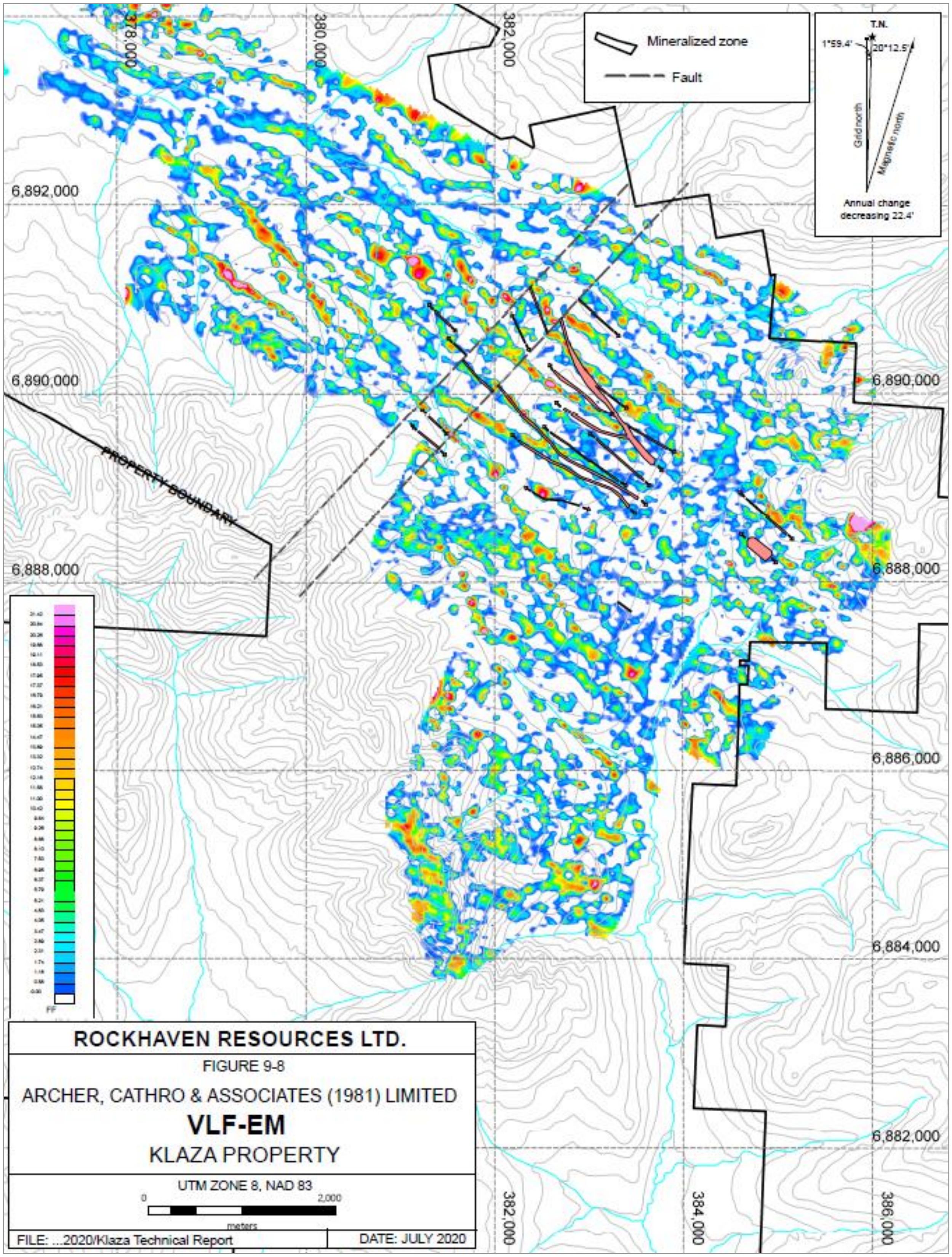
The experimental IP survey conducted by Ground Truth Exploration collected dipole-dipole extended, inverse Schlumberger and strong gradient array data on section lines 10+050 mE and 10+600 mE at the Klaza and BRX zones. Each of these lines was 415 m long (a single spread length for the arrays). Transceivers were placed 5 m apart along the lines, resulting in a very high signal to noise ratio and thus providing high quality resistivity data. The mineralized vein and breccia zones tested by the two lines show up as resistivity lows that coincide with chargeability highs.

Figure 9.7 Magnetic first vertical derivative – Work Area 2 Klaza Property



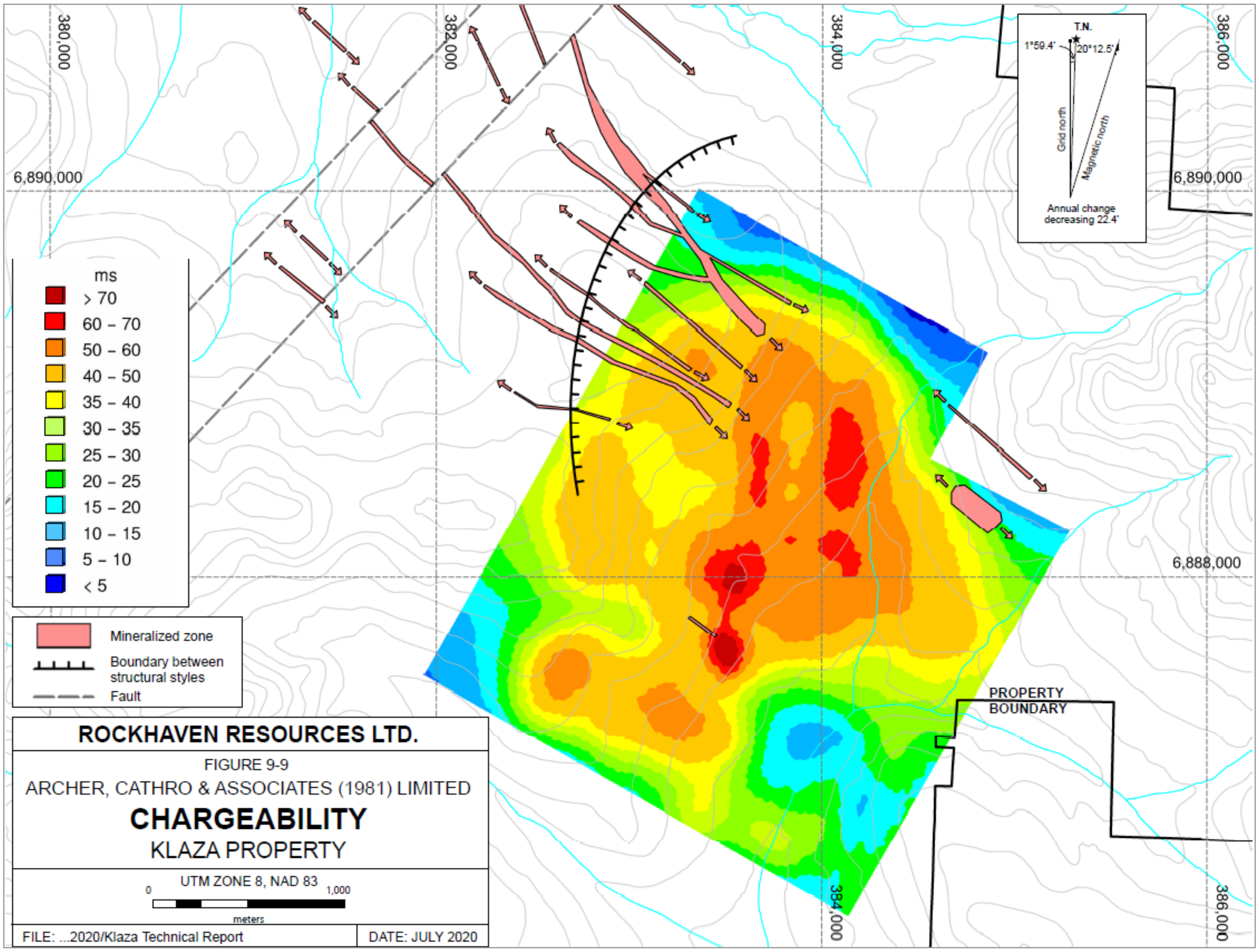
Source: Archer, Cathro & Associates (1981) Limited.

Figure 9.8 VLF-EM data – Work Area 2 Klaza Property



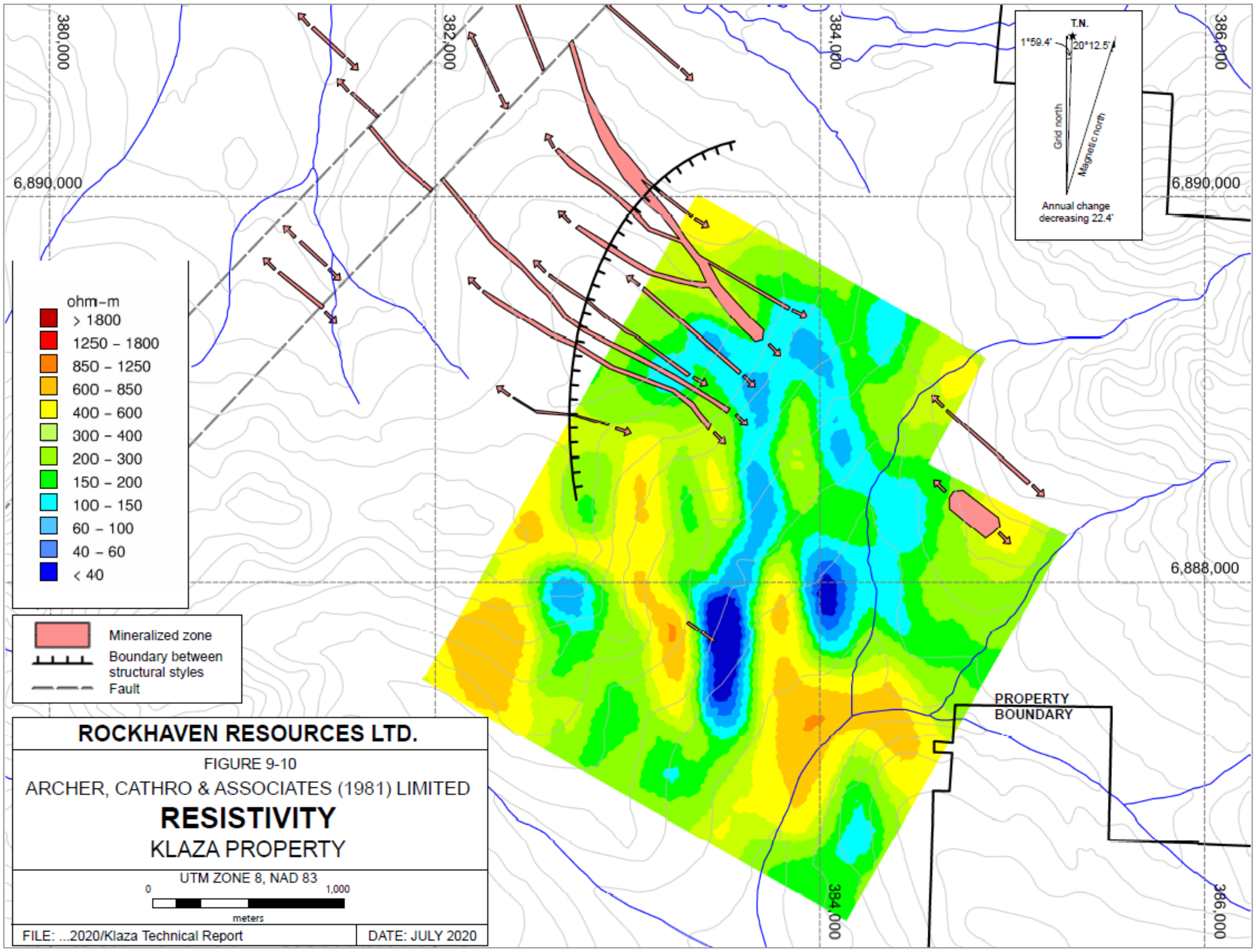
Source: Archer, Cathro & Associates (1981) Limited.

Figure 9.9 Chargeability – Work Area 2 Klaza Property



Source: Archer, Cathro & Associates (1981) Limited.

Figure 9.10 Resistivity – Work Area 2 Klaza Property



Source: Archer, Cathro & Associates (1981) Limited.

10 Drilling

The Mineral Resource estimation discussed in this report was determined using the data provided by diamond drilling completed by Rockhaven between 2010 and 2017. It does not include any of Rockhaven's percussion drill results or any historical drill data. It also does not include assays from trenching. Drilling in 2019 was conducted peripheral to the deposit area and is not included in the Mineral Resource estimation.

10.1 Diamond drilling summary

Between 2010 and 2019, a total of 100,200.85 m of exploration and definition drilling was done by Rockhaven in 468 diamond drillholes on the Property. Figure 10.1 shows the location of all 468 of the diamond drillholes.

The majority of diamond drillholes were collared at dips of -50° and had azimuths of 030° to 035° (north-northeast) as shown on Figure 10.1. Drilling was completed on section lines spaced roughly 50 m apart along much of the lengths of both the Klaza and BRX zones.

Some of the 2015 drilling was done in part for geotechnical and environmental purposes. To monitor seasonal water levels and frost variations, vibrating wireline piezometers were installed in four holes and a thermistor was installed in one hole. Five diamond drillholes, totalling 308.76 m, were drilled vertically, peripheral to the Mineral Resource areas, as water monitoring wells.

Core recovery was good, averaging 96%, excluding the top 10 m of the holes where core recovery was poor. The holes from the 2010 to 2012 programs were mostly sampled top to bottom (about 99% of core was sampled), while only visually mineralized or altered intervals and adjacent wallrocks were sampled in 2014 through 2017. No drilling was conducted in 2013.

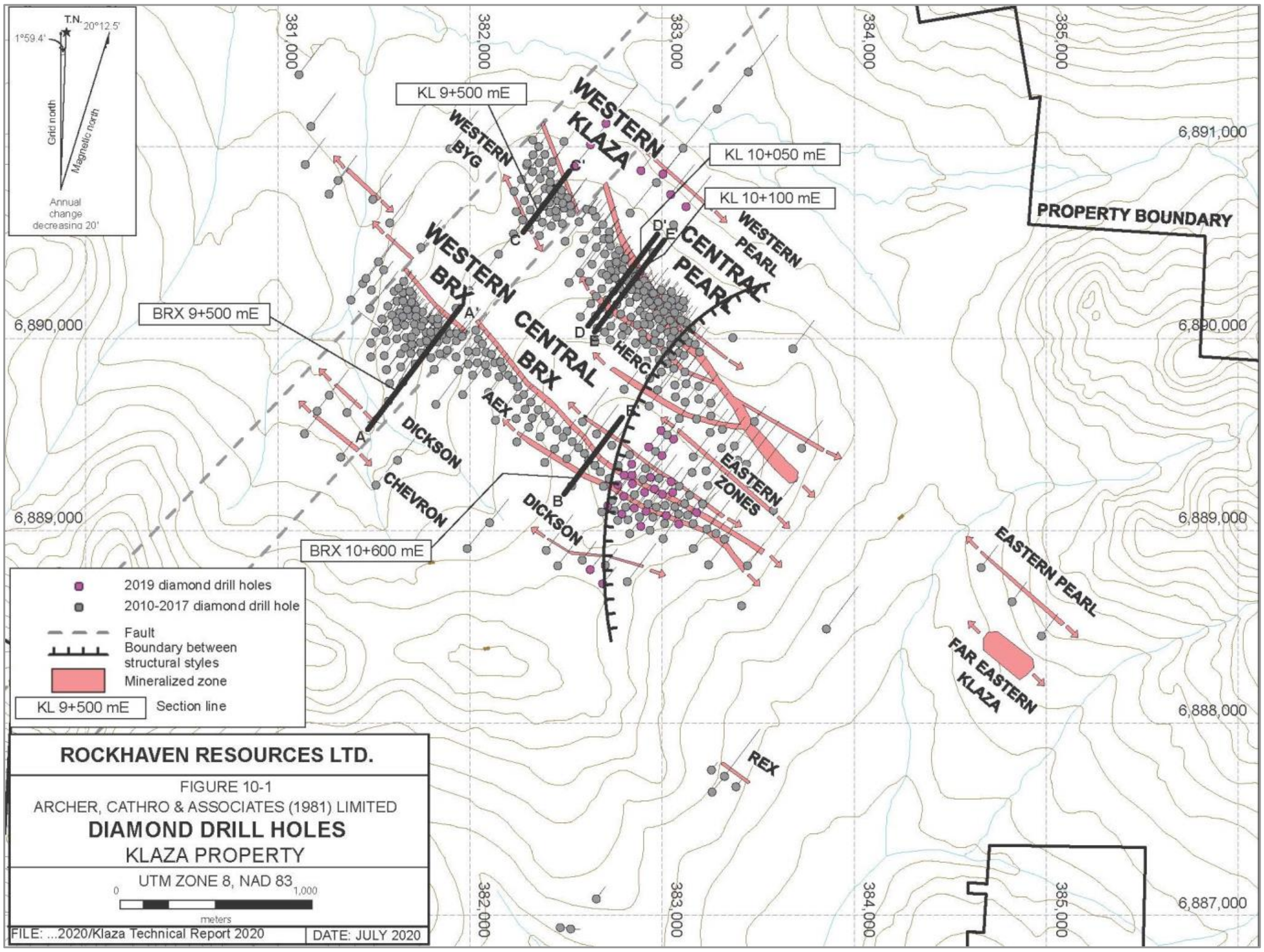
Final drillhole depths within the Klaza zone averaged 217.28 m, which included a maximum drillhole depth of 550.77 m. At the BRX zone, final drillhole depths averaged 213.32 m and reached a maximum of 567.26 m. The number of holes and total metres drilled on the Property each year between 2010 and 2019 are listed by zone in Table 10.1.

Table 10.1 2010 to 2019 diamond drilling summary

Target – year	Holes drilled	Total drilled (m)
Klaza zone – 2010	7	1,035.10
BRX zone – 2010	4	606.99
Klaza zone – 2011	39	11,211.85
BRX zone – 2011	9	1,717.25
Klaza zone – 2012	27	8,351.38
BRX zone – 2012	31	9,652.55
Klaza zone – 2014	34	6,709.97
BRX zone – 2014	58	10,140.73
Klaza zone – 2015	28	6,495.91
BRX zone – 2015	22	6,541.01
Klaza zone – 2016	25	4,204.19
BRX zone – 2016	12	2,591.74
Klaza zone – 2017	49	7,404.31
BRX zone – 2017	35	5,227.05
Nearby exploration – 2011 - 2019	88	18,310.82

Note: nearby exploration includes all holes drilled outside the BRX and Klaza zones.
Source: Archer, Cathro & Associates (1981) Limited.

Figure 10.1 Diamond drillhole locations – Klaza Property



Source: Archer, Cathro & Associates (1981) Limited.

Table 10.2 shows the drill confirmed strike length of each of the main zones and the maximum down-dip intersection depth in each zone.

Table 10.2 Data for main mineralized zones

Zone	Mineralized strike length (m)	Maximum down-dip drill intersection (m)
Western Klaza	450	250
Central Klaza	850	325
Eastern Klaza	1,100	180
Western BRX	500	520
Central BRX	1,900	400
Pika	740	250
AEX	1,650	310
BYG	800	150
Western BYG	250	310
Eastern BYG	450	100
Dickson	450	100
HERC	460	310
Chevron	250	90
Stroshein	450	280
Pearl	320	270

Source: Archer, Cathro & Associates (1981) Limited.

All of the mineralized zones listed above begin at surface and are open to expansion along strike and to depth.

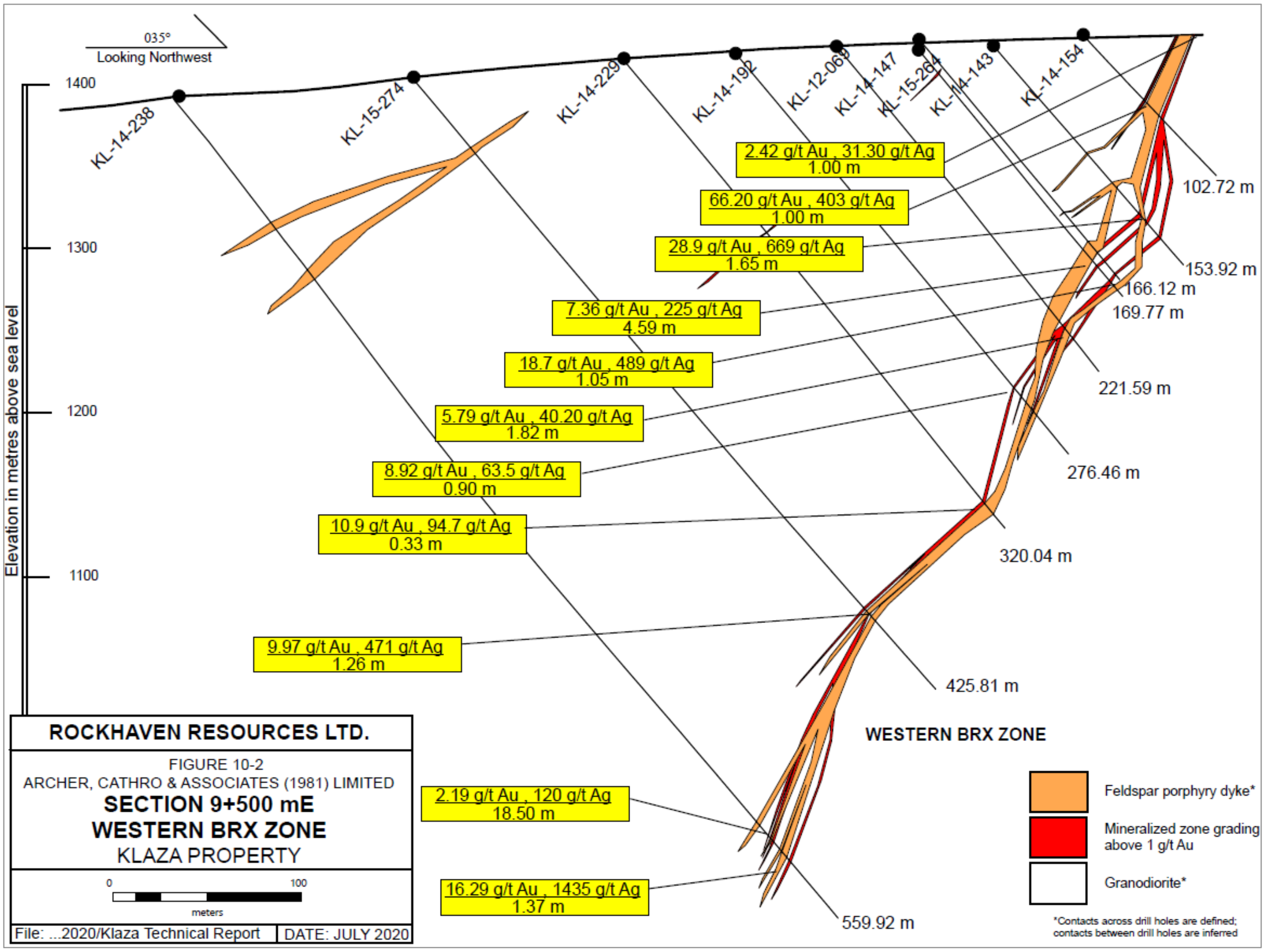
Although significant drill intersections have been obtained from all of the main mineralized zones, the focus of the most recent exploration has been the discrete high-grade veins within the BRX and Klaza zones. For the purposes of deposit modelling and Mineral Resource estimation, the BRX and Klaza zones have been subdivided as follows:

- BRX zone – Central BRX, and Western BRX zones.
- Klaza zone – Central Klaza, and Western Klaza zones.

The BRX zone has been traced for approximately 2,400 m along strike and been tested to a maximum depth of 520 m down-dip. Mineralization is associated with a laterally extensive north-west striking and moderately to steeply south-west dipping feldspar porphyry dyke. Veins occur on the margins of that dyke and, where the dyke bifurcates, the number of veins increases, which sometimes results in wider mineralized intervals.

The Western BRX zone consists of quartz veins and vein zones that contain pyrite, arsenopyrite, galena, sphalerite, chalcopyrite and sulphosalts. Carbonate gangue facies in these veins largely comprises manganiferous carbonates (rhodochrosite). Figure 10.2 illustrates the geometry of the mineralization defining this zone.

Figure 10.2 Section 9+500 mE Western BRX zone – Klaza Property



Source: Archer, Cathro & Associates (1981) Limited.

One of the best intersections in the Western BRX zone came from KL-14-238, which intersected multiple veins within an 18.5 m interval that averaged 2.19 g/t gold and 120 g/t silver. The best of the veins in that interval graded 16.29 g/t gold and 1,435 g/t silver over 1.37 m. At 520 m down-dip, this is the deepest intersection to date on the Property.

The Central BRX zone features veins and vein zones that are dominated by quartz, pyrite and iron-rich carbonates such as ankerite and siderite. Pyrite, sphalerite and galena are the main sulphide minerals, while arsenopyrite and sulphosalts are absent, or present in only minor quantities. A type section depicting the geometry of the mineralized veining relative to the dyke is shown in Figure 10.3.

The mineralogical differences between the Western BRX and Central BRX zones suggest some degree of vertical off-set along a major cross-fault, which separates the two segments of the zone.

The Klaza zone is on a parallel trend about 800 m north-east of the BRX zone. Drillholes have tested along the zone on section lines spaced approximately 50 m apart. The Klaza zone has been subdivided into three subzones - Western Klaza zone, Central Klaza zone and Eastern Klaza zone. The former two subzones are included in the Mineral Resource estimate and described below. The

Western and Central Klaza zones are off-set by the same cross-fault that separates the corresponding sections of the BRX zone.

The Western Klaza zone is defined by two narrow high-grade silver-gold veins (extending west from section KL 9+700). Unlike other zones, these veins are not emplaced alongside a feldspar porphyry dyke and they are not flanked by the type of sheeted veining seen elsewhere in the Klaza zone. The mineral assemblages in the Western Klaza zone contain higher proportions of arsenopyrite and sulphosalts than are common further east in the Klaza zone, and silver to gold ratios are higher.

Mineralization in the Central Klaza zone (east of section KL 9+700 m and west of section KL 10+600 m) is hosted within a laterally extensive complex of steeply dipping veins, breccias and sheeted veinlets, which are associated with a swarm of feldspar porphyry dykes. The strongest veins are typically found along dyke margins. Pyrite, arsenopyrite, galena and sphalerite are the main sulphide minerals in this subzone. The deepest hole at the Klaza zone, KL-12-133, intersected strong mineralization over 6.70 m suggesting the zone is open at depth.

Type sections for the Western Klaza and Central Klaza zones are shown on Figure 10.4 and Figure 10.5, respectively.

Mineralization within the eastern portions of the BRX and Klaza zones comprises a series of closely spaced, narrow, sub-parallel veins and vein zones dominated by quartz, pyrite and lesser chalcopyrite. Unlike the Central and Western zones, sulphide mineralization in the Eastern zones contains little arsenopyrite, galena, and sphalerite.

Exploration activities up to and including 2017 have assumed mineralization within the eastern zones behaves similarly to the discrete high-grade veins in the central and western zones. The drill spacing, and core sampling procedures used were designed for this style of target.

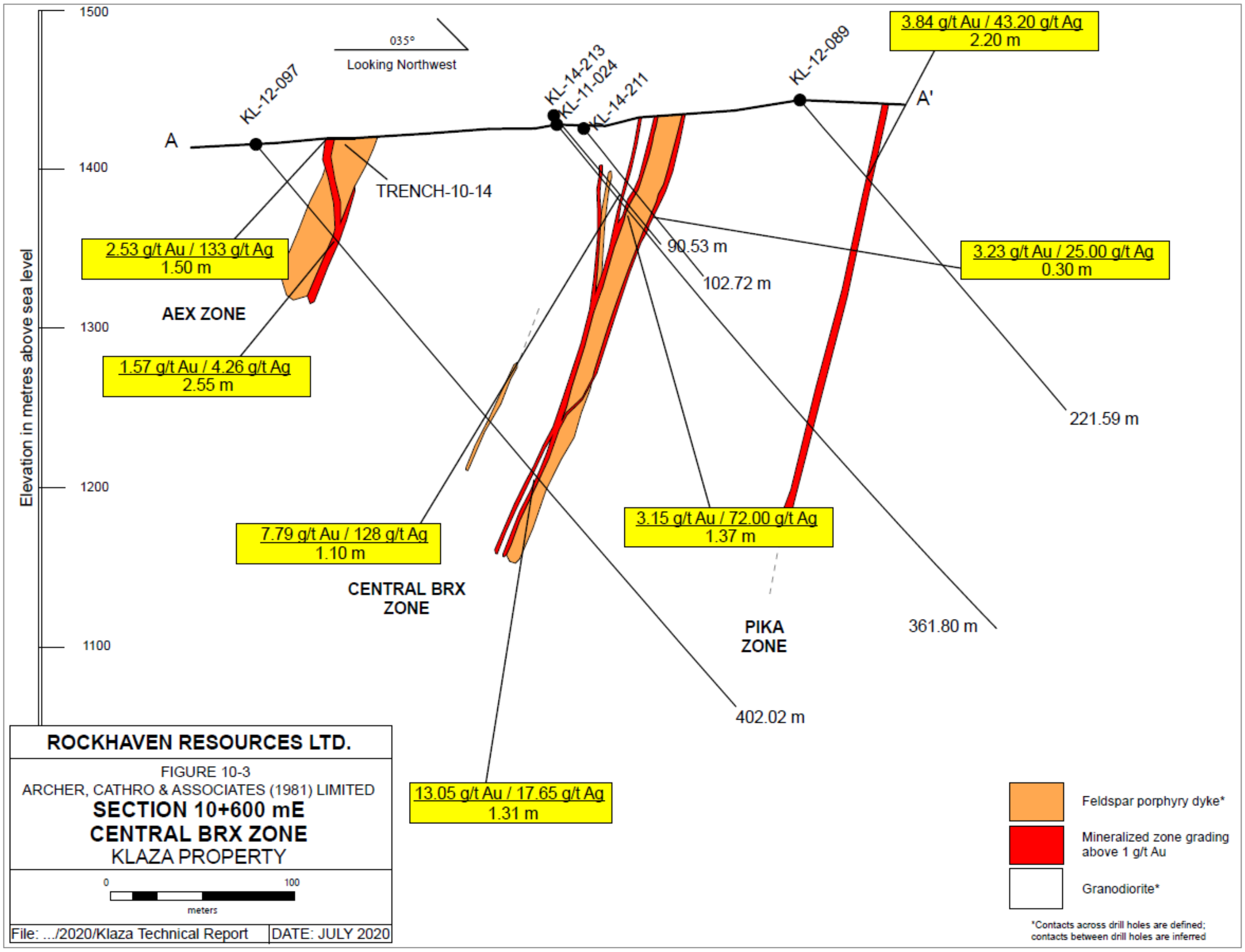
Diamond drilling in 2019 included 24 shallow infill drillholes, totalling 3,300.98 m, within the Eastern Zones (Willms and Turner 2020). These holes were designed to test stacked sub-parallel quartz veins and veinlets that form a broad, bulk-tonnage style vein zone.

Seven of the 2019 diamond drillholes tested the down dip and strike extension of the Central and Western Pearl zones, which were discovered in 2017.

In early 2018, results from metallurgical testing on core collected in 2017 highlighted the low-grade bulk tonnage potential of the eastern zones. The eastern zones were omitted from the 2018 Mineral Resource estimate because gaps within the current sampling record and the drill spacing do not adequately support a Mineral Resource estimate on this bulk tonnage target at this time.

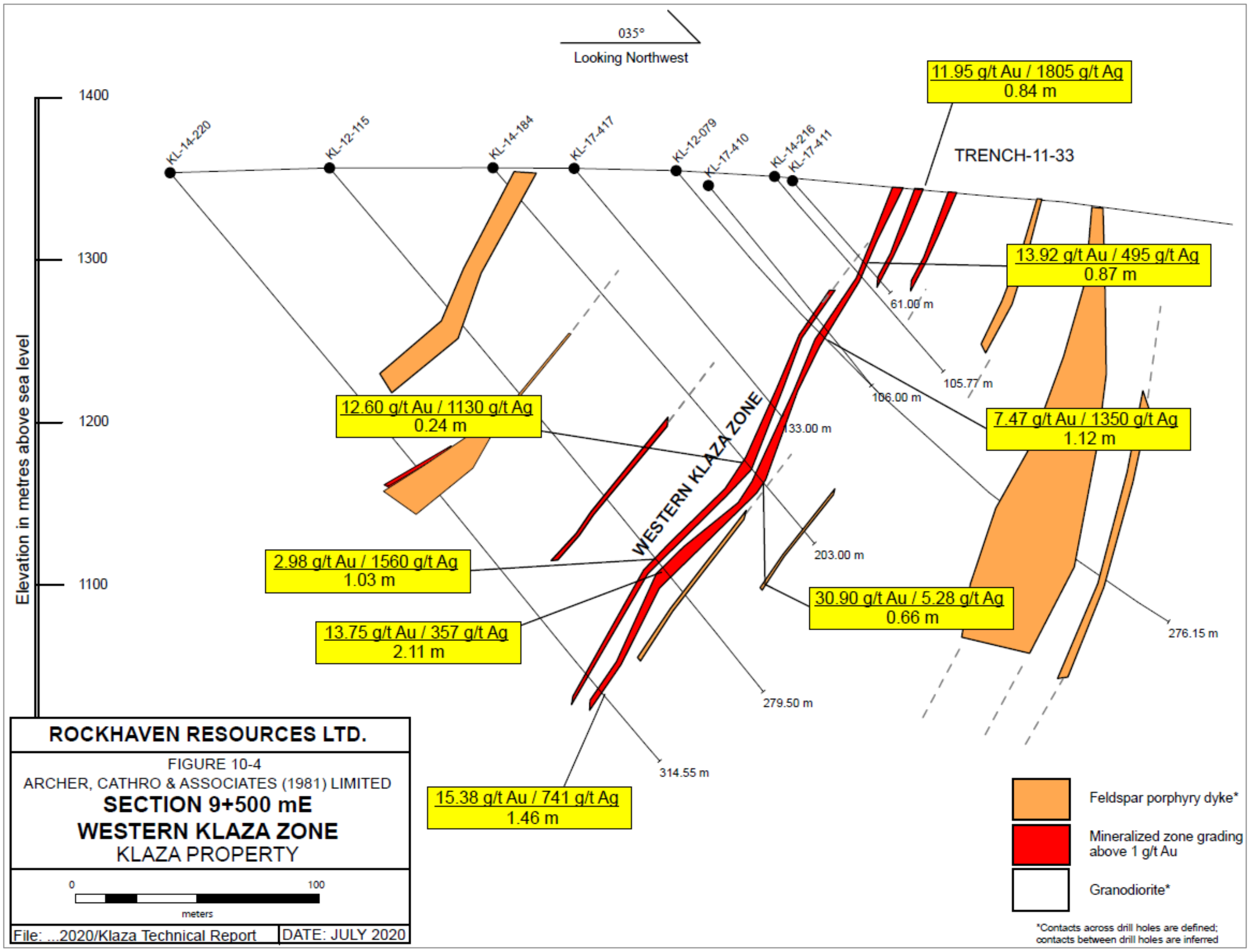
The QP does not know of any drilling, sampling or recovery factors that could materially impact the accuracy and reliability of the 2010 to 2019 results.

Figure 10.3 Section 10+600 mE Central BRX zone – Klaza Property



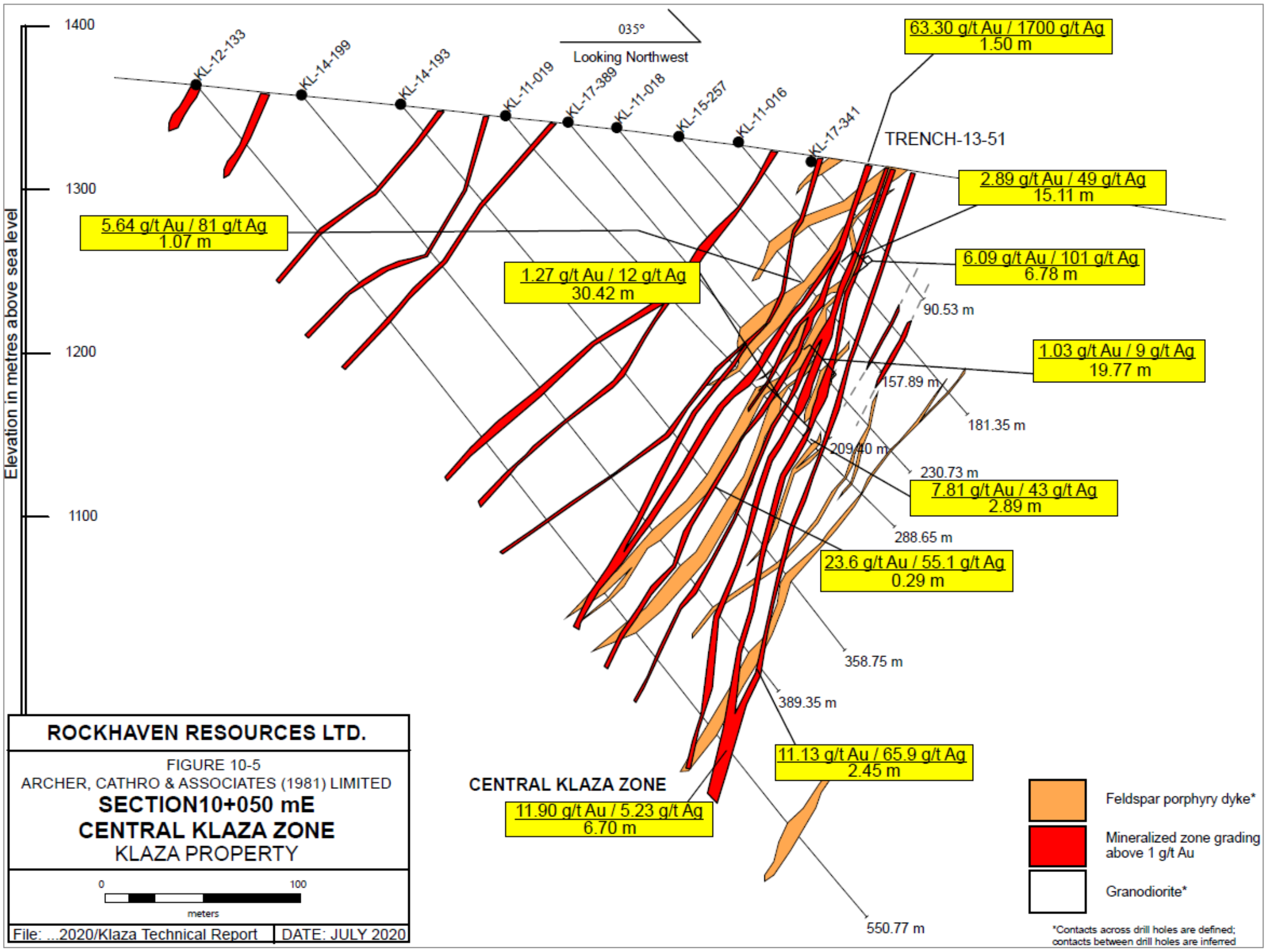
Source: Archer, Cathro & Associates (1981) Limited.

Figure 10.4 Section 9+500 mE Western Klaza zone – Klaza Property



Source: Archer, Cathro & Associates (1981) Limited.

Figure 10.5 Section 10+050 mE Central Klaza zone – Klaza Property



Source: Archer, Cathro & Associates (1981) Limited.

10.2 Diamond drilling specifications

In 2010, diamond drilling on the Property was contracted to Top Rank Diamond Drilling Ltd. of Ste Rose du Lac, Manitoba, and was done with two skid-mounted, diesel-powered JKS-300 drills using NTW and BTW equipment.

In 2011, diamond drilling on the Property was contracted to three companies: Swiftsure Diamond Drilling Ltd. of Nanaimo, British Columbia; Strike Diamond Drilling of Kelowna, British Columbia; and, Elite Diamond Drilling of Vernon, British Columbia. The work was done using two skid-mounted, diesel-powered A-5 drills and one skid-mounted, diesel-powered JKS-300 drill. The A-5 drills used HQ equipment while the JKS-300 used BTW equipment.

In 2012, diamond drilling on the Property was contracted to four companies: Swiftsure Diamond Drilling Ltd., Strike Diamond Drilling, Elite Diamond Drilling, and Platinum Diamond Drilling Inc. of Winnipegosis, Manitoba. The work was done using three skid-mounted, diesel-powered A-5 drills and one skid-mounted, diesel-powered JKS-300 drill. The A-5 drills used HQ and NQ equipment while the JKS-300 used BTW equipment.

In 2014 through 2017, diamond drilling on the Property was contracted to Platinum Diamond Drilling Inc. Most of the work was done using two skid-mounted, diesel-powered A-5 drills, with HQ and NQ equipment. A skid-mounted, diesel-powered Discovery II diamond drill using NQ equipment was also utilized in 2014.

Diamond drilling in 2019 was completed by Kluane Drilling Ltd of Whitehorse, Yukon using two skid mounted, diesel-powered KD1000 drills, with NTW equipment.

10.3 Drill collar and down-hole surveys

All drillhole collars were surveyed by Archer Cathro employees using a Trimble SPS882 and SPS852 base and rover Real Time Kinematic (RTK) GPS system. The collars are marked by individual lengths of drill rod that are securely placed into holes. A metal tag identifying the drillhole number is affixed to each drillhole marker.

Most drill collars were aligned at surface using a Brunton compass. In 2014, a Reflex North Finder APS, a GPS based compass, was used to align the later drillholes (KL-14-181 and higher). Beginning in 2015, all drillholes were aligned using the APS tool.

To determine the deflection of each drillhole, the orientation was measured at various intervals down the hole. In 2010 only, the dip was assessed by using an acid test taken at the bottom of the hole, while holes completed in 2011 and 2012 were measured every 50 feet (ft) (15 m) using a "Ranger Explorer" magnetic multi-shot tool provided by Ranger Survey Systems. Measurements taken and recorded were azimuth, inclination, temperature, roll angle (gravity and magnetic) plus magnetic intensity, magnetic dip and gravity intensity (for quality assurance). All readings were reviewed, and erroneous data were not used when plotting the final drillhole traces.

Drillholes completed during the 2014 through 2019 programs were routinely surveyed every 50 ft (15 m) using a Reflex EZ-Trac down-hole multi-shot magnetic survey instrument. At each survey station, this instrument recorded the drillhole azimuth and inclination as well as the magnetic intensity, temperature and other variables used for validating the readings.

Late in the 2014 season a manufacturing error with the magnetic sensors was discovered in one of the down-hole survey instruments used in 35 holes. Once identified, the faulty instrument was immediately replaced. To determine the orientation of the affected drillholes, data from reliable surveys was plotted on a scatter plot showing the rate of change (°/m) against the down-hole distance of the survey station. A best fit line was then passed through the data points and the equation of this line determined. This equation approximates the deviation of the drillholes and was used to calculate the deflection for the holes surveyed using the faulty instrument. These equations are presented in the following table, where c is equal to the rate of change (°/m) and d the down-hole distance.

Table 10.3 Downhole survey correction factor

HQ	c=0.00006 x d + 0.0027
NQ2	c=0.00006 x d + 0.0062

Source: Archer, Cathro & Associates (1981) Limited.

To calculate the azimuth at a given depth, the rate of change was calculated for each station. This was multiplied by the distance to the preceding station and added to the preceding azimuth. The surface orientation as recorded either by compass or with the APS, if available, was used as the initial azimuth at 0.00 m depth. The approximated azimuth values calculated using this equation for the drillholes surveyed only with the faulty instrument were determined to be adequate for further use and have been included in the drillhole database. While this approximation method is considered reliable for shallow holes, it should only be used where no other data exists and not be used for survey stations much beyond 300 m.

For a detailed description and validation of this calculation, please refer to the technical report entitled "Geology, Mineralization, Geochemical Surveys, Geophysical Surveys, Diamond and Percussion Drilling, Metallurgical Testing and Mineral Resources on the Klaza Property, Yukon, Canada" dated 11 March 2015 and amended 19 June 2015 (Wengzynowski et al. 2015).

10.4 Oriented core surveys

A Reflex ACT III downhole digital core orientation system was used in 2014 and 2015 to orient the core in a total of 46 holes.

In 2015, 18 of the 19 oriented drillholes were drilled using split tubes. The use of split tubes allowed orientation measurements to be collected across incompetent intervals or intervals with poor recovery.

Split tube intervals were oriented by Archer Cathro employees at the drill site. The core tube was first aligned by the driller's helper using the ACT III tool before the split tube was extracted from the core tube. Care was taken to not shift the core during this process. A line representing the top of the hole was marked down the length of the core by the Archer Cathro employees. Structural orientation measurements within the interval were taken prior to the core being transferred to core boxes.

11 Sample preparation, analyses, and security

11.1 Introduction

This section describes the sampling methods, sample shipment and security, analytical techniques, quality assurance / quality control (QA/QC) and data validation, followed during the 2010 to 2019 exploration programs. All exploration programs were supervised by Archer Cathro on behalf of Rockhaven.

Results of work completed during 2016 and 2017 are discussed in detail in this report. Results from 2010 – 2015 were discussed by AMC previously in "Technical Report and PEA for the Klaza Au-Ag Deposit, Yukon, Canada for Rockhaven Resources Ltd.", dated 26 February 2016.

Drilling from 2019 was conducted outside of the deposit area and is not discussed in detail in this section.

The QP is satisfied that the exploration approach and sample data from 2010 – 2017 are of sufficient quality for inclusion in resource evaluation studies. This data has been used for the Mineral Resource Estimation described in Section 14 of this report.

11.2 Sampling methods

11.2.1 Soil sampling methods

Soil sampling was conducted on the Property between 2010 and 2017. In the deposit area, grid soil samples were collected at 50 m intervals on lines spaced 100 m apart and oriented at 037°. All soil sample locations were recorded using hand-held GPS units. Sample sites were marked by aluminum tags inscribed with the sample numbers and affixed to 0.5 m wooden laths that were driven into the ground. Soil samples were collected from 30 cm to 80 cm deep holes dug with hand-held augers. They were placed into individually pre-numbered Kraft paper bags. Sampling was often hindered by permafrost on moss-covered, north-facing slopes. Samples were not collected from some locations due to poor sample quality.

11.2.2 Rock and trench sampling methods

All rock samples collected from the Klaza and BRX zones were taken from excavator trenches, because there are no naturally outcropping exposures of these zones.

Continuous chip samples were collected from excavator trenches in several parts of the Property during programs conducted between 2010 and 2017. The collection protocol for chip samples was as follows:

1 Trenches were excavated.

2 The walls of trenches were cleaned, where necessary, with a shovel.

3 Trenches were mapped, and sample intervals marked at geological breaks or at 1 to 10 m intervals depending on the intensity of alteration and mineralization.

4 Continuous chip samples were collected along one wall of the trench as close to the floor of the trench as slumping would allow using a geological hammer. The chips were collected either in a tub or on a sample sheet. Sample sizes averaged approximately 2.0 kilograms (kg) per linear metre sampled for intervals containing veins and about 1.5 kg per linear metre sampled for intervals comprised primarily of altered wallrock.

5 Samples were placed in doubled 6 mm plastic bags along with a pre-numbered sample tag, then two or three samples were placed in a fiberglass bag sealed with a metal clasp and sample numbers were written on the outside of that bag with permanent felt pen.

6 From 2011 to 2015, one blank and one Certified Reference Material (CRM standard) sample were randomly inserted into every batch. No quality control samples were inserted into batches in 2010.

7 In 2013, samples collected from trenches within the core of the BRX and Klaza zones were divided into batches comprising 31 trench samples plus one blank sample, one assay standard and one coarse reject duplicate sample.

11.2.3 Diamond drill core sampling methods

Geotechnical and geological logging was performed on all drill core from the 2010 to 2017 drill programs. Prior to 2015, all logging data were recorded as a hardcopy during the day and transcribed to digital format during the evenings. Beginning in 2015, drill logs were entered directly into a digital database.

Drill core samples were collected using the following procedures:

1 Core was reassembled, lightly washed and measured.

2 Core was wet photographed.

3 Core was geotechnically logged.

4 Magnetic susceptibility measurements were taken at 1 m intervals along the core.

5 Core was geologically logged, and sample intervals were designated. Sample intervals were set at geological boundaries, drill blocks, or sharp changes in sulphide content.

6 Core recovery was calculated for each sample interval.

7 From 2010 to 2011, visually promising core intervals were sawn in half using a rock saw and the remainder of the core was split with an impact core splitter. In 2012, all visually promising core intervals were sawn in half using a rock saw, while selected specimens of altered country rock were split using an impact core splitter. From 2014 onwards, all marked samples were cut using a rock saw. In each case, one half of the core was sampled, and the remaining half was placed back in the core box.

8 All samples were double bagged in 6 mm plastic bags, a pre-numbered sample tag was placed in each sample bag, then two or three samples were placed in a fiberglass bag sealed with a metal clasp and sample numbers were written on the outside of that bag with permanent felt pen. From 2012 onwards, the fibreglass bag was sealed with a numbered security tag.

9 Two blank and two assay standard samples were randomly included with every batch of 30 samples (prior to 2012, batches comprised 31 core samples).

10 One duplicate sample consisting of quarter-split core was included with every batch of 30 samples (prior to 2012, batches comprised 31 core samples).

11 Starting in 2012, one coarse reject duplicate sample was included with every batch of 30 core samples.

12 In 2019, batches comprised 35 samples, 1 blank sample, 2 standard samples, one field duplicate, and one coarse reject duplicate. Field duplicates consisted of two ¼ core duplicates prepared from the same ½ core.

A geotechnical log was filled out prior to geological logging of drill core and included the conversion of drill marker blocks from imperial to metric plus determinations of core, rock quality designations (RQD), hardness and weathering. From 2015 onwards, fracture frequency, joint sets, and joint set roughness, shape and infill were also recorded.

Within oriented intervals, alpha and beta angles were recorded for each joint along with the roughness, shape, and infill material and thickness.

A total of 172 point load measurements were taken on core in 2015 using an ELE International digital point load test apparatus (Model 77-0115). Both axial and diametral measurements were taken intermittently on all rock types except veins. The narrow nature of the veins and volume requirements of the apparatus prohibited testing of the veins.

Density measurements were systematically taken on core, throughout each of the drill programs except in 2010. A total of 2,644 density measurements were collected between 2011 and 2017 from a variety of holes and lithologies. Measurements are mostly from vein, porphyry dyke, fresh granodiorite and mineralized granodiorite, but also include aplite and mafic dyke material. Sample densities were determined by cutting a 10 cm long section of core and then determining its weight dry and its weight immersed in water. The data were then applied to the following formulas, as applicable, to establish the density of each of these samples:

$$\textit{Density} = \textit{weight in air} \div [\textit{Pi} \times (\textit{diameter of core} \div 2)^2 \times \textit{length of core}]$$

For samples that could not be cut, a graduated cylinder (filled with water) was used to calculate the volume of the core sample and in turn the sample’s density. Employing this technique, each sample was first weighed in air, and then its displacement was calculated using a volumetric cylinder. A second formula was then used to determine the density of each sample:

$$\textit{Density} = \textit{weight in air} \div (\textit{Final Volume} - \textit{Initial Volume})$$

In addition to density, the specific gravity was calculated using the following formula for each sample wherever possible.

$$\textit{Specific Gravity} = \textit{weight in air} \div (\textit{weight in air} - \textit{weight in water})$$

Density calculated using the volumetric method is the preferred value. Where this is unavailable, the calculated volume value is the second choice. Values derived from each of the three methods were compared against each other. Any significant discrepancies between methods were investigated and corrected. If no resolution was determined, the measurement was removed from the database.

Note that density measurements ignore the potential impact of pore space. As the rock is generally competent and contains minimal voids, the density measurements are considered to be a good approximation of bulk density.

Only vein zones and associated peripheral alteration were sampled beginning in 2014. Care was taken during all drill programs to ensure that the sample split was not biased to sulphide content and, therefore, the sampling should be reliable and representative of the mineralization.

11.3 Sample shipment and security

In 2010, all drill core was trucked to the Archer Cathro yard in Whitehorse for logging and splitting. Between 2011 and 2019, drill core was logged and sawn or split at a processing facility on the Property. Chip samples taken between 2010 and 2017 were collected and labelled at the trenches on the Property.

In 2010, Archer Cathro personnel were responsible for transporting all samples from Archer Cathro’s Whitehorse yard to ALS Minerals’ Whitehorse preparation facility. Between 2011 and 2019, Archer Cathro personnel were responsible for transporting all samples from the Property by truck to ALS Minerals’ facility in Whitehorse for preparation. ALS Minerals was responsible for shipping the prepared sample splits from Whitehorse to its North Vancouver laboratory, where they were analyzed. All samples were controlled by employees of Archer Cathro until they were delivered directly to ALS Minerals in Whitehorse.

In 2012 through 2019, Archer Cathro ensured that a Chain of Custody form accompanied all batches of drill core during transportation from the Property to the preparation facility. A unique security tag was attached to each individual fiberglass bag when the bag was sealed. The bags and security tags had to be intact in order to be delivered to ALS Minerals. If a security tag or bag arrived at the laboratory damaged, an investigation into the transportation and handling of that sample bag was undertaken by ALS Minerals and Archer Cathro and any affected samples were not processed until a resolution was reached regarding the security of the samples.

Prior to shipping, each individual sample was weighed. These weights were compared to weights recorded by ALS Minerals upon receiving the samples. Any discrepancies between the two weights were investigated.

11.4 Sample preparation and analysis

All samples were sent to ALS Minerals' laboratory in Whitehorse for preparation and then on to its laboratory in North Vancouver for analysis. ALS Minerals, a wholly owned subsidiary of ALS Limited, is an independent commercial laboratory specializing in analytical geochemistry services. Both ALS Minerals' Whitehorse and North Vancouver laboratories meet requirements of International Standards ISO/EC 17025:2017.

All soil samples were dried and screened to -180 microns. All rock, core and trench samples were dried, fine crushed to better than 70% passing -2 mm and then a 250 g split was pulverized to better than 85% passing 75 microns. Between 2014 and 2019, visually mineralized intervals and adjoining samples were prepared using a technique designed for samples where coarse gold and silver could be present. Using this technique, the sample is first dried and crushed to better than 90% passing 2 mm, then a 1,000 g split is taken and pulverized to better than 95% passing 106 microns.

In 2010 and 2011, all core and trench samples were initially analyzed for gold by fire assay followed by atomic absorption spectrometry (Au-AA24) and 35 other elements by inductively coupled plasma-atomic emission spectrometry (ME-ICP41). Over limit values for gold were determined by fire assay and gravimetric finish (Au-GRA22) and silver values were determined using inductively coupled plasma-atomic emission spectrometry (Ag-OG46). Sample pulps from mineralized intervals of drill core from 2011 were later reanalyzed for lead and zinc as well as 46 other elements using four acid digestion followed by inductively coupled plasma-atomic emission spectrometry and mass spectrometry (ME-MS61). Over limit values for silver, lead and zinc were determined by inductively coupled plasma-atomic emission spectrometry (Ag / Pb / Zn-OG62).

Beginning in 2012, all core and trench samples were routinely analyzed for gold by fire assay followed by atomic absorption spectrometry (Au-AA24) and 48 other elements by four acid digestion (ME-MS61). All over limit values were determined for gold by fire assay and gravimetric finish (Au-GRA22), and for silver, copper, lead, and zinc by inductively coupled plasma-atomic emission spectrometry (Ag / Cu / Pb / Zn-OG62).

Soil samples collected in 2010 were analyzed for gold by fire assay with inductively coupled plasma-atomic emission spectrometry finish (Au-ICP21) and for 35 other elements using aqua regia digestion and inductively coupled plasma-atomic emission spectrometry. Soil samples collected between 2011 and 2019, were analyzed for gold by fire assay fusion and atomic absorption spectrometry (Au-AA24) and for 35 other elements using aqua regia digestion and inductively coupled plasma-atomic emission spectrometry.

11.5 Quality Assurance and Quality Control

For all of its exploration programs, Rockhaven routinely inserted CRMs, blanks and duplicates into each batch. In the 2010 and 2011 drill programs, commercially available assay CRM samples for gold and silver were purchased. Six project specific assay CRMs were prepared from coarse reject material from the 2011 and 2012 core samples for use during the 2012 through 2019 programs. These assay standards were prepared, homogenized and packaged by CDN Resource Laboratories Ltd. of Delta, British Columbia. All assay CRMs were certified by Smee & Associates Consulting Ltd. of North Vancouver, British Columbia.

Starting in 2012 batches comprised 30 samples. Two CRMs and blanks were inserted into the sample sequence in each batch. CRMs were placed randomly, while blanks were placed following visually mineralized intervals where possible. One quarter-core duplicate was also inserted into each batch at random locations chosen by the geologist while logging. From 2012 forward, one sample in each batch was selected at random by the geologist and recorded on the sample submittal form as a coarse duplicate, instructing the preparation laboratory to collect a second sample following crushing. This coarse duplicate was processed and analyzed at the same time as the rest of the batch. Prior to 2012, batches comprised 31 samples, and did not include coarse reject duplicates.

The following table summarizes the number of QA/QC samples analyzed each year by Rockhaven. No drilling was conducted in 2013.

Table 11.1 QA/QC samples by year

Year	All samples	Core samples	CRMs (standards)	Blanks	Quarter-core duplicates	Coarse reject duplicates
2010	867	746	49	50	22	0
2011	7,469	6,419	423	422	205	0
2012	13,663	11,359	760	789	373	382
2013	0	0	0	0	0	0
2014	7,450	6,201	415	416	195	223
2015	4,998	4,166	277	277	132	146
2016	3,844	3,202	213	216	100	113
2017	5,954	4,959	331	331	157	176
2018	0	0	0	0	0	0
2019	4,993	4,242	285	156	159	151
Total	**49,238**	**41,294**	**2,753**	**2,657**	**1,343**	**1,191**

Note: No drilling was conducted in 2013 or 2018. Coarse reject duplicates, as defined in this report, are a second pulp prepared from the same coarse reject material as the main sample.
Source: Compiled by AMC Mining Consultants (Canada) Ltd. from data provided by Archer Cathro.

Results from the QA/QC program were reviewed immediately upon receipt of the assay certificate and on a regular basis to identify potential biases.

Following core processing, individual samples were segregated based on a visual geological assessment of mineralization into samples appearing to be mineralized and those appearing un-mineralized. Samples were then packaged accordingly into separate batches for dispatch to the laboratory. Mineralized samples were prepared using a different preparation technique, appropriate for coarse gold and silver as described in Section 11.4. A second coarse reject duplicate was requested for visually mineralized batches in place of a quarter-core duplicate to preserve core and enable further studies (i.e. metallurgy) on mineralized intervals.

Rockhaven's 2016 – 2017 QA/QC program comprised 9,798 samples. This comprised 8,161 core samples, 544 CRMs, 547 blanks, 257 quarter-core duplicates, and 289 coarse reject duplicates.

Results from Rockhaven's 2019 exploration program have passed all of the QA/QC checks; however, they do not impact the Mineral Resource and are not discussed in the following sections.

11.5.1 Assay results of Certified Reference Materials

A total of seven different CRMs was used during Rockhaven's 2016-2017 program. Six of these CRMs were prepared by CDN Resource Laboratories from coarse reject material taken from Rockhaven's 2012 drill program. One higher-grade CRM was obtained from CDN Resource Laboratories commercial inventory.

The CRMs contain standard, predetermined quantities of gold, silver, lead, zinc, and copper which are all monitored in Rockhaven's QA/QC process. Since gold has the greatest impact on potential value, the following tables and figures focus on the gold assays.

Table 11.2 tabulates the CRMs used for the 2016 and 2017 programs and the expected values.

Table 11.2 Recommended values of Certified Reference Materials

CRM	Au (g/t)	Standard deviation	Ag (g/t)	Standard deviation	Pb (%)	Standard deviation	Zn (%)	Standard deviation
KL1	4.46	0.155	119.8	2.15	0.960	0.026	1.590	0.04
KL2	2.23	0.08	25.3	1.05	0.283	0.008	0.544	0.018
KL3	0.766	0.036	9.49	0.475	0.136	0.005	0.300	0.006
KL4	0.355	0.02	4.6	0.35	0.049	0.001	0.087	0.004
KL5	0.880	0.036	11.9	0.55	0.128	0.005	0.236	0.010
KL6	1.72	0.067	38.9	1.05	0.166	0.004	0.332	0.017
CDN-ME-1402	13.9	0.4	131	3.5	2.48	0.055	15.23	0.335

Source: Compiled by AMC Mining Consultants (Canada) Ltd. from data provided by Archer Cathro.

CRMs are inserted to check the analytical accuracy of the laboratory. AMC advocates an insertion rate of at least 5% of the total samples analyzed. AMC notes that the CRMs used show a range of values for each economic mineral to monitor analytical accuracy at various metal levels. This is appropriate given the two sample streams that Rockhaven employs.

CRM performance is typically monitored using control charts. The performance of the CRMs is measured against the standard deviation (SD) values that are provided with the CRMs. AMC advocates re-assaying assay batches with two consecutive CRMs occurring outside two SDs, or one CRM occurring outside three SDs.

Rockhaven has provided data for CRMs used during the 2016 and 2017 programs. Rockhaven's CRMs represent over 5% of the total samples assayed. Rockhaven runs two separate sample streams based on visual indications of mineralization and has selected CRMs that cover a range of grades within the typical sample population in each respective sample stream.

Rockhaven monitors CRMs immediately upon receipt of assays. Where a CRM fails (returns a result ±3 times the SD from the expected value), or there are two consecutive warnings (±2 times the SD from the expected value), the assay batch is re-run.

Table 11.3 shows a summary of the CRM assay results from the 2016 and 2017 programs for gold.

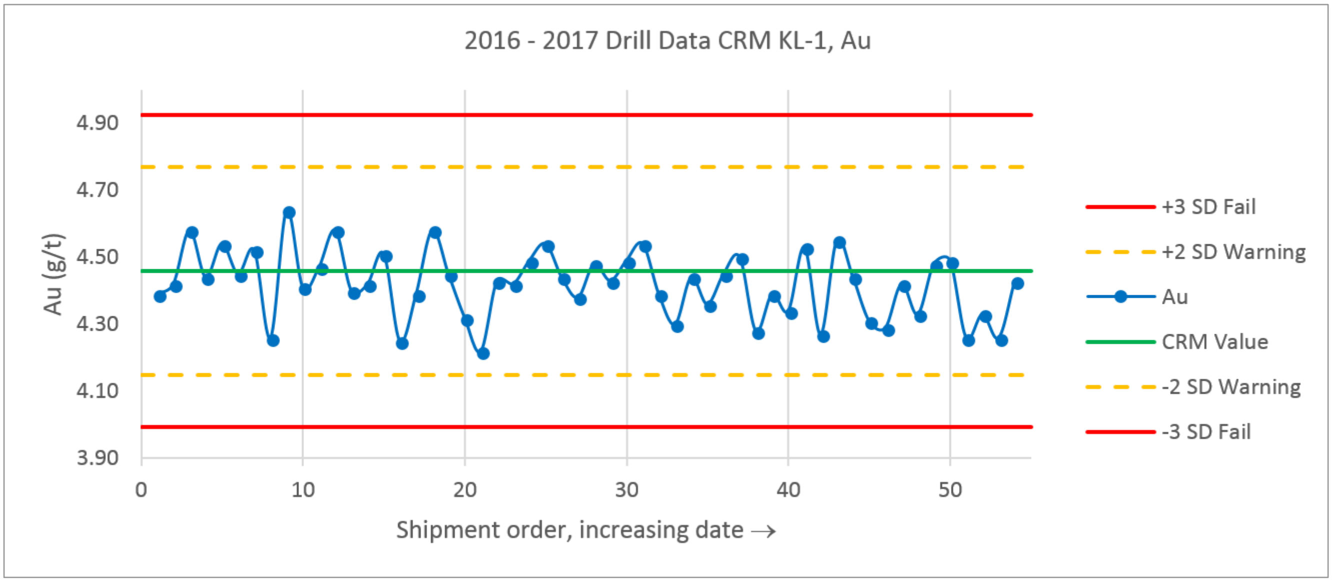
Table 11.3 2016 – 2017 Assay results of CRMs for gold

CRM	Expected Au value (g/t)	No. of assays	Warnings (±2 SD)	Fails (± 3 SD)	Sample switch	True fails
KL1	4.46	54	0	0	0	0
KL2	2.23	32	2	2	0	2
KL3	0.766	74	2	1	0	1
KL4	0.355	101	4	1	0	1
KL5	0.88	121	6	2	1	1
KL6	1.72	148	11	2	0	2
CDN-ME-1402	13.9	14	0	1	0	1
Total	**-**	**544**	**25**	**9**	**1**	**8**

Note: A sample switch is when the geologist has inserted a CRM into the sample bag, different from what he / she recorded on the sample sheet.
Source: Compiled by AMC Mining Consultants (Canada) Ltd. from data provided by Archer Cathro.

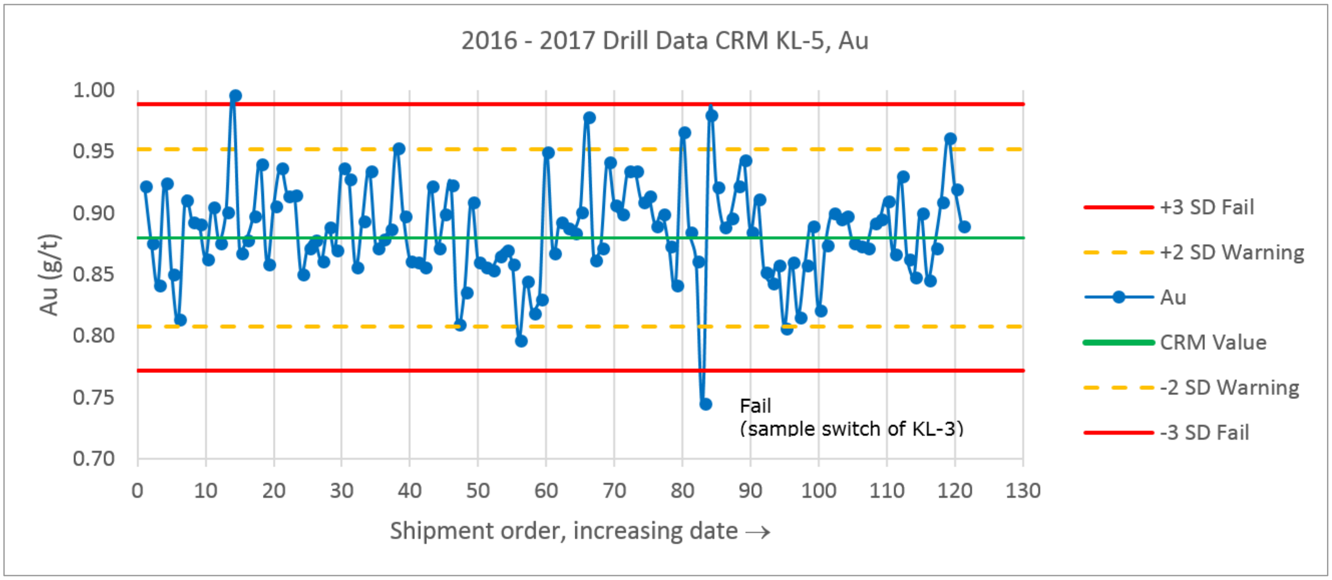
Figure 11.1 and Figure 11.2 show the results of 54 assays on CRM KL1 and 121 assays on CRM KL5.

Figure 11.1 Control chart for 2016 – 2017 KL1



Note: Warning limit is outside two SDs. Failure limit is outside three SDs.
Source: Compiled by AMC Mining Consultants (Canada) Ltd. from data provided by Archer Cathro.

Figure 11.2 Control chart for 2016 – 2017 KL5



Note: Warning limit is outside two SDs. Failure limit is outside three SDs.
Source: Compiled by AMC Mining Consultants (Canada) Ltd. from data provided by Archer Cathro.

AMC considers the CRMs chosen by Rockhaven to be appropriate to monitor laboratory accuracy within the respective sample streams at this stage of project development.

Rockhaven monitors CRM performance on a batch by batch basis and addresses batch issues using procedures that match industry standards.

11.5.2 Assay results of blank samples

Coarse blanks test for contamination during both the sample preparation and assay process. In AMC's opinion, the "pass mark" requirement for blanks is that 80% of coarse blank assays should be less than twice the detection limit for that element.

Rockhaven routinely inserts two blanks into each batch sent to the laboratory. Blanks are prepared from commercially available marble and are kept in bags in the core shack, away from any possible sources of contamination. Blank samples are prepared in advance and are weighed to ensure the total mass of each blank is close to the average mass of samples submitted.

A total of 547 blank samples were inserted into the sample sequence during Rockhaven's 2016 – 2017 programs. A total of 20 blank samples returned values gold values greater than two times the detection limit. When a blank fails, new pulps are prepared for the whole batch and the whole batch is re-assayed.

As 96% of the coarse blanks assays were less than twice the detection limit for gold, AMC considers the assay results of the blank materials to be acceptable.

11.5.3 Assay results of duplicates

Coarse, uncrushed duplicate samples monitor sampling variance (including that arising from crushing), analytical variance, and geological variance.

In AMC's opinion, duplicates should constitute around 5% of the samples submitted to the laboratory. Unmineralized samples should not be sent as duplicates because assays near the detection limit are commonly inaccurate.

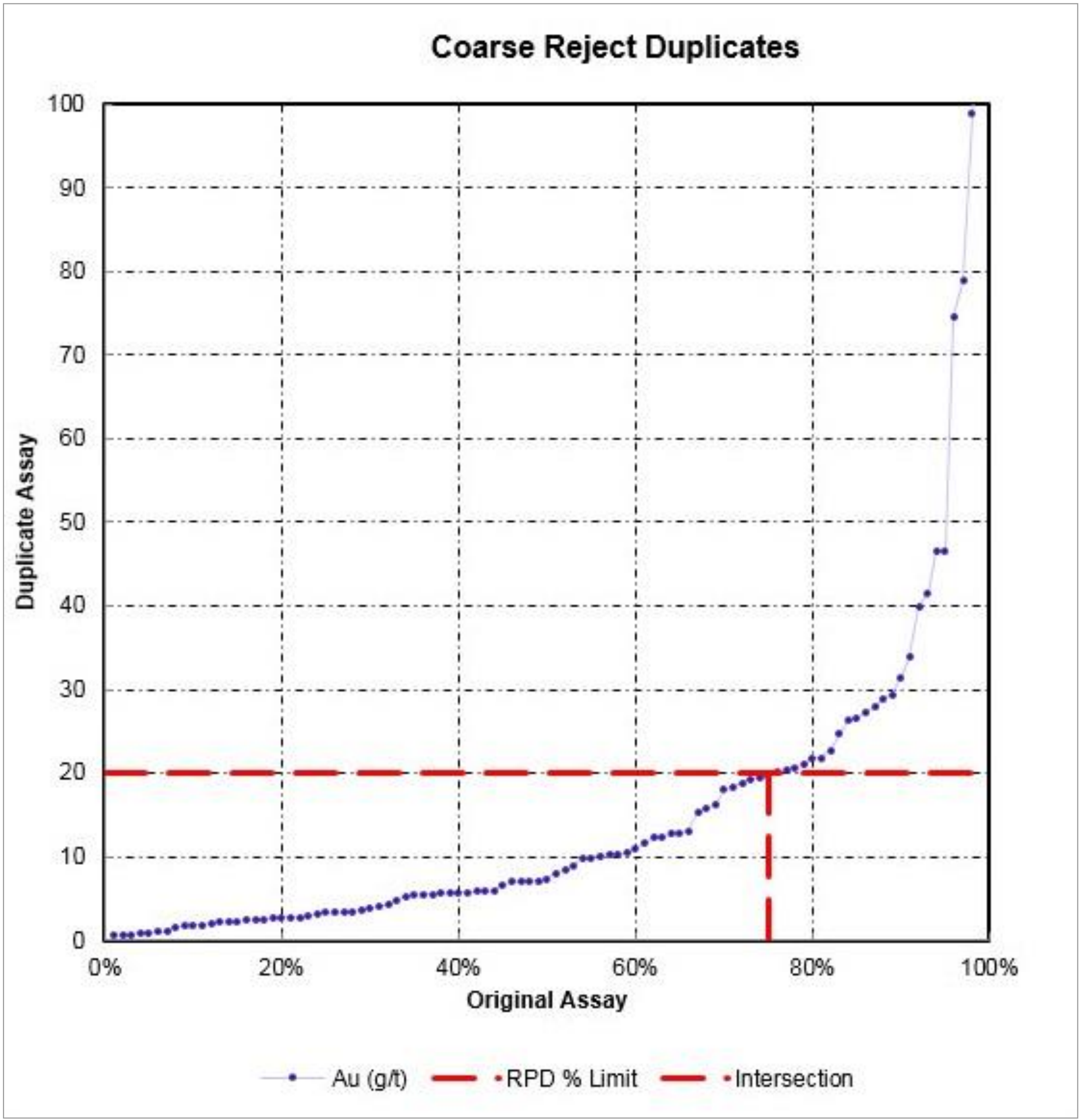
Duplicate data can be viewed on a scatterplot but should also be compared using the relative paired difference (RPD) plot. This method measures the absolute difference between a sample and its duplicate. It is desirable to achieve 80 to 85% of the pairs having less than 20% RPD between the original assay and check assay if it is a coarse duplicate (Stoker 2006). Sample pairs should be excluded from the analysis if the combined mean of the pair is less than 15 times the detection limit (Kaufman and Stoker 2009). Removing the low values ensures that there is no undue influence on the RPD plots due to the higher variance of grades likely near to the detection limit, where precision becomes poorer (Long et al. 1997).

In 2016 and 2017 drill programs, Rockhaven collected 289 coarse reject duplicates and 257 quarter-core (field) duplicates to test for repeatability. These samples were not based solely on gold assay results. As a result, only 110 coarse reject duplicates and 45 field duplicates were greater than the 15 times detection limit. RPD plots are presented in Figure 11.3 for the coarse reject duplicate dataset.

AMC makes the following observations based on the coarse reject duplicate results:

- 75% of the coarse reject duplicate pairs were less than 20% RPD, which, though less than desirable, is an acceptable result.
- No significant bias is observed between the original and duplicate assays.

Figure 11.3 Relative paired difference plot for coarse reject duplicates



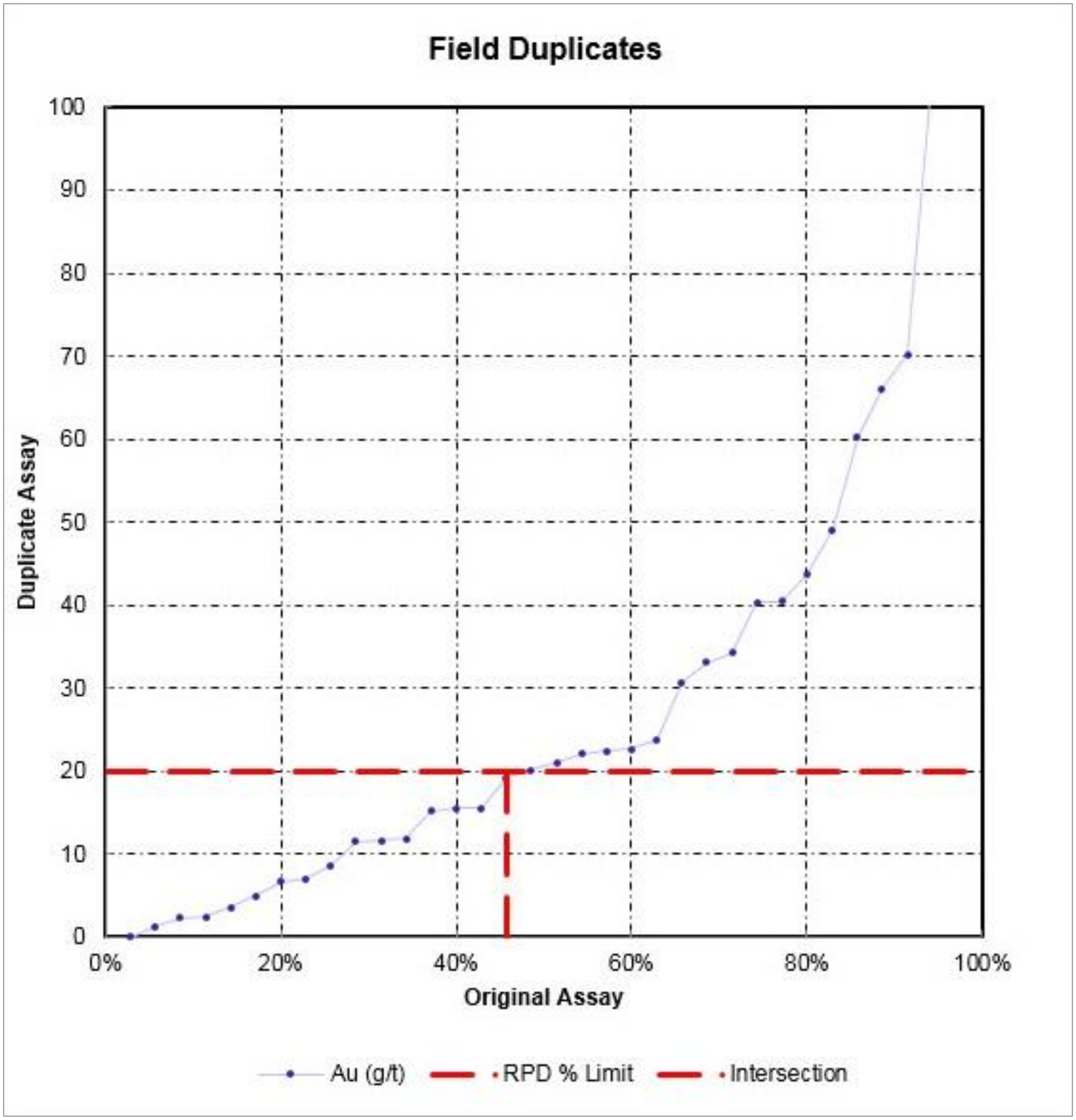
Source: AMC Mining Consultants (Canada) Ltd.

RPD plots are presented in Figure 11.4 for the field duplicate datasets.

AMC makes the following observations based on the field duplicate results:

- 46% of the field duplicate pairs were less than 20% RPD, which, is understandable given the inherent variability in field duplicate sampling in gold systems.
- No significant bias is observed between the original and duplicate assays.
- Given the limited number of samples (< 100) that were > 15 x detection limit for the field duplicates, no conclusions can be drawn from the data.

Figure 11.4 Relative paired difference plot for field duplicates



Source: AMC Mining Consultants (Canada) Ltd.

AMC recommends that additional duplicate samples be taken from mineralized material to provide sufficient data for future RPD analyses.

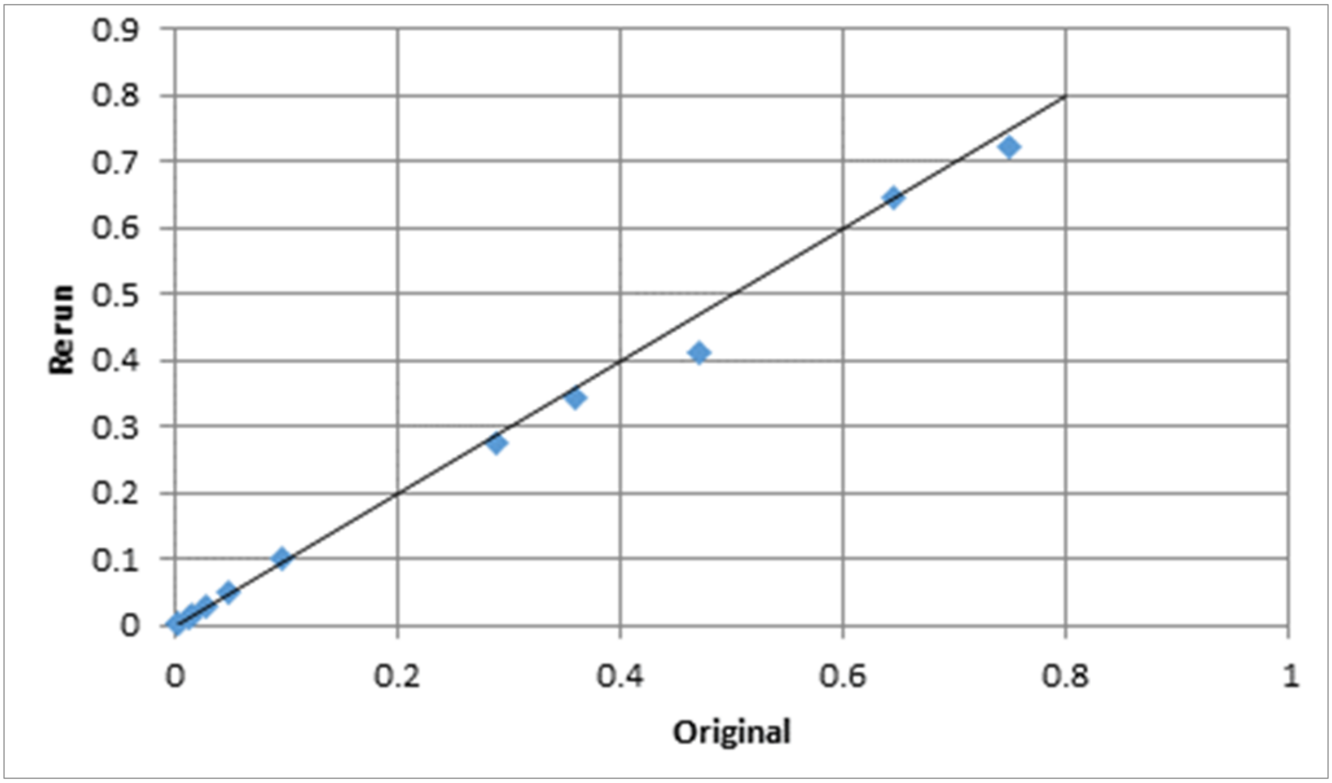
11.5.4 Results of external check assays

Umpire laboratory duplicates are pulp samples sent to a separate laboratory to assess the accuracy of the primary laboratory (assuming the accuracy of the umpire laboratory). Umpire duplicates measure analytical variance and pulp sub-sampling variance. Umpire duplicates should comprise around 5% of assays.

A total of 140 core samples analyzed in 2015 by ALS Minerals were randomly selected for check analysis. These samples represent approximately 3% of the samples analyzed in 2015. Pulp rejects from these samples were submitted to SGS in Burnaby, BC to be analyzed for gold by fire assay followed by atomic absorption (GE FAA313) and 33 elements by four acid digestion followed by inductively coupled plasma-atomic emission spectrometry (GE ICP40B). SGS is an ISO certified laboratory and is independent of Rockhaven.

Results from the SGS assays are consistent with the assays completed by ALS Minerals (Figure 11.5).

Figure 11.5 Scatterplot showing ALS Minerals versus SGS analysis in 2015



Source: AMC Mining Consultants (Canada) Ltd.

Rockhaven did not complete any umpire laboratory duplicate sample testing of drillhole samples completed in 2016 and 2017.

AMC recommends that further umpire duplicate sampling be completed.

11.6 Data validation

M.R. Dumala, P.Eng., of Archer Cathro, has supervised the exploration programs at the Property from 2013 through 2019. He has helped establish the data collection and quality control procedures used since 2010. At the beginning of each field season, he has provided on-site training to field personnel. During subsequent visits, he reviewed data collection procedures and inspected selected drill intervals.

Over the duration of each field program, sample information, drillhole surveys, drill logs and other collected data were forwarded to Mr Dumala on a daily basis. The data were reviewed, and corrections immediately made if necessary. Any changes to the collection procedure were made or additional training was provided as needed.

Drillhole locations, downhole surveys and mineral intersections were plotted as they became available. These were inspected and compared to the existing geological model. Any discrepancies identified were investigated further and addressed as needed. In addition to the QA/QC procedures outlined in Section 11.5 above, assay data stored in the drill database were routinely spot checked against the original ALS assay certificates.

Prior to commencing the updated Mineral Resource estimate in Fall 2015, geotechnical, geological, sample, mineralization, and density logs were reviewed by two Archer Cathro employees operating independently of each other. Intervals were checked for missing data, overlaps and data entry errors. Spot checks were performed against original paper logs where available. Any erroneous data were reported, and steps were taken to either correct the errors or remove the affected data from further use. Data captured after 2015 was validated in real-time by Archer Cathro’s inhouse database software as described in Section 11.6.1. Data is also reviewed in 3D and cross checked against the most recent geological interpretation.

The following sub-sections provide details of Archer Cathro’s data validation procedures for data collection primarily focusing on data associated with the diamond drilling.

11.6.1 Database verification

Prior to 2014, geological and geotechnical logging was initially recorded as a hardcopy and then transcribed into MS Excel®. In 2014, logging was recorded as hardcopy and then entered into a MS-SQL Server® database (the Database). Starting in 2015 drill logs were entered directly into the Database. All of the pre-2014 data has been verified and transferred to the Database.

Algorithms within the database automatically check all data as it is entered to ensure accuracy and consistency. These checks include interval checks that alert the user if overlapping or missing intervals are detected. Alerts are also generated if a downhole depth has been entered that is greater than the final hole depth. Drop-down menus and internal libraries ensure consistency between users by requiring the use of pre-approved lithological units, minerals and other logging codes.

11.6.2 Collar location verification

Prior to 2017, all drillhole collars were surveyed by Archer Cathro employees using a Trimble SPS882 and SPS852 base and rover RTK GPS system. In 2017 and 2019, drill collars were surveyed using a Leica GS15 base and rover RTK GPS.

All drillhole collars were re-surveyed in 2017 using the Leica system and, where necessary, survey data collected in previous years was corrected. It was found that the differences between surveys increased as the distance from the base station increased. At the furthest point, approximately 1.75 km from the base station, the horizontal difference averaged approximately 6 m. Nearer to the base station, the difference averaged approximately 1 – 2 m. Differences between the Leica surveys and the earlier Trimble surveys are explained by inclusion of an updated scaling factor and corrected geoid model.

Elevation data obtained during the RTK GPS survey were compared to elevation data calculated from low level orthorectified photographs. Any discrepancies identified were investigated and corrected, if possible. If no resolution to a discrepancy was immediately apparent, an additional RTK GPS survey was conducted.

11.6.3 Down-hole orientation verification

Prior to 2011, no down-hole azimuth measurements were made, and dip deviations were measured using an acid tube at the bottom of each hole. This practice did not follow industry standards, but due to the limited number of holes (11), shallow depths (up to 273.12 m) and good ground conditions, this is not considered to be a significant issue.

Original survey data collected between 2011 and 2017 were obtained from the down-hole survey tools in CSV format and imported directly into the Database. Data were visually inspected, and erroneous data were not used during the interpretation process.

11.6.4 Assay verification

Digital assay certificates, for all of the drilling, were obtained from ALS Minerals in CSV format and imported directly into the Database.

Internal algorithms built into the Database ensure that the correct assay data were matched with the correct sampling data. Errors detected by the Database were inspected and corrected. Spot checking of data within the Database against hard copy certificates issued by ALS Minerals was also implemented and did not reveal any issues.

11.7 Conclusions

In the opinion of the QP, the sampling, sample preparation, security, and analytical procedures adopted by Rockhaven for its exploration programs meet accepted Industry standards. The QA/QC results confirm that the assay results may be relied upon for Mineral Resource estimation purposes.

12 Data verification

On 18 and 19 August 2015, full-time AMC employee Dr Adrienne Ann Ross, P. Geo., visited the Property to undertake the following data verification steps:

Discussion with site personnel regarding:

- Sample collection
- Sample preparation
- Sample storage
- QA/QC
- Data validation procedures
- Survey procedures
- Geological interpretation
- Exploration strategy

Inspections of trenches, core shed, and drill core intersections were conducted. Table 12.1 lists the inspected drillholes.

Table 12.1 Inspected drillholes

Zone	Drillhole No.
Central BRX	KL-14-196
Central Klaza	KL-15-255
Central Klaza	KL-15-270
Central Klaza	KL-15-257
Central Klaza	KL-11-050
Western BRX	KL-14-192
Western BRX	KL-15-291
Western Klaza	KL-14-178
Western Klaza	KL-14-182
Western Klaza	KL-14-179

Source: AMC Mining Consultants (Canada) Ltd.

Dr Ross reviewed the processes used in the data collection and handling.

Under the supervision of Dr Ross, Mary Alejo, P.Eng. of AMC undertook random cross-checks of 7% of the 2015 assay results in the database with original assay results on the assay certificates returned from ALS Laboratories and found no errors. Further details of this verification are described in the NI 43-101 technical report titled "Technical Report and PEA for the Klaza Au-Ag Deposit, Yukon Canada for Rockhaven Resources Ltd." dated 26 February 2016.

Under the supervision of Dr Ross, Simeon Robinson, P.Geo. of AMC undertook random cross-checks of assay results in the Rockhaven drillhole database provided by Archer Cathro with 2016 and 2017 assay certificates received directly from ALS.

Of the 9,798 samples sent to ALS for analysis in 2016 and 2017, 494 samples representing 5% of the assay data were reviewed and assay results for Au, Ag, Pb, and Zn in the database were compared to the original assay certificates.

No errors were detected between the drillhole database and assay certificates. This result was anticipated as Archer Cathro has in-house software that automatically imports data from the lab into the database.

AMC makes the following observations based on the data verification that was conducted in 2015 and from discussions on the work since:

- Site geologists are appropriately trained.
- Procedures for data collection and storage are well-established and adhered to.
- QA/QC procedures are adequate and give confidence in the assay results.
- Cross-checking a sample set of the database with the original assay results uncovered no errors.

The QP considers the database fit-for-purpose and in the QP's opinion, the geological data provided by Archer Cathro for the purposes of Mineral Resource estimation were collected in line with industry best practice as defined in the CIM Exploration Best Practice Guidelines and the CIM Mineral Resource, Mineral Reserve Best Practice Guidelines. As such, the data are suitable for use in the estimation of Mineral Resources.

13 Mineral processing and metallurgical testing

Several rounds of tests have been conducted on samples from the Klaza project since 2014. Early work was done by SGS, however since 2015 most of the testing has been done by Blue Coast Research Ltd. Much of the key work was done in 2015 and early 2016, when the baseline flowsheet was developed. Additional work since 2016 investigated (a) the potential for gravity pre-concentration ahead of milling, (b) precious metal recoveries from the Eastern BRX zone, and (c) the potential to make a saleable gold-rich arsenopyrite concentrate. As this work does not impact the current PEA, it has only been described in limited detail at the end of this section.

The metallurgical forecast and mill design are largely based on metallurgical testwork conducted at Blue Coast Research Limited in the years prior to 2016.

The mean head assays of the composites used for flowsheet development are shown below. These composites included the "Project-wide Composite", which was created as a blend of all zones based on the grades and tonnages of each zone as described in the 19 June 2015 NI 43-101 Technical Report issued by Rockhaven.

Table 13.1 Composite head assays

Composite	Pb (%)	Zn (%)	Au (g/t)	Ag (g/t)	As (%)
Western BRX	0.98	1.16	6.27	97	1.11
Western Klaza	0.67	0.94	5.56	262	0.88
Central Klaza	0.80	1.52	4.84	70	1.00
Project-wide	0.79	1.28	5.35	111	1.00

13.1 Mineralogical analysis

Subsamples of the Western and Central Klaza Variability Composites, as well as the Western BRX were submitted to Process Mineralogical Consultants (PMC) in Maple Ridge, British Columbia. Each sample was ground to passing of 80% (P_{80}) of approximately 100 µm and sized to produce a +53 µm and -53 µm fraction. The samples were analyzed via quantitative scanning electron microscopy (TIMA) to determine mineral abundance, liberation and grain size.

The modal abundance of the three samples is summarized below.

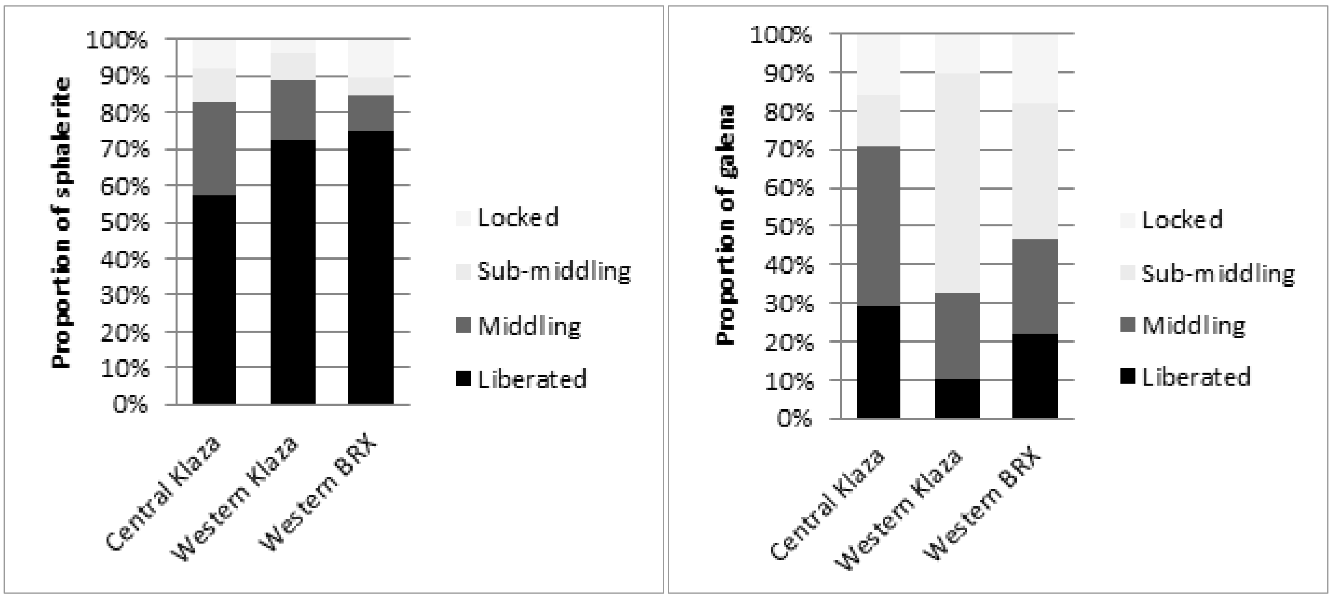
Table 13.2 Summary of modal abundance for Central and Western Klaza, Western BRX

	Central Klaza	Western Klaza	Western BRX
Chalcopyrite	0	0.02	0.09
Sphalerite	2.62	3.90	2.33
Pyrite	3.64	4.59	5.80
Galena	0.52	0.82	0.89
Arsenopyrite	0.52	1.08	1.68
Other sulphides	0	0.08	0.15
Calcite	4.5	2.31	3.42
Quartz	51.5	48.6	41.2
Feldspars	13.01	13.5	10.9
Muscovite	13.4	17.5	23.1
Pyroxene-Amphibole	2.11	1.04	0.83
Other	8.18	6.56	9.61

All three composites are dominated by quartz / feldspar / muscovite, which represent 75% – 85% of the mineral mass of the samples. Zinc is essentially present as sphalerite (assaying 57% – 60% Zn, and 4.8% – 7.4% Fe), lead as galena and arsenic as arsenopyrite. Carbonates are not abundant, while there is no evidence of the presence of preg-robbing carbonaceous matter.

The liberation characteristics of galena and sphalerite, at a grind of P_{80} of 100 microns, are summarized in the graphs below.

Figure 13.1 Sphalerite and galena liberation characteristics for Central and Western Klaza, Western BRX

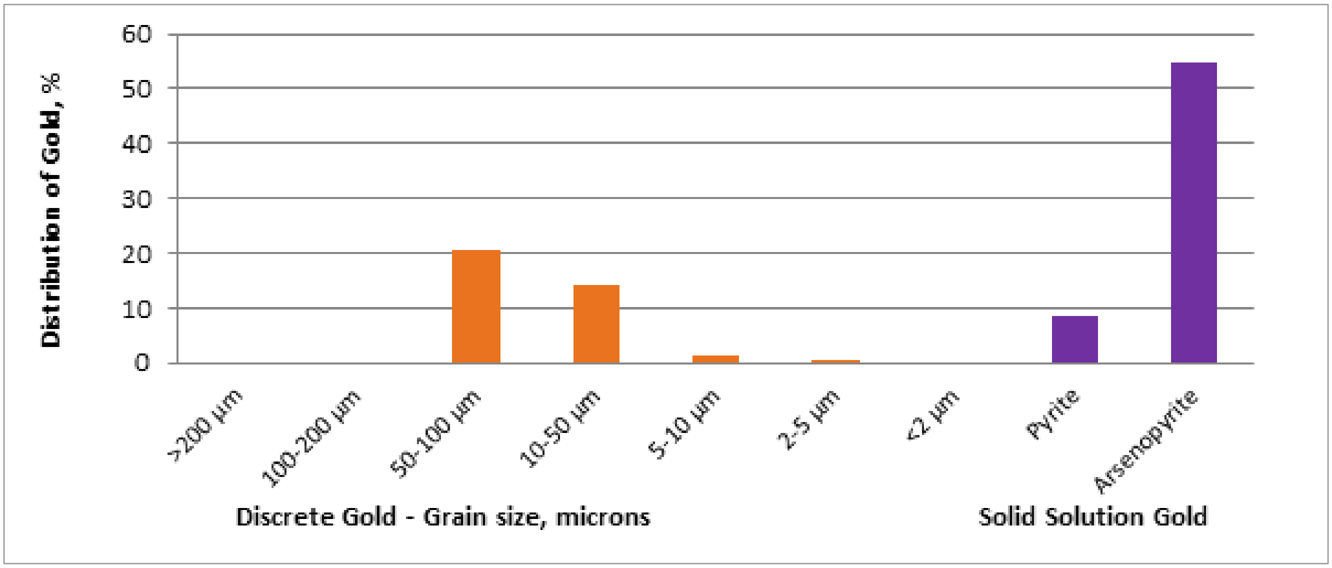


Note: Liberated: >80% free, Middling: 50 – 80% free, Sub-middling: 20 – 50% free, Locked: <20% free.
Source: BCM.

Galena is more finely disseminated than sphalerite, and at a grind size of 100 microns is somewhat under-liberated for good rougher flotation recoveries. Sphalerite is more completely liberated and would be expected to float well at this grind. Accordingly, amongst the base metals, galena liberation drives the primary grind size which can be expected to be substantially below 100 microns.

Gold occurs both as discrete grains and in solid solution in both arsenopyrite and pyrite. Roughly 55% of the gold in the project-wide composite is refractory (solid-solution) gold contained in arsenopyrite, and a further approximately 8% is in pyrite. The remainder, roughly 37%, is discrete gold, slightly more than half of which is above 50 microns in size.

Figure 13.2 Gold occurrence in the Project-wide composite



Note: orange = discrete; purple = solid solution.
Source: BCM.

There is a tendency for a higher proportion of the gold to occur in refractory form in the Klaza zones, and in discrete form in the BRX zones (especially Eastern BRX).

13.2 Comminution testwork

Grindability testing on material from the Klaza and BRX zones, yielded the following indices. The material has moderate resistance to grinding either by semi-autogenous grinding (SAG) or ball milling:

- Bond Ball Mill Work Index (BWi): 15.3 kWh/tonne (2 samples)
- Bond Rod Mill Work Index (RWi): 15.3 kWh/tonne (1 sample)
- SAG Mill Comminution (SMC) Test: A: 68.1, b: 0.71, Axb: 48.4, ta: 0.45 (1 sample)

13.3 Flotation

The deposit contains potentially economic quantities of gold, silver, lead, and zinc, with, at current prices, more than three quarters of the in-situ value contained in gold. Early work at SGS had included testwork aimed at recovering only the precious metals but gravity concentration and leach recoveries were generally poor, while bulk flotation led to a product that would be hard to market.

Accordingly, the focus shifted to creating a flowsheet comprising sequential lead, zinc, and bulk sulphide flotation. Developed at Blue Coast Research, this flowsheet creates saleable lead and zinc concentrates, and a gold-rich sulphide concentrate that may be processed economically on site or has the potential to be sold.

The developed flowsheet employs conventional processes that have all been used on mines in Northern Canada. The metallurgical program itself was conventional in nature for such projects, using testwork to identify the primary grind size, the selection and dosage of zinc, pyrite, and arsenopyrite depressants in lead flotation, then conventional zinc activation using copper sulphate, and zinc flotation while still keeping the other sulphides depressed, prior to flotation of the remaining sulphides. Collector doses were established, while the pH regime for each stage of flotation was developed. Both lead and zinc flotation responded well to standard treatment approaches, consistent with what had been expected from the mineralogy.

Much of the latter part of the program focused on developing a process to produce a refractory gold-bearing sulphide concentrate that would maximize the financial return from either sale to a third party or on-site processing.

In parallel with the early Pb-Zn flotation work, a greater understanding of the deportment of the gold was developed through mineralogical studies conducted at Surface Science Western in London, Ontario, who recognized that almost all of the refractory gold was tied up in arsenopyrite. Consequently, a process was developed to float arsenopyrite selectively from pyrite. High pH levels and starvation collector doses combine to ensure pyrite flotation is controlled, and essentially selectivity between arsenopyrite and pyrite can be dialed-in by modifying the pH and collector dose. This process is driven by downstream economics and environmental factors, whereby enough pyrite is allowed to float to provide iron for stabilization of arsenic in the autoclave, while not too much is floated as this represents an added net cost in autoclaving (pyrite-hosted gold recovered after autoclaving is not enough to cover the cost of the autoclaving process).

Flotation was aimed at achieving a mass ratio of iron to arsenic in the concentrate of 4:1, a mass ratio widely seen as suitable for the precipitation of arsenic as environmentally stable scorodite. The process was tested in locked cycle mode, yielding the following metallurgy.

Table 13.3 Summarized metallurgy from locked cycle testing of the Project-wide composite

	Grades				Recoveries (%)			
	Pb (%)	**Zn (%)**	**Ag (g/t)**	**Au (g/t)**	**Pb**	**Zn**	**Ag**	**Au**
Pb conc	60	3.1	5957	130	85	3	62	26
Zn conc	2.0	48	1318	13.5	6	85	27	5
Au / sulphide conc	0.3	1.0	73	30.7	5	10	8	65

The Au / sulphide concentrate from the locked cycle test was subjected to autoclaving. High sulphide oxidation was achieved in one hour and cyanidation of the residue recovered 98% of the gold, meaning that combined gold recovery to flotation concentrates and doré was 96%.

While the entire process was only tested on a single project-wide composite, batch flotation testing on composites from Western Klaza, Central Klaza, and Western BRX all indicated that the process would be effective for these resources.

Finally, lead concentrate was subjected to intensive cyanidation. Despite high lead levels cyanide consumption was modest and 85% of the gold was extracted. Including this process, shifts 22% of the gold from lead concentrate to doré, leaving 4% of the Au remaining in the lead concentrate, 5% of the Au in the zinc concentrate and 87% of the gold produced as doré. Only 15% of the silver in the lead concentrate leached.

13.4 Process development for Eastern BRX

The Eastern BRX zone is not included as a resource in this PEA, however a brief description of metallurgical work on Eastern BRX is included for the sake of completeness.

13.4.1 Flotation testing

The sample provided to the laboratory for metallurgical testing assayed 2.06 g/t Au, 26 g/t Ag, 0.11% Pb, 0.12% Cu, and 0.26% Zn. Limited flotation testing indicated that while sequential Cu / Pb and Zn concentrates could be floated, concentrate base metal grades were poor owing to the very low head grades.

13.4.2 Cyanide leaching

The Eastern BRX composite was direct leached using 1 g/L sodium cyanide for 48 hours, yielding 81% gold and 34% silver extraction. Cyanide consumption was 2.3 kg/ton.

The composite was also floated to produce a high-grade gold concentrate, gold cleaner tails sample and a bulk sulphide composite. This approach was tested as it offered a way to process the Eastern BRX material using, as much as possible, the same equipment as was scoped for the rest of the project. The gold concentrate was intensively leached, and the bulk sulphide concentrate was subjected to a fine regrind and then leached. The rougher flotation tail was not leached. Floating these concentrates and subsequent leaching of the concentrates yielded 80% gold and 36% silver extractions, based on the mill feed. Cyanide consumption equated to 1.4 kg/ton of primary mill feed across the entire flowsheet.

13.5 Pre-concentration testing

Gravity pre-concentration of a crushed product is commonly used to upgrade run-of-mine (ROM) material after crushing and ahead of grinding and downstream processing. It is a useful way to reduce the tonnage capacity of the mill, and / or drop the effective mining cut-off grade as the marginal material only has to carry the cost of crushing, sizing, and separation (as long as the concentrated material can be economically processed downstream).

In 2017 a program of tests was conducted to explore the response to pre-concentration by samples from each of the deposits. In each case suites of variability samples with different head grades were subjected to pre-concentration testing using heavy liquid separation. Each sample was crushed to 9.5 mm and minus 1.4 mm fines were removed ahead of the test, with the fines being added to the pre-concentrate. Metal recoveries to pre-concentrates representing between 32% and 53% of the ROM mass are shown below.

Central Klaza, Eastern BRX, and Western BRX all respond quite well to pre-concentration, with gold recoveries ranging from 95% to 98%. Western Klaza, however, responded poorly with only 74% of the gold being recovered to pre-concentrate. Lead and zinc recoveries are also shown below. They tend to be lower than gold.

Table 13.4 Mean recoveries from pre-concentration of samples

	Mass pull	Au recovery	Pb recovery	Zn recovery
Central Klaza, > 1 g/t Au	45%	96.3%	93.4%	94.9%
Western Klaza, >1 g/t Au	32%	74.3%	85.8%	70.6%
Eastern BRX, > 0.5 g/t Au	46%	94.8%	91.0%	86.3%
Western BRX, > 1 g/t Au	53%	98.0%	93.9%	92.9%

Source: Blue Coast Metallurgy Ltd.

Recoveries tended to drop for lower grade samples.

Limited flotation work has been conducted on pre-concentrated material where the indicated recovery of gold from the preconcentrate was 85% – 90%.

13.6 Flotation to create a potentially saleable refractory gold concentrate

Arsenopyrite at Klaza typically hosts about 140 g/t – 170 g/t gold so a high-grade arsenopyrite concentrate would contain considerable gold values. Testing on Western BRX material yielded concentrates assaying over 70% of the arsenopyrite. Such concentrates could feasibly contain 100 g/t gold. The high arsenic contents hinder the marketability of such concentrates, but this remains a possible option to on-site hydrometallurgy.

13.7 Adopted metallurgical recoveries for the PEA

The mine plan adopted for the 2020 Klaza PEA includes feed that is a blend of material from the Klaza and BRX zones. While not a perfect homologue to the Project-wide composite it is quite similar, so the basic metallurgy as indicated from the Project-wide Composite testing is deemed sufficient representative to the currently envisaged blend of feed materials.

One exception is in the lead and zinc feed grades, which are significantly lower than the Project-wide composite. Accordingly, recoveries of these two metals have been scaled down to 80% to reflect the lower grades. The gold grade is higher; however, it has been assumed that recoveries would be the same.

Table 13.5 Metallurgical grades and recoveries for 2020 Klaza PEA study

	Grades				Recoveries (%)			
	Pb (%)	Zn (%)	Ag (g/t)	Au (g/t)	Pb	Zn	Ag	Au
Feed	0.65	0.73	95	4.4				
Lead concentrate	60	3.1	5,957	130	80	3	53	4
Zinc concentrate	2	48	1,318	13.5	6	80	27	5
Gold from hydrometallurgy	N/A	N/A	N/A	N/A	N/A	N/A	1	64
Gold from lead conc leach	N/A	N/A	N/A	N/A	N/A	N/A	9	22
Total to payable concentrate					**80**	**80**	**90**	**95**

14 Mineral Resource estimates

14.1 Introduction

The Mineral Resource for the Klaza deposit has been estimated by Dr A. Ross., P.Geo. Principal Geologist of AMC, who takes responsibility for the portions of the model estimated by ordinary kriging (OK) and Mr Ingvar Kirchner, Principal Geologist of AMC Consultants Pty Ltd (Perth, Australia) who takes responsibility for the portions of the model estimated by multiple indicator kriging (MIK), local multiple indicator kriging (LMIK), and restricted ordinary kriging (ROK).

AMC is not aware of any known environmental, permitting, legal, title, taxation, socioeconomic, marketing, political, or other similar factors that could materially affect the stated Mineral Resource estimate. The Yukon is a mining friendly territory within a stable jurisdiction.

This estimate is dated 5 June 2018 and supersedes the previous estimate outlined in the "Technical Report and PEA of the Klaza Au-Ag Deposit, Yukon, Canada" with an effective date of 26 February 2016.

The data used in the 5 June 2018 estimate (AMC 2018 estimate) include results of all drilling carried out on the Property to 31 December 2017. The estimation work was carried out in Datamine™ software. Grade interpolation was completed using a combination of OK, MIK, LMIK, and ROK. The interpolation methodology utilized for each area is detailed in Table 14.1.

Table 14.1 Resource by area and estimation method

Zone	Area	Estimation method
BRX	Western BRX	OK
	Central BRX	OK
Klaza	Western Klaza	OK
	Central Klaza	OK, LMIK, MIK, and ROK

Source: AMC Mining Consultants (Canada) Ltd.

The results of the AMC 2018 estimate are summarized in Table 14.2 and expanded in Table 14.3.

Table 14.2 Summary of Mineral Resources as of 5 June 2018

Resource classification	Tonnes (kt)	Grade					Contained metal				
		Au (g/t)	Ag (g/t)	Pb (%)	Zn (%)	AuEq (g/t)	Au (koz)	Ag (koz)	Pb (klb)	Zn (klb)	AuEq (koz)
Indicated	4,457	4.8	98	0.7	0.9	6.3	686	14,071	73,268	92,107	907
Inferred	5,714	2.8	76	0.6	0.7	3.9	507	13,901	77,544	89,176	725

Notes:
- CIM Definition) Standards (2014) were used for the Mineral Resource.
- Estimate includes drill results to 31 December 2017.
- Near surface Mineral Resources are constrained by an optimized pit shell at metal prices of $1,400/oz, $19/oz Ag, $1.10/lb Pb, and $1.25/lb Zn.
- Cut-off grades applied to the pit constrained and underground resources are 1.0 g/t AuEq and 2.3 g/t AuEq respectively.
- Gold equivalent values were calculated using parameters outlined in Table 14.4.
- Numbers may not add due to rounding.
- All metal prices are quoted in US$ at an exchange rate of US$0.80 to C$1.00.
- Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

Source: AMC Mining Consultants (Canada) Ltd.

Table 14.3 Mineral Resource estimate by mining method as of 5 June 2018

Resource classification	Tonnes (kt)	Grade					Contained metal				
		Au (g/t)	Ag (g/t)	Pb (%)	Zn (%)	AuEq (g/t)	Au (koz)	Ag (koz)	Pb (klbs)	Zn (klbs)	AuEq (koz)
Indicated											
Pit constrained	2,447	5.3	90	0.7	1.0	6.7	414	7,096	39,143	52,935	529
Underground	2,010	4.2	108	0.8	0.9	5.8	272	6,974	34,125	39,172	378
Indicated total	**4,457**	**4.8**	**98**	**0.7**	**0.9**	**6.3**	**686**	**14,071**	**73,268**	**92,107**	**907**
Inferred											
Pit constrained	1,754	2.6	43	0.4	0.5	3.3	147	2,429	14,897	18,599	187
Underground	3,960	2.8	90	0.7	0.8	4.2	359	11,472	62,647	70,578	538
Inferred total	**5,714**	**2.8**	**76**	**0.6**	**0.7**	**3.9**	**507**	**13,901**	**77,544**	**89,176**	**725**

Notes:
- CIM Definition Standards (2014) were used for the Mineral Resource.
- Estimate includes drill results to 31 December 2017.
- Near surface Mineral Resources are constrained by an optimized pit shell at metal prices of $1,400/oz Au, $19/oz Ag, $1.10/lb Pb, and $1.25/lb Zn.
- Cut-off grades applied to the pit constrained and underground resources are 1.0 g/t AuEq and 2.3 g/t AuEq respectively.
- Gold equivalent values were calculated using parameters outlined in Table 14.4.
- Numbers may not add due to rounding.
- All metal prices are quoted in US$ at an exchange rate of US$0.80 to C$1.00.
- Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

Source: AMC Mining Consultants (Canada) Ltd.

Table 14.4 Gold equivalent (AuEq) formula

Zone	Au	Recovery %				AuEq (g/t)
		Au	Ag	Pb	Zn	
Central Klaza	>1.0 g/t	92	86	79	81	AUEQ = 1 x AU + AG/94.40 + PB/3.38 + ZN/3.21
	<1.0 g/t	78	76	73	71	AUEQ = 1 x AU + AG/90.56 + PB/3.10 + ZN/3.11
Western Klaza	All	96	91	85	85	AUEQ = 1 x AU + AG/93.09 + PB/3.28 + ZN/3.20
Western BRX & Central BRX	>1.0 g/t	94	87	80	79	AUEQ = 1 x AU + AG/95.34 + PB/3.41 + ZN/3.37
	<1.0 g/t	64	57	49	42	AUEQ = 1 x AU + AG/99.08 + PB/3.79 + ZN/4.31

Note: AuEq values assume metal prices of US$1,400/oz Au, US$19/oz Ag, US$1.10/lb Pb, and US$1.25/lb Zn and payable values of 97% Au, 81% Ag, 62% Pb, and 52% Zn.
Source: AMC Mining Consultants (Canada) Ltd.

14.1.1 Drillhole database

The data used in the AMC 2018 estimate consisted of surface diamond drillhole data held in a MS SQL Server® database, which was provided to AMC as Microsoft Excel® files. The data type and number of holes in the resource area are shown in Table 14.5.

Table 14.5 Drillhole data used in the Mineral Resource estimate

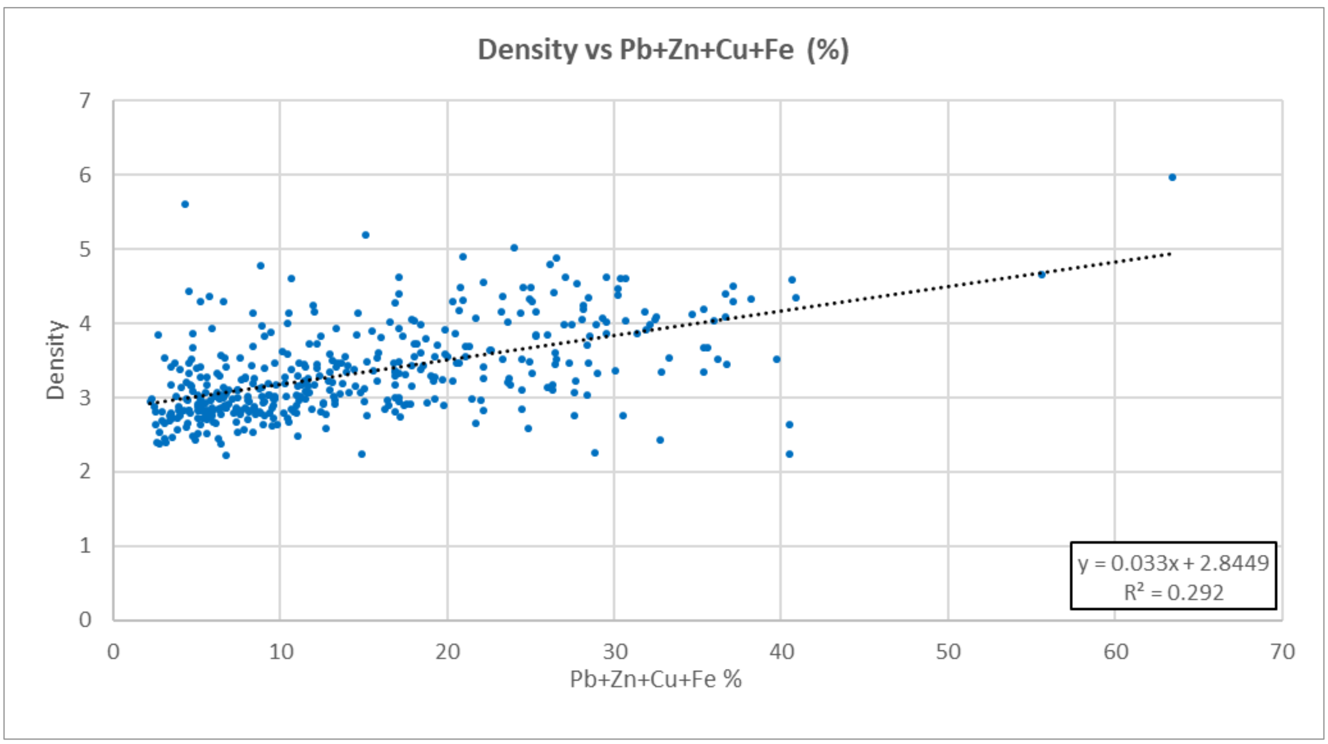
Year	No. of drillholes	No. of assays	Metres drilled (m)
2010	6	447	961
2011	48	6,193	12,929
2012	42	6,813	13,462
2014	88	5,296	16,071
2015	40	3,416	10,955
2016	24	1,540	3,830
2017	74	3,561	10,741
Total	**322**	**27,266**	**68,948**

Note: All drillholes are surface diamond drillholes.
Source: AMC Mining Consultants (Canada) Ltd.

14.1.2 Bulk density

The collection of density measurements is described in Section 11.2.3. Mineralized veins usually contain significant concentrations of sulphide minerals, including pyrite and galena which have much higher specific gravities than normal rock forming minerals. Examination of correlation coefficients demonstrates a strong relationship between measured density and a sum of the base metal grades. A scatterplots of density versus combined lead, zinc, copper, and iron assay values was produced as shown in Figure 14.1.

Figure 14.1 Scatter plot of density versus percentage lead-zinc-copper-iron



Source: AMC Mining Consultants (Canada) Ltd.

For this Mineral Resource estimate the mineralized portions of the block model were assigned a density based on the combined estimated grades of lead, zinc, copper, and iron using the following equation:

$$\textit{Density } (t/m^3) = 0.033 \times (Pb\%+Zn\%+Cu\%+Fe\%) + 2.85$$

Barren rock, for the purpose of delineating a pit constrained resource, was assigned a valued of 2.80 t/m^3 (which approximates the average measured density of the granodiorite).

Note that density measurements ignore the potential impact of pore space. As the rock is generally competent rock that contains minimal voids, the density measurements are considered to be a good approximation of bulk density.

14.2 Ordinary kriging domaining and estimation parameters

14.2.1 Domain modelling OK

For the purpose of the Mineral Resource estimate, the Klaza deposit has two main zones, the BRX and Klaza zones. These two zones are further subdivided into the Western BRX, Central BRX, Western Klaza, and Central Klaza subzones (zones).

Matthew Dumala, P.Eng., of Archer, Cathro & Associates (1981) Limited, built 32 three dimensional solids to constrain the mineralization within the BRX (Western and Central) zones and Klaza (Western and Central). Mineralized solids define the better known and higher grade continuous structures / veins. Dyke intersections were used as a marker to help constrain the orientation and position of mineralized structures. In the previous estimate, as much as possible, high-grade solids were built to capture only vein mineralization and often consisted of only one or two samples per drillhole. In this estimate, the three-dimensional solids were wider and incorporated waste.

The number of mineralization domains varied between the subzones. There were 14 domains at Western BRX and four domains at Central BRX. In the Klaza zone, Archer Cathro provided two domains for the Western Klaza and 12 domains for Central Klaza. After reviewing the data, 7 domains in the Central Klaza that represented cross cutting structures were removed from the OK estimation and included in the MIK-LMIK-ROK estimation.

Domains are summarized in Table 14.6.

Table 14.6 Domain nomenclature

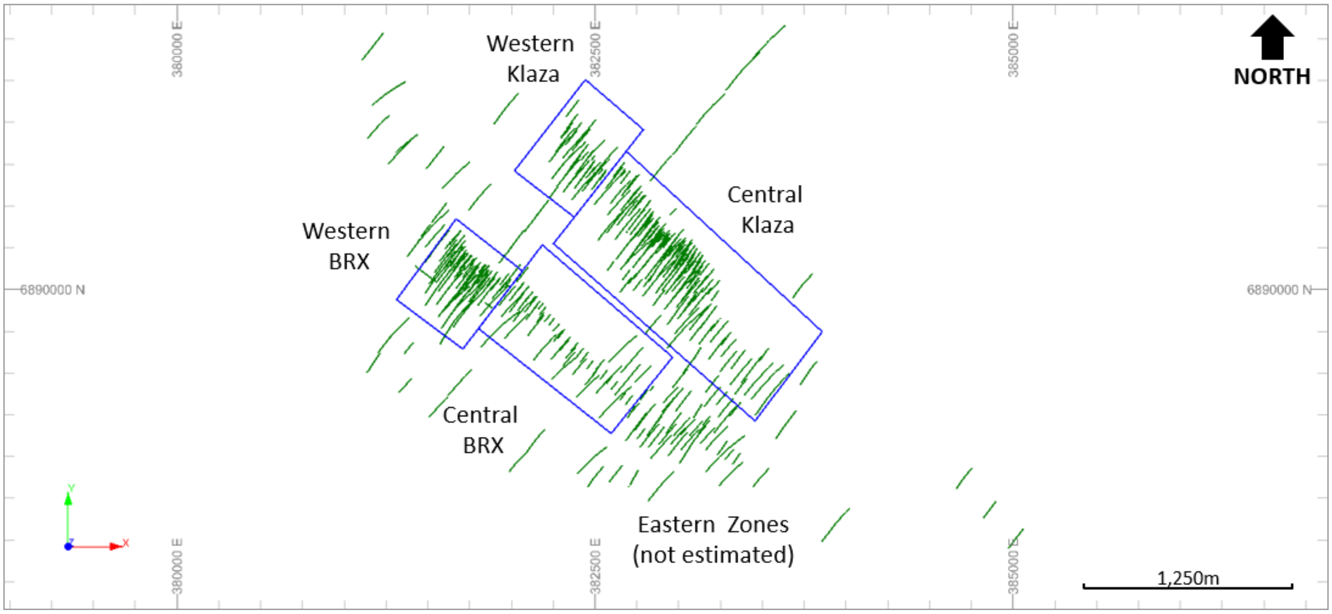
Zone	Area	No. of domains for OK
BRX	Western BRX	14
	Central BRX	4
Klaza	Western Klaza	2
	Central Klaza	5

Source: AMC Mining Consultants (Canada) Ltd.

In total, 25 domains were used for the OK Mineral Resource estimate. On completion of the domain modelling, visual checks were carried out to ensure that the constraining wireframes respected the raw data.

Figure 14.2 shows the location of the zones in which the domains were modelled.

Figure 14.2 Zone locations with drillholes

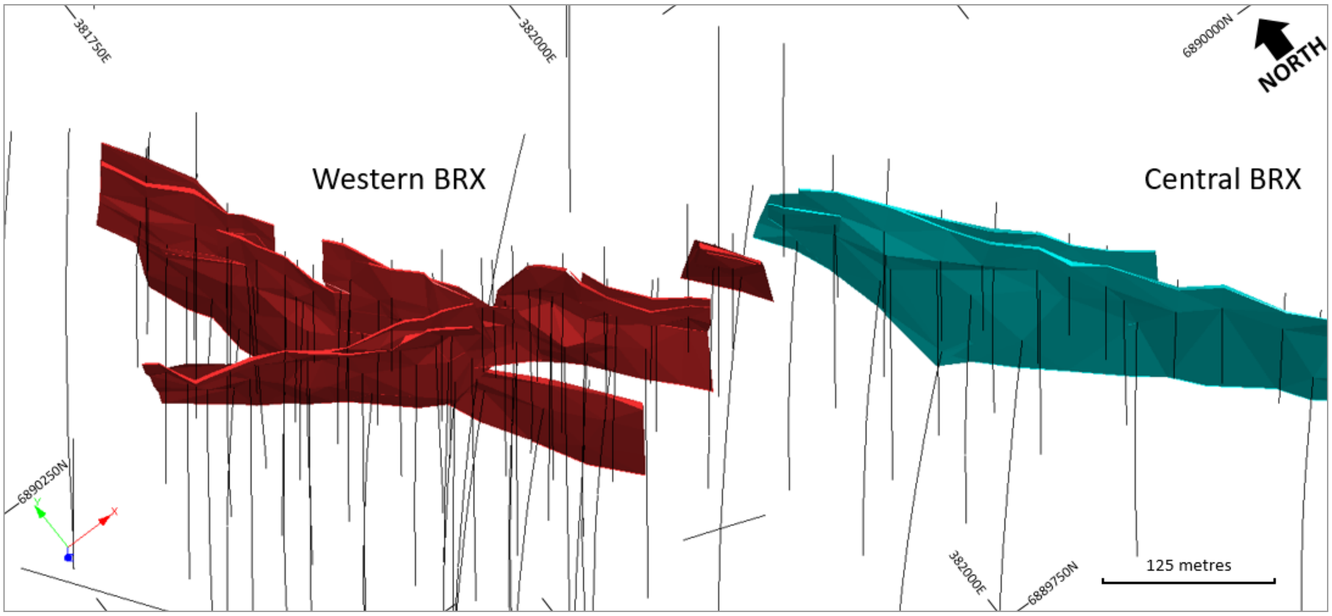


Notes:
- Cross cutting structures in the Central Klaza area were estimated using MIK-LMIK-ROK estimation.
- All other areas were estimated by OK.

Source: AMC Mining Consultants (Canada) Ltd.

Figure 14.3 shows an isometric view of the mineralized domains in Western BRX. It highlights the local complexity of the Klaza deposit.

Figure 14.3 Isometric view of the Western BRX zone looking north-east



Source: AMC Mining Consultants (Canada) Ltd.

14.2.2 Statistics and compositing OK

AMC selected a compositing interval of 1 m, which is the most common sample length in the database. The second most common sample length is 3 m. As a result of this the median sample length is 1.5 m. In some cases, compositing increased the number of samples in a domain. The 1 m length also gave the appropriate selectivity for the narrow-vein style of this mineralization. Assays within the wireframe domains were composited as close to 1 m intervals as possible which approximates the mean sample length. Compositing started at the first mineralized wireframe boundary from the collar and was reset at each new wireframe boundary.

Composited assay data for gold, silver, lead, zinc, copper, arsenic, and iron were then examined on probability plots for each of the 25 domains, and outliers examined. This resulted in top cuts as shown in Table 14.7. The main domains required little to no capping. It is important to note that in some domains the upper limit of copper, iron, and arsenic was already defined by the upper detection limit of the analytical method.

Table 14.7 Top cutting ranges by element

Element	No. of domains with top cut	% of domains with top cut	Top cut range
Gold	9	36	1.5 - 60 g/t
Silver	8	32	15 - 700 g/t
Lead	9	36	0.2 – 7%
Zinc	8	32	0.7 – 4%
Copper	9	36	0.01 - 1%
Iron	5	20	10 - 30%
Arsenic	5	20	300 - 12,000 ppm

Notes:
- Copper and iron were estimated to provide a regression line for the density correlation.
- Arsenic was estimated for metallurgical assessment.

Source: AMC Mining Consultants (Canada) Ltd.

Table 14.8 to Table 14.11 show the statistics of raw, composite, and capped assay data from the main mineralization domains for each zone.

Table 14.8 Statistics of raw, capped, and composite assay data – Western BRX

Raw							
Element	**Au (g/t)**	**Ag (g/t)**	**Pb (%)**	**Zn (%)**	**As (ppm)**	**Cu (%)**	**Fe (%)**
Number	67	67	67	67	67	67	67
Minimum	0.00	0	0.00	0.01	18	0.00	2.72
Maximum	196.00	919	9.06	8.99	151500	2.02	33.20
Mean	**10.90**	**126**	**0.82**	**1.22**	**14679**	**0.13**	**7.85**
Std Dev	27.96	216.84	1.57	1.85	31174.41	0.30	6.11
Coeff Var	2.57	1.73	1.91	1.52	2.12	2.27	0.78
Composited							
Element	**Au (g/t)**	**Ag (g/t)**	**Pb (%)**	**Zn (%)**	**As (ppm)**	**Cu (%)**	**Fe (%)**
Number	56	56	56	56	56	56	56
Minimum	0.01	0	0.00	0.01	18	0.00	2.98
Maximum	131.45	748	5.51	5.45	133500	1.70	29.10
Mean	**10.90**	**126**	**0.82**	**1.22**	**14679**	**0.13**	**7.85**
Std Dev	21.57	184.94	1.16	1.50	26111.81	0.25	4.97
Coeff Var	1.98	1.47	1.42	1.23	1.78	1.90	0.63
Capped							
Element	**Au (g/t)**	**Ag (g/t)**	**Pb (%)**	**Zn (%)**	**As (ppm)**	**Cu (%)**	**Fe (%)**
Number	56	56	56	56	56	56	56
Minimum	0.01	0	0.00	0.01	18	0.00	2.98
Maximum	131.45	600	5.51	3.00	133500	1.70	29.10
Mean	**10.90**	**122**	**0.82**	**1.02**	**14679**	**0.13**	**7.85**
Std Dev	21.57	172.73	1.16	1.06	26111.81	0.25	4.97
Coeff Var	1.98	1.42	1.42	1.03	1.78	1.90	0.63

Notes:
- Std Dev = Standard Deviation; Coeff Var = Coefficient of Variation. All statistics are length-weighted.
- Statistics reported for Domain 5310 (wireframe WBRX01).

Source: AMC Mining Consultants (Canada) Ltd.

Table 14.9 Statistics of raw, capped, and composite assay data – Central BRX

Raw							
Element	**Au (g/t)**	**Ag (g/t)**	**Pb (%)**	**Zn (%)**	**As (ppm)**	**Cu (%)**	**Fe (%)**
Number	167	167	167	167	167	167	167
Minimum	0.00	0	0.00	0.00	16	0.00	2.15
Maximum	25.70	1600	12.20	18.10	51600	5.66	38.20
Mean	**1.57**	**83**	**0.61**	**0.79**	**4084**	**0.11**	**8.04**
Std Dev	2.76	158.75	1.40	1.72	7139.78	0.43	6.85
Coeff Var	1.76	1.92	2.29	2.18	1.75	3.83	0.85
Composited							
Element	**Au (g/t)**	**Ag (g/t)**	**Pb (%)**	**Zn (%)**	**As (ppm)**	**Cu (%)**	**Fe (%)**
Number	168	168	168	168	168	168	168
Minimum	0.00	0	0.00	0.00	17	0.00	2.44
Maximum	13.05	978	10.10	8.78	32000	4.07	31.15
Mean	**1.57**	**82**	**0.61**	**0.79**	**4075**	**0.11**	**8.02**
Std Dev	2.16	131.53	1.16	1.28	5335.02	0.36	5.89
Coeff Var	1.37	1.60	1.91	1.62	1.31	3.20	0.73
Capped							
Element	**Au (g/t)**	**Ag (g/t)**	**Pb (%)**	**Zn (%)**	**As (ppm)**	**Cu (%)**	**Fe (%)**
Number	168	168	168	168	168	168	168
Minimum	0.00	0	0.00	0.00	17	0.00	2.44
Maximum	13.05	700	10.10	8.78	32000	4.07	25.00
Mean	**1.57**	**80**	**0.61**	**0.79**	**4075**	**0.11**	**7.94**
Std Dev	2.16	120.33	1.16	1.28	5335.02	0.36	5.62
Coeff Var	1.37	1.50	1.91	1.62	1.31	3.20	0.71

Notes:
- Std Dev = Standard Deviation; Coeff Var = Coefficient of Variation. All statistics are length-weighted.
- Statistics reported for Domain 5110 (wireframe CBRX01).

Source: AMC Mining Consultants (Canada) Ltd.

Table 14.10 Statistics of raw, capped, and composite assay data – Western Klaza

Raw							
Element	**Au (g/t)**	**Ag (g/t)**	**Pb (%)**	**Zn (%)**	**As (ppm)**	**Cu (%)**	**Fe (%)**
Number	127	127	127	127	127	127	127
Minimum	0.00	0	0.00	0.00	11	0.00	1.69
Maximum	49.50	1890	6.06	9.16	64900	0.42	13.90
Mean	**3.17**	**163**	**0.48**	**0.63**	**5403**	**0.02**	**4.36**
Std Dev	5.68	340.28	1.01	1.23	10319.75	0.05	2.14
Coeff Var	1.79	2.09	2.09	1.96	1.91	2.36	0.49
Composited							
Element	**Au (g/t)**	**Ag (g/t)**	**Pb (%)**	**Zn (%)**	**As (ppm)**	**Cu (%)**	**Fe (%)**
Number	106	106	106	106	106	106	106
Minimum	0.00	0	0.00	0.00	17	0.00	2.25
Maximum	23.44	1560	5.95	4.67	55400	0.42	13.90
Mean	**3.15**	**163**	**0.48**	**0.63**	**5376**	**0.02**	**4.36**
Std Dev	4.55	283.26	0.88	0.96	8478.40	0.05	1.95
Coeff Var	1.44	1.74	1.81	1.53	1.58	2.18	0.45
Capped							
Element	**Au (g/t)**	**Ag (g/t)**	**Pb (%)**	**Zn (%)**	**As (ppm)**	**Cu (%)**	**Fe (%)**
Number	106	106	106	106	106	106	106
Minimum	0.00	0	0.00	0.00	17	0.00	2.25
Maximum	23.44	1560	5.95	4.67	55400	0.42	13.90
Mean	**3.15**	**163**	**0.48**	**0.63**	**5376**	**0.02**	**4.36**
Std Dev	4.55	283.26	0.88	0.96	8478.40	0.05	1.95
Coeff Var	1.44	1.74	1.81	1.53	1.58	2.18	0.45

Notes:
- Std Dev = Standard Deviation; Coeff Var = Coefficient of Variation. All statistics are length-weighted.
- Statistics reported for Domain 7410 (wireframe WKZA01).

Source: AMC Mining Consultants (Canada) Ltd.

Table 14.11 Statistics of raw, capped, and composite assay data – Central Klaza

Raw							
Element	**Au (g/t)**	**Ag (g/t)**	**Pb (%)**	**Zn (%)**	**As (ppm)**	**Cu (%)**	**Fe (%)**
Number	448	448	448	448	448	448	448
Minimum	0.00	0	0.00	0.00	8	0.00	1.85
Maximum	46.80	1100	15.85	29.80	69300	1.34	26.30
Mean	**2.68**	**40**	**0.43**	**0.75**	**3066**	**0.05**	**4.94**
Std Dev	5.40	95.08	1.26	2.07	6214.87	0.11	2.79
Coeff Var	2.02	2.40	2.96	2.75	2.03	2.41	0.57
Composited							
Element	**Au (g/t)**	**Ag (g/t)**	**Pb (%)**	**Zn (%)**	**As (ppm)**	**Cu (%)**	**Fe (%)**
Number	497	497	497	497	497	497	497
Minimum	0.00	0	0.00	0.00	12	0.00	1.90
Maximum	38.00	635	9.80	15.37	40887	0.71	19.30
Mean	**2.68**	**40**	**0.43**	**0.75**	**3069**	**0.05**	**4.94**
Std Dev	4.52	75.97	1.05	1.66	4905.57	0.09	2.38
Coeff Var	1.69	1.91	2.46	2.20	1.60	1.99	0.48
Capped							
Element	**Au (g/t)**	**Ag (g/t)**	**Pb (%)**	**Zn (%)**	**As (ppm)**	**Cu (%)**	**Fe (%)**
Number	497	497	497	497	497	497	497
Minimum	0.00	0	0.00	0.00	12	0.00	1.90
Maximum	38.00	635	9.80	15.37	40887	0.20	19.30
Mean	**2.68**	**40**	**0.43**	**0.75**	**3069**	**0.04**	**4.94**
Std Dev	4.52	75.97	1.05	1.66	4905.57	0.06	2.38
Coeff Var	1.69	1.91	2.46	2.20	1.60	1.50	0.48

Notes:
- Std Dev = Standard Deviation; Coeff Var = Coefficient of Variation. All statistics are length-weighted.
- Statistics reported for Domain 7110 (wireframe CKZA01).

Source: AMC Mining Consultants (Canada) Ltd.

14.2.3 Block model OK

14.2.3.1 Block model parameters

The parent block size for the model was 25 m by 12.5 m by 5 m with sub-blocking employed. Sub-blocking resulted in minimum cell dimensions of 0.25 m by 0.5 m by 0.5 m. The block model is not rotated. The block model extents are shown in Table 14.12.

Table 14.12 Block model parameters

Parameter	**X**	**Y**	**Z**
Origin (m)	381,525	6,888,840	795
Rotation angle (deg.)	0	0	0
No. of blocks	80	180	43

Source: AMC Mining Consultants (Canada) Ltd.

14.2.3.2 Variography and grade estimation

Variography was carried out on the Klaza and BRX zones to ensure sufficient sample density. The purpose of the variograms was to produce inputs for the estimate.

The OK interpolation method was used for the estimate. The dimensions of the search radius for the domains are shown in Table 14.13.

A number of passes were employed, each using different search distances and multiples as follows:

- Pass 1 = 1 x search distance
- Pass 2 = 1.5 x search distance
- Pass 3 = 4 x search distance

Parameters used for the pass 1 search are summarized in Table 14.13. Note, there was no rotation of the search ellipse along the X axis.

Table 14.13 Pass 1 search ellipse parameters

Zone	Element	X (m)	Y (m)	Z (m)	Z-Axis rotation (degrees)	Y-axis rotation (degrees)	Minimum No. of samples	Maximum No. of samples	Minimum No. of drillholes
Klaza	Au	70	50	20	40	60	4	10	2
	Ag	65	125	25	30	60	4	10	2
	Pb	150	70	70	30	60	4	10	2
	Zn	100	60	60	30	60	4	10	2
	As	70	60	40	40	60	4	10	2
	Cu	180	60	60	30	70	4	10	2
	Fe	80	100	80	30	60	4	10	2
BRX	Au	75	90	45	30	50	4	10	2
	Ag	85	75	95	40	50	4	10	2
	Pb	120	50	40	40	50	4	10	2
	Zn	90	35	40	40	50	4	10	2
	As	280	60	60	30	40	4	10	2
	Cu	90	130	90	40	50	4	10	2
	Fe	70	80	70	50	60	4	10	2

Notes:
- Z-axis rotation describes the rotation of the ellipse about the Z-axis in a counter clockwise direction.
- Y-axis rotation describes the rotation of the ellipse about the Y-axis in a clockwise direction.

Source: AMC Mining Consultants (Canada) Ltd.

Elements estimated were gold, silver, lead, zinc, copper, iron, and arsenic. Gold, silver, lead, and zinc are of economic importance and are reported in the Mineral Resource tables. Copper and iron were estimated to provide a regression line for the density correlation. Arsenic was estimated for metallurgical assessment.

14.2.4 Mineral Resource classification OK

AMC classified the Mineral Resource with consideration of the narrow-vein style of mineralization, the observed gold grade continuity and the drillhole spacing. Additional confidence in geological continuity also came from surface mapping and excavator trenches.

The criteria for the Indicated classification were based on 2/3rd of the range of the gold variogram. This resulted in a required nominal drillhole sample spacing in longitudinal projection of approximately 30 m by 30 m. Inferred classification was generally constrained by the wireframes.

14.2.5 Block model validation OK

The block models were validated in three ways. First, visual checks were carried out to ensure that the block grades respected the raw gold assay data and were constrained by the domain wireframes. Secondly, swath plots were reviewed. Lastly, the results were statistically compared to the composite gold assay data.

14.3 MIK / LMIK / ROK domaining and estimation parameters

Central Klaza splay-hosted mineralization has complex mineralization and relatively limited drilling for the style of Au-Ag-Pb-Zn mineralization. Therefore, the resource model for this area was generated using MIK incorporating COS to produce a selective mining unit (SMU) model for Au and Ag. ROK was used to estimate other ancillary elements (Pb, Zn, As, Cu, Fe). The MIK models were converted to simplified, localized versions of the MIK models (LMIK) that replicate the recoverable panel models on the smaller SMU block dimensions while incorporating the ROK estimates in the hybrid model as required.

14.3.1 Domain modelling MIK / LMIK / ROK

A new mineralization envelope was constructed by AMC to accommodate both the anticipated estimation method (MIK / LMIK) and to manage the significant issues related to the geological and grade continuity of mineralization. The new envelopes were based primarily on the grade distribution, but also tie in with the general geology. These envelopes are “soft” interpretations as they are intended to generally capture most of the mineralization in a reasonable manner, while not using strict criteria for unmineralized material incorporated into the shapes. Coherent zones of unmineralized material incorporated into the mineralization envelopes are treated in the MIK estimation process as internal dilution and should be represented as coherent unmineralized zones in the final LMIK model.

The mineralization envelope was constructed on nominal oblique southwest-northeast (037°) cross sections using a nominal 0.3 g/t Au (or AuEq) mineralization boundary on the raw grade data to define the edges of mineralized zones. Definition of quantities of internal dilution material and minimum thickness are not relevant to the estimation method. Strings were snapped to drillholes and used for subsequent wireframes of the mineralization.

At Central Klaza, a broad mineralization envelope was defined. The mineralization envelope and relation to drilling and other interpreted lodes are shown in Figure 14.4.

Mineralization is not constrained on all margins by drilling (notably at depth and along strike in some areas).

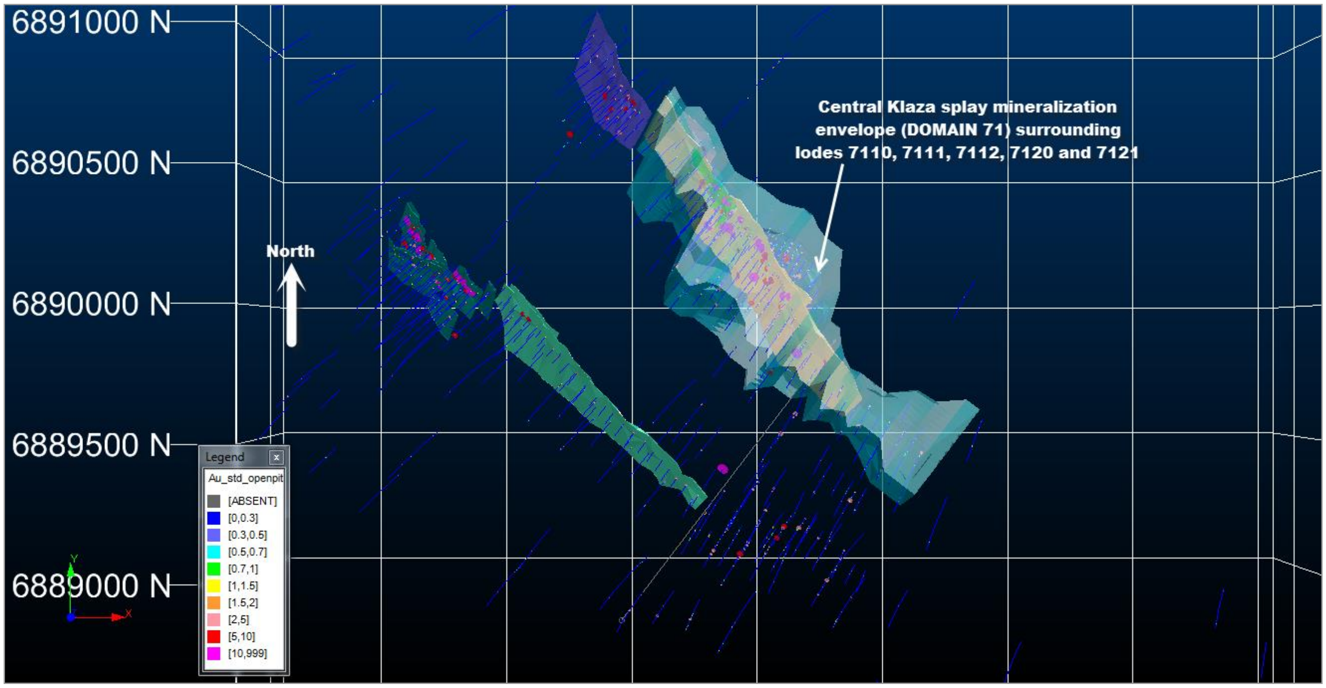
The DOMAIN values used to flag the mineralization envelopes at Central Klaza are defined in Table 14.14.

Table 14.14 DOMAIN flags used for the Central Klaza LMIK model

DOMAIN (numeric)	Area	Description
71	Central Klaza	Multiple sheeted splay lode-hosted mineralized zones with variable orientations and continuity.

Source: AMC Mining Consultants (Canada) Ltd.

Figure 14.4 The Central Klaza MIK / LMIK mineralization domain



Notes:
- Domain 71 is Central Klaza LMIK domain.
- 7110, 7111, 7112, 7120, 7121 domains were estimated by OK.

Source: AMC Mining Consultants (Canada) Ltd.

14.3.2 Statistics and compositing MIK / LMIK / ROK

The drillhole database, coded with interpreted mineralization zones via the DOMAIN field, was composited to a regular 3 m downhole composite length as a means of achieving a uniform sample support. Any missing, unassayed (assumed waste) intervals were patched and given default waste multi-element grades.

The decision to use 3 m composites considered the common raw sampling intervals in the drillhole data, the amount of data available for the domains, definition of mineralization, the parent cell sizes used for modelling, and mining considerations which form decisions regarding the SMU dimensions and LMIK model block size. Most of the assay data for the mineralized drillhole intersections are from core samples collected for a range of intervals, with common 0.5 m, 1 m, 2 m, and 3 m downhole lengths (Figure 14.5). A 3 m composite interval was selected as appropriate and minimized the decompositing of original sample intervals.

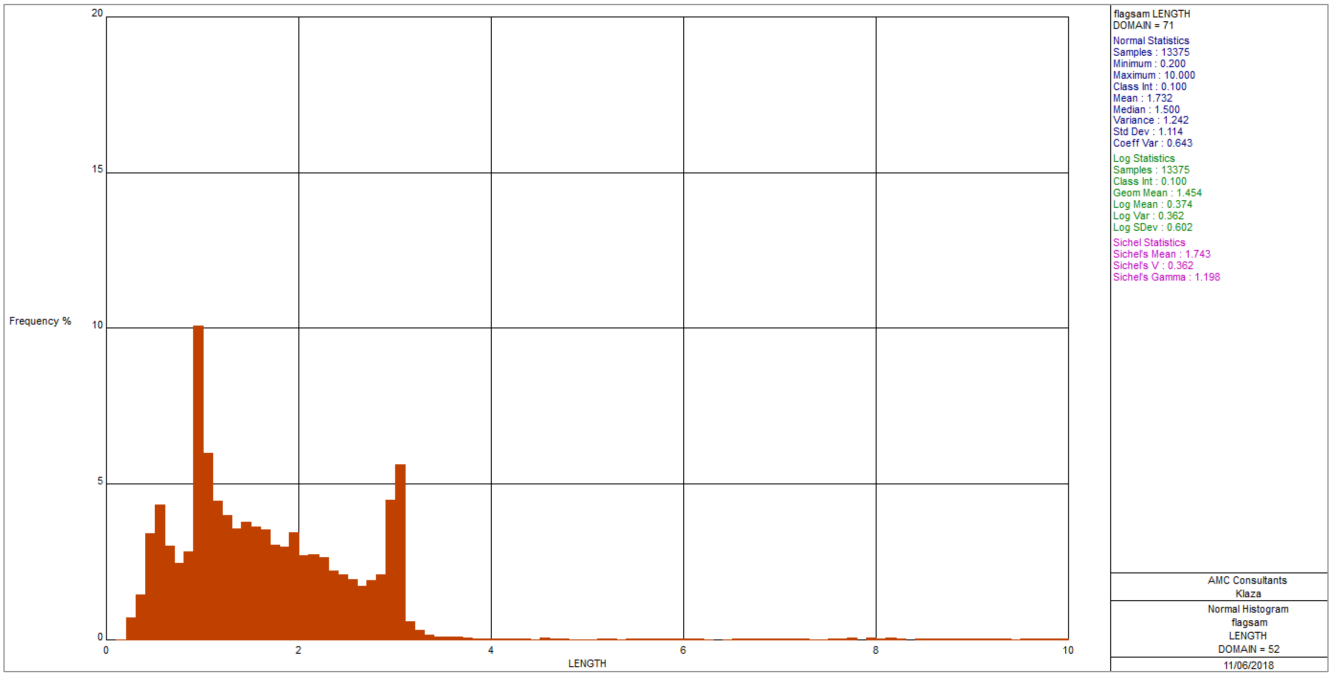
This is a larger composite size than for the OK portions of the model which used 1 m composites.

A residual retention compositing methodology with was used by AMC to prevent loss of data at zone or domain boundaries.

No composites were generated that crossed DOMAIN boundaries. For Central Klaza (DOMAIN 71), the compositing process generated 9,489 composites, with all composite intervals having at least the default waste multi-element values (0.001).

Multi-element data issues during compositing was managed via assignment of the default waste values for unassayed intervals.

Figure 14.5 Raw sample interval lengths for Central Klaza (DOMAIN 71) drillhole data



Source: AMC Mining Consultants (Canada) Ltd.

Where the non-linear LMIK estimation method is being used for the Au and Ag model, technically, high-grade capping of data is not required or used directly. High-grade capped data is used indirectly for both the Au and Ag grade variography and global estimates (used to calibrate the final SMU model). High-grade capped data was potentially considered for any portions of the remaining model that were estimated using ROK (Pb, Zn, Cu, As, Fe). Therefore, assessment of the high-grade composites was completed to determine the requirement for high-grade cutting.

The approach taken to the assessment of the high-grade composites and outliers is summarized as follows:

- Detailed review of histograms and probability plots, with significant breaks in populations used to interpret possible outliers.
- The spatial distribution of high-grade data was reviewed.

High-grade caps were applied only to Au and Ag composites for Central Klaza splay mineralization and are supplied in Table 14.15. The high-grade caps are light and affect minimal data.

Table 14.15 Statistics of raw, capped, and composite assay data – Central Klaza (DOMAIN 71)

Raw							
Element	**Au (g/t)**	**Ag (ppm)**	**Pb (ppm)**	**Zn (ppm)**	**As (ppm)**	**Cu (ppm)**	**Fe (%)**
Number	13,633	13,633	13,633	13,633	13,633	13,633	13,633
Minimum	0.001	0.001	0.001	0.001	0.001	0.001	0.001
Maximum	182	981	195,500	338,100	75,300	18,350	35
Mean	**0.29**	**5.87**	**726**	**1,306**	**385**	**89**	**3.3**
Std Dev	2.14	28.93	4,144	6,435	1,854	469	1.79
Coeff Var	7.3	4.9	5.7	4.9	4.8	5.3	0.5
Composited							
Element	**Au (g/t)**	**Ag (ppm)**	**Pb (ppm)**	**Zn (ppm)**	**As (ppm)**	**Cu (ppm)**	**Fe (%)**
Number	9,489	9,489	9,489	9,489	9,489	9,489	9,489
Minimum	0.001	0.001	0.001	0.001	0.001	0.001	0.001
Maximum	37.1	353	39,159	38,599	12,065	5,844	14
Mean	**0.13**	**2.6**	**323**	**586**	**177**	**44**	**2.4**
Std Dev	0.60	9.05	1,215	1,620	563	173	1.51
Coeff Var	4.6	3.5	3.8	2.8	3.2	3.9	0.6
Capped							
Element	**Au (g/t)**	**Ag (ppm)**	**Pb (ppm)**	**Zn (ppm)**	**As (ppm)**	**Cu (ppm)**	**Fe (%)**
Number	9,489	9,489	9,489	9,489	9,489	9,489	9,489
Minimum	0.001	0.001	0.001	0.001	0.001	0.001	0.001
Maximum	12.0	190	39,159	38,599	12,065	5,844	14
Mean	**0.13**	**2.6**	**323**	**586**	**177**	**44**	**2.4**
Std Dev	0.47	8.50	1,215	1,620	563	173	1.51
Coeff Var	3.7	3.3	3.8	2.8	3.2	3.9	0.6

Notes:
- Std Dev = Standard Deviation; Coeff Var = Coefficient of Variation.
- Pb, Zn, Cu, As, and Cu have been evaluated and initially estimated in units of ppm rather than %.
- For DOMAIN 71, only Au and Ag data had high-grade-caps applied.

Source: AMC Mining Consultants (Canada) Ltd.

De-clustered zonal statistics are used for both the intra-class (bin) mean grades applied for the MIK process and the global estimate used to tune the SMU model for the final LMIK resource estimate. The de-clustering results are only relevant for Au and Ag data from Central Klaza (DOMAIN 71).

The results for a range of declustering cell sizes were compared with the OK Au (capped) results from the first pass estimates to determine a practical optimal de-clustered mean grade and cell dimension. The declustering was done using Isatis software.

Conditional statistics were generated for Au and Ag data from Central Klaza (DOMAIN 71).

For this domain and these elements, MIK / LMIK was the appropriate estimation method. The conditional statistics were used to determine the appropriate indicator thresholds and the intra-class means, thus allowing post-processing. The indicator thresholds were based on analysis of the percentile statistics of Au grades, metal, and cumulative metal for the top 50% of the composite data. Twelve indicator thresholds were selected for the mineralized zones. The indicator thresholds were selected to discretize both the sample distribution and the metal in key ranges within the domains investigated.

The indicator thresholds were determined using the 3 m composites and are shown in Table 14.16.

Table 14.16 Indicator thresholds and intra-class grades and probabilities used for the MIK estimates

Area / domain	Element	Indicator	Threshold (g/t or ppm)	Intra-class mean grade[1] (g/t or ppm)	Number	Probability	Cumulative probability
Central Klaza	Au	1	0.10	0.017	7555	79.35	79.35
		2	0.20	0.143	729	8.09	87.45
		3	0.30	0.246	327	3.55	91.00
		4	0.40	0.349	209	2.25	93.24
		5	0.50	0.443	130	1.44	94.68
		6	0.60	0.543	104	1.04	95.72
		7	0.80	0.695	119	1.31	97.03
		8	1.00	0.887	73	0.65	97.68
		9	1.40	1.193	85	0.80	98.48
		10	2.00	1.641	67	0.61	99.08
		11	3.00	2.433	47	0.49	99.57
		12	5.00	3.857	32	0.31	99.88
		Top	–	7.968[3]	12	0.12	100.00
	Ag	1	0.50	0.155	5001	52.00	52.00
		2	1.00	0.714	1301	14.66	66.66
		3	2.00	1.420	1104	12.21	78.87
		4	3.00	2.445	546	5.46	84.34
		5	4.00	3.440	295	3.05	87.39
		6	6.00	4.918	367	3.90	91.30
		7	10.00	7.638	314	3.37	94.66
		8	15.00	12.154	201	2.13	96.79
		9	20.00	17.041	128	1.13	97.92
		10	30.00	24.666	98	0.86	98.77
		11	50.00	36.626	80	0.67	99.44
		12	90.00	67.286	38	0.39	99.84
		Top	-	115.22[2]	16	0.16	100.00

Notes:
[1] Intra-class grades are based on capped de-clustered 3 m composite data.
[2] Where specified, the final bin grade used was a median value for the bin.
[3] Where specified, the final bin grade used was a mean value for the bin.
Source: AMC Mining Consultants (Canada) Ltd.

14.3.3 Block model MIK / LMIK / ROK

14.3.3.1 Block model parameters MIK / LMIK / ROK

Three-dimensional block models were generated for the Central Klaza deposit to enable grade estimation via MIK and OK with subsequent conversion to LMIK. The final block model sizes were selected to represent the available data, the data characteristics (variability as defined by variography), expected mining selectivity (medium scale open cut), and final LMIK SMU model dimensions.

SMU dimensions were discussed with the project geologists. Anticipated bench heights are assumed to be 5 m using moderate sized mining equipment.

Bench heights and mining parameters will need to be tested and finalized during feasibility studies.

Key assumptions and inputs for the LMIK model at Central Klaza are:

- Moderately selective mining by open pit method.
- The MIK model panel block dimensions are 25 mE by 12.5 mN by 5 mRL.
- Change of support (COS) in the MIK SMU model utilized a 6.25 mE by 3.125 mE by 5 mRL SMU dimension.
- The final LMIK SMU block dimension is 6.25 mE by 3.125 mE by 5 mRL.

Mineralized domain coding (DOMAIN) was established in the block model, based on the modelled mineralization wireframe constraints.

The parent block size was selected to be approximately half of the current resource definition drillhole data spacing in well drilled areas. The final SMU block dimensions allows for significant resolution of grade distribution within the mineralized zones. Sub-blocking was used, allowing effective volume representation of the interpretation based wireframes, incorporation of narrow lodes in Central Klaza, and definition of surface topography. Model blocks are not rotated for general ease of use and transfer between different mining software.

The parent cell dimensions, model origins, and model extents are shown in Table 14.17.

Table 14.17 Block model parameters

Deposit	Field	Easting (X)	Northing (Y)	RL (Z)
Central Klaza	Model origin coordinates	381,525	6888715	795
	Model extent (m)	2,300	2,375	715
	Panel – parent cell dimensions (m)	25	12.5	5
	Panel – number of parent cells	92	190	143
	SMU – parent cell dimensions (m)	6.25	3.125	5
	SMU – number of parent cells	368	760	143

Source: AMC Mining Consultants (Canada) Ltd.

14.3.3.2 Estimation methods MIK / LMIK / ROK

Ordinary kriging

OK, one of the more common geostatistical methods for estimating block grades, has been used for estimation of the ancillary grade variables (Pb, Zn, Cu, As, and Fe) for the final project models. OK was used to generate whole block Au and Ag panel estimates utilized for both the data declustering process and as a check for the MIK-derived E-type mean. OK was also used to generate service variables estimates for the localization step in the LMIK process.

Multiple indicator kriging

The MIK technique is implemented by completing a series of OK estimates of binary transformed data. A composite datum, which is equal to or above a nominated cut-off or threshold, is assigned a value of 1, with those below the nominated indicator threshold being assigned a value of 0. Variography is computed and modelled on these binary transformed datasets to determine kriging parameters, with a series of OK estimates then undertaken for each of the nominated indicator thresholds using the transformed datasets.

The indicator estimates, with an inclusive range between 0 and 1, represent the probability the point will exceed the indicator cut-off grade. The probability of the points exceeding a cut-off can also be considered equivalent to the proportion of a nominated block that will exceed the nominated cut-off grade.

The estimation of a complete series of indicator cut-offs allows the reconstitution of the local histogram or conditional cumulative distribution function (CCDF) for the estimated block. This allows the investigation of a series of local or block properties, such as the block mean of the grade (termed the E-type mean) and proportion (tonnes), above or below a nominated cut-off grade. However, the primary function of MIK is to produce a CCDF or local histogram for each block, conditional to or dependent on the data collected in the sample search. The local histogram is constructed by combining the data from each of the estimates, produced using the range of indicator cut-offs, into a CCDF representing the entire block. The estimated indicator cut-offs represent the probability (or proportion) of that block exceeding the nominated cut-off for the given sample support or size. Order relation corrections are applied to probabilities for individual indicator variable estimates where those estimates have deviated from logical relative trends between the consecutive indicator variables. The CCDF results are used to validate the initial MIK estimates prior to COS.

An important function of the estimation method is that a support correction can be applied to generate a model with grade-tonnage characteristics representing mining on a level of selectivity defined by a chosen SMU.

A range of techniques are known to produce a support correction and therefore allow for SMU emulation. The common features of the support correction are:

- Maintenance of the mean grade of the histogram (E-type mean).
- Adjustment of the histogram variance by a variance adjustment factor (f).
- Assumption of grade distribution shape.

The variance adjustment factor, used to reduce the histogram or CCDF variance, can be calculated using the determined variogram model from the dataset, and is often adjusted based on similar deposits for which close-spaced grade control data are available.

In simplest terms, the variance adjustment factor considers the known relationship derived from the dispersion variance:

- Total variance = variance of samples within blocks + variance between blocks.

The variance adjustment factor is calculated as the ratio of the variance between the blocks and the variance of the samples within the blocks, with a small ratio (e.g. 0.10) indicating a large adjustment of the CCDF variance and large ratio (e.g. 0.80) representing a small shift in the CCDF.

The Indirect Lognormal (ILN) support correction incorporating a post adjustment variance correction was applied for this Mineral Resource estimate.

Discrete Gaussian models (DGM; global models) have been used to check the MIK support correction results.

The product, at this stage (prior to localization), is the MIK SMU model incorporating the COS.

Local MIK

LMIK has been used for estimation of Au and Ag in the Central Klaza splay-hosted mineralization domain for the final project model.

Localization of the MIK model results involves conversion of the MIK SMU model panel estimates to SMU blocks within the panel having incremental fixed tonnages. Grades are allocated to the SMU blocks on a ranking basis in such a way that the MIK SMU model panel grade-tonnage characteristics are replicated by the fixed tonnage bins or SMU blocks within the panel.

The ability to replicate the MIK SMU model grade-tonnage characteristics is dependent on the number of cut-offs used in the MIK SMU model and then the SMUs available within the MIK model panel.

14.3.3.3 Variography and grade estimation MIK / LMIK / ROK

The variography generated for Central Klaza splay-hosted mineralization was based on the 3 m composite data coded with the mineralization domains. Variography was generated using Isatis software. The correlogram was selected as the best spatial measure to deal with the heteroscedastic characteristics of the data (clustering of data and zonal anisotropy in grades).

For the ancillary elements where ROK was used as the estimation method (Pb, Zn, Cu, As, and Fe), downhole and directional grade correlograms were generated as appropriate. For the major elements where MIK / LMIK was used as the estimation method (Au and Ag), both grade and selected indicator downhole and directional correlograms were generated. Two structure spherical models were used to model the correlograms. Variogram orientations and anisotropies approximately reflect geological and visible data trends.

Results of the grade and indicator variography for the following sections are supplied in Table 14.18.

ROK and MIK estimates have been completed for the domains, applying the relevant grade and indicator variograms. A set of ancillary service variables were used to record estimation statistics to the block model (e.g. number of samples used per block estimate, minimum distance of data from block centroid, pass in which the estimate was generated etc.).

For the mineralized zones, a first pass search ellipse of 120 m by 120 m by 20 m (major axis by semi-major axis by minor axis respectively) was used, reflecting the observed trends for the Au and

Ag mineralization and some average anisotropies indicated by the variogram models. All search ellipses are oriented according to the variogram model orientations.

The search strategy used in the models is provided as Table 14.19, and is summarized as follows:

- A two-pass estimation strategy was implemented for the mineralized zones wherein each successive estimate is completed with expanded sample searches and relaxed composite collection criteria.
- To minimize the effect of data clustering a limit of 7 composites per drillhole was implemented for any panel-type estimates and 3 for any SMU block estimates.
- For the major mineralization zones estimated using MIK, block discretization of 4 by 3 by 3 points was used for the panel estimates used for indicator variables prior to COS and localization. Block discretization for the ancillary grade variables estimated by ROK directly into the SMU blocks, block discretization of 2 by 2 by 3 points was used.
- Because of the SMU sized blocks, the ancillary grade variables estimated via ROK used a deliberately restricted neighbourhood with limited composites to retain local variability in the block estimates.

The estimation process output Au and Ag indicator probability values for the parent cell volumes (i.e. sub-cells have identical grades to each other and the notional parent cell for that area). The Au and Ag grades are also estimated via OK in both panel and local SMU models, but these estimates are used only for validation purposes and service variables at several stages of the MIK process and are later discarded.

The tables of variogram model and parameters used for the estimates are in Table 14.18 and Table 14.19 respectively.

Table 14.18 Variogram model parameters

Deposit	Domain	Variable	Used for	Major axis		Semi-major axis		Minor axis		Relative nugget (C_0%)	Sill 1 (C_1%)	Range structure 1 (m)			Sill 2 (C_2%)	Range structure 2 (m)		
				Dip (°)	Azimuth (°)	Dip (°)	Azimuth (°)	Dip (°)	Azimuth (°)			Major axis	Semi-major axis	Minor axis		Major axis	Semi-major axis	Minor axis
Central Klaza	71	AUCUT	MIK	0	125	-50	215	-40	35	75	17	50	60	5	08	180	180	20
	71	INDAU1	MIK	0	125	-50	215	-40	35	60	30	50	50	8	10	300	300	40
	71	INDAU2	MIK	0	125	-50	215	-40	35	60	30	50	50	8	10	300	300	40
	71	INDAU3	MIK	0	125	-50	215	-40	35	60	30	50	50	8	10	300	300	30
	71	INDAU4	MIK	0	125	-50	215	-40	35	65	27	50	50	8	08	300	300	30
	71	INDAU5	MIK	0	125	-50	215	-40	35	65	27	50	50	8	08	300	300	30
	71	INDAU6	MIK	0	125	-50	215	-40	35	70	24	50	50	8	06	250	250	25
	71	INDAU7	MIK	0	125	-50	215	-40	35	70	24	50	50	8	06	250	250	25
	71	INDAU8	MIK	0	125	-50	215	-40	35	75	15	50	50	6	10	130	130	10
	71	INDAU9	MIK	0	125	-50	215	-40	35	75	15	50	50	6	10	130	130	10
	71	INDAU10	MIK	0	125	-50	215	-40	35	80	11	50	50	6	09	130	130	10
	71	INDAU11	MIK	0	125	-50	215	-40	35	90	06	15	15	5	04	120	120	10
	71	INDAU12	MIK	0	125	-50	215	-40	35	90	06	15	15	5	04	120	120	10
	71	AGCUT	MIK	0	125	-50	215	-40	35	65	25	25	50	7	10	100	150	14
	71	INDAG1	MIK	0	125	-50	215	-40	35	55	33	40	40	10	12	220	220	50
	71	INDAG2	MIK	0	125	-50	215	-40	35	55	33	40	40	10	12	220	220	50
	71	INDAG3	MIK	0	125	-50	215	-40	35	60	30	35	35	8	10	220	220	40
	71	INDAG4	MIK	0	125	-50	215	-40	35	60	33	30	30	8	07	220	220	20
	71	INDAG5	MIK	0	125	-50	215	-40	35	60	33	30	30	8	07	220	220	20
	71	INDAG6	MIK	0	125	-50	215	-40	35	65	30	30	30	6	05	180	180	12
	71	INDAG7	MIK	0	125	-50	215	-40	35	65	30	30	30	6	05	180	180	12
	71	INDAG8	MIK	0	125	-50	215	-40	35	70	23	30	30	6	07	100	100	12
	71	INDAG9	MIK	0	125	-50	215	-40	35	70	23	30	30	6	07	100	100	12
	71	INDAG10	MIK	0	125	-50	215	-40	35	75	20	30	30	6	05	80	80	12
	71	INDAG11	MIK	0	125	-50	215	-40	35	85	10	20	20	6	05	45	45	12
	71	INDAG12	MIK	0	125	-50	215	-40	35	85	10	20	20	6	05	45	45	12

Deposit	Domain	Variable	Used for	Major axis		Semi-major axis		Minor axis		Relative nugget (C_0%)	Sill 1 (C_1%)	Range structure 1 (m)			Sill 2 (C_2%)	Range structure 2 (m)		
				Dip (°)	Azimuth (°)	Dip (°)	Azimuth (°)	Dip (°)	Azimuth (°)			Major axis	Semi-major axis	Minor axis		Major axis	Semi-major axis	Minor axis
Central Klaza	71	PB	ROK	0	125	-50	215	-40	35	60	28	30	40	8	12	115	200	20
	71	ZN	ROK	0	125	-50	215	-40	35	65	27	25	25	7	08	200	200	25
	71	CU	ROK	0	125	-50	215	-40	35	55	40	25	25	8	05	100	150	20
	71	AS	ROK	0	125	-50	215	-40	35	65	23	25	25	7	12	170	200	25
	71	FE	ROK	0	125	-50	215	-40	35	20	60	40	30	20	20	150	220	60

Notes:
- Orientations for the major, semi-major and minor axes are supplied as dip and azimuths.
- Spherical models were applied to the experimental correlograms.
- MIK=multiple indicator kriging into initial 25 mE by 12.5 mN by 5 mRL panels. Converted through localization process to LMIK grades in the SMU blocks.
- ROK=restricted ordinary kriging directly into SMU blocks 6.25 mE by 3.125 mN by 5 mRL.

Source: AMC Mining Consultants (Canada) Ltd.

Table 14.19 Search neighbourhood parameters used for estimation of the Central Klaza splay mineralization domain

Domain	Variable	Used for	Search range (m)			First pass		Second pass			Maximum number of composites per drillhole
			Major	Semi-major	Minor	Min. No. of comps used	Max. No. of comps used	Search volume factor	Min. No. of comps used	Max. No. of comps used	
71	AUCUT	MIK	120	120	20	21	28	2.0	7	28	7
	AGCUT	MIK	120	120	20	21	28	2.0	7	28	7
	Au and Ag Indicators	MIK	120	120	20	21	28	2.0	7	28	7
	PB	ROK	120	120	20	9	12	3.0	6	12	3
	ZN	ROK	120	120	20	9	12	3.0	6	12	3
	CU	ROK	120	120	20	9	12	3.0	6	12	3
	AS	ROK	120	120	20	9	12	3.0	6	12	3
	FE	ROK	120	120	20	9	12	3.0	6	12	3

Notes:
- Search ellipse axes are oriented in accordance with the major, semi-major and minor variogram model axes in Table 14.18.
- The MIK estimates are for the large panel blocks and later adjusted to the smaller SMU block sizes through the localization process.
- AUCUT and AGCUT estimates for the large panel blocks are used for validation purposes only and are not retained in the final LMIK model.
- All parameters used for the ancillary grade variables above are used for ROK directly into the final SMU block model.

Source: AMC Mining Consultants (Canada) Ltd.

14.3.4 Mineral Resource classification MIK / LMIK / ROK

The criteria used to categorize the Mineral Resource in Central Klaza splay-hosted mineralization include the robustness of the input data, the confidence in the geological interpretation including the predictability of both structures and grades within the mineralized zones, the distance from data, and amount of data available for block estimates within the mineralized zone. An Inferred Mineral Resource for open pit mining has been defined.

Final resource classification was set in the model using wireframes to flag the blocks for the mineralized zones within closed 3D solids defining Inferred Resource material. By default, any other material in these domains is assumed to be unclassified and is not reported as part of the Mineral Resource.

14.3.5 Block model validation MIK / LMIK / ROK

The primary MIK panel estimates have been validated by comparing the average CCDF of the domain with the CCDF, based on both the raw and de-clustered composites. Good correlation between the CCDF (MIK estimates) and the de-clustered composites is evident for the domain, with satisfactory replication of the de-clustered composite data grades and distribution by the MIK estimates.

The SMU models (both interim MIK model and final LMIK model) were reported and compared against global COS estimates generated using DGM. Trial SMU models with variance adjustment factors of 1 (essentially no change from point support) were generated as a check for “fit” with the de-clustered point data used for the DGM global estimates. Actual panel grade estimates were also used to cross-check the DGM results. The LMIK results were cross checked against the both the theoretical DGM SMU curves and the MIK SMU model results.

Visual comparison of the LMIK and OK model with the composite data supports the statistical validation.

14.4 Mineral Resource estimate

The Mineral Resource estimate consists of pit constrained and underground Mineral Resources for four zones at the Klaza Deposit (Western BRX, Central BRX, Western Klaza, Central Klaza). Pit constrained Mineral Resources are reported between a base-of-weathering surface and a conceptual pit shell based on parameters as outlined in Table 14.20 and Table 14.21. The base of weathering was built using a topographic surface and displacing it downwards by 3 m. Assumptions considered for the conceptual pit shell included mining costs, processing costs and recoveries obtained from this report and comparable industry situations. These are summarized below in Table 14.20 and Table 14.21

A cut-off of 1.0 g/t gold equivalent was applied for reporting the pit constrained Mineral Resources. This overall cut-off grade was derived after reviewing the cut-off grades for each zone.

Representative cross sections through the Klaza deposit are shown in Section 10 of this report in Figure 10.2 to Figure 10.5.

Table 14.22 and Table 14.25 report the Mineral Resource estimates by zone.

Table 14.20 Conceptual pit shell parameters applied to all zones

Item	Pit optimization parameters	Unit
Gold price	1,400	US$/oz
Silver price	19	US$/oz
Lead price	1.10	US$/lb
Zinc price	1.25	US$/lb
Exchange rate	0.80	C$ to US$
% Payable gold [1]	97	%
% Payable silver [1]	81	%
% Payable lead [1]	62	%
% Payable zinc [1]	56	%
Royalties	0.0	%
Waste Mining cost	3.56 plus 0.03 / 5 m bench (@ 1425 RL)	C$/tonne
Ore Mining cost	4.38 plus 0.03 / 5 m bench (@ 1425 RL)	C$/tonne
Grade Control cost	0.50	C$/tonne
ROM rehandling cost	0.75	C$/tonne
Supervision / technical services cost	4.31	C$/tonne
Processing costs	Variable - See Table 14.21	
Processing G & A	9.60	C$/tonne
Preliminary overall slope angles	44 – 48	degrees
Dilution	5	%
Plant rate	2,000	Tonnes per day
Aueq formula	Variable - See Table 14.21	
Metallurgical recovery	Variable - See Table 14.21	

Note: [1] Payables account for all refining, concentrate and selling costs.
Source: AMC Mining Consultants (Canada) Ltd.

Table 14.21 Conceptual pit shell parameters applied to specific zones

Item		Central Klaza		Western Klaza	Western & Central BRX		Unit
Gold grade		> 1.0 g/t	<1.0 g/t	all	>1.0 g/t	<1.0 g/t	g/t Au
Processing cost		33.14	33.14	39.45	33.14	33.14	C$/tonne
Metallurgical recovery	Gold	92	78	96	94	64	%
	Silver	86	76	91	87	57	%
	Lead	79	73	85	80	49	%
	Zinc	81	71	85	79	42	%
AuEq formula		AUEQ = 1 x AU + AG/94.40 + PB/3.38 + ZN/3.21	AUEQ = 1 x AU + AG/90.56 + PB/3.10 + ZN/3.11	AUEQ = 1 x AU + AG/93.09 + PB/3.28 + ZN/3.20	AUEQ = 1 x AU + AG/95.34 + PB/3.41 + ZN/3.37	AUEQ = 1 x AU + AG/99.08 + PB/3.79 + ZN/4.31	AuEq g/t

Source: AMC Mining Consultants (Canada) Ltd.

Table 14.22 Mineral Resources as of 5 June 2018 (Western BRX)

Resource classification		Kt	Au (g/t)	Ag (g/t)	Pb (%)	Zn (%)	AuEq (g/t)	Au (koz)	Ag (koz)	Pb (klbs)	Zn (klbs)	AuEq (koz)
Indicated	Pit constrained	759	9.5	109	0.9	1.0	11.2	232	2,671	15,312	16,201	273
	Underground	510	6.7	122	0.9	1.1	8.5	110	2,001	10,341	11,920	140
	Total	**1,269**	**8.4**	**115**	**0.9**	**1.0**	**10.1**	**341**	**4,672**	**25,653**	**28,120**	**414**
Inferred	Pit constrained	369	4.7	54	0.5	0.6	5.6	56	644	3,915	4,884	67
	Underground	974	3.7	93	0.7	0.9	5.2	116	2,906	16,044	19,976	162
	Total	**1,343**	**4.0**	**82**	**0.7**	**0.8**	**5.3**	**172**	**3,550**	**19,958**	**24,859**	**228**

Notes:

- CIM Definition Standards (2014) were used for the Mineral Resource.
- Estimate includes drill results to 31 December 2017.
- Near surface Mineral Resources are constrained by an optimized pit shell at metal prices of $1,400/oz Au, $19/oz Ag, $1.10/lb Pb, and $1.25/lb Zn.
- Cut-off grades applied to the pit constrained and underground resources are 1.0 g/t AuEq and 2.3 g/t AuEq respectively.
- Gold equivalent values were calculated using parameters outlined in Table 14.4.
- Numbers may not add due to rounding.
- All metal prices are quoted in US$ at an exchange rate of US$0.80 to C$1.00.
- Mineral resources that are not Mineral Reserves do not have demonstrated economic viability.

Source: AMC Mining Consultants (Canada) Ltd.

Table 14.23 Mineral Resources as of 5 June 2018 (Central BRX)

Resource classification		Kt	Au (g/t)	Ag (g/t)	Pb (%)	Zn (%)	AuEq (g/t)	Au (koz)	Ag (koz)	Pb (klbs)	Zn (klbs)	AuEq (koz)
Indicated	Pit constrained	289	2.9	150	0.8	1.3	5.1	27	1,396	5,029	8,141	47
	Underground	424	1.9	106	1.0	1.1	3.6	26	1,443	9,793	10,171	50
	Total	**713**	**2.3**	**124**	**0.9**	**1.2**	**4.2**	**53**	**2,838**	**14,822**	**18,312**	**97**
Inferred	Pit constrained	238	1.9	99	0.7	0.8	3.4	15	760	3,795	3,993	26
	Underground	1,187	1.7	102	1.0	0.9	3.3	64	3,896	26,215	23,401	126
	Total	**1,425**	**1.7**	**102**	**1.0**	**0.9**	**3.3**	**79**	**4,656**	**30,010**	**27,394**	**152**

Note: See footnotes on Table 14.22.
Source: AMC Mining Consultants (Canada) Ltd.

Table 14.24 Mineral Resources as of 5 June 2018 (Western Klaza)

Resource classification		Kt	Au (g/t)	Ag (g/t)	Pb (%)	Zn (%)	AuEq (g/t)	Au (koz)	Ag (koz)	Pb (klbs)	Zn (klbs)	AuEq (koz)
Indicated	Pit constrained	139	4.8	234	0.8	0.9	7.8	21	1,043	2,388	2,722	35
	Underground	361	4.0	181	0.6	0.7	6.4	47	2,099	4,440	5,773	74
	Total	**500**	**4.2**	**195**	**0.6**	**0.8**	**6.8**	**68**	**3,142**	**6,828**	**8,495**	**109**
Inferred	Pit constrained	2	1.8	118	0.3	0.6	3.3	0	8	12	26	0
	Underground	227	3.9	170	0.5	0.8	6.1	28	1,240	2,440	3,817	44
	Total	**229**	**3.8**	**169**	**0.5**	**0.8**	**6.0**	**28**	**1,248**	**2,452**	**3,842**	**44**

Note: See footnotes on Table 14.22.
Source: AMC Mining Consultants (Canada) Ltd.

Table 14.25 Mineral Resources as of 5 June 2018 (Central Klaza)

Resource classification		Kt	Au (g/t)	Ag (g/t)	Pb (%)	Zn (%)	AuEq (g/t)	Au (koz)	Ag (koz)	Pb (klbs)	Zn (klbs)	AuEq (koz)
Indicated	Pit constrained	1,260	3.3	49	0.6	0.9	4.3	133	1,987	16,414	25,871	173
	Underground	715	3.9	62	0.6	0.7	5.0	90	1,432	9,550	11,309	114
	Total	**1,976**	**3.5**	**54**	**0.6**	**0.9**	**4.5**	**223**	**3,419**	**25,964**	**37,180**	**288**
Inferred	Pit constrained	1,145	2.1	28	0.3	0.4	2.6	76	1,017	7,176	9,696	94
	Underground	1,572	3.0	68	0.5	0.7	4.1	151	3,431	17,948	23,385	206
	Total	**2,717**	**2.6**	**51**	**0.4**	**0.6**	**3.4**	**227**	**4,448**	**25,124**	**33,081**	**301**

Note: See footnotes on Table 14.22.
Source: AMC Mining Consultants (Canada) Ltd.

The results of the pit constrained Mineral Resource estimates for the BRX and Klaza deposits are shown in Table 14.26 and Table 14.27 at a range of cut-off grades. The base case cut-off grade derived from parameters outlined in Table 14.20 and Table 14.21 is highlighted in bold.

Table 14.26 Klaza and BRX deposits pit constrained Indicated Mineral Resource estimate (all zones) cut-off grade sensitivity

Cut-off grade (g/t AuEq)	Tonnes (kt)	Au (g/t)	Ag (g/t)	Pb (%)	Zn (%)	AuEq (g/t)	Metal (Au koz)	Metal (Ag koz)	Metal (Pb klb)	Metal (Zn klb)	Metal (AuEq koz)
0.75	2,471	5.2	90	0.7	1.0	6.7	414	7,113	39,238	53,082	530
1.00	**2,447**	**5.3**	**90**	**0.7**	**1.0**	**6.7**	**414**	**7,096**	**39,143**	**52,935**	**529**
1.25	2,399	5.3	92	0.7	1.0	6.8	412	7,066	38,928	52,665	527
1.50	2,303	5.5	95	0.8	1.0	7.1	409	6,999	38,428	52,029	523
1.75	2,216	5.7	97	0.8	1.1	7.3	406	6,916	37,881	51,418	518
2.00	2,147	5.8	99	0.8	1.1	7.5	403	6,838	37,391	50,838	514
2.50	1,951	6.3	106	0.8	1.1	8.0	393	6,621	36,121	48,570	500
3.00	1,789	6.7	111	0.9	1.2	8.4	383	6,381	34,781	46,493	486

Notes:
- CIM Definition Standards (2014) were used for the Mineral Resource.
- Estimate includes drill results to 31 December 2017.
- Near surface Mineral Resources are constrained by an optimized pit shell at metal prices of $1,400/oz Au, $19/oz Ag, $1.10/lb Pb, and $1.25/lb Zn.
- Cut-off grades applied to the pit constrained are 1.0 g/t AuEq.
- Gold equivalent values were calculated using parameters outlined in Table 14.4.
- Numbers may not add due to rounding.
- Mineral resources that are not Mineral Reserves do not have demonstrated economic viability.

Source: AMC Mining Consultants (Canada) Ltd.

Table 14.27 Klaza and BRX deposits pit constrained Inferred Mineral Resource estimate (all zones) cut-off grade sensitivity

Cut-off grade (g/t AuEq)	Tonnes (kt)	Au (g/t)	Ag (g/t)	Pb (%)	Zn (%)	AuEq (g/t)	Metal (Au koz)	Metal (Ag koz)	Metal (Pb klb)	Metal (Zn klb)	Metal (AuEq koz)
0.75	2,071	2.3	38	0.3	0.4	2.9	154	2,543	15,803	19,851	196
1.00	**1,754**	**2.6**	**43**	**0.4**	**0.5**	**3.3**	**147**	**2,429**	**14,897**	**18,599**	**187**
1.25	1,560	2.8	47	0.4	0.5	3.6	142	2,344	14,276	17,662	180
1.50	1,352	3.1	52	0.5	0.6	3.9	135	2,250	13,569	16,468	171
1.75	1,223	3.3	55	0.5	0.6	4.2	129	2,180	13,090	15,605	165
2.00	1,092	3.5	59	0.5	0.6	4.5	123	2,081	12,450	14,553	157
2.50	887	3.9	67	0.6	0.7	5.0	111	1,907	11,458	13,033	142
3.00	683	4.4	77	0.7	0.7	5.7	97	1,682	10,005	10,984	124

Note: See footnotes on Table 14.26.
Source: AMC Mining Consultants (Canada) Ltd.

The underground Mineral Resources were reported outside of the conceptual pit shells. No allowances were made for crown pillars. The cut-off applied to the underground Mineral Resources was 2.30 g/t gold equivalent for all zones.

Assumptions made to derive a cut-off grade included mining costs, processing costs, and recoveries were obtained from this report and comparable industry situations.

The results of the underground Mineral Resource estimates for the Klaza and BRX deposits are shown in Table 14.28 and Table 14.29 at a range of cut-offs, with the selected cut-offs shown in bold.

Table 14.28 Klaza and BRX deposits underground Indicated Mineral Resource estimate (all zones) cut-off grade sensitivity

Cut-off grade (g/t AuEq)	Tonnes (kt)	Au (g/t)	Ag (g/t)	Pb (%)	Zn (%)	AuEq (g/t)	Metal (Au koz)	Metal (Ag koz)	Metal (Pb klb)	Metal (Zn klb)	Metal (AuEq koz)
2.00	2,174	4.0	103	0.7	0.9	5.6	280	7,205	35,170	40,824	389
2.30	**2,010**	**4.2**	**108**	**0.8**	**0.9**	**5.8**	**272**	**6,974**	**34,125**	**39,172**	**378**
2.50	1,900	4.4	112	0.8	0.9	6.1	266	6,815	33,248	37,910	370
2.75	1,731	4.6	117	0.8	0.9	6.4	257	6,521	31,830	35,784	355
3.00	1,604	4.8	121	0.9	1.0	6.7	249	6,251	30,369	34,129	344
4.00	1,134	5.9	142	1.0	1.1	8.0	214	5,176	24,126	26,613	291
5.00	854	6.8	160	1.0	1.1	9.1	187	4,397	19,751	21,372	251
6.00	670	7.7	172	1.1	1.2	10.2	165	3,703	15,773	17,232	219

Notes:
- CIM definition (2014) standards were used for the Mineral Resource.
- Estimate includes drill results to 31 December 2017.
- Cut-off grades applied to the underground are 2.3 g/t AuEq.
- Gold equivalent values were calculated using parameters outlined in Table 14.4.
- Numbers may not add due to rounding.
- Mineral resources that are not Mineral Reserves do not have demonstrated economic viability.

Source: AMC Mining Consultants (Canada) Ltd.

Table 14.29 Klaza and BRX deposits underground Inferred Mineral Resource estimate (all zones) cut-off grade sensitivity

Cut-off grade (g/t AuEq)	Tonnes (kt)	Au (g/t)	Ag (g/t)	Pb (%)	Zn (%)	AuEq (g/t)	Metal (Au koz)	Metal (Ag koz)	Metal (Pb klb)	Metal (Zn klb)	Metal (AuEq koz)
2.00	4,739	2.6	83	0.7	0.8	3.9	395	12,621	69,344	78,921	592
2.30	**3,960**	**2.8**	**90**	**0.7**	**0.8**	**4.2**	**359**	**11,472**	**62,647**	**70,578**	**538**
2.50	3,573	3.0	94	0.7	0.8	4.4	340	10,817	59,041	65,336	508
2.75	3,137	3.1	98	0.8	0.9	4.7	318	9,929	54,243	59,255	472
3.00	2,676	3.4	105	0.8	0.9	5.0	289	9,029	49,215	52,537	429
4.00	1,507	4.2	130	1.0	1.1	6.2	204	6,276	33,752	35,669	300
5.00	938	5.0	148	1.2	1.2	7.2	150	4,471	23,860	24,774	219
6.00	612	5.7	163	1.2	1.2	8.2	113	3,212	16,118	16,711	161

Note: See footnotes on Table 14.28.
Source: AMC Mining Consultants (Canada) Ltd.

14.5 Comparison with previous Mineral Resource estimate

As discussed in Section 2 this PEA is based on the 2018 Mineral Resource outlined above. As such there is no comparison to a previous estimate.

The 2018 Mineral Resource estimate on the Property was originally published in the AMC report titled "Technical Report Describing Updated Diamond Drilling, Metallurgical Testing and Mineral Resources on the Klaza Property, Yukon, Canada" with an effective date of 5 June 2018.

15 Mineral Reserve estimates

There are no Mineral Reserve estimates to report for the Property.

16 Mining methods

The PEA considered vein systems contained in two distinct zones – Klaza and BRX. Each zone can be further broken down by relative location: West, Central and East. The two zones lend themselves to open pit mining as the mineralized veins are located close to surface. The surrounding topography is moderately steep with sufficient flat areas suitable for the placement of waste dumps and stockpiles. The climate is favourable to open pit mining, with relatively light precipitation, predominantly in the summer.

Mineralization has been identified through exploration drilling below the potential open pits to a depth of approximately 450 m below surface; both zones remain open at depth and along strike. Both the BRX and Klaza vein systems are amenable to mining by underground methods. The West and Central zones are separated by a fault system. The West zones for both Klaza and BRX are the highest grade. The eastern extent of the Central Klaza zone would be accessed by an independent decline system from the pit. As the mineralization tends to lower in grade and thickness towards the eastern extents of both deposits, this material is only considered as upside potential that warrants further study and is not included in this PEA update.

16.1 Hydrological parameters

In anticipation of a future application under the YESAA and the Waters Act as part of the mine permitting process, Tetra Tech EBA was retained by Rockhaven to install a groundwater monitoring well network in the area of the mineralized zones at Klaza and to conduct a preliminary hydrogeological assessment.

A preliminary network consisting of five nested monitoring wells was installed, with well locations downgradient of the main mineralized zones, including Western, Central, and Eastern BRX and Western and Central Klaza.

In addition to the monitoring wells, four observations wells were installed with vibrating wire piezometers (VWPs) to monitor pore pressures at three different depths in each of the wells. These wells were installed in the Western, Central, and Eastern BRX areas, and Western and Central Klaza. One shallow piezometer and one deep piezometer were installed at each location. The deep piezometers target the sub-permafrost aquifer whereas the shallow piezometers were installed within the anticipated active zone to monitor seasonal supra-permafrost groundwater.

Ground temperatures and permafrost conditions were assessed using subsurface temperature data from the VWPs and one thermistor cable installed to a depth of about 100 m in the Central Klaza zone.

The main conclusions of the preliminary hydrogeological assessment can be summarized as follows:

- Permafrost appears to act as a confining layer for the deeper bedrock aquifer. However, some uncertainty remains with respect to the permafrost temperature, extent, and interaction with groundwater that requires further data collection.
- The groundwater flow regime at the site is controlled by the steep terrain, with flow from areas at higher elevations on the mountain slopes toward the valley bottoms and generally mimicking the topography.
- Hydraulic conductivities of the granodiorite bedrock aquifer were inferred from packer tests conducted in all four VWP observation wells and hydraulic response tests conducted in select monitoring wells. Inferred hydraulic conductivities ranged over several orders of magnitude from about 4 × 10-10 m/s to 1 × 10-5 m/s, with an average hydraulic conductivity of 2 × 10-8 m/s. Hydraulic conductivity in bedrock is inferred to be largely controlled by fracture density and permeability.

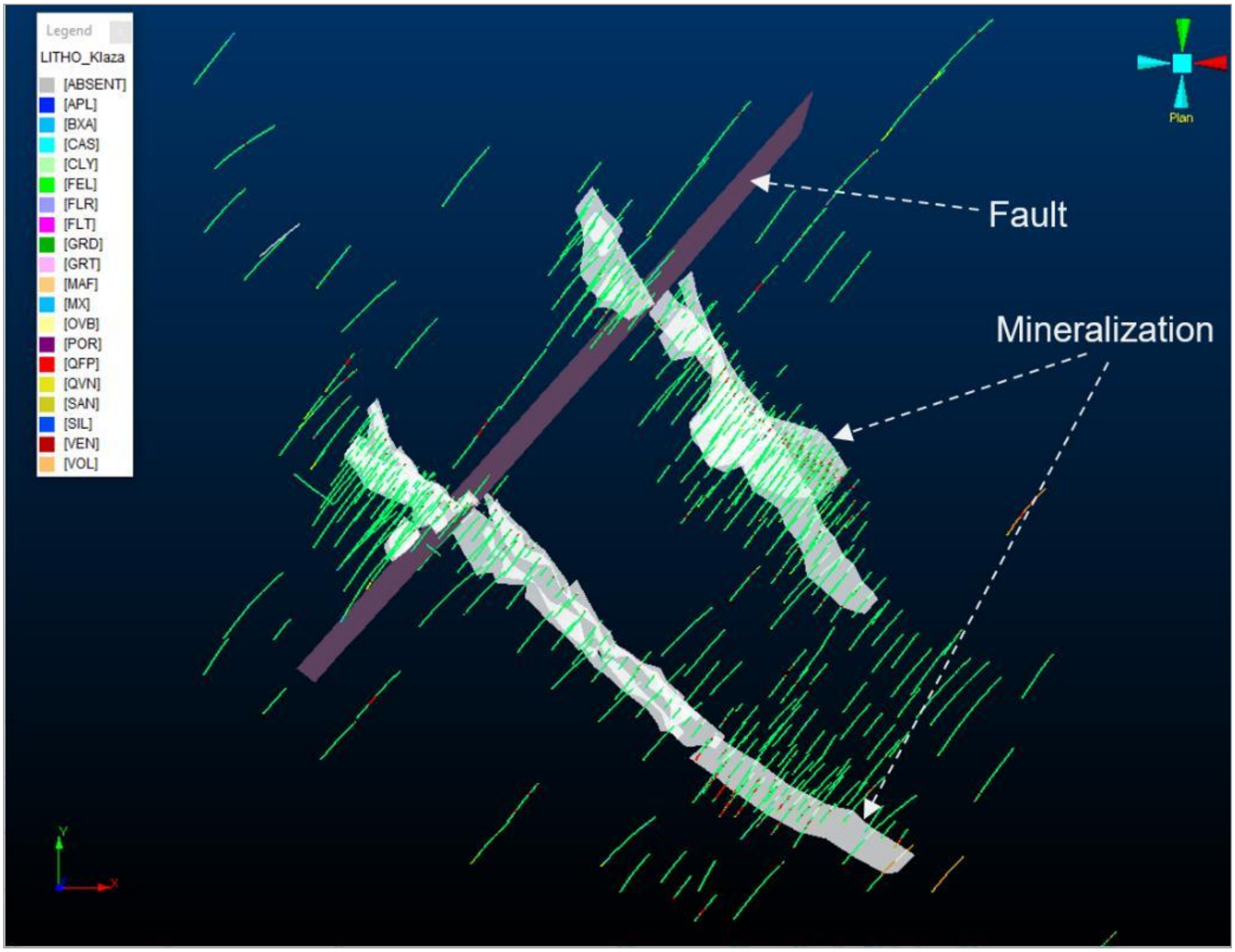
- Groundwater samples for chemical analysis were collected from all monitoring wells in August 2015. The results show a similar hydrogeochemical composition with slight differences between samples due to sample location. All groundwater samples are of a calcium and / or magnesium dominant cation type, and bicarbonate and / or sulphate anion type.
- All groundwater samples, including those collected in the area of the mineralized zones, showed a near neutral to slightly basic pH (between 7 and 8).

Based on the above conclusions, AMC has assumed a relatively low groundwater inflow that can be adequately pumped from the potential mine workings using submersible pumps and, in the underground case, a four-inch (100 mm) discharge pipeline. The majority of the discharge water will be service water for operating equipment with minor inflow from ground water.

16.2 Geotechnical parameters

Rockhaven provided drilling information (from 2010 to 2019) for the Property. Figure 16.1 shows drilling locations with lithology information. Granodiorite (GRD) represents 88% of the total drilled length.

Figure 16.1 Drillhole locations and lithological units: Klaza and BRX zones (2010 – 2019)



Source: AMC Mining Consultants (Canada) Ltd.

Based on the available drillhole data, geotechnical design parameters were determined by AMC for the potential open pit and underground mines, these are summarized in the following sections.

16.2.1 Local lithology

It is evident that the dominant lithological unit in the mineralized areas is granodiorite (88% of the total drill metres). The other lithologies make up the remaining 12% (Table 16.1).

Table 16.1 Total drillhole length within lithological units

Rock type		Total drillhole length (m)	% freq.
Aplite	APL	29	0.03
Breccia	BXA	348	0.35
Casing	CAS	714	0.71
Felsic Rock	FEL	1404	1.4
Fault	FLT	230	0.23
Granodiorite	GRD	88362	88.38
Mafic Rock	MAF	360	0.36
Overburden	OVB	1255	1.25
Porphyry	POR	549	0.55
Quartz Feldspar Porphyry	QFP	4679	4.68
Sand	SAN	4	0
Siliceous Dyke	CLY	108	0.11
Vein	VEN	336	0.34
Volcanic	VOL	1597	1.6
Total		**99,974**	**100.00**

16.2.2 Rock quality designation (RQD)

RQD is a fundamental input into several rock mass classification schemes and is generally regarded as a basic indicator of ground conditions. The RQD is calculated as the ratio of the sum of the lengths of all core sticks greater than 10 cm in length, to the total length of the drill core run, expressed as a percentage. Values and descriptions of RQD are presented in Table 16.2.

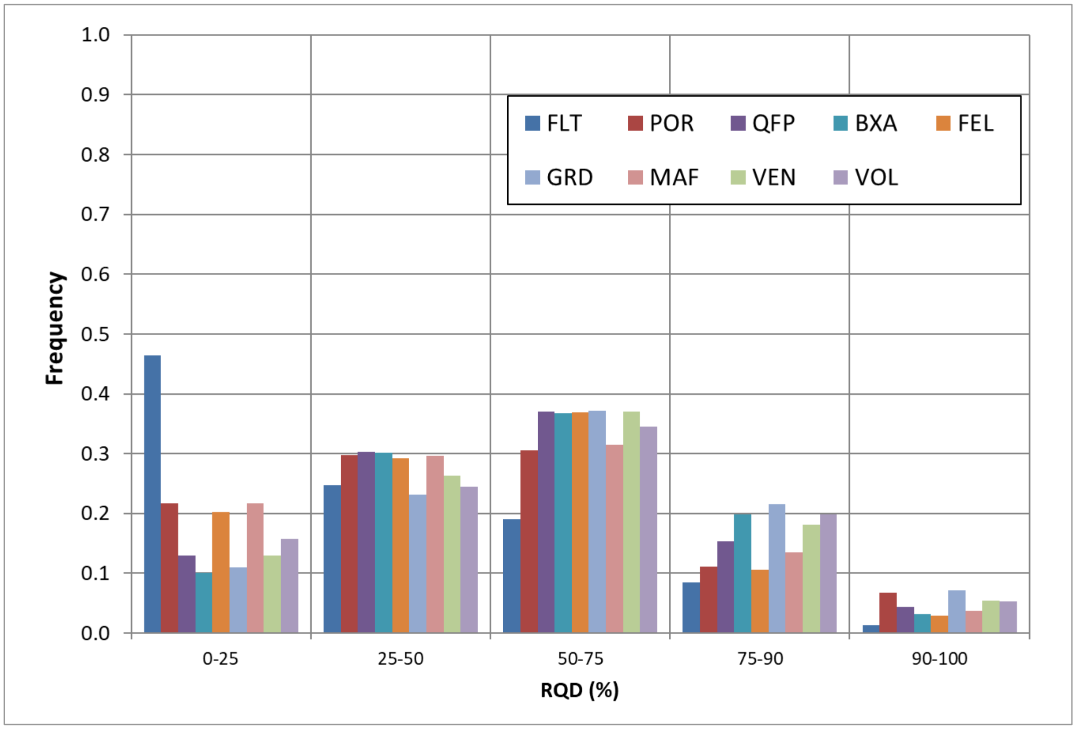
Table 16.2 RQD values and descriptions

RQD qualitative description	RQD %
Very poor	0 to 25
Poor	25 to 50
Fair	50 to 75
Good	75 to 90
Excellent	90 to 100

Source: Deere 1964.

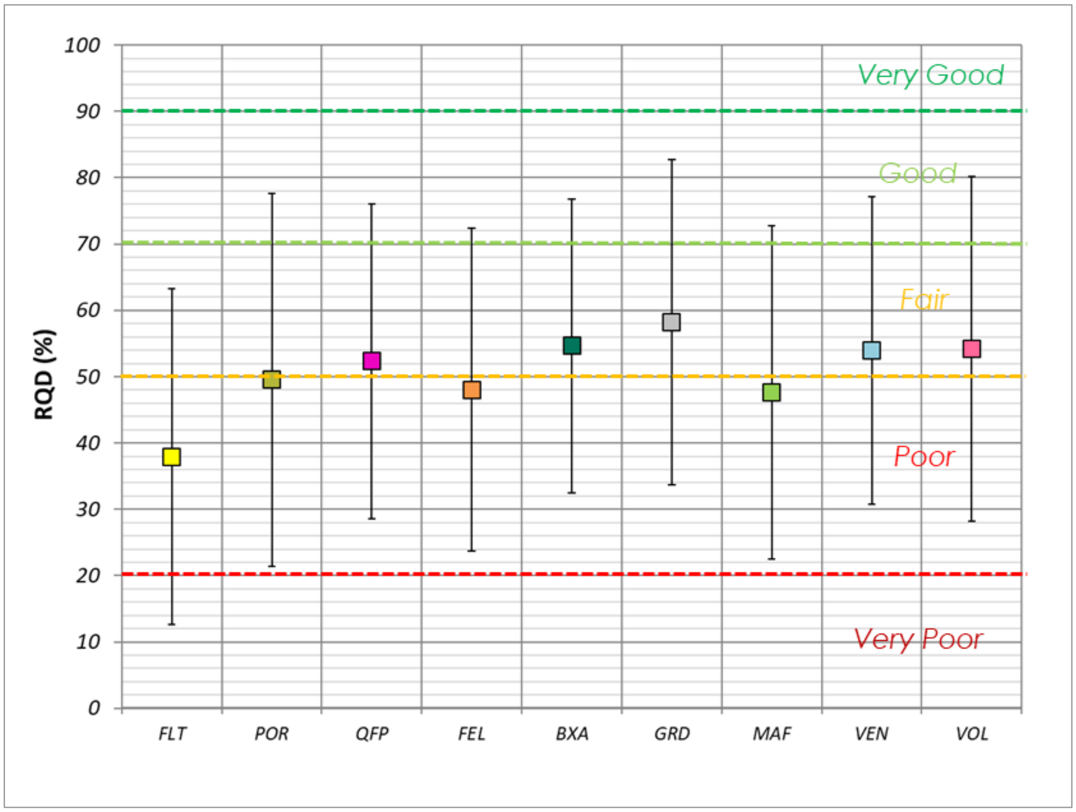
Figure 16.2 and Figure 16.3 present the statistical results of this assessment for each lithological unit, respectively. In general, the RQD assessment indicates that most lithologies identified at Klaza have a wide range of RQD values, typically from Poor to Good.

Figure 16.2 RQD frequencies by lithological domain



Source: AMC Mining Consultants (Canada) Ltd.

Figure 16.3 RQD by lithological domain (average +/- standard deviation)



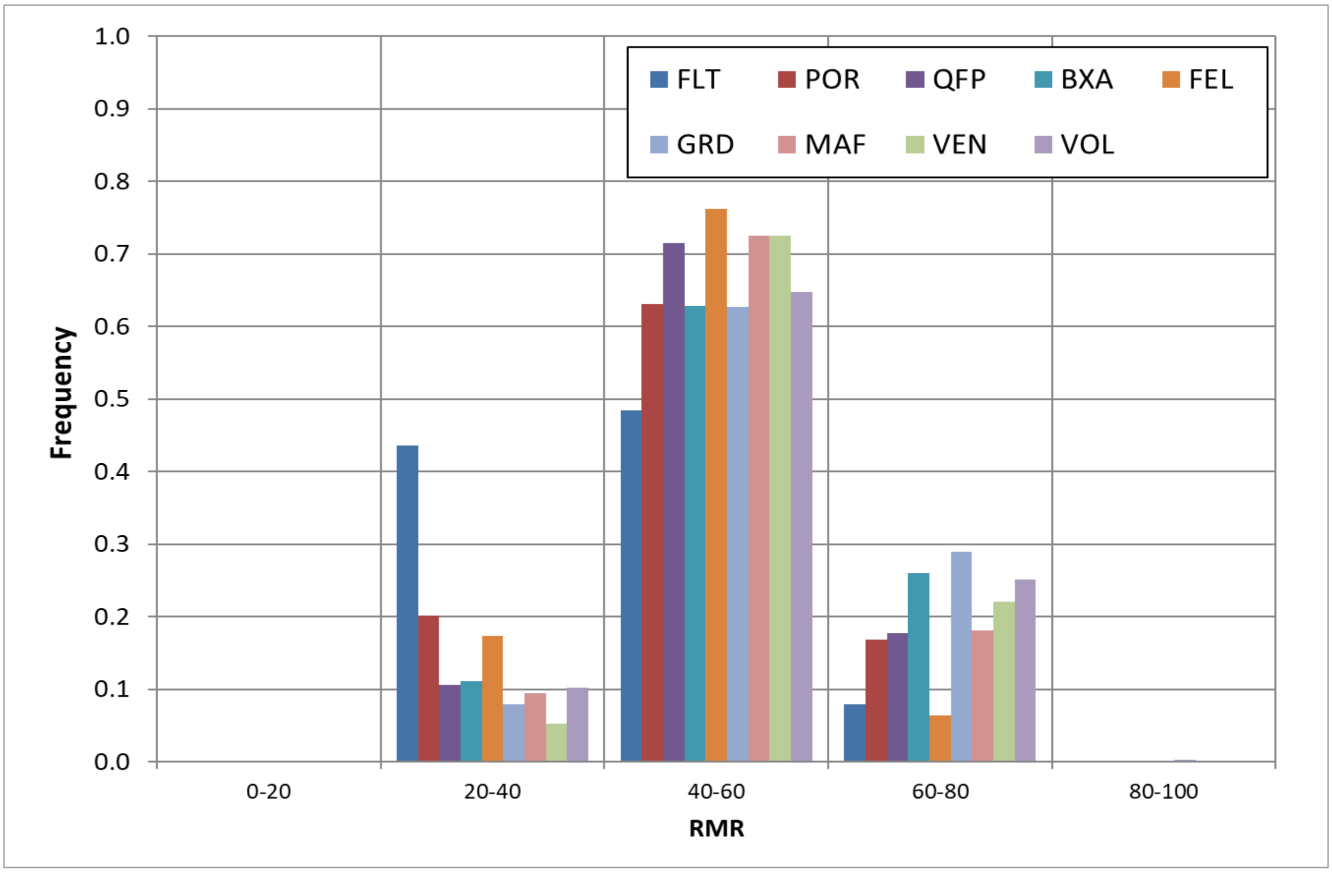
Source: AMC Mining Consultants (Canada) Ltd.

16.2.3 Rock mass characterization

Rock mass characterization was carried out using two main classification systems: RMR (Rock Mass Rating, Bieniawski 1976) and Q-system (Tunnelling Quality Index, Barton et al. 1974). RMR is typically used for slope stability analysis. The Q' (Modified Q with both SRF and JW being unity) are used for the assessment of open stope stability and ground support for underground excavations.

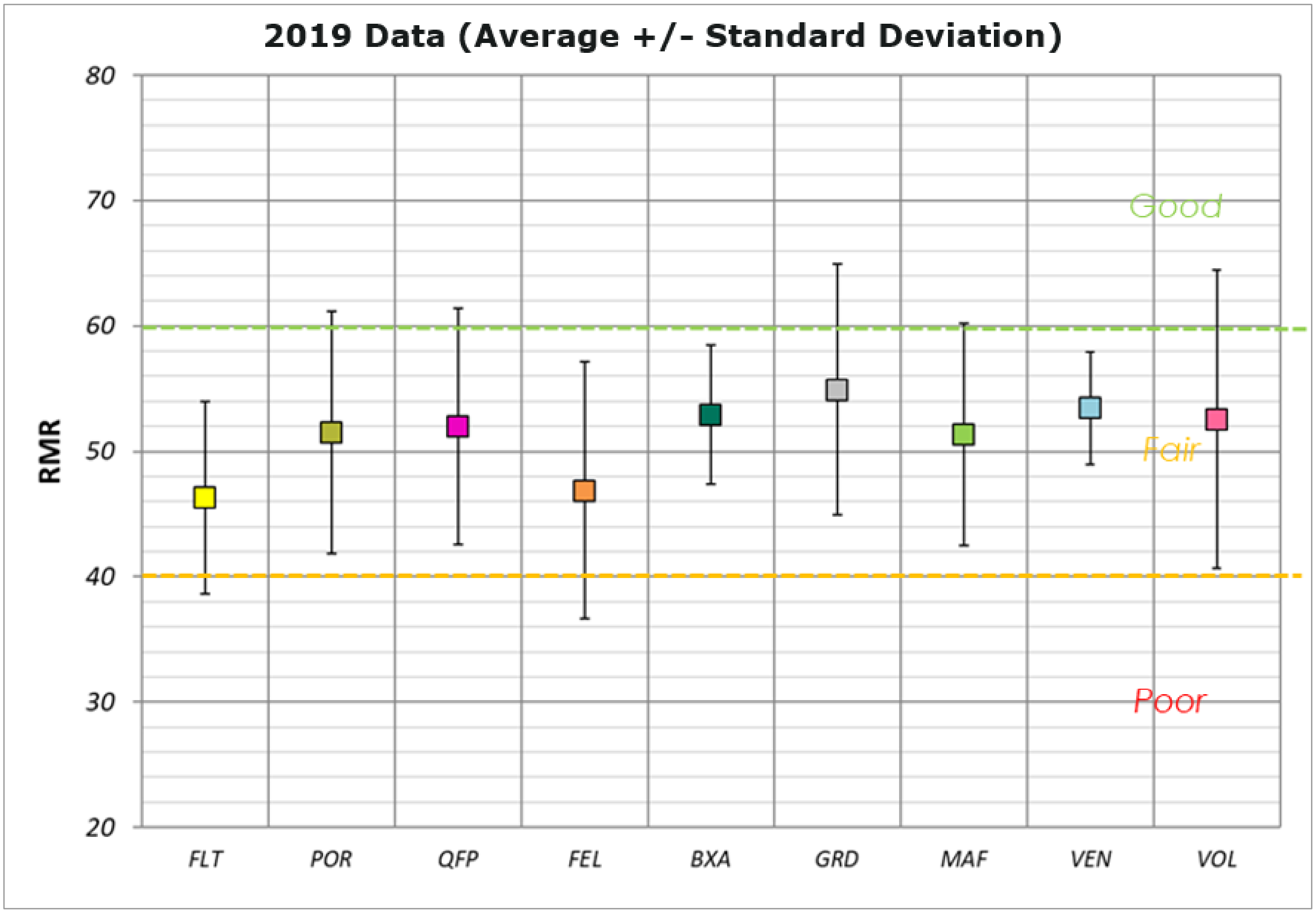
The RMR frequency distribution histogram and average ratings (with +/- standard deviation) for main lithologies are presented in Figure 16.4 and Figure 16.5, respectively. In summary, the rock mass quality of the host rock (primarily GRD) and ore is anticipated to be Fair and Good as per RMR classification. There would be some areas where poor ground conditions are expected, majority of these being in the vicinity of faults.

Figure 16.4 RMR_{76} by lithological domain



Source: AMC Mining Consultants (Canada) Ltd.

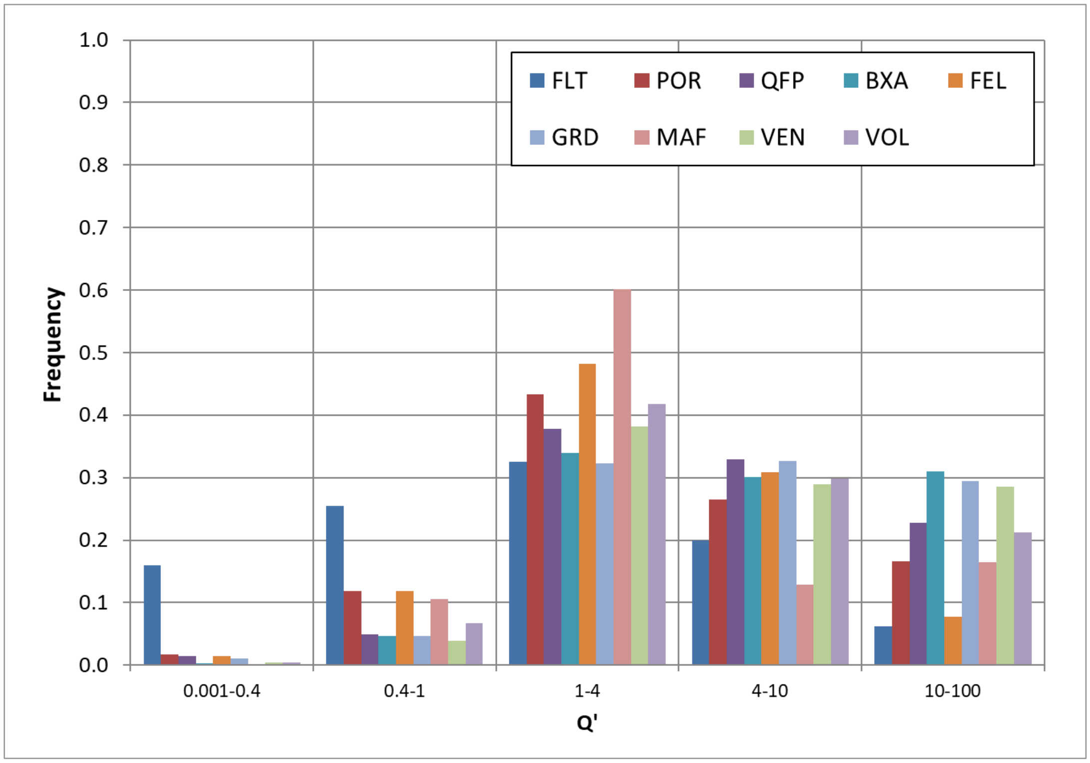
Figure 16.5 RMR76 by lithological domain (average +/- standard deviation)



Source: AMC Mining Consultants (Canada) Ltd.

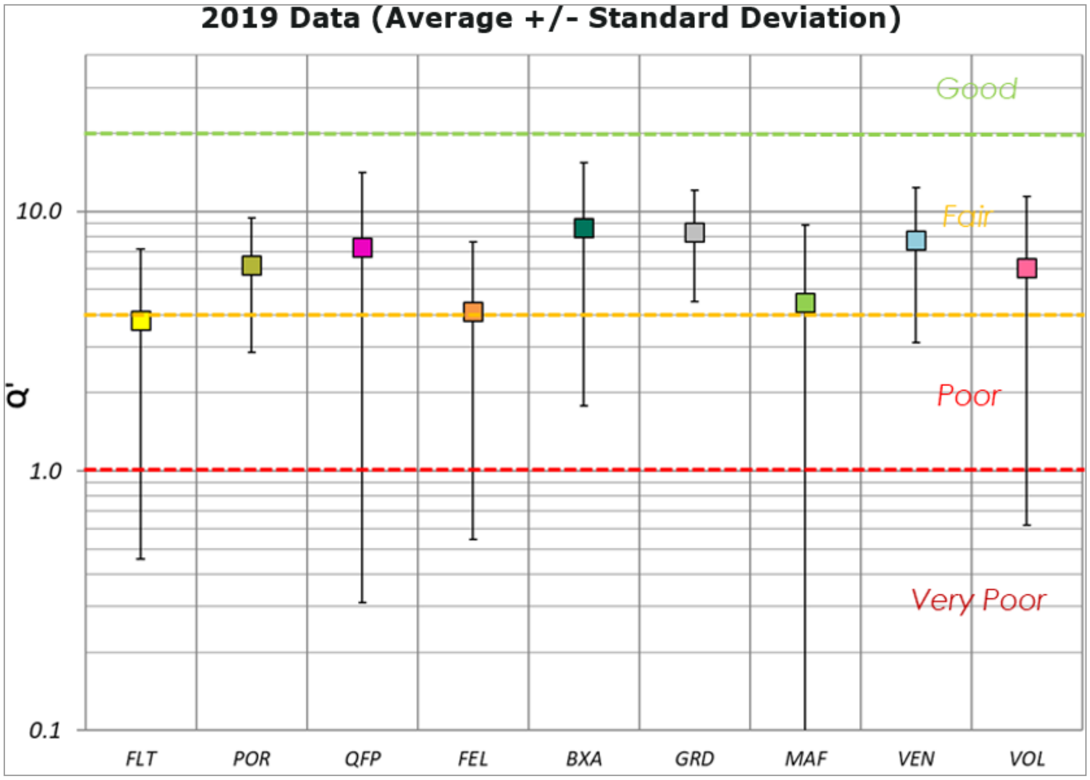
The Q′ frequency distribution histogram and average Q′ (with +/- standard deviation) for main lithologies are presented in Figure 16.6 and Figure 16.7, respectively. Based on the Q system, the rock mass quality of major lithologies are classified as Poor to Fair.

Figure 16.6 Q’ frequencies by lithological domain



Source: AMC Mining Consultants (Canada) Ltd.

Figure 16.7 Q’ by lithological domain (average +/- standard deviation)



Source: AMC Mining Consultants (Canada) Ltd.

16.2.4 Open pit geotechnical considerations

16.2.4.1 Open pit slope design

A number of slope stability analyses were carried out for the proposed open pits:

- Empirical analysis.
- 2D limit equilibrium (LE) analysis using Slide v6 (Rocscience 2015).
- Probabilistic kinematic assessment.

The design criteria adopted were based on factor of safety (FOS) and probability of failure (POF) calculations (Read & Stacey 2009). Typical values of FOS used in pit design are indicated in Table 16.3.

Table 16.3 Typical FOS and POF acceptance criteria values

Slope scale	Consequence of failure[2]	Acceptance criteria[1]		
		FOS (min) (static)	FOS (min) (dynamic)	POF (max) P (FOS </= 1)
Bench	Low-high	1.1	NA	25 – 50%
Inter-ramp	Low	1.15 – 1.2	1.0	25%
	Medium	1.2	1.0	20%
	High	1.2 – 1.3	1.1	10%
Overall	Low	1.2 – 1.3	1.0	15 – 20%
	Medium	1.3	1.05	5 – 10%
	High	1.3 – 1.5	1.1	$\leq$5%

Notes:
[1] Needs to meet all acceptance criteria.
[2] Semi-quantitatively evaluated.
Source: AMC Mining Consultants (Canada) Ltd.

16.2.4.2 Empirical analysis

An initial assessment of slope design parameters can be obtained from an empirical analysis based on rock mass characterization data. The benefit of using empirical methods is that they are based on experience gained from real cases of stable and failed slopes, with available information on rock mass quality obtained from the rock mass classification systems.

An empirical slope design technique commonly used in the mining industry is the Haines-Terbrugge approach (Haines and Terbrugge 1991). The analysis is based on MRMR rock mass classification system (Laubscher 1990), which is a modification of RMR76 (Bieniawski 1976) with adjustment factors for joint orientation, weathering potential, blast damage, presence of water, and stress effects. The adjustments are empirical with the values obtained from multiple observations in the field. The estimated MRMR values for the host rock (GRD) and the mineralization (QFP) of the Property are presented in Table 16.4. Estimated slope heights for these geotechnical domains are also provided.

Table 16.4 MRMR and slope height by geotechnical domain

Domain	RMRav	Weathering	Blasting	Joints	Water	Stress	MRMRav	Slope height, m
GRD	55	0.96	0.94	0.8	0.9	1	41	100
QFP	52	0.96	0.94	0.8	0.9	1	39	100

Source: AMC Mining Consultants (Canada) Ltd.

The summary values can be plotted on the Haines-Terbrugge chart to determine the inter ramp slope angles, which are summarized in Table 16.5.

Table 16.5 Summary of open pit slope angles based on empirical analyses

Domain	**Haines-Terbrugge**	
	FOS=1.5	**FOS=1.2**
GRD	44	54
QFP	43	53

Source: AMC Mining Consultants (Canada) Ltd.

16.2.4.3 Limit equilibrium analysis

A series of LE analyses by the "method of slices" was carried out for various slope geometries using Slide software (Rocscience 2015). The method of slices involves discretizing the slope geometry into a number of slices. Multiple iterations are carried out analyzing different failure surfaces. For each iteration, force components are calculated based on the slice geometry above the failure surface, as well as resultant forces from interaction with adjacent slices. A FOS is calculated for each failure surface as follows:

$$FOS = \frac{\sum Resisting\ Forces}{\sum Driving\ Forces}$$

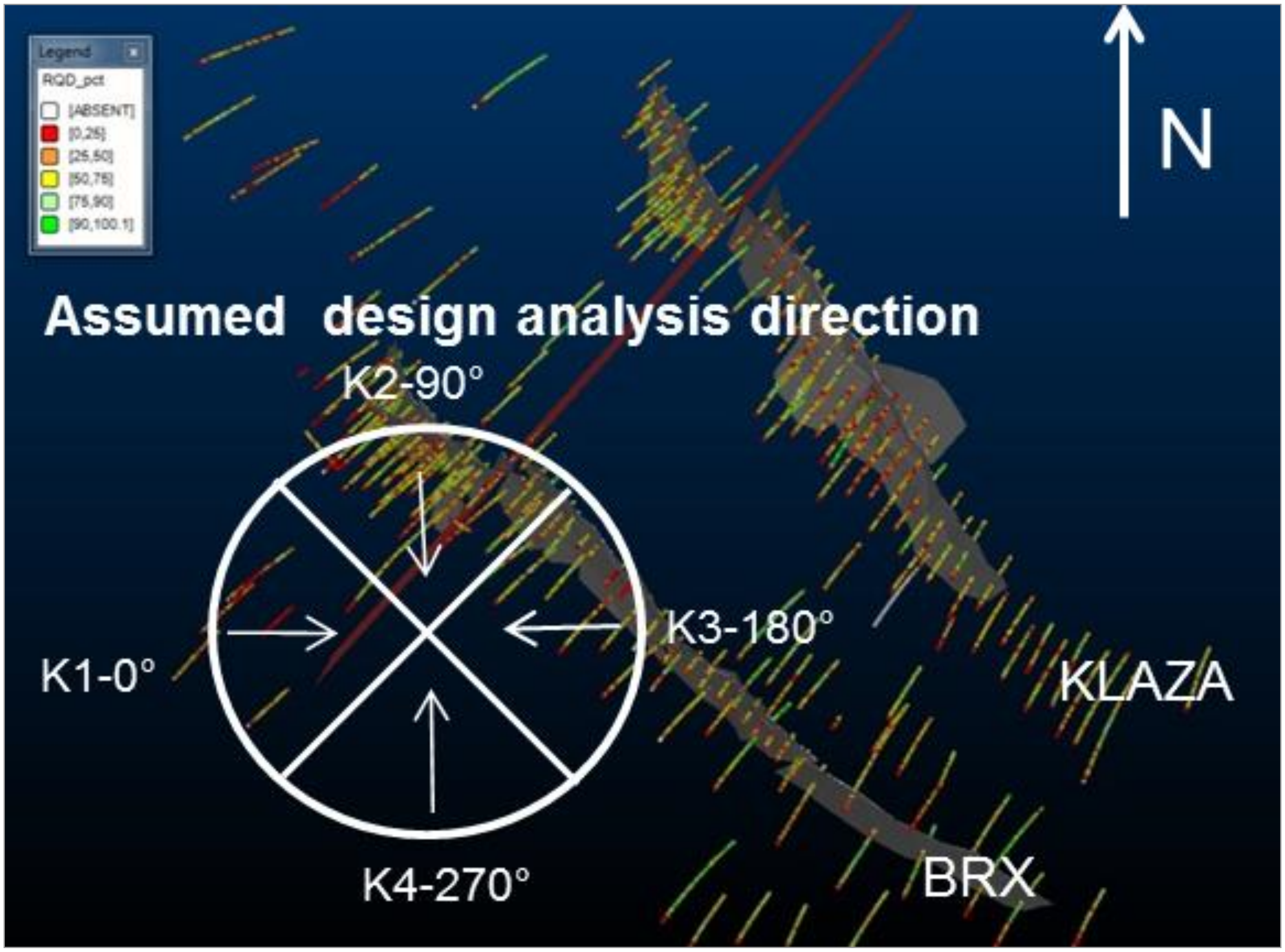
where the resisting forces are related to the rock mass strength components resisting failure, and driving forces are related to the weight of the slices and resultant forces promoting failure.

In the LE modelling, the intact rock uniaxial compressive strength (UCS) is taken as 35 MPa and the GSI of rock mass is taken as 55 MPa which were assumed based on the geotechnical data. The modelling results show that the FOS for overall slope angles (45° to 55°) is in a range of 3.4 to 2.8.

16.2.4.4 Kinematic analysis

Kinematic analyses were carried out for open pits to assess the potential for planar, wedge, and toppling failure. The preliminary pit shells were used as the basis for sectoring. The pits were sub-divided into the design sectors assumed on the four wall orientations (Figure 16.8).

Figure 16.8 Design sectors of open pit for kinematic analysis



Source: AMC Mining Consultants (Canada) Ltd.

Several assumptions were required for the kinematics:

- The slopes were assumed dry – without water pressure acting on the discontinuity planes.
- The role of tension cracking was ignored.
- The discontinuity planes are assumed persistent (continuous) frictional surfaces.
- The role of cohesion and rock bridging was ignored.
- Friction angle Ø = 30° for the probabilistic analyses.

Probabilistic kinematic assessments were carried out for each design sector to assess the potential of the various failure mechanisms. The probabilistic assessment used the measured discontinuity data per feature to assess the cumulative probability of planar or wedge failure.

The POF can be related to the cumulative frequency (CF) in the kinematic assessments, whereby POF = 50% would correspond to CF = 50% (on a bench scale the acceptable POF is in the range of 25% to 50%). However it should be noted that the CF is based on kinematic analyses by individual pit sector, and does not consider the overall likelihood of failure based on other rock mass factors such as major structure or low rock mass strength.

The wall angles associated with the CF of 20% and 50% are given in the kinematic tables below. The Bench Face Angle (BFA) and Inter-ramp Angle (IRA) criteria are based on the acceptable maximum POF of 50% (BFA) and 20% (IRA). The IRA criteria in conjunction with a "High" likelihood of kinematic instabilities will be the control for the overall slope design. The BFA criteria will also require "constructability" considerations. The results of the analysis for each sector (K1 to K4) are provided in Table 16.6 and Table 16.7 for BRX and Klaza respectively.

Table 16.6 Probabilistic kinematic assessment results for open pit (BRX)

	BFA (°)		IRA (°)	
	Low	**High**	**Low**	**High**
K1	54	75	42	48
K2	60	68	46	50
K3	54	64	39	50
K4	53	61	41	46

Source: AMC Mining Consultants (Canada) Ltd.

Table 16.7 Probabilistic kinematic assessment results for open pit (Klaza)

	BFA (°)		IRA (°)	
	Low	**High**	**Low**	**High**
K1	59	74	46	50
K2	59	69	44	50
K3	52	64	39	46
K4	55	62	39	46

Source: AMC Mining Consultants (Canada) Ltd.

16.2.5 Pit wall design parameters

Pit slope design recommendations are shown in Table 16.8 and Table 16.9 for BRX and Klaza respectively. The design parameters are based on the probabilistic kinematic analyses. The bench height was limited to 10 m. The berm width of 6.0 m is considered to be sufficient to ensure retention of bench-scale size failures. BFA exceeds the kinematic design acceptance (POF) criteria for all cases. Some bench overbreak is expected. Therefore the design strategy will be to maintain adequate catchment by maximizing berm width.

Identification and characterization of geological structure is one deficiency in the current design that should be addressed in future work.

Table 16.8 Open pit design parameters for BRX

	BFA (°)		IRA (°)		Bench height (m)	Bench width (m)
	Weathered	**Fresh**	**Weathered**	**Fresh**		
K1	60	75	39	48	10	6.5
K2	60	70	39	50	10	6.5
K3	60	65	39	50	10	6.5
K4	60	65	39	46	10	6.5

Source: AMC Mining Consultants (Canada) Ltd.

Table 16.9 Open pit design parameters for Klaza

	BFA (°)		IRA (°)		Bench height (m)	Bench width (m)
	Weathered	Fresh	Weathered	Fresh		
K1	60	75	39	50	10	6.5
K2	60	70	39	50	10	6.5
K3	60	65	39	46	10	6.5
K4	60	65	39	46	10	6.5

Source: AMC Mining Consultants (Canada) Ltd.

16.2.6 Underground geotechnical considerations

16.2.6.1 Stable stope spans

AMC reviewed and updated the stope stability design based on the updated mine design and updated rock mass classification data using the same empirical stability graph method (ESGM, after Potvin 1988, Nickson 1992, and Hadjigeorgiou et. al. 1995). This approach is widely practiced in Australia and North America to obtain a first-pass estimate of stable stope spans when little or no local stoping experience is available. Stability of each stope face is assessed separately.

Under the ESGM, the stability number N' is calculated using the following expression:

$$N' = Q' \times A \times B \times C$$

Where:

A is rock stress factor, determined by the ratio of maximum induced stress of stope face to the intact rock UCS (taken as 45 MPa); the maximum stress in the back was estimated by doubling the pre-mining horizontal stress perpendicular to the ore strike proposed by Yves and Hadjigeorgiou (2001). As no in situ stress measurement is undertaken, a low to moderate in situ stress regime is assumed. (K_0 = 2); For stope walls, *A* is assumed to be 1, indicating wall relaxation;

B is joint orientation factor, determined based on the orientation of dominant joints relative to the stope surface;

C is gravity adjustment factor, determined based on dip of stope face.

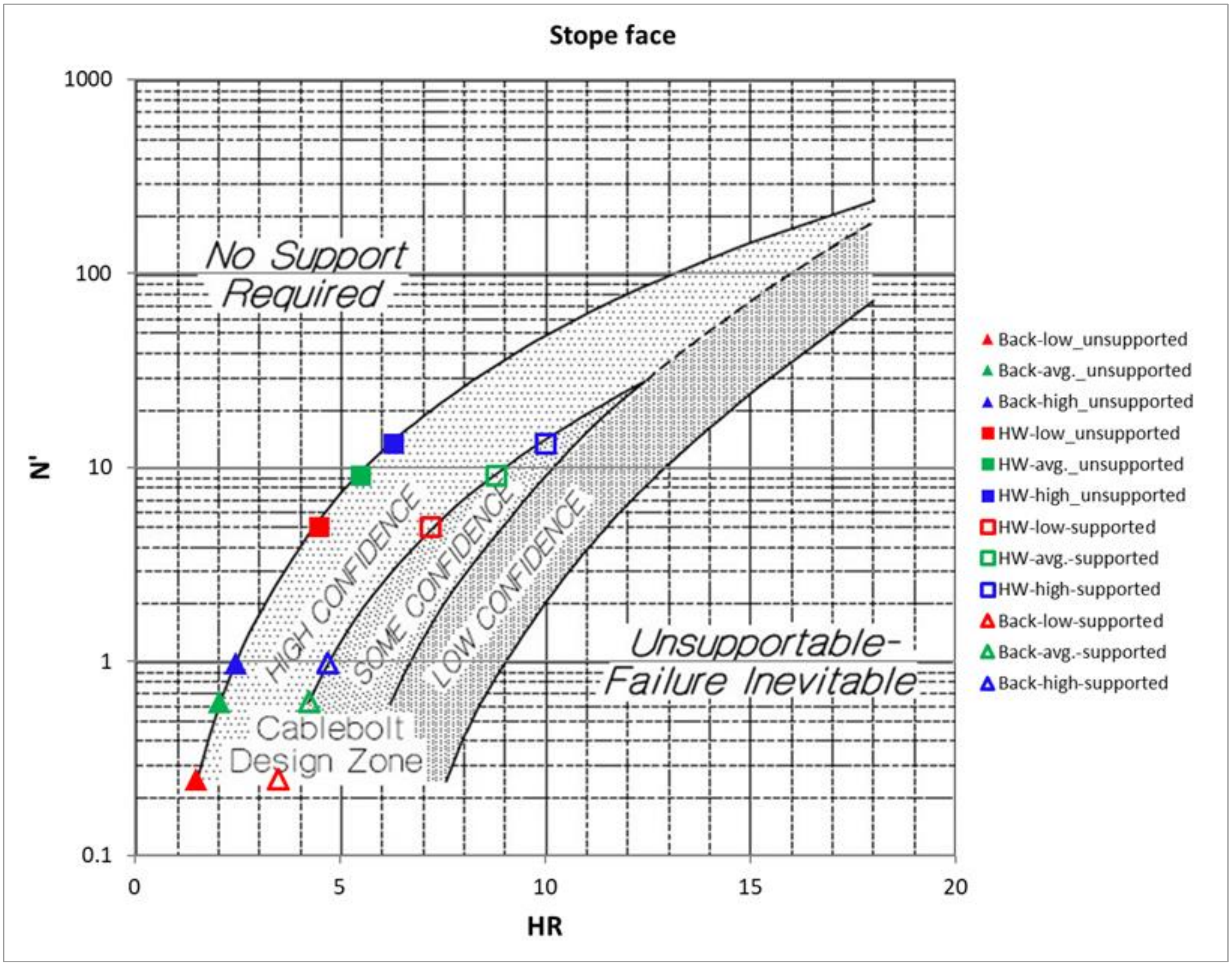
The hydraulic radius (*HR*) of a stope face is given by:

$$HR = \frac{area}{perimeter} = \frac{w \times l}{2(w + l)}$$

Where *w* and *l* are surface width and surface length, respectively.

Based on the stability number of a stope face (HW / FW and back), the hydraulic radius of a stable stope face with and without support, can be determined from the stability graph. Figure 16.9 shows the stable HR limits with and without support of an open stope (25 mH X 4 mW, HW dipping 65°) for anticipated rock mass conditions (upper, typical, and lower bound).

Figure 16.9 Stability graph results for the stope face (vein dip - 65°)



Source: AMC Mining Consultants (Canada) Ltd.

Based on the *HR* limits, the maximum stable unsupported and supported strike lengths for both HWs and backs have been projected for an open stope with a sublevel interval of 25 m and a width of 4 m. Table 16.10 summarizes the HR design limits and associated maximum strike length for both unsupported and supported cases for rock mass conditions anticipated.

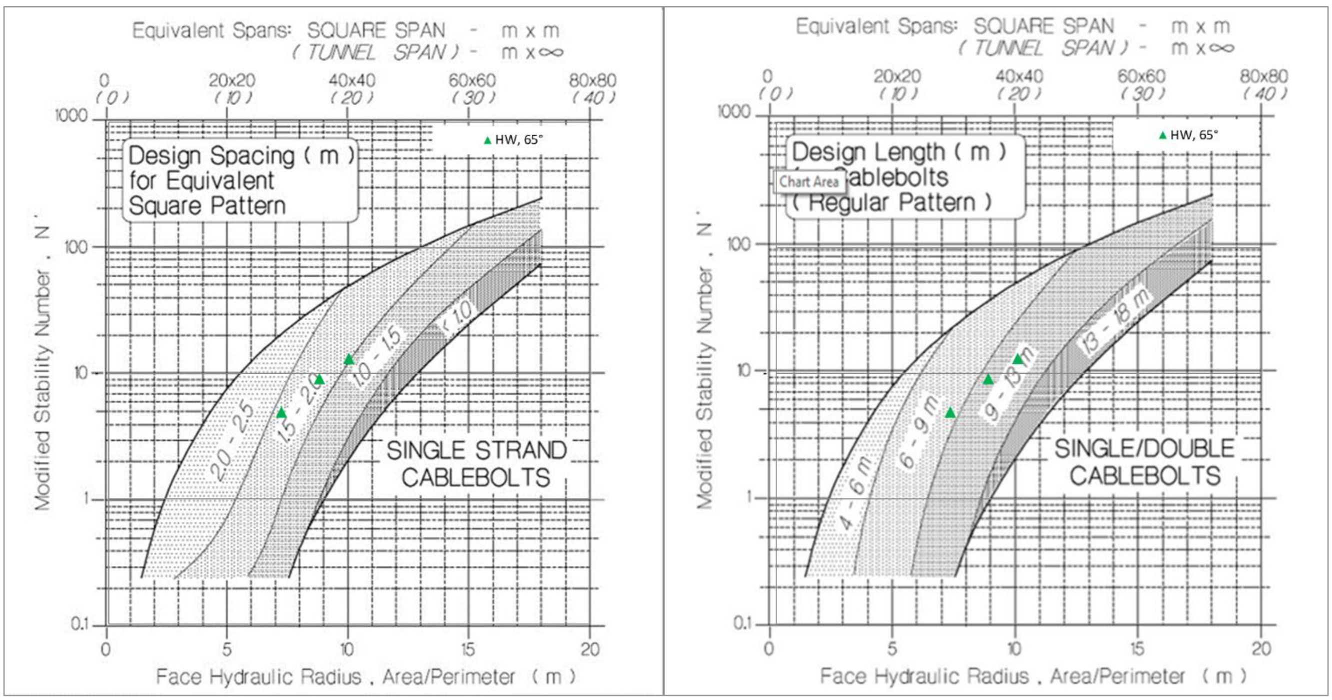
Table 16.10 Design limits for a stable open stope

Rock mass quality	Stope face	Q'	A	B	C	N'	Unsupported		Supported	
							HR (m)	Max strike length (m)	HR (m)	Max strike length (m)
Upper	HW, 65°	12.1	1	0.2	5.5	13.2	6.3	23	10.0	72
	Back, 0°	12.3	0.1	0.4	2.0	1.1	2.5	Infinite	4.9	Infinite
Typical	HW, 65°	8.3	1	0.2	5.5	9.1	5.5	18	8.8	48
	Back, 0°	7.7	0.1	0.4	2.0	0.6	2.1	Infinite	4.2	Infinite
Lower	HW, 65°	4.5	1	0.2	5.5	4.9	4.5	13	7.2	30
	Back, 0°	3.1	0.1	0.4	2.0	0.2	1.5	12	3.0	Infinite

The results are for the HR threshold that AMC considers is applicable for this level of assessment. The assessment suggests that for typical rock mass conditions, stope walls with 18 m unsupported strike length stope or 48 m supported strike length, and backs at typical envisaged width (3 m to 4) would be stable at projected stope heights (25 m) and 65˚ dip.

Recommended cable bolt spacing and length can also be derived from empirical data (Hutchinson 1996). Figure 16.10 shows recommended spacing and minimum lengths of cable bolts for HWs at their design limits (maximum strike lengths) for anticipated rock mass conditions (lower, typical, and upper bound). The suggested range of cablebolt length is 9 m to 13 m, and the spacing should be a minimum of 1.5 m to 2.0 m.

Figure 16.10 Recommended spacing and minimum length for cable bolts

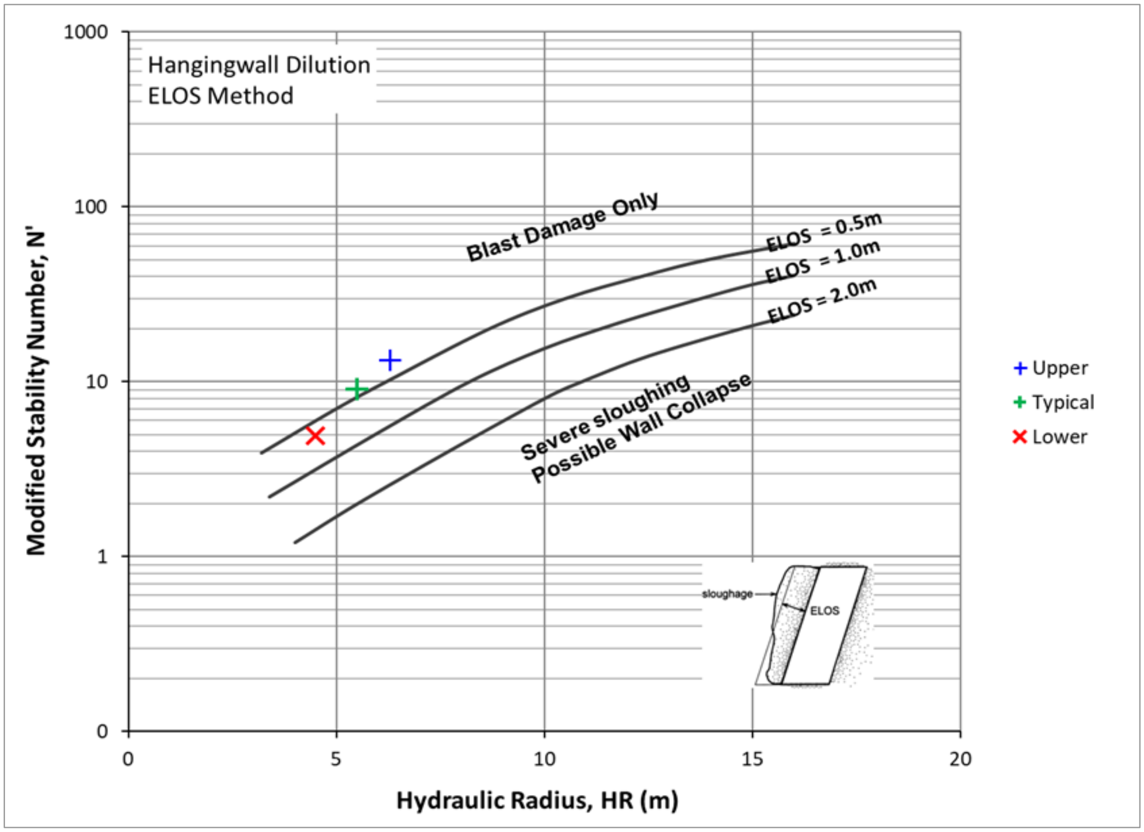


Source: AMC Mining Consultants (Canada) Ltd.

16.2.6.2 Stope dilution estimation

A preliminary estimate of dilution for longhole open stoping (LHOS) was made using the equivalent linear overbreak slough (ELOS) technique (Clark and Pakalnis 1997). This empirical method estimates the overbreak based on recorded case histories and established design curves, relating the modified stability number N’ (vertical axis) and the hydraulic radius (horizontal axis). Figure 16.11 shows the ELOS estimation of stope walls at their unsupport design limits (Table 16.10) for anticipated rock mass conditions; indicating a less than 0.5 m of ELOS for typical and better rock mass conditions, and slightly higher ELOS for poorer rock mass condition. It should be noted that the stability graph method is approximate in ELOS estimate. Early stoping should be carefully monitored, and designs should be adjusted in response to actual performance.

Figure 16.11 ELOS estimation of stope walls at the maximum unsupported strike lengths



Source: AMC Mining Consultants (Canada) Ltd.

16.2.6.3 Preliminary ground support estimates

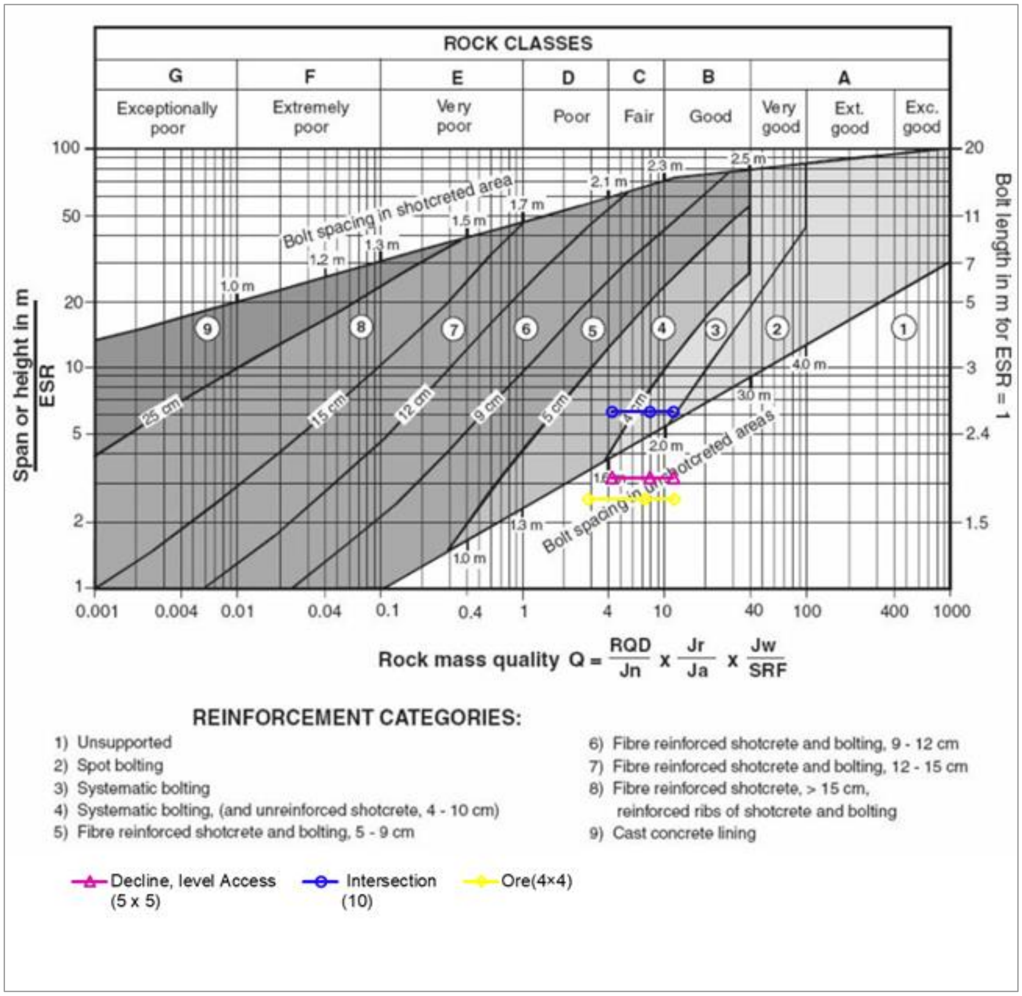
Preliminary ground support requirements have been assessed using the Q-system chart for tunnel support guideline (Grimstad and Barton 1993), which relates rock quality, excavation span, and service life to support requirements. The method converts the span (width / height) of the excavation to an equivalent dimension, given by:

$$D_e = \frac{Span}{ESR},$$

Where ESR is excavation support ratio, which takes into account the function and service life of the excavation.

The required ground support for anticipated rock mass conditions are presented in Figure 16.12. As seen from the chart, all excavations are falling in the 'unsupported' category except intersection area. Based on state of the art industry practices, such ground conditions will require minimum ground support installations. The proposed ground support for lateral development is presented in Table 16.11

Figure 16.12 Ground support requirment based on Q-System chart



Source: Grimstad and Barton 1993.

Table 16.11 Proposed primary ground support for permanent lateral development

Support class (SC)	Ground support requirements with span less than 6 m
Q 10 - 40	- Minimum 9 Ga wire mesh or chain-link equivalent to springline - 2.4 m long #6 fully encapsulate rebar bolt or equivalent bolt on 2 x 2 m spacing in back, - 1.8 m long #6 fully encapsulate rebar bolt or equivalent bolt in walls as required
Q 4 – 10	- Minimum 9 Ga wire mesh and six 1.2 m long split sets at the face - Minimum 9 Ga wire mesh or chain-link equivalent to 1.8 m above sill - 2.4 m long #6 fully encapsulate rebar bolt or equivalent bolt on 1.5 x 1.5 m spacing in back - 1.8 m long #6 fully encapsulate rebar bolt or equivalent bolt in walls as required
Q 1 – 4	- Minimum 9 Ga wire mesh and six 1.2 m long split sets at the face - Minimum 9 Ga wire mesh or chain-link equivalent to 1.8 m above sill - 2.4 m long #6 fully encapsulate rebar bolt or equivalent bolt on 1.2 x 1.2 m spacing in back - 1.8 m long #6 fully encapsulate rebar bolt or equivalent bolt in walls as required
Q < 1	- Minimum 9 Ga wire mesh and six 1.2 m long split sets at the face - 2-3" shotcrete, full coverage - Minimum 6 Ga wire mesh or chain-link equivalent to 1.5 m above sill - 2.4 m long #6 fully encapsulate rebar bolt or equivalent bolt on 1.2 x 1.2 m spacing in back and wall

Secondary ground support would be required for large spans (up to 10 m) such as intersections. For an intersection of 10 m span, minimum 6 m long 0.6 inch to 0.7 inch (15 mm to 18 mm) twin-strand bulbed cablebolts on 1.5 m to 2.5 m square pattern or an approved equivalent would be required.

For rock mass conditions of 'Very Poor' or 'Extremely Poor', spiling as pre-support and other excavation and support measures may be required for long term stability, the ultimate requirements must be determined on-site and in consideration of actual ground conditions encountered.

16.3 Mineral Resource model for mining

AMC used the AMC 2018 estimate block model for the Klaza and BRX zones (reference: klaclamd_w_pit) for evaluation of mining potential. The data used in the evaluation includes results of all drilling carried out on the Property to December 2017. The evaluation work was carried out in Datamine™ software.

16.4 Cut-off value

The cut-off value is based on NSR value, which accounts for all downstream processing costs. A net payable recovery for each metal was determined by H. M. Hamilton & Associates Inc. (HMH); the payable recovery is based on marketing research that takes into account likely smelter terms and penalties, transport, treatment and refining costs. AMC considers the assumptions used for determining net payable recovery to be reasonable. The NSR cut-off value is based on the assumptions shown in Table 16.12.

Table 16.12 NSR cut-off value assumptions

Mining factors	Unit	Open pit	Underground
Mining dilution Klaza	%	25	17
Mining dilution BRX	%	25	17
Mining recovery	%	95	90
Operating costs			
Mining cost	C$/t	4.50	58.40
Fixed & throughput processing cost (BRX and Klaza)	C$/t	50.03	50.03
BRX and Klaza G&A	C$/t	12	12
Processing recovery			
Gold	%	96	96
Silver	%	91	91
Lead	%	85	85
Zinc	%	85	85
Lead concentrate grade	%	60	60
Zinc concentrate grade	%	48	48
Revenue			
% Payable gold*	%	97.0	97.0
% Payable silver*	%	81.0	81.0
% Payable Lead*	%	62.0	62.0
% Payable zinc*	%	52.0	52.0
Exchange rate	C$/US$	1.389	1.389
Gold price	US$/oz	1,450.00	1,450.00
Silver price	US$/oz	17.00	17.00
Lead price	US$/lb	0.95	0.95
Zinc price	US$/lb	1.0	1.0
BRX and Klaza NSR cut-off value	C$/tonne	62.03[1]	120

Notes:
* Payable gold, silver, lead, and zinc take into account all downstream costs for transport, port handling, ocean freight, and treatment charges.
[1] Open pit marginal cut-off excludes mining cost.
Source: AMC Mining Consultants (Canada) Ltd.

16.5 Open pit

16.5.1 Dilution and mining recovery factors

The thickness of the mineralized veins generally varies between 1.0 m and 3.7 m with an average within the pit shells estimated at 2.3 m. However, the mineralization is visually distinguishable from the surrounding waste rock, which will help in dilution control during excavation. Mining dilution was estimated by evaluating the thickness of the veins within the proposed open pits and assuming a 50 cm dilution skin, representative of blast movement mixing and the selectivity of a 80 t excavator. The analysis resulted in an overall estimated dilution of 30% for BRX and 28% for Klaza. A mining recovery factor of 95% was applied. The mining dilution and recovery were applied as factors during the pit optimization process and to estimate mill feed tonnes in the schedule. The dilution material is assumed to have zero value. Dilution and recovery factors are summarized in Table 16.14.

Table 16.13 Open pit mining dilution and recovery

	Mining dilution (%)	Mining recovery (%)
BRX	30	95
Klaza	28	95

Source: AMC Mining Consultants (Canada) Ltd.

16.5.2 Pit optimization and selection

The Lerchs-Grossmann pit optimization algorithm was used to define the ultimate pit shell for the BRX and Klaza zones. The selected pit shells were then used to produce pit designs and the open pit mining schedule.

16.5.3 Open pit to underground interface

In order to optimize the open pit to underground interface, AMC compared the incremental value of sucessive shells to the cost of underground mining. As the shells get bigger, the stripping ratio increases and at some point it becomes more cost effective to take the ore by undergound mining. The shell where this cross-over transition occurs is not as big as the revenue factor (RF) 1 shell. If there was no underground mining, then the pit could expand further (to the RF 1 shell). Table 16.14 provides the Whittle results for the Klaza main zone.

Table 16.14 Klaza pit optimization results

Pit shell	Revenue factor	Cashflow undiscounted ($M)	Processed tonne (kt)	Waste tonnage (kt)	Total tonnes (kt)	Strip ratio	Grade input Au (g/t)	Grade input Ag (g/t)	Value input NSR ($/t)	Incremental mining cost ($/t of ore)	Incremental value ($/t of ore)	Cumulative mining cost ($/t of ore)
11	0.50	54.3	338	1,758	2,096	5.2	3.23	61.42	253.79	27.0	85.1	31.3
12	0.52	57.0	369	1,952	2,322	5.3	3.17	59.31	248.36	36.8	90.1	31.8
13	0.54	63.1	445	2,407	2,852	5.4	3.02	55.91	236.16	35.4	79.5	32.4
14	0.56	66.5	503	2,610	3,113	5.2	2.90	52.73	225.55	22.9	58.7	31.3
15	0.58	68.8	530	2,880	3,410	5.4	2.88	53.25	224.55	56.5	87.4	32.6
16	0.60	71.1	560	3,168	3,728	5.7	2.87	52.58	222.90	54.3	77.1	33.8
17	0.62	78.7	679	4,041	4,720	6.0	2.77	48.31	213.07	41.6	63.4	35.1
18	0.64	85.3	789	5,055	5,845	6.4	2.70	47.14	207.58	51.9	59.9	37.5
19	0.66	92.4	898	6,333	7,231	7.1	2.70	45.02	205.76	65.0	65.4	40.8
20	0.68	97.1	976	7,237	8,213	7.4	2.67	44.64	204.19	64.1	60.2	42.7
21	0.70	101.7	1,061	8,223	9,284	7.7	2.66	43.39	202.16	63.7	53.2	44.4
22	0.72	102.3	1,073	8,411	9,484	7.8	2.66	43.55	202.29	92.7	60.4	44.9
23	0.74	103.6	1,109	8,673	9,782	7.8	2.64	42.69	200.19	41.5	34.9	44.7
24	0.76	107.2	1,154	10,093	11,246	8.7	2.68	46.81	204.63	173	79.8	49.7
25	0.78	113.6	1,321	12,195	13,516	9.2	2.64	43.94	200.19	69.2	38.3	52.2
26	0.80	119.6	1,473	14,840	16,313	10.1	2.61	46.54	199.81	95.1	39.5	56.6
27	0.82	129.8	1,762	19,583	21,345	11.1	2.58	47.12	197.91	90.8	35.4	62.2
28	0.84	130.2	1,782	19,719	21,501	11.1	2.57	46.92	197.08	40.4	20.1	62.0
29	0.86	130.6	1,800	19,950	21,749	11.1	2.57	46.69	196.69	71.9	23.3	62.1
30	0.88	131.7	1,863	20,450	22,313	11.0	2.54	45.94	194.21	45.5	16.2	61.5
31	0.90	134.1	2,019	22,668	24,687	11.2	2.52	43.67	191.12	76.4	15.7	62.7
32	0.92	136.1	2,137	25,199	27,337	11.8	2.52	43.50	191.40	117.2	17.0	65.7
33	0.94	136.2	2,148	25,373	27,521	11.8	2.52	43.66	191.26	89.9	11.2	65.8
34	0.96	136.6	2,214	25,939	28,153	11.7	2.49	43.07	189.00	48.0	5.3	65.3
35	0.98	136.7	2,242	26,198	28,440	11.7	2.48	43.12	188.09	51.2	3.8	65.1
36	1.00	136.8	2,295	26,935	29,230	11.7	2.46	42.96	187.00	77.7	1.2	65.4
37	1.02	136.7	2,322	27,276	29,598	11.7	2.46	42.69	186.37	70.7	-1.7	65.4

Up to shell 24 the incremental mining cost to produce a tonne of ore is generally less than the underground mining cost of approximately $65/t. At the RF1 shell (#36), the cumulative open pit mining cost is $65.4/t ore produced and the incremental shells are over $70/t ore. If there was no underground mining alternative, shell 36 would be an appropriate shell to choose for the ultimate pit. In the present case, for Klaza, shell 24 was selected as the transition point to end open pit mining and commence underground mining.

The Whittle results for BRX are shown in Table 16.15.

Table 16.15 BRX pit optimization results

Shell	Factor	Undiscounted ($M)	Processed tonne (kt)	Waste tonnage (kt)	Total tonnes (kt)	Strip ratio	Input Au (g/t)	Input Ag (g/t)	Input NSR ($/t)	Mining cost ($/t of ore)	Value ($/t of ore)	Mining cost ($/t of ore)
1	0.30	156.6	362	4,608	4,970	12.7	8.46	72.36	569.24			74.8
2	0.32	188.3	465	6,145	6,610	13.2	8.06	71.51	544.46	87.8	307.7	77.6
3	0.34	197.4	499	6,629	7,129	13.3	7.91	71.37	535.20	82.3	264.9	78.0
4	0.36	209.4	541	7,380	7,921	13.6	7.80	72.28	529.30	106.4	289.8	80.2
5	0.38	224.8	606	8,378	8,984	13.8	7.58	70.63	514.45	90.3	238.0	81.2
6	0.40	226.0	614	8,429	9,043	13.7	7.51	71.64	510.97	36.0	151.7	80.6
7	0.42	250.2	711	10,476	11,187	14.7	7.33	71.18	500.18	121.2	248.8	86.2
8	0.44	255.7	741	10,873	11,614	14.7	7.19	72.88	492.70	74.1	180.4	85.7
9	0.46	258.1	757	11,073	11,830	14.6	7.11	73.76	488.65	73.1	159.6	85.4
10	0.48	261.6	780	11,397	12,177	14.6	7.00	75.35	482.80	78.7	151.5	85.2
11	0.50	274.4	854	12,730	13,584	14.9	6.78	76.33	470.15	103.5	171.5	86.8
12	0.52	276.3	874	12,861	13,735	14.7	6.67	76.81	463.72	37.0	94.9	85.7
13	0.54	277.0	881	12,942	13,822	14.7	6.65	76.78	462.20	72.8	119.0	85.6
14	0.56	287.6	952	14,509	15,461	15.2	6.46	79.21	452.38	122.4	147.1	88.3
15	0.58	294.3	993	15,607	16,600	15.7	6.41	78.97	449.37	153.9	163.5	91.0
16	0.60	295.9	1,020	15,723	16,743	15.4	6.27	79.47	441.34	24.8	60.7	89.3

It can be seen that globaly at BRX, none of the shells mine for cheaper than underground mining cost of US$58.40/t milled. Therefore at first, there were no BRX pits designed. However, it was later determined that it was too costly to extend underground development to the southern end of BRX. Therefore open pit designs were prepared to slavage some surface ore from the southern most BRX areas. These selected small zones are economic as designed.

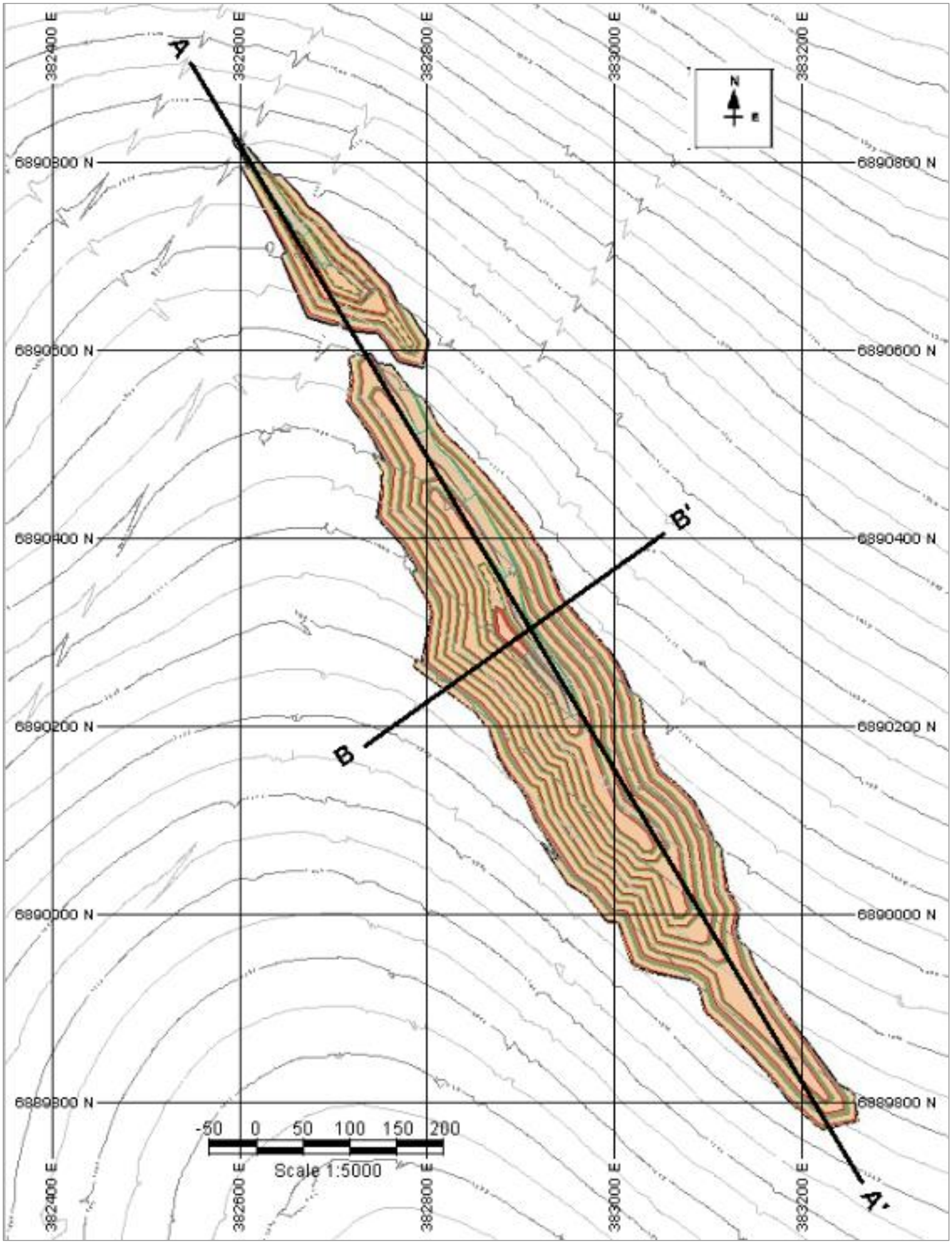
After selection of the optimal depth of the pit relative to underground stopes, a combined value is determined to confirm that the selected depth generates the maximum value. Stopes are then clipped to the pit design to determine tonnes and grade for the underground mine.

As part of the updated PEA, AMC completed a trade-off study between the cost of building the Tailings Storage Facilities (TSF) all on surface versus disposing tailings in the TSF for the first few years and then in-pit in the remaining years. The results showed minimal variance between value of the additional ore recovered from the crown pillar versus the additional capital cost of building the larger TSF on surface. However, other benefits including the ability to construct the portal at the bottom of the pit, reducing development capital cost as well as the increased recovery of the Mineral Resource, led to the decision to build the entire TSF on surface. Therefore, the crown pillars will be mined beneath the Central Klaza pit, as well as the BRX pit with an assumed 90% mining recovery. The Klaza North pit does not have a crown pillar or underground stopes beneath it.

16.5.3.1 Pit design

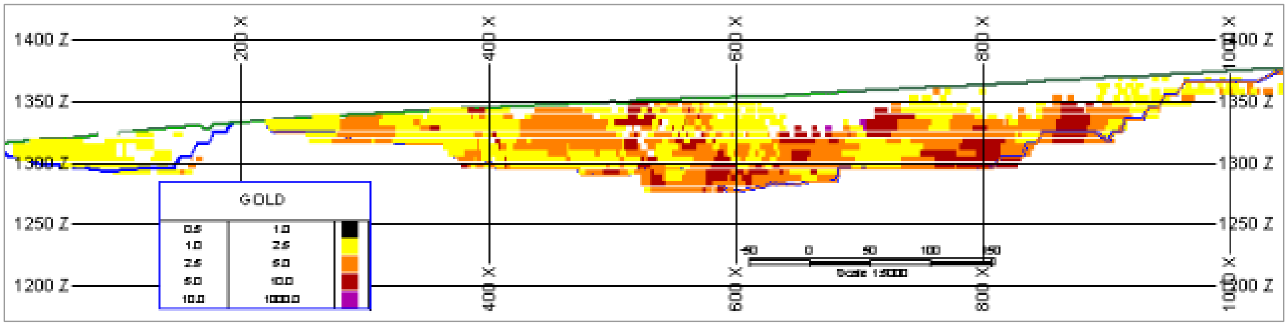
Conceptual pits were designed based on the selected pit optimization shells. Wall design conforms to the geotechnical recommendations (see Table 16.8 and Table 16.9). Haul roads are nominally 12 m wide at 10% grade with final “goodbye” cuts narrowing to 8 m and with grades to 12%. The conceptual designs for the Western BRX and Klaza zones are presented in Figure 16.13 and Figure 16.15 respectively. Representative pit design sections for Klaza are shown in Figure 16.14.

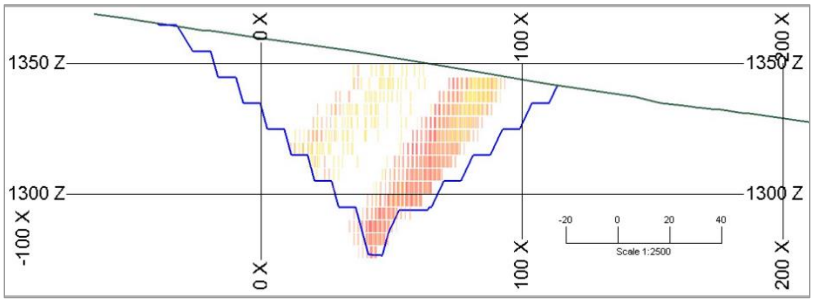
Figure 16.13 Klaza pit design



Source: AMC Mining Consultants (Canada) Ltd.

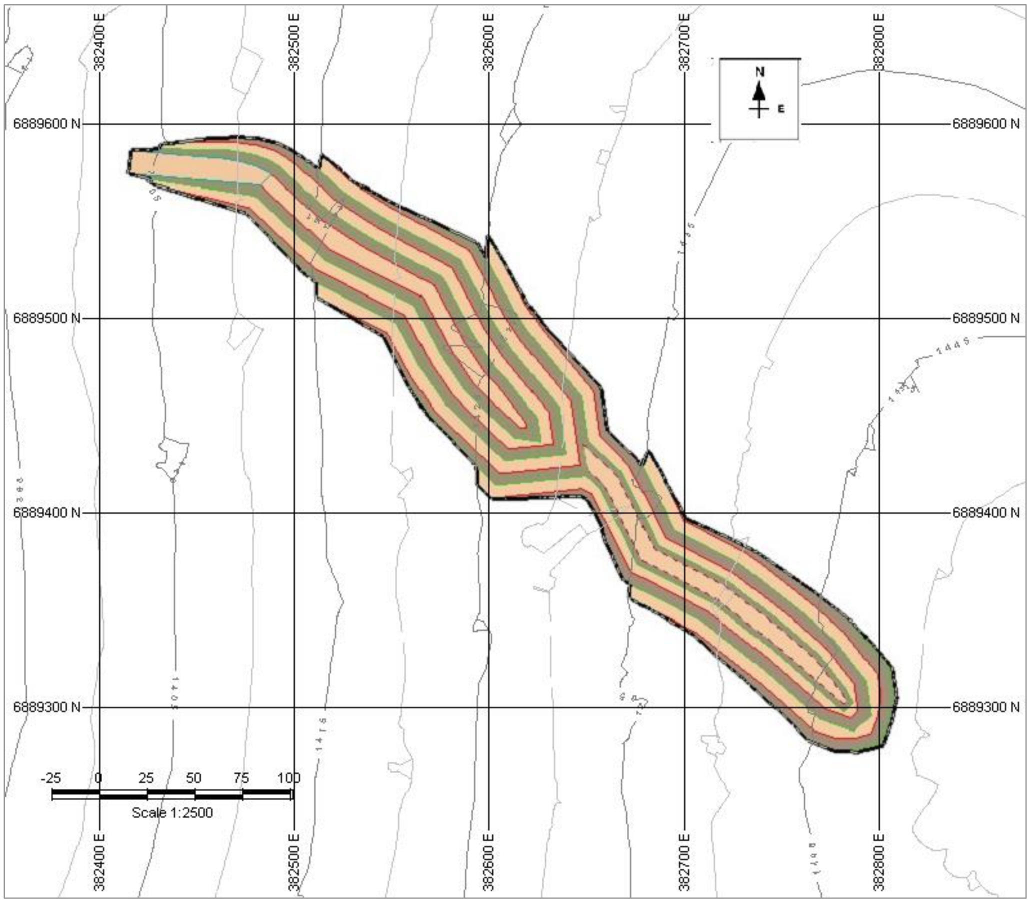
Figure 16.14 Klaza pit sections A-A′ and B-B′ with gold grade





Source: AMC Mining Consultants (Canada) Ltd.

Figure 16.15 Southern BRX pit design



Source: AMC Mining Consultants (Canada) Ltd.

Indicative tonnes and grades contained within the conceptual pit designs are presented in Table 16.16

Table 16.16 Open pit projected tonnes and grades

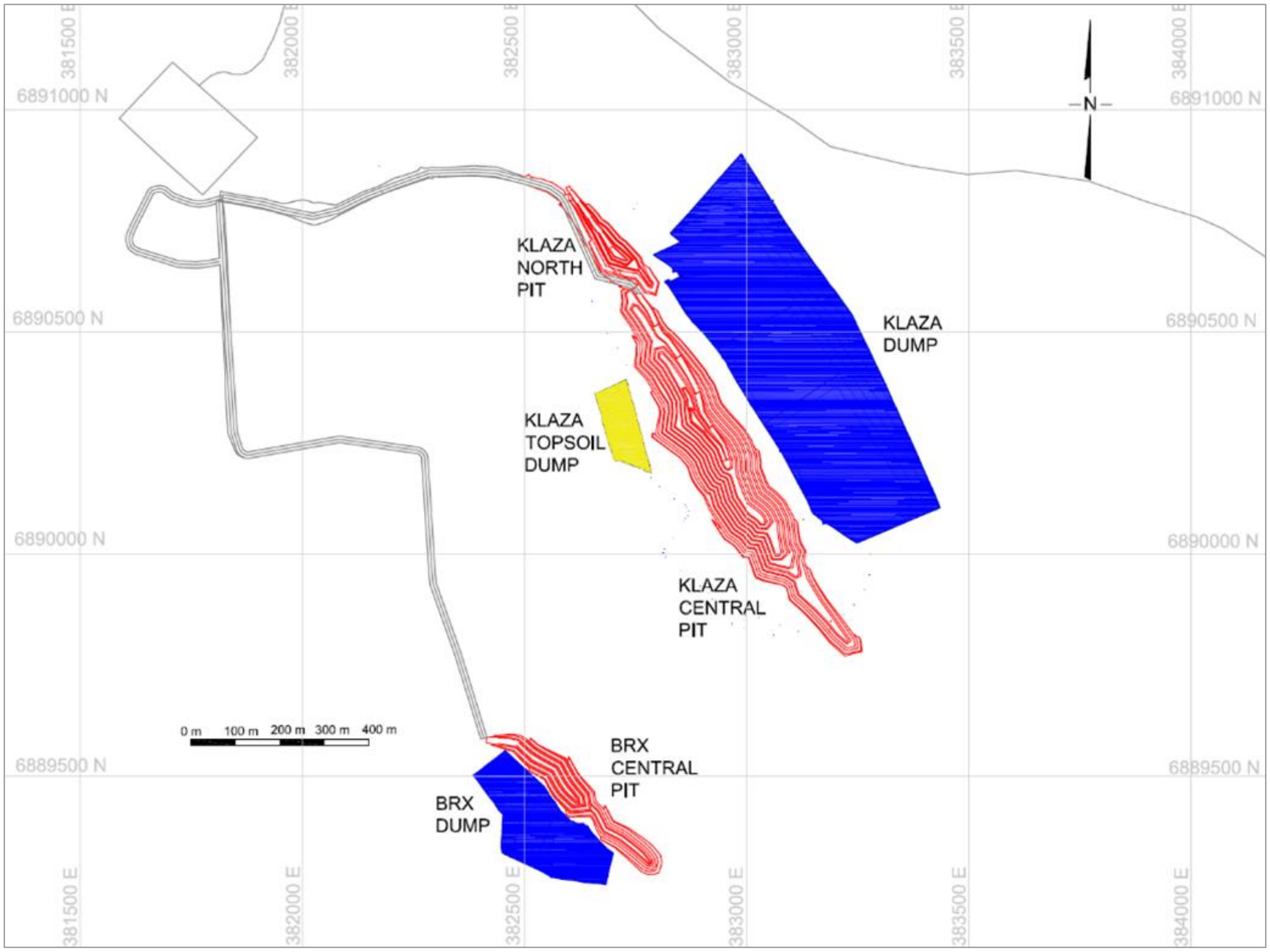
Zone	Mineralized material tonnes (Mt)	Waste material tonnes (Mt)	ADNSR (C$)	Au (g/t)	Ag (g/t)	Pb (%)	Zn (%)	Arsenic (ppm)
Klaza	1.07	7.96	182	2.5	31.6	0.4	0.7	2,759
Western BRX	0.11	0.78	141	1.5	67.9	0.4	0.6	5,200
Total	**1.18**	**8.74**	**178**	**2.4**	**35.1**	**0.4**	**0.7**	**2,994**

Source: AMC Mining Consultants (Canada) Ltd.

16.5.3.2 Layout of other open pit mining related facilities

The general site layout for BRX and Klaza is shown in Figure 16.16. Waste dumps have been designed to accommodate the totality of the waste mined from the pits; a proportion of the waste material may be used for building the tailings dam wall and backfilling underground workings. The waste dumps have been designed based on a 2:1 overall slope angle and an overall toe to crest maximum height of 80 m for the Klaza dump. Top soil stockpiles have been designed to handle the volume generated by removing 30 cm from the surface area of the pits. Connecting haul roads are maximum 10% grade and 14 m wide.

Figure 16.16 Layout of open pit infrastructure



Source: AMC Mining Consultants (Canada) Ltd.

16.5.4 Mining method

AMC proposes to mine the open pits using conventional truck and excavator mining methods. A mining contractor operation is presumed. At this stage, AMC has assumed that a 10 m bench height would be adopted and mineralized material mined in two 5 m flitches to increase mining selectivity. Due to the small pits sizes, none of the pits are phased and mining will be by a simple desending, full bench, top down sequence.

16.5.4.1 Drill and blast

The majority of the material will require blasting. Proposed drilling parameters for 10 m bench heights are presented in Table 16.17. Standard, midsized down the hole hammer drill rigs are envisioned. The rigs would be equipped with blasthole sample equipment to collect samples for grade control. Explosives would be emulsion or anfo blends. Drilling and explosive supply including loading and shooting, are assumed to be provided by contractors.

Table 16.17 Open pit drilling parameters

Parameter	Value	Unit
Bench height	10	m
Burden	4.25	m
Spacing (equilateral triangle)	4.9	m
Hole size	140	mm
Collar	2.7	m
Subdrill	1.0	m
Explosive density	1.0	kg/m^3
Rock density	2.5	kg/m^3
Powder factor	0.25	kg/t

Source: AMC Mining Consultants (Canada) Ltd.

Due to the projected short life of the open pit mines and the shallow mining depth, AMC has assumed that no presplit blasting would be required. Minimizing dilution from blast movemnet and mixing will be critical. Choked blasts and reduced powder factors creating harder digging for the excavators will need to be traded off against dilution.

Assuming 10% re-drills, 18 m/operating hour penetration rate, 72% utilization and 3,744 operating hours per year, 2 drill rigs would be required. Shortfalls in drill and blast inventory can partly be made up by adding overtime shifts (beyond the 5*10 h/day roster).

16.5.4.2 Load and haul

Up to four hydraulic excavators equipped with 5.5 m^3 buckets (similar to CAT 390 machines) would be required to mine waste and mineralized material. They would load into a fleet of 36 t road trucks (such as Mercedes Actros) or articulated dump trucks (ex. CAT 740 ADT). Waste hauls are shorter but largely up hill while ore hauls are longer but down hill. Overall, annual excavator productivity is estimated at approximatley 350 t/h and trucks at 150 t/h. Excavators have been estimated to operate just under 4,000 h/a and trucks 3,500 h/a. As with drills, periodic production shortfalls can be made up by adding overtime shifts.

16.5.4.3 Stockpile rehandling

Direct dumping into the crusher by ore trucks during mine operating hours is presumed. Crusher operations outside of the mine operating hours would be fed by a front-end loader (FEL) rehandling material from the ROM stockpile. During direct dumping shifts or when the FEL is not required at the crusher, the FEL would be availale to assist in the mine.

16.5.4.4 Ancillary equipment

Ancillary mobile equipment includes dozers, graders, water truck and pickups. This standard equipmnet is used to maintain roads and dumps and transport staff and personnel respectively.

16.5.5 Open pit equipment

The open pit contractor operations are projected to work on a five day, 10 h/day roster. Two shifts (day and night) are envisioned. Therefore 100 h/week are scheduled over 52 weeks per year for 5,200 h/a.

Based on the production schedule (see Table 16.28), roster schedule, and equipment productivity estimates, the required equipment list is as shown in Table 16.18.

Table 16.18 Equipment estimate

Equipment	Y1	Y2
Excavators	4	3
Trucks	11	7
Loader	1	1
Drills	2	2
Grader	1	1
Water truck	1	1
Dozers	3	2
Pickups	10	7

Source: AMC Mining Consultants (Canada) Ltd.

16.5.6 Open pit labour and staff

The open pit mining contractor is presumed to provide all equipment operators, maintenance workers and shift supervisors. The owners team is assumed to provide, mine engineers, geologists, survey and ore control staff. Numbers include a small supplement to account for redundancy in case of absenteeism, training etc. An estimate of approximately 80 staff and labour is estimated at peak production rates for the open pit. Open pit labour and staff, camp, other management and administration, and the underground workforce are covered in Section 16.6.10.

16.6 Underground mining

16.6.1 Underground mining method

The mining method selection criteria are based on:

- Vein geometry – Depth, shape, thickness, and plunge.
- Rock quality – Mineralized rock and host rock competency (structures, stress, and stability).
- Vein variability – Vein uniformity, continuity, and grade distribution.
- Economics – Metal recovery, value attributed to mineralization, productivity, capital and operating costs, and safety.

The BRX and Klaza zones consist of several near vertical veins averaging 0.5 m to 3 m in width. The strike length averages approximately 1,200 m for both zones and the vertical extent is approximately 450 m below surface.

The vein geometry is most suited to the following mining methods:

- Longhole or Sublevel Stoping – Medium to steep dip, competent to fair ground.
- Shrinkage – Medium to steep dip, variable ground conditions.
- Cut and Fill – Medium to steep dip, variable ground conditions, high selectivity.

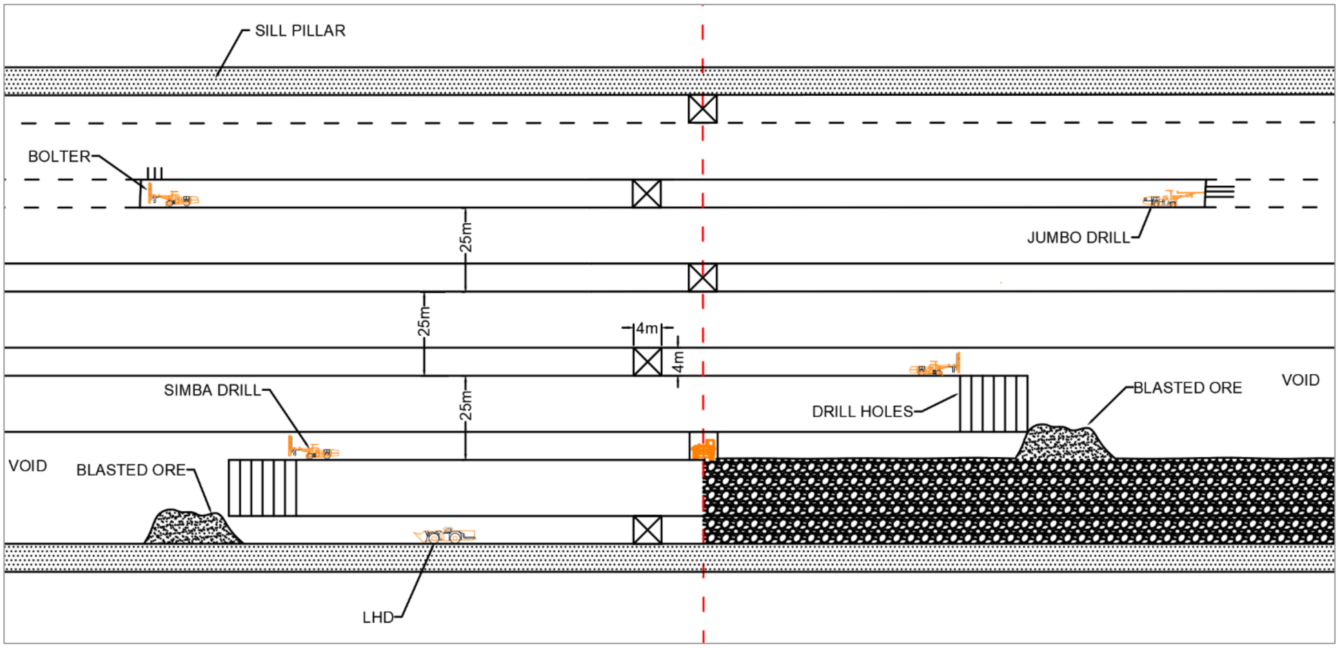
Relative to current overall knowledge of the Klaza deposits, AMC considers that the most appropriate method is mechanized LHOS. LHOS methods generally provide relatively high productivity within reasonable cost limits. It is also the most common method applied to narrow veins requiring a fair amount of selectivity, whilst maintaining reasonable production rates.

Based on geotechnical recommendations, an inter-level spacing of 25 m is selected for the longhole benches. Relative to the anticipated vein width, AMC notes that longhole drilling accuracy will be very important in mining operations. Figure 16.17 shows the general arrangement of the longhole mining method.

Access to each underground zone is via a single ramp (5 m by 5 m). Crosscuts are developed from the ramp to each level. Development along the strike of the vein is 4 m by 4 m.

Waste sourced from underground development and open pit mining will be used to fill the stopes as they are mined. A maximum open stope length of 24 m prior to filling with waste rockfill is assumed for supported stopes.

Figure 16.17 Longhole mining method



Source: AMC Mining Consultants (Canada) Ltd.

16.6.2 Dilution and mining recovery factors

Geotechnical evaluation of dilution for longhole stopes using the ELOS technique indicates less than 0.5 m from the HW. There are two main sources of dilution in narrow vein stopes:

- Planned dilution. This is the dilution required to achieve the designed stope shape. Designed dilution can result from waste included:
  - To achieve minimum mining width.
  - To achieve a viable mining shape.
- Unplanned dilution. This is dilution that is outside of the designed stope shape. Depending on the mining method, it may include both overbreak and floor dilution.
  - Overbreak is typically a result of blasting practices and geotechnical conditions.
  - Floor dilution is the result of mucking rock backfill from the floor and end wall.

AMC has applied a dilution of 0.25 m to the HW and 0.1 m to the footwall (FW), and an additional 5% of unplanned dilution for backfill mucking and / or overbreak from blasting / stope wall sloughing. A mining recovery factor of 90% has been applied to the stopes.

Stope wireframes were generated using Mine Stope Optimizer (MSO), a function of the Datamine software at a cut-off NSR value of $120 for the BRX and Klaza zones. When generating stopes, a minimum stoping width of 3 m was used. For MSO design, a dilution skin of 0.25 m on the HW and 0.1 m on the FW was added to the stope width. All dilution was assumed to have zero grade. The stope wireframes were then evaluated against the Mineral Resource block model to determine tonnes and grade. The unplanned dilution was then added to the reported tonnes and grade.

16.6.3 Production rate analysis

In order to determine an appropriate production rate that can be supported by the deposit, AMC has used a combination of Taylor`s rule of thumb and maximum vertical tonnes per metre to project production ranges.

Production rate based on Taylor's rule of thumb, is estimated at 378 ktpa for BRX and 307 ktpa for Klaza for a combined annual production rate of 685 ktpa.

$\text{Annual Production Rate} = 5 * \text{Mineable mineralization}^{0.75}$

Most successful narrow vein mines do not exceed 30 to 40 vertical metres / annum. Based on the mineralization by level, this would be equivalent to 225 to 300 ktpa for BRX and 290 to 390 ktpa for Klaza, for a total annual production rate of 515 ktpa to 690 ktpa.

AMC recommends that the BRX and Klaza deposits be considered for mining as four virtually independent operations at a combined production rate of approximately 690 ktpa. This production rate is well supported by the detailed production scheduling.

16.6.4 Stope design and selection

Stope wireframes were generated using MSO on a 3 m increment. Once the stopes were generated, a check was made to remove any outlying stopes that would not be economic when the cost of access development was included. Tonnes and grades by level were then used in the determination of the optimum interface between the open pit and the underground.

The cost of access development was then determined for each level and each level was evaluated to determine if the value was sufficient to pay for its access. Once the projected economic stopes were selected, the wireframes were combined into stopes 24 m in length. Stopes overlapping with the pit were removed and crown pillars for the BRX and Klaza zones were included in the design. For Central Klaza zone, AMC has assumed that the crown pillar will be partially extracted (90%) at the end of the mine life.

The mineralization associated with the projected economic stopes is summarized in Table 16.19 by zone. The total underground mineralization is estimated to be 6.3 Mt at grades of 3.59 g/t Au, 87 g/t Ag, 0.6% Pb, and 0.7% Zn with an NSR value of C$285.

Table 16.19 Underground mineralization

Zone	Tonnes	NSR (C$/t)	Au (g/t)	Ag (g/t)	Pb (%)	Zn (%)	Arsenic (ppm)
Western BRX	2,404,014	396	5.42	87.01	0.68	0.76	8,003
Central BRX	1,301,690	179	1.64	96.62	0.88	0.92	4,305
Western Klaza	726,340	266	2.90	138.79	0.42	0.55	4,708
Central Klaza	1,850,741	224	2.87	60.27	0.54	0.65	3,759
Total	**6,282,785**	**285**	**3.59**	**87.11**	**0.65**	**0.74**	**5,606**

Source: AMC Mining Consultants (Canada) Ltd.

16.6.5 Underground development

The BRX and Klaza zones are approximately parallel and 800 m apart, and as such separate declines were designed for both zones. Access to the Klaza zone underground mine would be via two independent 5 m by 5 m declines, one for the Western Klaza zone and one for the Central Klaza zone with crosscuts on each level. The Central Klaza zone also has a pit portal to access one level of stopes. Levels are spaced at a vertical distance of 25 m floor to floor. Development (4 m by 4 m) was designed to follow the vein along strike from a central access crosscut. Main access to the BRX zone has a similar design with two independent (5 m by 5 m) access declines for the Western and Central BRX zones. The proposed development required by zone is summarized in Table 16.20.

Table 16.20 Development physicals

Area	Decline (m)	Access drives (m)	Ventilation access (m)	Ventilation raises (m)	Mineralized vein development (m)	Waste vein development (m)
Klaza	5,087	981	468	695	10,174	3,632
BRX	6,711	1,390	922	867	13,186	3,397
Total	**11,799**	**2,372**	**1,390**	**1,562**	**23,360**	**7,029**

Source: AMC Mining Consultants (Canada) Ltd.

Key underground design parameters are summarized in Table 16.21.

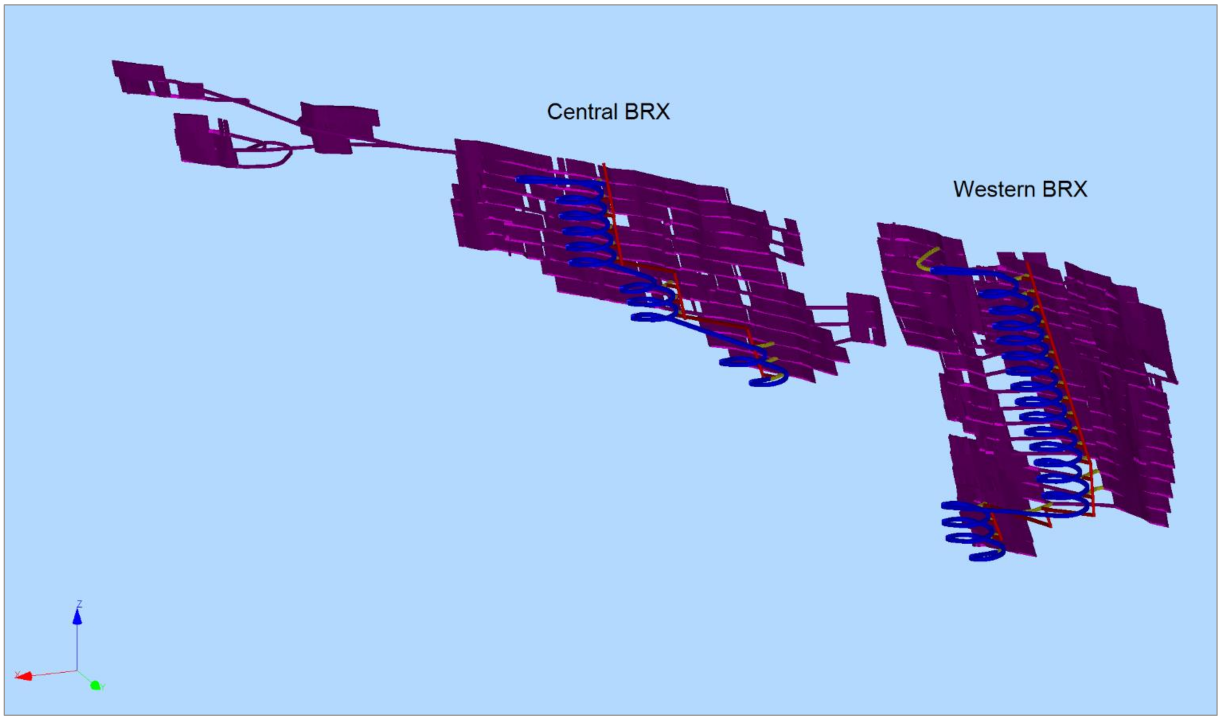
Table 16.21 Key underground design parameters

Parameter	Assumption
Waste development dimensions	5 m by 5 m
Vein development dimensions	4 m by 4 m
Decline gradient	15%
Decline radius of curvature	25 m
Minimum stand-off distance to vein	34 m
Raises to surface for ventilation	4 m diameter
Allowance for passing bays	10% of decline length (15 m every 150 m)

Source: AMC Mining Consultants (Canada) Ltd.

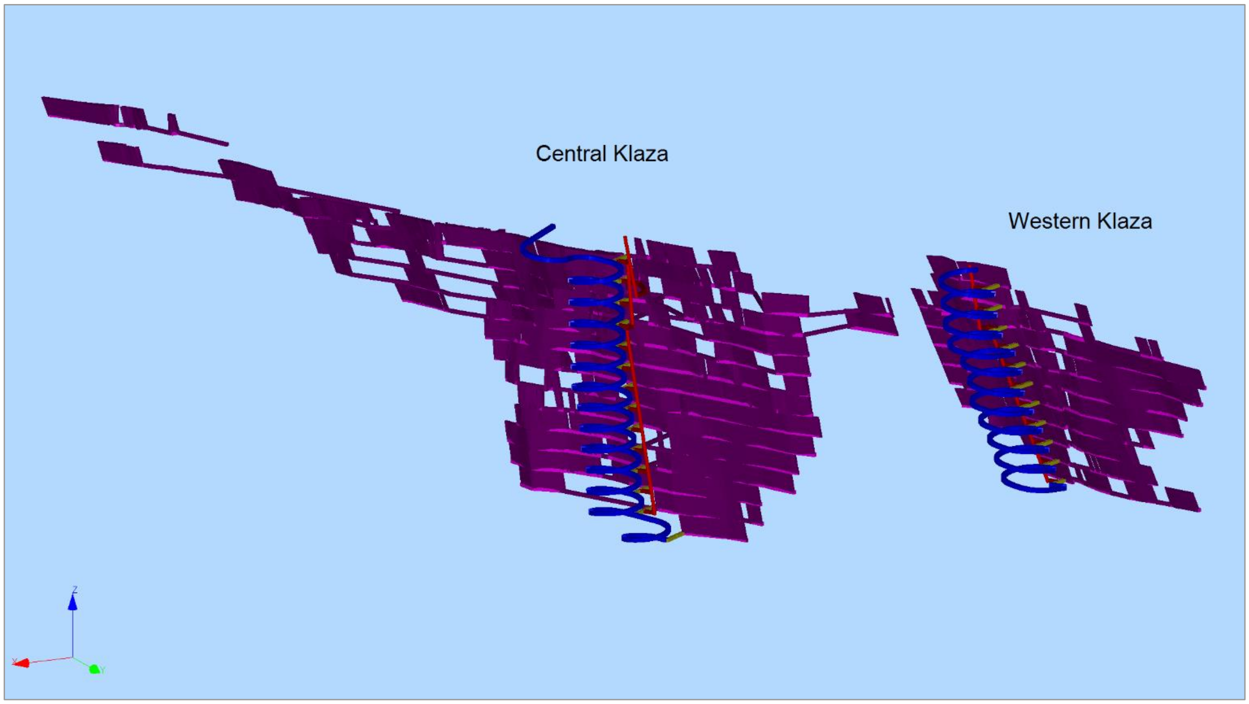
Isometric views of the BRX and Klaza proposed underground mines are provided in Figure 16.18 and Figure 16.19.

Figure 16.18 BRX underground mine design



Source: AMC Mining Consultants (Canada) Ltd.

Figure 16.19 Klaza underground mine design



Source: AMC Mining Consultants (Canada) Ltd.

16.6.6 Proposed infrastructure

See Section 18 for details of the underground infrastructure.

16.6.7 Ventilation

AMC has undertaken a preliminary estimate of the ventilation requirements based on the underground equipment rating and activities being concurrently undertaken. This estimate has been checked against benchmark data for ventilation quantities for each of the mine areas. The function of the ventilation system is to dilute / remove airborne dust, diesel emissions, explosive gases, and to maintain temperatures at levels necessary to ensure safe production throughout the life of the mine.

For both zones, the mine will be ventilated by a “Pull” or exhausting type ventilation system. That is, the primary mine ventilation fans will be located at the primary exhaust airways of the mine. Fresh air will enter each mine via the decline portals with exhaust to the surface via a dedicated return airway. In winter, air will be heated by direct propane gas fired heaters located at each portal, with the heat ducted to the intake airflow.

Each level will have its own ventilation circuit. Fresh air will enter each level from the decline with contaminated air entering the return air raise (RAR) system via a regulator installed in the access to the RAR on each level. The proposed ventilation system has been modelled using Ventsim software to check air velocities and practicality of the overall system. Based upon the equipment projected to be required and the activities being concurrently undertaken, the following maximum airflows are planned for each of the areas of the the BRX and Klaza underground mines, primarily to provide adequate dilution of exhaust emissions from the planned underground diesel equipment fleet and ensure sufficient supply of fresh air for personnel.

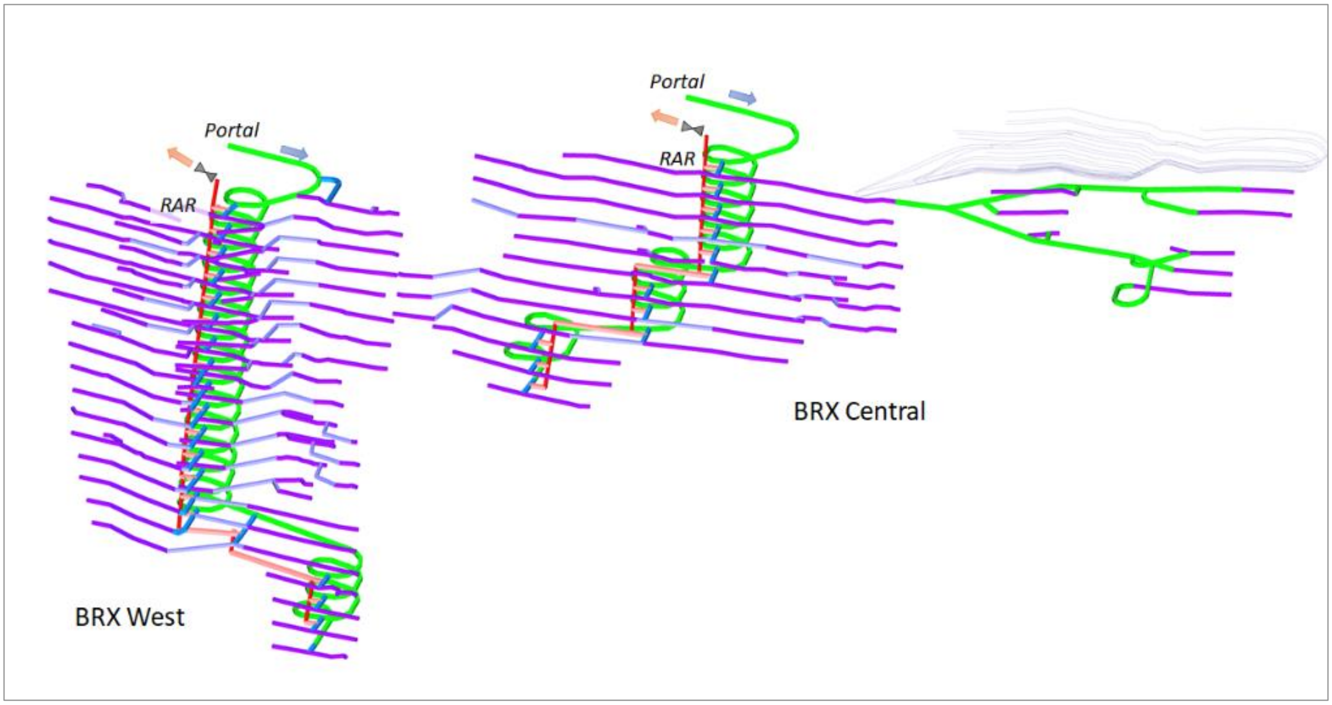
- Central Klaza: 159 m^3/s in Year 6
- Western Klaza: 159 m^3/s in Year 3
- Central BRX: 189 m^3/s in Year 9
- Western BRX: 217 m^3/s in Year 3

The primary surface exhaust fans required for the ventilation system are as follows:

- Central Klaza: 400 horsepower (hp)
- Western Klaza: 400 hp
- Central BRX: 600 hp
- Western BRX: 500 hp

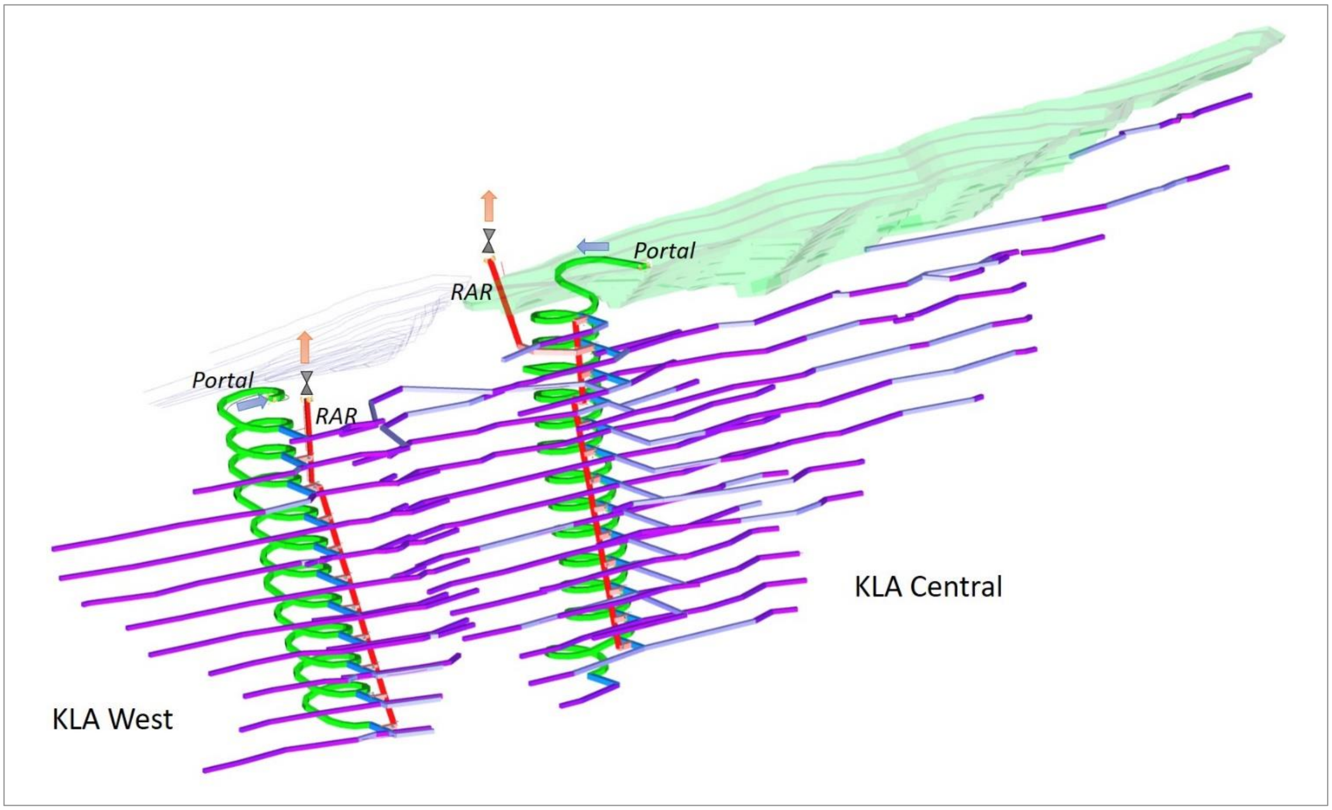
The proposed ventilation strategies for the BRX and Klaza zones are shown in Figure 16.20 and Figure 16.21.

Figure 16.20 Proposed BRX zone ventilation system



Source: AMC Mining Consultants (Canada) Ltd.

Figure 16.21 Proposed Klaza zone ventilation system



Source: AMC Mining Consultants (Canada) Ltd.

16.6.8 Secondary egress

On all levels the planned main escape route is either the main decline or the RAR. RARs will be equipped with ladderways for personnel egress. Refuge stations will be placed strategically in the underground mine. Refuge stations will be portable for flexibility of location, although lunch rooms will also be equipped as refuge stations.

16.6.9 Underground equipment

AMC has based its equipment selection and estimate of equipment numbers on operations of similar production rates. AMC has assumed that there will be some synergy between the two independent operations that will allow for sharing of any spare capacity equipment. The combined total equipment requirement estimated for BRX and Klaza to support a combined production rate of 690 ktpa is summarized in Table 16.22.

Table 16.22 Equipment selection

Underground mobile equipment	No of units
Production drill rig	3
2 Boom jumbo drill - development	4
Diesel LHD - production	3
Diesel LHD - development	3
UG haul truck (40 t)	5
Bolter	2
Cable bolter	2
Explosives loader	2
Personnel carrier	4
Scissor lift	4
Boom truck	2
Utility vehicle	5
Grader	1
Lubrication service truck	2

Source: AMC Mining Consultants (Canada) Ltd.

16.6.10 Mine site management, supervision, administration and labour

Labour will be sourced from the local community and area around Klaza, as well as fly-in fly-out (FIFO) workers. The FIFO workforce will be housed in the onsite camp facilities. Daily bus service will also be provided to employees that live in Carmacks, which is 73 km to the mine and back.

AMC has estimated labour and supervision requirements at the full production rate. Staff is assumed to work on a weekly roster while mining labour is assumed to work on a two weeks on one week off basis. AMC has assumed two 12 hour shifts per day.

AMC has allowed for a combined open pit and underground technical services team in consideration of the size of the operation as a whole. The technical services team is summarized in Table 16.23.

Table 16.23 Combined open pit and underground technical services

Combined OP and underground technical services	Number
Tech Services Superintendent	1
Scheduling Engineer	1
Mining Technologist (OP)	1
Geotechnical Engineer	1
Design / planning Engineer	2
Production Engineer	2
Ventilation Engineer	1
Senior Surveyor	1
Surveyor	2
Senior Geologist	1
UG Mine Geologist	2
Technicians (OP)	2
Subtotal	**17**

Source: AMC Mining Consultants (Canada) Ltd.

Open pit is planned to be mined using a contractor, however for manning numbers AMC has estimated the labour and maintenance requirement seperately. It is noted that the underground manning will only reach peak numbers when the open pit is depleted. A summary of the open pit labour estimate is provided in Table 16.24.

Table 16.24 Open pit labour and supervision

OP labour	Number
Mine Foreman	2
Drill Operator	2
Drill Helper	3
Blaster	1
Blaster Helper	1
Shovel / loader Operator	12
Haul Truck Operator	22
Dozer Operator	5
Water Truck Operator	2
Grader Operator	2
Mine Labourer	8
Subtotal	**60**
OP maintenance	Number
Mine Maintenance Foreman	2
Maintenance Planner	1
Mechanic	3
Mechanic Heavy Equipment	4
Electrician	2
Welder	2
Serviceman	2
Maintenance Labourer	4
Subtotal	**20**

The total labour and supervision personnel numbers including technical services are summarized in Table 16.25.

Table 16.25 Underground labour and supervision

Underground management	**Number**
Mining Manager	1
Administration Assistant	1
Clerk	2
Supervision	
Mine General Foreman	2
Shift Supervisor	12
Subtotal	**18**
UG labour	**Number**
Workplace Trainer / Safety	3
Jumbo Operator	12
Production Driller	9
Charge / Scale	12
Bolters / Cable bolters	12
Loader operator	18
Service Crew	15
Grader Operator	3
Truck Driver	15
Nipper	6
Subtotal	**105**
UG maintenance	**Number**
Senior Maintenance Engineer	1
Maintenance Planner	2
Mechanical Foreman	2
Electrical Foreman	2
Leading Hand Mechanics	3
Shift Mechanics	9
Millwright	3
Shift Serviceman	6
Electrical Leading Hand	3
Shift Electricians	6
Welders	3
Subtotal	**40**
Total (including technical services)	**180**

Source: AMC Mining Consultants (Canada) Ltd.

The total workforce for processing, administration and support, camp housing and food service, and spares are summarized in Table 16.26.

Table 16.26 Processing, adminstration and support, camp operation, and spare labour

Admin & support	**Number**
General Manager	1
Comptroller / Accountant	2
Purchassing Agent	3
Warehouse operator	6
First Aid / Safety nurse	3
Operators	12
Clerks	6
Subtotal	**33**
Camp	**Number**
Camp Manager	1
Chief Cook	2
Cooks	9
Cleaners / Laundry	9
Maintenance	3
Clerks	3
Subtotal	**27**
Mill	**Number**
Operators	47
Maintenance	12
Management	5
Subtotal	**64**
Total	**124**

Source: AMC Mining Consultants (Canada) Ltd.

The total underground mining labour and supervision requirements were estimated to be 180 employees (Table 16.25); this excludes contractor labour for the open pit. A further 124 employees are required for processing, administration and support, and camp housing and food service (Table 16.26). The total workforce is 304 employees. There will be an average of 202 employees on the mine site at any given time.

16.7 Projected LOM development and production schedule

16.7.1 Mine sequence optimization

In order to optimize the overall value of the project and the sequence of mining, AMC has estimated revenue for each pit and each underground zone. The areas were then ranked in order of value after accounting for mining costs. The projected revenue from each source and consideration of practical scheduling constraints provided a basis for the order in which the pits and underground zones are scheduled. The focus was to mine the open pits in the first three years to provide feed to the mill while the declines for Western BRX and Klaza zones are advancing for the underground ramp up to production. The pit and underground areas are ranked by value as shown in Table 16.27.

Table 16.27 Ranking of zone value

Zone	Value (C$/t)	Ranking
Central Main Klaza and North East Klaza Pits	177	3
South End BRX Pit	136	5
Western Klaza UG	208	2
Central Klaza UG	165	4
Western BRX UG	338	1
Central BRX UG	121	6

Source: AMC Mining Consultants (Canada) Ltd.

Using the rankings shown above and the other referenced considerations, the planned initial development and mining sequence is to mine out Central Main Klaza and Klaza North East Pits while the two declines to the Western BRX and Klaza zones underground mines are developed. Underground development will target opening up the high-grade Western BRX and Klaza zones as the priority. The declines for Central Klaza and BRX zones will be deferred due to their lower grade in the underground and as indicated by the Ranking. Similarily, South End BRX pit is deferred due to the lower grade and Ranking. Underground mineralization is projected to be mined at a rate that, together with open pit production, will generally fill the mill from Year 2 through to Year 10 at approximately 690 ktpa.

16.7.2 Conceptual open pit production schedule

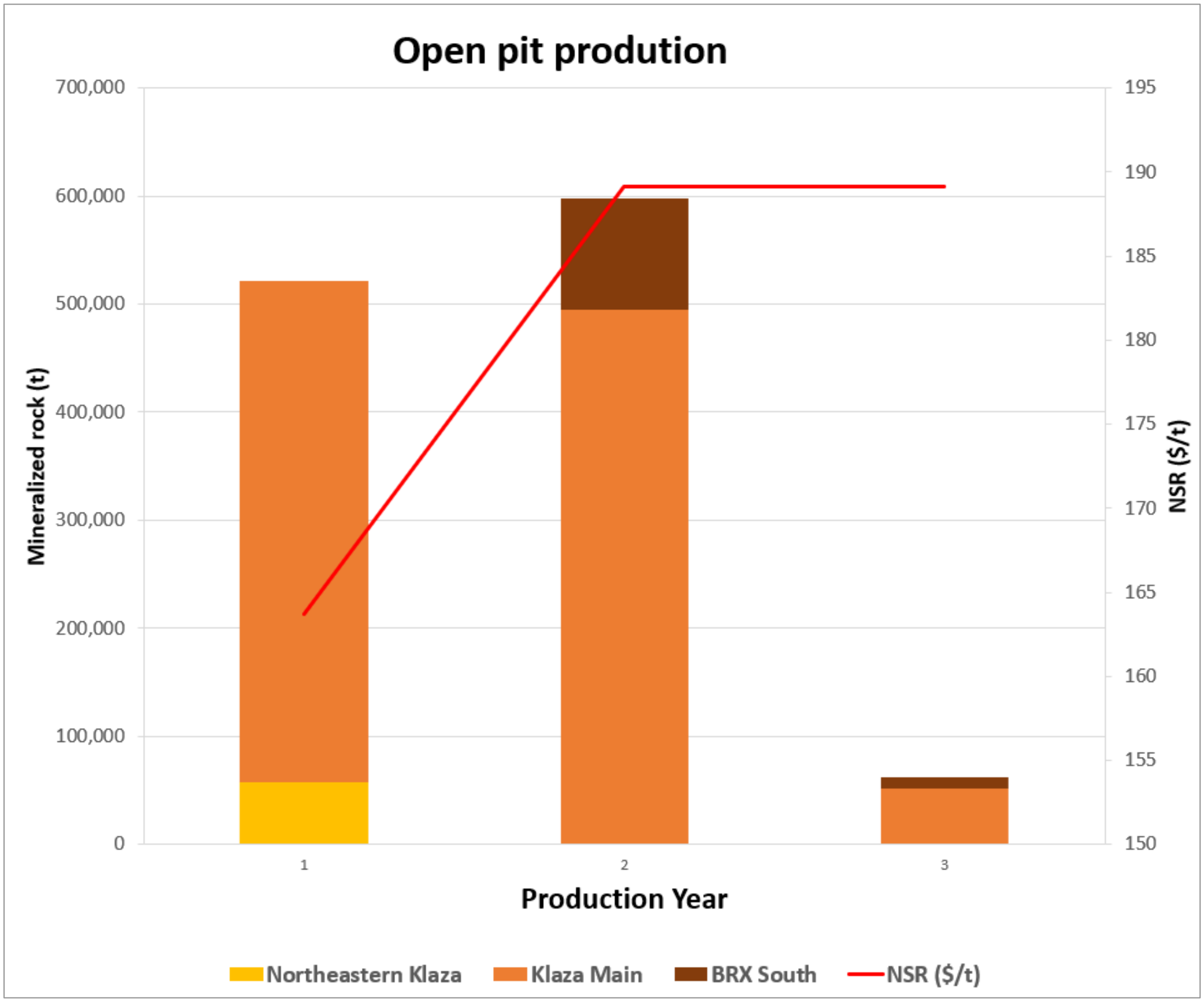
The proposed open pit production schedule extends over a three year period and is summarized in Table 16.28 and Figure 16.22. It is assumed that construction of the process plant and surface infrastructure will take place in Year 0.

Table 16.28 Conceptual open pit production schedule

OP production	YR0	YR1	YR2	YR3	OP totals
Waste (Mt)		5.5	2.9	0.3	8.7
Mineralized rock (kt)		521	598	62	1,181
NSR ($/t)		178	164	189	171
Au (g/t)		2.2	2.5	2.5	2.4
Ag (g/t)		29.7	39.4	39.4	35.1
Pb (%)		0.3	0.5	0.5	0.4
Zn (%)		0.6	0.7	0.7	0.7
As (ppm)		2,411	3,455	3,455	2,994

Source: AMC Mining Consultants (Canada) Ltd.

Figure 16.22 Projected open pit production



Source: AMC Mining Consultants (Canada) Ltd.

16.7.3 Projected underground development schedule

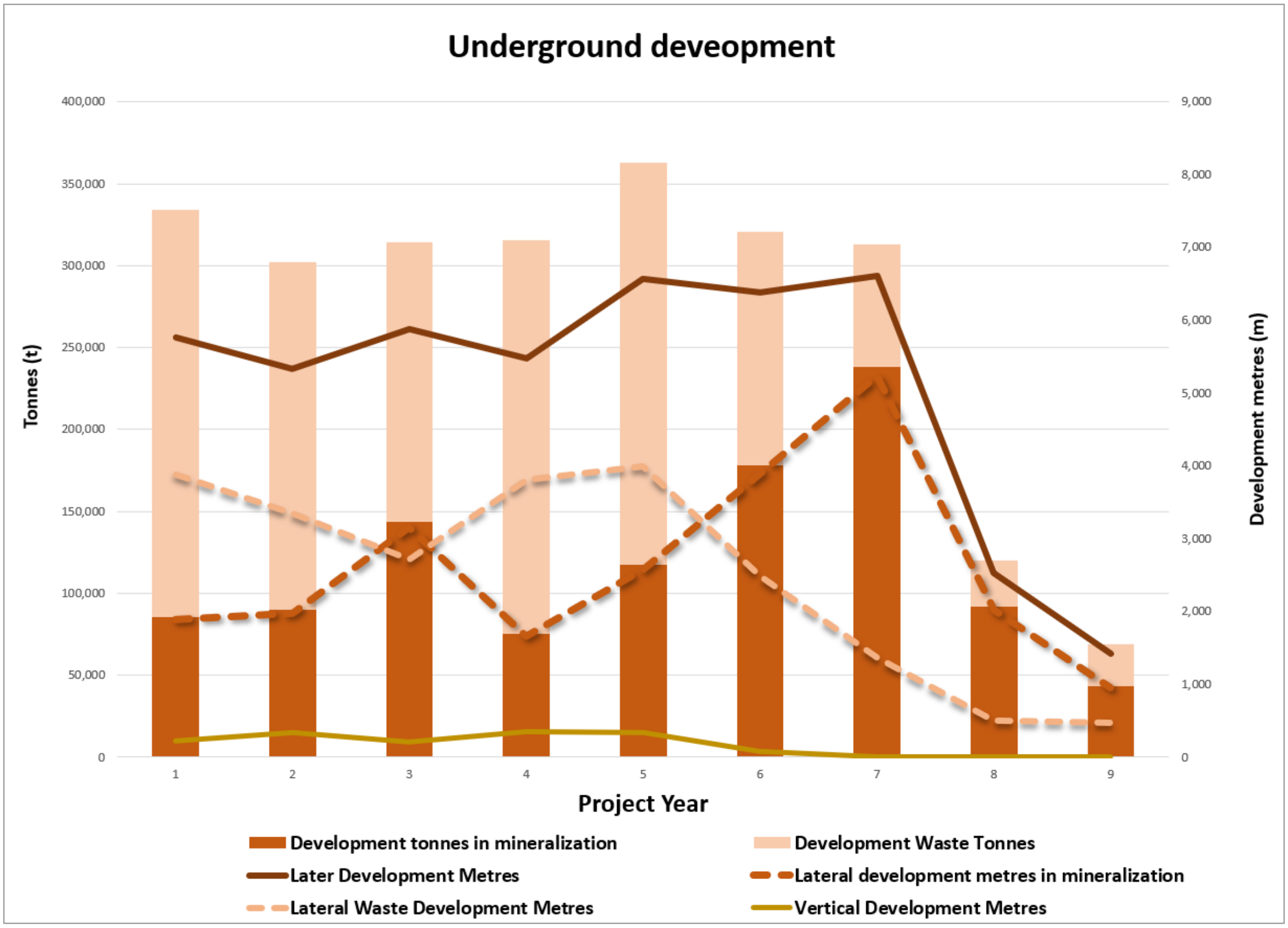
The projected underground development schedule is summarized in Table 16.29 and Figure 16.23.

Table 16.29 Proposed underground development schedule

Description	**YR0**	**YR1**	**YR2**	**YR3**	**YR4**	**YR5**	**YR6**	**YR7**
Access development (m)		2,819	2,200	1,690	2,475	2,116	499	
Vein development (m)		2,941	3,134	4,184	2,997	4,453	5,883	6,612
Vertical development (m)		227	338.6	214	354	346	83	
Description	**YR8**	**YR9**						**Total**
Access development (m)								11,799
Vein development (m)	2,528	1,418						34,150
Vertical development (m)								1,562

Source: AMC Mining Consultants (Canada) Ltd.

Figure 16.23 Projected underground development



Source: AMC Mining Consultants (Canada) Ltd.

16.7.4 Conceptual underground production schedule

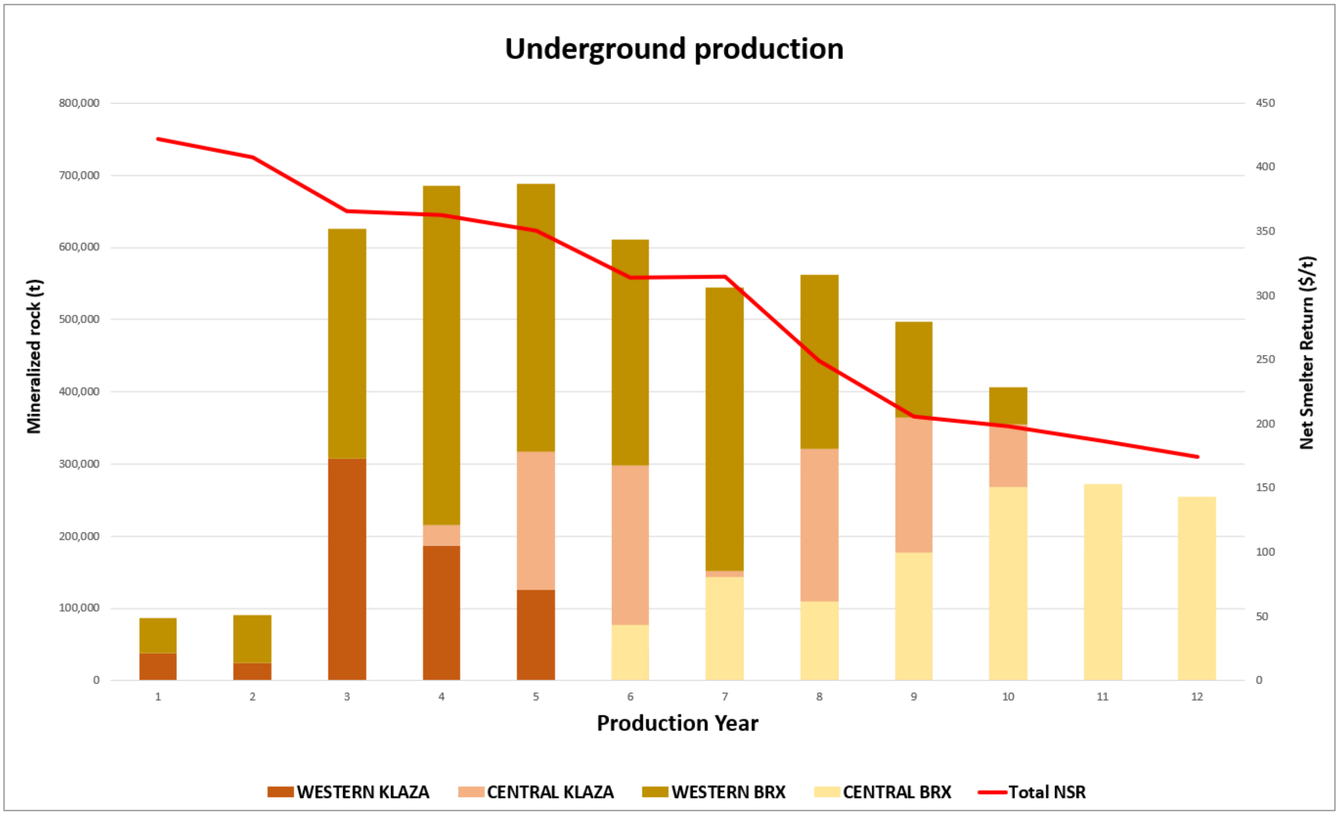
The projected underground production schedule is summarized in Table 16.30 and Figure 16.24.

Table 16.30 Projected underground production schedule

UG production	YR0	YR1	YR2	YR3	YR4	YR5	YR6	YR7
Mineralized rock (kt)		86	90	656	686	688	688	688
Au (g/t)		5.69	5.50	4.77	4.83	4.60	4.04	3.93
Ag (g/t)		111.85	100.70	108.32	93.30	95.79	88.75	96.79
Pb (%)		0.53	0.63	0.55	0.62	0.65	0.67	0.77
Zn (%)		0.63	0.73	0.65	0.68	0.71	0.77	0.85
As (ppm)		8,219	7,721	6,892	6,881	6,820	6,100	6,320
UG production	**YR8**	**YR9**	**YR10**	**YR11**	**YR12**			**Total**
Mineralized rock (kt)	672	675	675	456	255			6,283
Au (g/t)	3.14	2.50	2.27	1.96	1.67			3.59
Ag (g/t)	72.75	65.24	71.68	84.14	93.18			87.11
Pb (%)	0.60	0.60	0.67	0.72	0.69			0.65
Zn (%)	0.74	0.68	0.76	0.81	0.80			0.74
As (ppm)	4,987	3,669	4,033	3,936	4,757			5,606

Source: AMC Mining Consultants (Canada) Ltd.

Figure 16.24 Projected underground production



Source: AMC Mining Consultants (Canada) Ltd.

16.7.5 Waste rockfill

Stopes will be backfilled with waste rock on a sequential basis (stopes blasted nominally 24 m along strike and then filled). Approximately 1.4 Mt of waste is planned to be produced from underground over the life of the mine. Where possible, waste rock generated underground will be backfilled directly into stopes, with excess waste stockpiled underground or transported to surface waste dumps. The total waste required for backfilling the planned underground stopes is 3.6 Mt over the life of the mine. Particularly in the latter stages of underground mine production it is planned that waste will be back-hauled underground from the waste dumps for backfill.

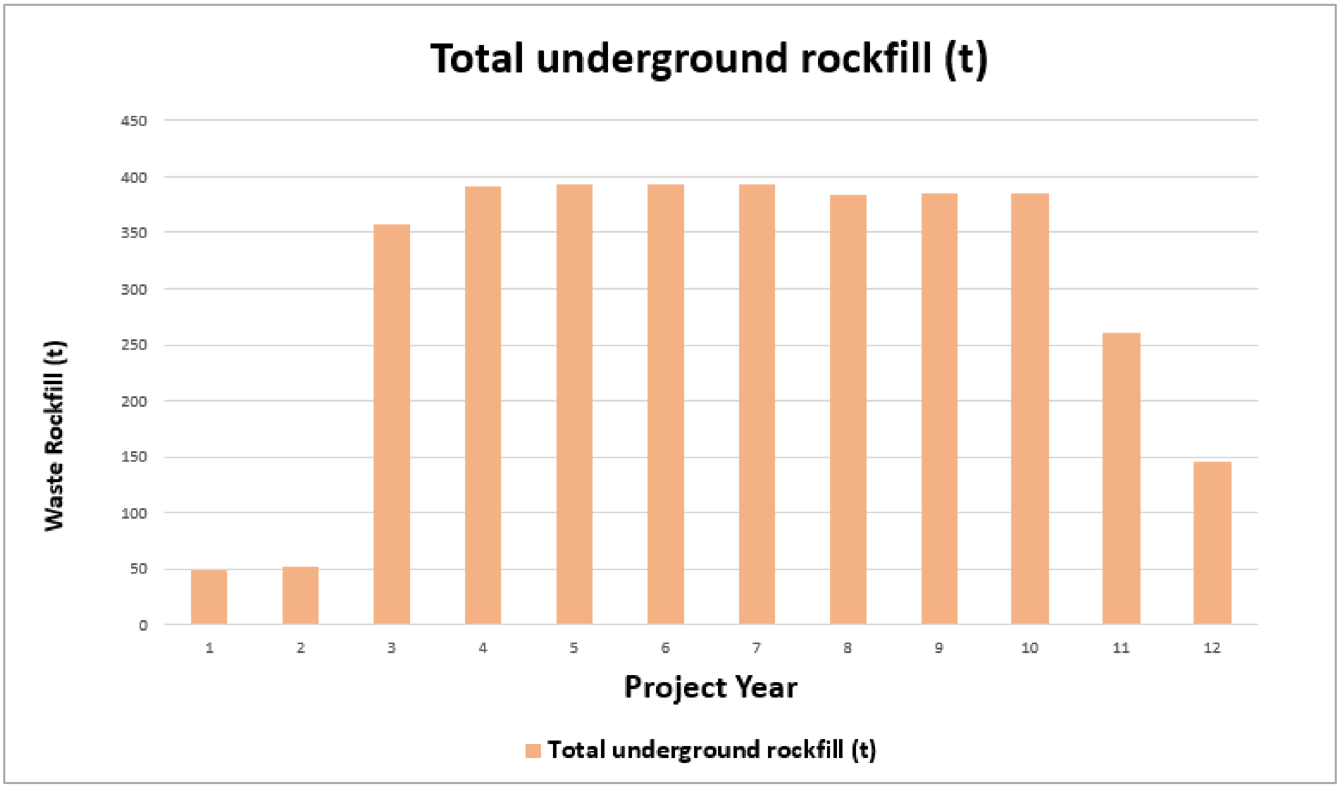
The projected annual backfill requirements are summarized in Table 16.31 and shown in Figure 16.25.

Table 16.31 Projected annual underground waste rock fill schedule

Quantity	**YR0**	**YR1**	**YR2**	**YR3**	**YR4**	**YR5**	**YR6**	**YR7**
Waste rock fill (kt)		49	52	357	392	393	393	393
Quantity	**YR8**	**YR9**	**YR10**	**YR11**	**YR12**			**Total**
Waste rock fill (kt)	384	385	386	261	146			3,590

Source: AMC Mining Consultants (Canada) Ltd.

Figure 16.25 Projected underground waste rock fill requirements



Source: AMC Mining Consultants (Canada) Ltd.

16.7.6 Conceptual combined production schedule

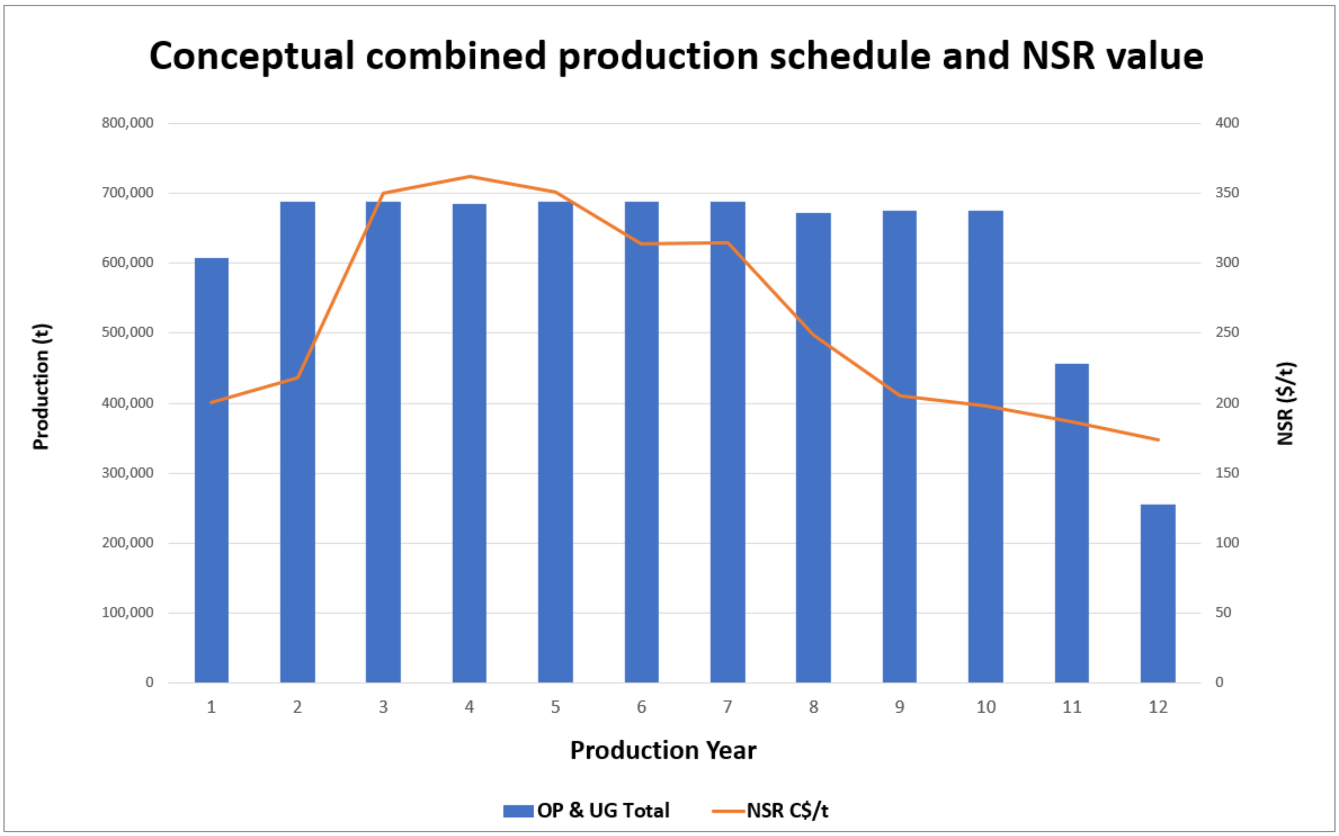
The projected combined life-of-mine (LOM) production schedule for the open pit and underground is summarized in Table 16.32. The conceptual schedule is shown together with the NSR value in Figure 16.26. The projected combined open pit and underground mine design is shown in Figure 16.27.

Table 16.32 Conceptual LOM production schedule

Production	YR0	YR1	YR2	YR3	YR4	YR5	YR6	YR7
Waste (kt)		5,749	3,149	475	240	245	143	75
Mineralized rock (kt)		607	688	688	686	688	688	688
NSR ($/t)		200	218	350	362	351	314	315
Production	**YR8**	**YR9**	**YR10**	**YR11**	**YR12**			**Total**
Waste (kt)	28	26						10,130
Mineralized rock (kt)	672	675	675	456	255			7,464
NSR ($/t)	249	206	198	187	174			268

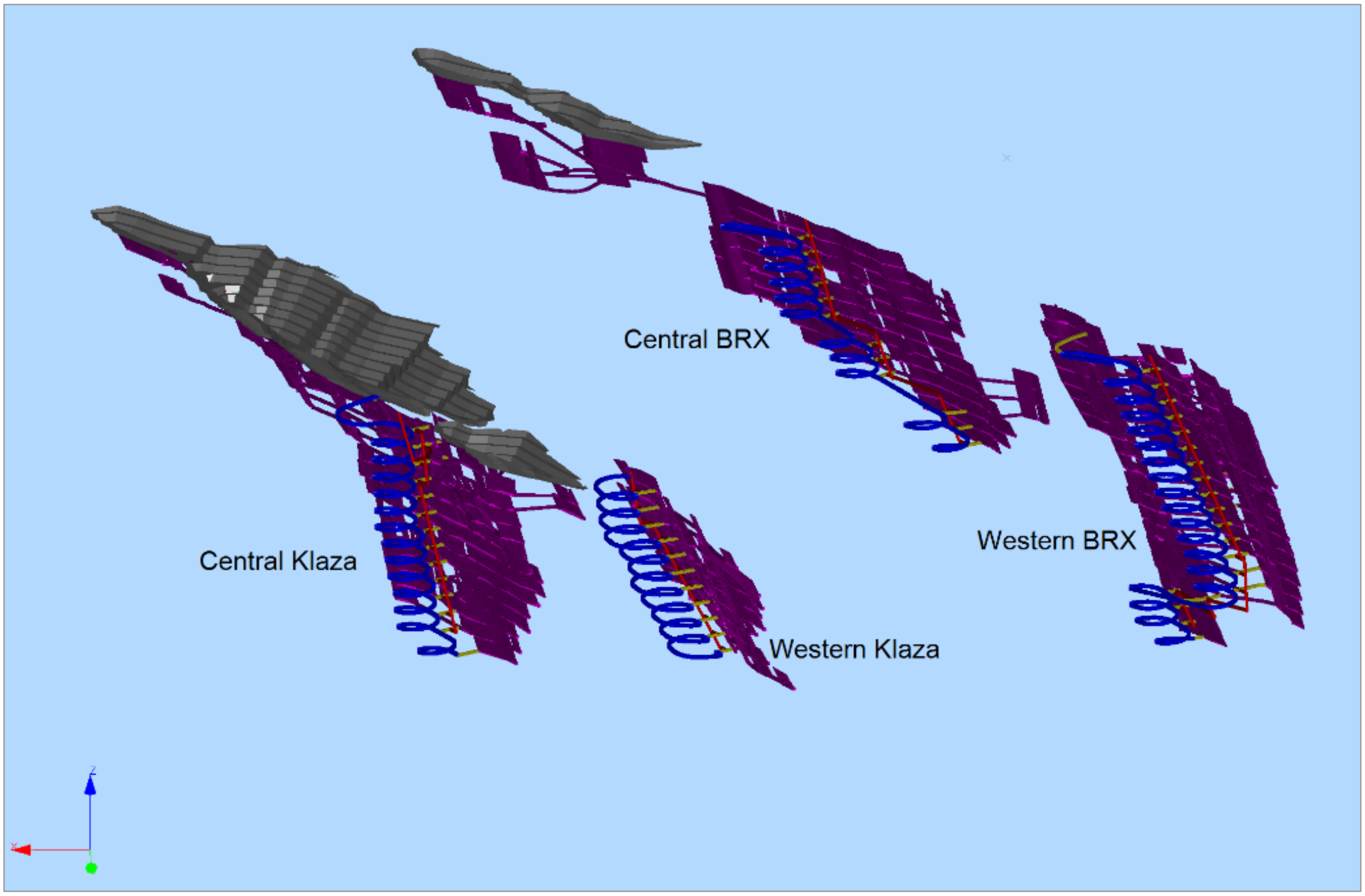
Source: AMC Mining Consultants (Canada) Ltd.

Figure 16.26 Conceptual LOM production schedule and NSR value



Source: AMC Mining Consultants (Canada) Ltd.

Figure 16.27 Projected open pit and underground mine design



Source: AMC Mining Consultants (Canada) Ltd.

16.7.7 Projected process plant feed schedule

The construction of the process plant will take place in YR0 and there will be no production from the open pit and underground during construction. It has been assumed that the process plant would be capable of reaching 88% capacity (607 ktpa) during YR1 and 100% capacity (688 ktpa) in YR2.

The conceptual process feed is the same schedule as the conceptual LOM production schedule and is summarized in Table 16.33.

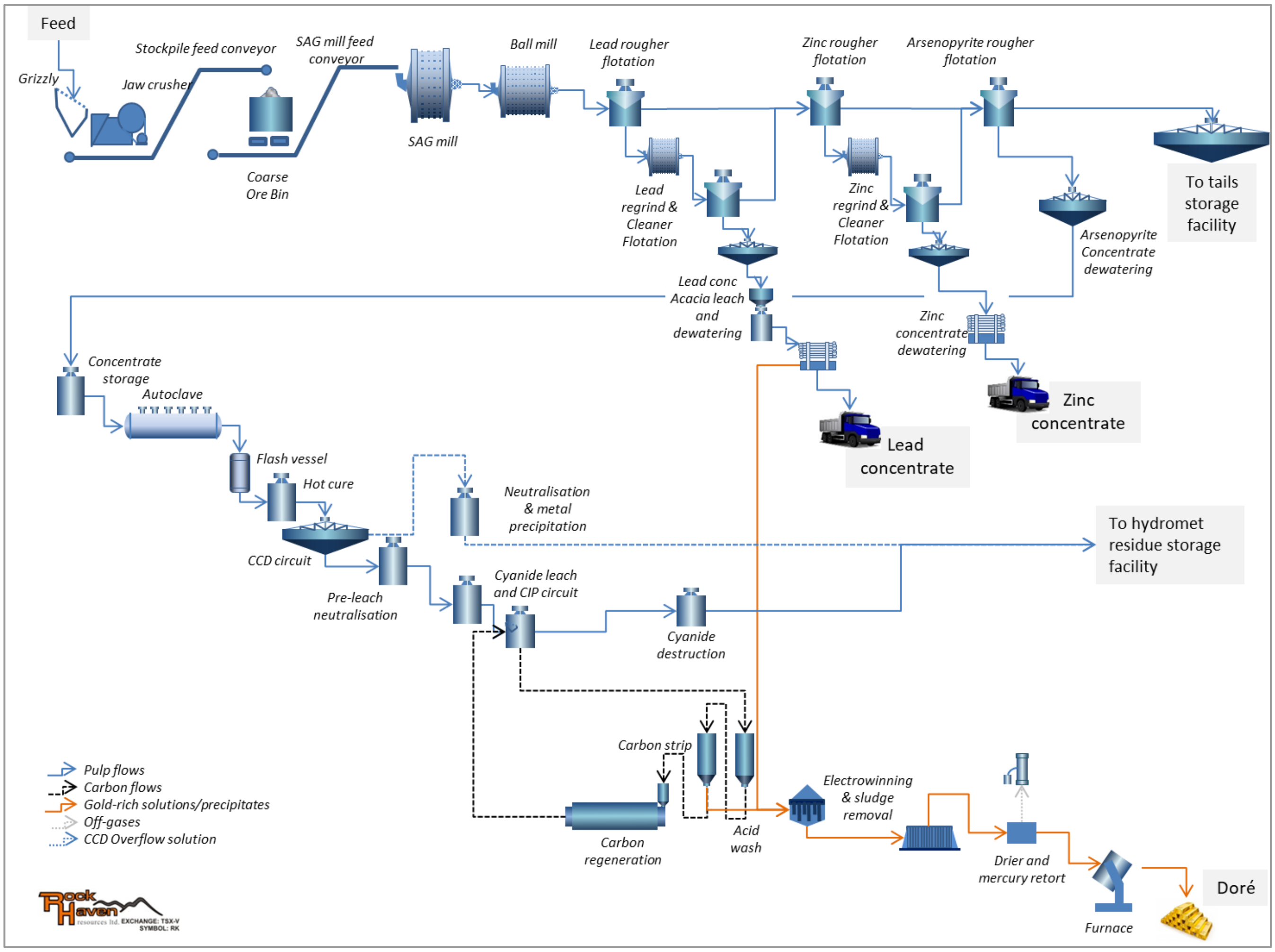
Table 16.33 LOM process plant feed schedule

Production	YR0	YR1	YR2	YR3	YR4	YR5	YR6	YR7
Mineralized rock (kt)		607	688	688	686	688	688	688
NSR ($/t)		200	218	350	362	351	314	315
Au (g/t)		2.70	2.87	4.57	4.83	4.60	4.04	3.93
Ag (g/t)		41.33	47.45	102.11	93.30	95.79	88.75	96.79
Pb (%)		0.36	0.52	0.55	0.62	0.65	0.67	0.77
Zn (%)		0.63	0.74	0.66	0.68	0.71	0.77	0.85
As (ppm)		3,233	4,015	6,582	6,881	6,820	6,100	6,320
Production	**YR8**	**YR9**	**YR10**	**YR11**	**YR12**			**Total**
Mineralized rock (kt)	672	675	675	456	255			7,464
NSR ($/t)	249	206	198	187	174			268
Au (g/t)	3.14	2.50	2.27	1.96	1.67			3.40
Ag (g/t)	72.75	65.24	71.68	84.14	93.18			78.88
Pb (%)	0.60	0.60	0.67	0.72	0.69			0.61
Zn (%)	0.74	0.68	0.76	0.81	0.80			0.73
As (ppm)	4,987	3,669	4,033	3,936	4,757			5,192

17 Recovery methods

This section describes the process approach with some of the key design data tabulated. The overall process flowsheet is shown in Figure 17.1.

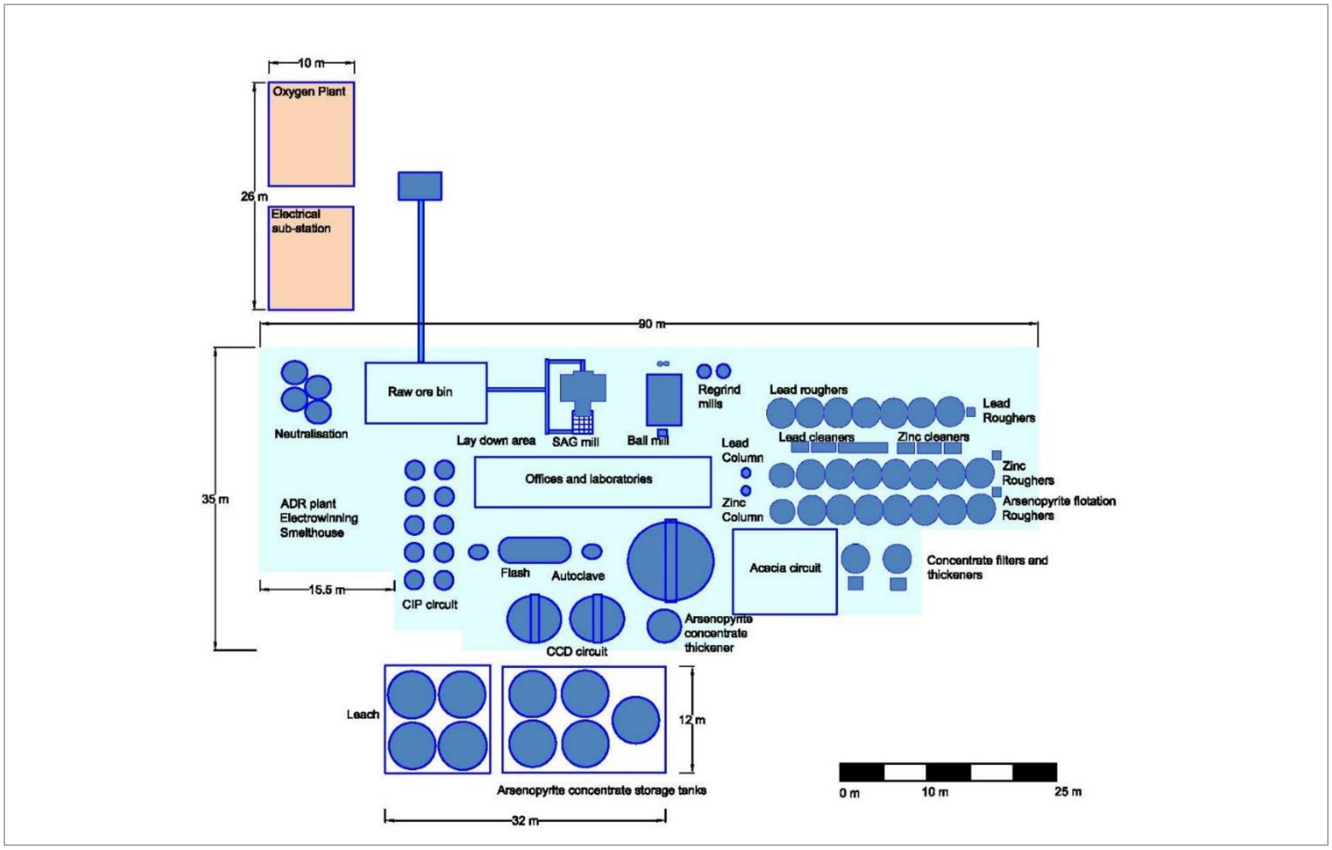
Figure 17.1 Overall process flowsheet



Source: BCM.

The layout for the metallurgical facility is shown below.

Figure 17.2 Plant layout



Source: BCM.

17.1 Ore characteristics

For the basis of the overall process design, the ore characteristics have been assumed as follows.

Table 17.1 Site conditions and ore characteristics

Parameter	Unit	Design value	Reference
Specific gravity of ore	t/m^3	2.80	BCR-G
Bulk density of crushed ore	t/m^3	1.8 – 2.0	BCM
Moisture content of ROM ore	% w/w	2	BCM
Angle of repose	degrees	38	BCM

17.2 Crushing and stockpiling

The crushing plant consists of a single primary crushing stage for top size control. Material from the mine is hauled by truck and loaded onto a static grizzly. The primary crusher consists of a jaw crusher, and would crush the grizzly oversize. Grizzly undersize and jaw crusher product are combined and delivered to an ore bin with 24 hours' live capacity (1,900 t). Material from the ore bin is withdrawn using one of two vibrating feeders, which will feed onto a SAG mill feed conveyor belt, equipped with a weightometer.

17.3 Milling and classification

The entire mineral processing circuit has been designed with an assumed availability of 92%, indicating a mean operating throughput of 86 t/hour.

Crushed ore is reclaimed from the mill feed storage bin and fed to a single SAG Mill driven by a 1 MW motor. Lime and a pre-mixed sodium cyanide / zinc sulphate complex is added to the mill feed chute, to promote selective lead flotation in the downstream lead rougher circuit. SAG Mill discharge is screened, with oversize, returning by conveyor to the SAG mill feed without crushing, although space is allowed for a crusher should it be needed in the future. Screen undersize is pumped to the ball mill pump box. Material from the ball mill pump box is pumped to a cluster of hydrocyclones for classification, at a cut size of 90 microns. Cyclone underflow, operating at a 250% circulating load ratio, launders to a 1400 kW ball mill. Cyclone overflow launders to the lead rougher float. Balls would be added by overhead crane into the SAG and ball mills using bottom discharging kibbles.

17.4 Lead flotation

A lead flotation circuit is employed to concentrate lead and silver from the ore. Cyclone overflow from the hydrocyclone cluster is laundered to the rougher where it is combined with flotation reagents:

- Diaryl dithiophosphate Cytec Aero 241 (Aero 241) collector.
- Dithiophosphinate Cytec 3418A (3418-A) collector.
- Methyl Isobutyl Carbinol frother in the feed well.

Concentrate from the lead rougher is collected in a launder system. Tailings from the rougher launders to the lead rougher tails pump box. The concentrate from the lead rougher is pumped to the lead regrind circuit consisting of a mill operating in closed circuit with hydrocyclones at a circulating load ratio of 350%. The regrind mill, sized at 2.4 m x 4.3 m, is powered by a 200 kW motor. It is combined with the lead regrind mill discharge to be pumped to a hydrocyclone cluster where it is classified, with underflow laundered back to the regrind mill feed chute and overflow laundered to the lead first cleaner circuit. Lime and a pre-mixture of the depressants sodium cyanide and zinc sulphate is added to the lead regrind mill feed chute.

Reground concentrate, at a target product grind of 80% passing 28 microns, is added to the lead first cleaner where Aero 241, 3418A, and F-160-10 are added. Concentrate from this bank is laundered to the lead first cleaner concentrate pump box, where it is pumped to the feed box of the lead second cleaner bank. Tailings from the lead first cleaner bank is directed to the lead rougher tails pump box. At this point lime is added to raise the pH to 11, as measured through a pH probe in the zinc conditioning tank.

Additional Aero 241, 3418A, and MIBC is added in the lead first cleaner concentrate pump box prior to pumping to the bank of second cleaner cells. Concentrate from these cells is laundered to the lead second cleaner concentrate pump box, where it is pumped to the feed box of the lead cleaner column. Tails from the lead second cleaner flotation bank is pumped to the first cleaner bank.

Additional Aero 241, 3418A, and MIBC is added in the lead first cleaner concentrate pump box prior to pumping to the lead column cell. Concentrate from the lead column cell is laundered to the lead final concentrate pump box and then pumped to the lead concentrate thickener where it is mixed with flocculant. Tailings from the lead column cell reports to a column scavenger for operating flexibility, with column scavenger concentrates returning to the column feed. Column scavenger tailings is pumped back to the second cleaner bank.

Thickener overflow is recycled to the process water tank and the thickener underflow is pumped to an agitated surge tank. Thickened lead concentrate is fed to a batch operated Acacia leach plant to recover the gold and silver. See Section 17.11.

17.5 Zinc flotation

A zinc flotation circuit is employed to concentrate zinc from the ore. Lead rougher and first cleaner tails, together with lime, is pumped from the lead rougher tails pump box to the zinc conditioner where it is combined with lime to raise the pH to 11.0 and copper sulphate to activate the zinc minerals. Overflow from the conditioner launders into the zinc rougher flotation cells, where sodium isopropyl xanthate (SIPX) and the modified thionocarbamate, Cytec Aero 5100 (Aero 5100), is added to the feed box of the first cell.

Concentrate from the zinc rougher cells is collected in a launder system. Tailings from the zinc rougher will launder to the zinc rougher tails pump box, where lime is added to raise the pH to 11.8. The concentrate from the zinc rougher is pumped to the zinc regrind circuit, consisting of a 2.4 m diameter x 4.3 m long Effective Grinding Length (EGL) regrind mill, powered by a 200 kW motor and operating in closed circuit with hydrocyclones at a circulating load ratio of 350%. It is combined with the zinc regrind mill discharge to be pumped to a hydrocyclone cluster where it is classified, with underflow laundered back to the regrind mill feed chute and overflow laundered to the zinc first cleaner circuit. The target product size for the zinc regrind circuit is 80 percent passing 32 microns.

Lime is added to the mill discharge pump box to maintain a pH of 11.8. Copper sulphate is added to the zinc regrind feed. Reground concentrate is added to the zinc first cleaner bank where Aero 5100 and SIPX is added. Concentrate from this bank is laundered to the zinc first cleaner concentrate pump box, where it is pumped to the feed box of the zinc cleaner column. Tailings from the zinc first cleaner bank are directed to the zinc rougher tails pump box.

Additional Aero 5100 and SIPX is added in the zinc first cleaner concentrate pump box prior to pumping to the zinc column cell. Concentrate from the zinc column cell is laundered to the zinc final concentrate pump box, where it is pumped to the zinc concentrate thickener and mixed with flocculant. Tailings from the zinc column report to a zinc column scavenger bank for operating flexibility with column scavenger tailings then pumped back to the zinc first cleaner feed box.

Zinc thickener overflow is recycled to the process water tank and the thickener underflow is pumped to an agitated surge tank. Zinc concentrate is fed to a batch operated plate and frame pressure filter to remove water. The pressure filter is directly above a concentrate conveyor which will discharge filter cake onto the zinc concentrate stockpile.

17.6 Arsenopyrite flotation

An arsenopyrite flotation circuit is employed to concentrate gold contained in arsenopyrite (and to a lesser extent pyrite) from the ore. Zinc rougher and first cleaner tails, together with lime, is pumped from the zinc rougher tails pump box to the arsenopyrite conditioner where it is combined with copper sulphate to activate the arsenopyrite. Overflow from the conditioner is laundered into the arsenopyrite rougher flotation cells, where SIPX is added to the feed box of the first cell. Further doses of SIPX are added twice down the bank of cells.

Concentrate from the arsenopyrite rougher cells is collected in a launder system. Tailings from the arsenopyrite rougher is laundered to the final flotation tails pump box, where it is pumped to the tails thickener. The concentrate from the arsenopyrite rougher is pumped to the pressure oxidation (POX) circuit feed thickener, designed to thicken the concentrate to 55% solids so reducing the required capacity of the concentrate surge tanks as well as helping to keep the mineral processing process water system separate from the hydrometallurgical water reticulation system.

17.7 Flotation tailings

The arsenopyrite flotation tailings is transferred to a high rate thickener where flocculant is added to aid settling of solids and clarification of overflow water. Overflow water will gravity feed into the transfer tank to be pumped to the milling process water facility for recycle. Underflow at 50% solids would be pumped to the TSF. Water is returned from the TSF to supplement process water requirements, as required.

17.8 Pressure oxidation circuit

Underflow from the arsenopyrite thickener will continuously feed a set of surge capacity tanks providing up to 5 days (120 hours) of surge capacity storage, ahead of the POX circuit. This is to allow up to 5 days of autoclave downtime for maintenance and repairs, while the mill-flotation plant continues to operate. The storage tanks will operate with continuous overflow tank to tank. Discharge from storage tank (#5) into a POX feed tank, where return water from the POX CCD circuit would be added to decrease the percent solids to 15%. The circuit downstream from these tanks has been sized to operate at an effective 85% availability.

The autoclave would be fed by a positive displacement pump. It would operate at 220 degrees centigrade and 2,965 kPaG (430 Psig). The four compartment autoclave would be agitated using four agitators, one agitator per compartment.

Table 17.2 Pressure oxidation design criteria

Parameter	Unit	Design value	Reference
Autoclave			
Feed flow rate	tph operating	10.0	BCM
	m^3/h	68.7	BCM
Feed slurry density	% solids w/w	15	CM-Solutions
Retention time	mins	60	Autec-1
pH	-	1.2	Autec-1
Slurry density	% solids w/w	12	CM-Solutions
Temperature	Degrees C	220	AuTec-1
Pressure	Psig	430	AuTec-1
Oxygen supply	t/d	76.5	CM-Solutions

The product from the autoclave will flow through a single flash vessel, to reduce temperature to <100°C and produce steam. The autoclave – flash vessel discharge slurry is pumped to the hot cure section.

17.9 Hot cure, counter-current decantation, neutralization, and CIP leach

The hot solution (90°C) from the flash vessels is treated through a hot cure stage to cure the basic iron sulphates (BIS) solid. The hot cure acid-conditions the POX product for 6 hours by breaking down the BIS to ferric sulphate and iron hydroxide into the aqueous liquor phase. Steam generated from the autoclave is used to maintain temperature in the hot cure tanks.

After hot curing, counter-current decantation is used to separate the acid and soluble salts from the solid residue. It comprises two stages of thickening using high capacity thickeners. The first thickener would be constructed of stainless steel, the second thickener of rubber-lined mild steel.

Overflow from the first thickener (hot cure liquor) is directed to a neutralization process comprising several tanks, providing 6 hours residence time. Neutralization of this liquor is completed using cheaper limestone to substantially reduce the need for lime (substituting it with limestone) and reduce cost. The neutralized liquor is directed to a lined Hydromet Residue Storage Facility (HRSF). Overflow from this pond will feed into the main TSF, to capture gypsum and iron precipitates and neutralize any residual acid.

The second thickener underflow would be neutralized with lime to pH 10 and the slurry transferred to a series of cyanide leach tanks (4), followed by a set of carbon-in-pulp (CIP) pump cell tanks. The total leach residence time is 24 hours. The CIP circuit is operated in carousel fashion by rotating the feed to each of the 10 x 16 m^3 KEMIX pump cell tanks. The total CIP residence time is 12 hours. Each pump cell tank has its own contained carbon load (0.8 t), with only the slurry passing cascading through each cell. Sequentially, as the carbon in leading tank cell reaches its target gold loading, the tank cell is by-passed and the cell's carbon removed. The pump-cell is brought back into the circuit as the first feed tank, with fresh or regenerated carbon added. This allows the slurry, carbon counter-current process to be achieved. The slurry is transferred from the first to last tank cell, with carbon retained in each cell. A carbon safety screen would be included on the CIP tails before pumping to cyanide destruction.

CIP tailings, acid wash water is treated to detoxify the cyanide using the INCO Air / SO_2 process before pumping to the residue storage (HRSF), reducing the WAD cyanide concentration to less than 5 ppm. Detoxification is carried out in two stages with two-hour residence time in each stage. Sodium metabisulphite (SMBS) is used to replace SO_2 as this presents fewer transportation and handling hazards.

17.10 Carbon handling, elution, electrowinning, and refining

Loaded carbon from POX - CIP plant circuit, at an estimated 4,000 g/t gold, is acid washed in a wash column with nitric acid solution. The acid wash column is fitted with a gas scrubbing system to capture hydrogen cyanide. After acid washing, the loaded carbon is washed with water then caustic solution to raise the pH. The carbon is then stripped of gold in two elution columns with a solution containing 2% NaOH and 2% NaCN. The eluate is stored in a tank from which the electro-winning circuit is fed.

Carbon is periodically regenerated in an electric kiln at 650°C to 700°C to remove organic contaminants and reactivate the carbon load sites. The reactivated carbon is quenched in water and stored ahead of being returned to the CIP circuit.

The gold containing solution (eluate) is circulated through electrowinning cells where the dissolved gold is deposited on the cathodes and removed using high pressure water. The collected gold and silver precipitate (sludge) are filtered and placed in a calcining oven with retort system to remove any mercury before smelting in an induction furnace.

The furnace is tapped into a series of casting molds, in which the slag separates from the gold. Gold doré bars are removed from the molds, cleaned and weighed prior to labelling and storage in the gold room safe, ready for dispatch.

17.11 Lead concentrate leaching, elution, and electrowinning

The thickened lead concentrate is pumped to a 12 hour intensive cyanide leach – electrowinning Acacia plant, to recover the gold and silver. The Acacia modular plant is provided as a package system by Consep – Innovative Process Systems. The modular plant is operated batch-wise, on 18 t Pb concentrate per day, and consists of:

- Concentrate Storage Cone
- Consep Acacia Dissolution Module
- Slimes Recovery Module
- Electrowinning Module
- Cathode Wash Module

The Acacia plant modules are controlled using a Programmable Logic Controller (PLC) system, with its own control centre and MCC connected to all the modules and their instrumentation and drives. High gold recoveries from the Pb lead concentrate are expected, at 85% or higher. A leach aid (proprietory to Consep) is added to the leach dissolution process.

Electrowon gold and silver (sludge) from the cathode wash module is bagged and transported to the Gold room for drying and smelting. The leached lead concentrate residue is fed to a Larox pressure filter for thorough washing and water removal from the solids. The filter filtrate solutions are pumped to the plant's cyanide detoxification section for final CN destruction. The clean filtered lead concentrate cake (7% moisture), is sampled, weighed and bagged for dispatch to a lead smelter.

17.12 Reagent storage and mixing

Reagents required are lime, zinc sulphate, sodium cyanide, Aero 3418A, Aero 241, Aero 5100, copper sulphate, SIBX, MIBC, flocculant, caustic, hydrochloric acid, and sodium metabisulphite. In most cases each reagent will have a make-up tank and a holding tank, and is distributed by pump. A slaking system has been budgeted for the preparation of hydrated lime from quicklime.

17.13 Services and utilities

Dual process water holding systems (one for mineral processing and one for hydrometallurgy) have been included. Process water is obtained from various points throughout the facility (such as thickener overflows and TSF reclaim water) and stored in the respective system, while fresh water, used for reagent mixing, gland water and in the POX plant, would be obtained from the mine fresh water supply storage tank.

17.14 Major instrumentation and sampling

The major instrumentation for the plant will include the following items:

- Mineral processing:
  - Process control system
  - Coarse ore stockpile feed weightometer
  - On-Stream analyser
  - Particle size analyser on PCOF
  - Cross-stream samplers
  - Froth cameras

- Hydrometallurgy:
  - Central control system
  - Feed sulphur analysis
  - Slurry density, POX Feed, hot cure, CCD underflows
  - Flow measurements, POX Feed, Hot Cure, CCD underflows
  - Autoclave pressure
  - Temperature, POX, hot cure, CCD, neutralization
  - Mass flowmeter systems

The control philosophy to be implemented for the Klaza project is typical of those used in modern small scale mineral processing and hydrometallurgical operations. Field instruments provide inputs to a set of PLCs. Process control cubicles are located in the Motor Control Centres (MCC's) and contain the PLC hardware, power supplies, and I/O cards for instrument monitoring and loop control.

The PLCs perform the control functions by:

- Collecting status information of drives, instruments, and packaged equipment.
- Providing drive control and process interlocking.
- Providing proportional-integral-derivative (PID) control for process control loops.

Standard personal computers (PC's) would be located in the Main Control Room (MCR). The PC's are networked to the PLCs and operate a Supervisory Control and Data Acquisition (SCADA) program that provides an interface to the PLCs for control and monitoring of the plant.

The SCADA is configured to provide outputs to alarms and control the process functions. This allows central control of the plant process areas, with some "roving" operators checking the metallurgical process areas on the plant.

17.15 Laboratories

The major equipment in the support laboratories will include Inductive Coupled Plasma (ICP), atomic absorption (AA), LECO element analyzers, and two furnaces for the assay laboratory, and bench-top flotation machine, bottle roll set-up and all supporting equipment for the metallurgical laboratory.

17.16 Tailings management

17.16.1 General

The principal design objectives for tailings management are protection of the regional groundwater and surface waters both during operations and in the long-term (after closure), and to achieve effective reclamation at mine closure. The design of the tailings management strategy has taken into account the following requirements:

- Permanent, secure and total confinement of all solid waste materials within engineered disposal facilities.
- Control, collection and removal of free-draining liquids from the tailings during operations for recycling as process water to the maximum practical extent.
- Collection and diversion of water from upstream of the tailings storage sites, open pits, and mill site areas during operations.
- The inclusion of monitoring features for all aspects of the facility to verify performance goals are achieved and design criteria and assumptions are met.
- Staged development of the facility over the life of the project.

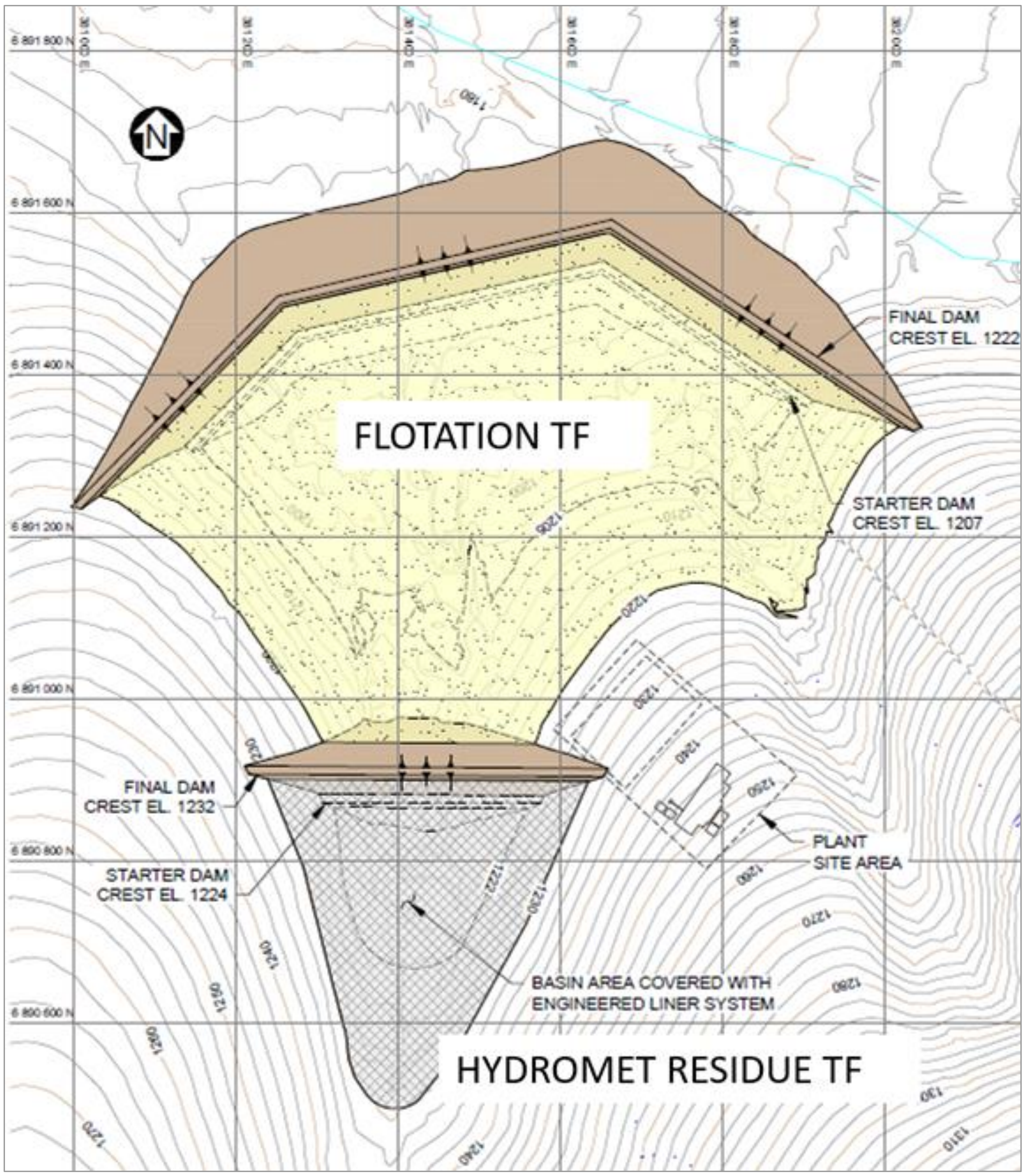
The tailings management strategy has been developed to manage the different tailings streams in separate facilities based on the tailings geochemical properties. Two tailings streams requiring disposal will be produced:

- A hydromet residue.
- A flotation tailings.

Hydromet residue tailings and flotation tailings will be managed in separate Tailings Facilities sized to contain the production schedule volumes. The Flotation Tailings Facility (FTF) will be located downstream of the Hydromet Residue Tailings Facility (HRTF). Detailed descriptions of the tailings facilities are provided in the section below.

The HRTF and FTF layouts are shown in Figure 17.3.

Figure 17.3 Hydromet residue and flotation tailings facilities overall layout



Source: Knight Piésold Ltd
Approximate site of process plant shown.

17.16.2 Hydromet residue tailings facility

The HRTF is optimally positioned adjacent to the plant and directly upgradient from the FTF. This location was the most viable and practical based on a number of economic and operational factors.

The hydromet residue tailings are expected to be relatively reactive and will be contained within a lined engineered containment to allow for the tailings solids to settle and consolidate behind a confining embankment.

A foundation drainage system will be constructed to collect any seepage and dewater any potential groundwater seeps in the foundation. The foundation drains will comprise a spine drain configuration of Corrugated Polyethylene Tubing (CPT) pipes installed in a gravelly sand matrix. The foundation drainage system will flow to a collection manhole at the downstream toe of the tailings embankment, where it will be pumped into the FTF. Flow rates and water quality in the foundation drain will be monitored to verify the integrity of the liner system.

The downstream method of embankment construction is proposed. The embankment is planned with a 2.5-horizontal to 1-vertical (2.5H:1V) upstream slope, and a 2.5H:1V downstream slope. Development will be in two stages as follows:

- Stage 1 – Elevation 1,224 m – Start-up (tailings storage Years 1 to 5).
- Stage 2 – Elevation 1,232 m - Constructed in Year 5 (tailings storage Years 6 to 12).

The embankment will be constructed and operated as a geosynthetic-faced rockfill dam. The starter embankment will be approximately 12 metres high and require an initial volume of 75,000 m^3 of fill. The embankment will comprise a rockfill dam with a geosynthetic liner system installed on the upstream face of the dam and extended over the entire basin area. The impoundment basin will be re-graded so that the floor of the impoundment drains towards the north and the side slopes of the facility will be re-graded to a maximum slope of 3H:1V. The lined area for the starter facility is 45,000 m^2.

The Stage 2 embankment configuration features a downstream raise to the dam roughly 20 m high and 400 m in length, with a total fill volume estimated to be 243,000 m^3. Waste rock will be used for the majority of the construction, as the short overhaul from the waste rock dump will cost less than constructing the dam with local borrow material. A thin bedding layer will be placed over the rough-graded surface to prepare the surface to receive geosynthetics. The lined area for the Stage 2 expansion of the facility is 55,000 m^2

Minimizing water loss is a key aspect of the conceptual design. This will involve keeping the supernatant pond surface area to a minimum, controlling the wet beach area, capturing runoff, and recycling seepage water. The facility is situated up-gradient from the FTF allowing for a simple, safe, and environmentally sound site water management plan.

Tailings will be deposited by gravity around the perimeter of the facility by means of sequential discharge through a series of spigot offtakes using subaerial / subaqueous deposition techniques. Reclaim water from the supernatant pond will be recycled to the mill from a floating barge located in the southern portion of the facility closest to the plant site.

Closure of the facility will involve a progressive capping of the facility with a waste rock and overburden blanket.

17.16.3 Flotation tailings facility

The FTF is optimally positioned downgradient and adjacent to the plant. This location was the most viable and practical based on a number of economic and operational factors.

The flotation tailings are expected to be relatively benign and will be contained within a solids retention facility to allow for the tailings solids to settle and consolidate behind a confining embankment.

A foundation drainage system will be constructed to collect any seepage and dewater any potential groundwater seeps in the foundation. The foundation drains will comprise a spine drain configuration of CPT pipes installed in a gravelly sand matrix. This drainage system will enhance the consolidation of the tailings solids by providing two-way drainage of the tailings deposit. The foundation drainage system will flow to a collection manhole at the downstream toe of the tailings embankment, where it will be pumped back to the TSF. Flow rates and water quality in the foundation drain will be monitored to ensure the integrity of the liner system.

The downstream method of embankment construction is proposed. The embankment is planned with a 2.5H:1V upstream slope, and a 2.5H:1V downstream slope. Development will be staged in two-year increments as follows:

- Stage 1 – Elevation 1,207 m – Start-up (tailings storage Years 1 and 2).
- Stage 2 – Elevation 1,211 m - Constructed Year 2 (tailings storage Years 3 and 4).
- Stage 3 – Elevation 1,215 m - Constructed Year 4 (tailings storage Years 5 and 6).
- Stage 4 – Elevation 1,219 m - Constructed Year 6 (tailings storage Years 7 and 8).
- Stage 5 – Elevation 1,222 m - Constructed Year 8 (tailings storage Years 9 to 12).

The embankment will be constructed and operated as a water-retaining structure. The embankment will be comprised of a zoned structure having a low-permeability core zone with appropriate filter and transition zones to prevent downstream migration of the tailings. The core zone will be keyed into the low-permeability overburden foundation or bedrock. The starter embankment height will be 17 metres and require a fill volume of roughly 594,000 m^3 of engineered fill.

The ongoing embankment raises expands the facility to a dam roughly 32 m high and 1.2 km in length, with a total fill volume estimated to be 2,191,000 m^3. Waste rock will be used for the majority of the construction, as the short overhaul from the waste rock dump will cost less than constructing the dam with local borrow material. Core zone and drainage materials will be sourced from local borrow areas and the placer-mining workings.

Minimizing water loss is a key aspect of the conceptual design. This will involve keeping the supernatant pond surface area to a minimum, controlling the wet beach area, capturing runoff, and recycling seepage water. The facility is down-gradient from all of the other mine site facilities, allowing for a simple, safe and environmentally sound site water management plan.

Tailings will be deposited by gravity around the perimeter of the facility by means of sequential discharge through a series of spigot offtakes using subaerial deposition techniques. Reclaim water from the supernatant pond will be recycled to the mill from a floating barge located in the southern portion of the facility closest to the plant site.

Closure of the facility will involve a progressive capping of the facility with a waste rock and overburden blanket.

17.16.4 Water management

The key facilities for the water management plan are:

- Open pits.
- Underground mine dewatering.
- Mill (including fresh and process water tanks).
- TSF.
- Diversion and water management structures.
- Fresh water supply.
- Sediment and erosion control measures for the facilities.

The water management strategy utilizes water within the project area to the maximum practical extent. The plan involves collecting and managing site runoff from disturbed areas and maximizing the recycle of process water. Site runoff water will be stored on site within the FTF. The water supply sources for the project are as follows:

- Precipitation runoff from the mine site facilities.
- Water recycle from the tailings supernatant ponds.
- Groundwater from open pit and underground dewatering and depressurization.
- Fresh water supply from the Klaza River.
- Treated black and grey water, in small quantities, from the camp.

An overall high-level average monthly site water balance assessment was carried out to determine the preliminary water management strategy and process makeup water requirements for the project.

The results from the water balance modelling are summarized below:

- There will be no surface water discharge from the tailings facilities to the environment during operations.
- Under average precipitation conditions the site will operate in a water-deficit condition with the need for additional makeup water.
- The average annual runoff into the hydromet and FTFs will be approximately 67,000 m^3 per year, which includes runoff from undisturbed catchments, TSF beaches, and ponds.
- Pit runoff and pit dewatering contributes on average 60,000 m^3 of water per year under average precipitation conditions.
- Mine dewatering provides approximately 50,000 m^3 annually.
- The average volume of water retained in the tailings voids, and hence not available for water recycle to the processing plant is approximately 250,000 m^3 annually. This volume will be replaced by pumping from the Klaza River at a rate of 685 m^3/hr.
- The potable water requirement is approximately 7,000 m^3 annually.
- Site road watering utilizes approximately 14,000 m^3 annually in the summer months.
- Underground water consumption for the drills and washdown consumes 38,000 m^3 annually.

18 Project infrastructure

18.1 Surface infrastructure

18.1.1 Mine offices

The mine offices will be an assembly of standard construction industry grade, portable trailers. The trailer complex will provide for a perimeter of offices, a common area in the center, meeting rooms, first aid room, mine rescue room, and training rooms. The structure will be insulated to suit the climate and tie into the site distribution systems for heating and power.

The offices would be particularly suitable as construction offices for the process plant and such use would reduce the indirect construction cost of the plant.

18.1.2 Camp

A camp facility to accommodate 240 persons will be constructed. The camp facility will contain a kitchen, lunch room, laundry facilities, recreation facilities and the mine dry. The camp facilities will also contain a potable water plant and a sewage treatment plant sized for the accomodations.

In the event there is insufficient accommodation at the camp during maintenance for construction, accommodation will be arranged in Carmacks and a bus will be provided to bring personnel in and out of the mine during those times.

18.1.3 Garbage incineration

Garbage will be sorted into various categories such as clean wood, metals, recyclables, and mixed. Clean wood and paper may be burnt at site in accordance with the operating permit. All other garbage will be removed from site to the local municipality and disposed of in accordance with best environmental practice.

18.1.4 Power

18.1.4.1 Generation

A trade-off study was conducted to investigate providing electric power using diesel generators or grid power from the territorial utility. The results indicated that grid electrical power provides more value to the project over the life of the mine.

The grid power option would require a transmission power line to be constructed from Carmacks to the mine-site along the existing Mount Nansen road and upgraded placer access roads (see Figure 18.2). The cost of the power line was estimated using unit rates, budget quotations, and published productivities. It was also benchmarked against other projects. A unit cost for electricity was obtained from the publications of the power utility. This cost was then discounted over the life of the mine. The discounted (5%) total cost (capital plus operating) of power over the LOM using the local grid was estimated to be $100M.

The key power assumptions were:

- Discount rate of 5%.
- Overall cost of utility power $0.105 per kW-hr (includes allowance for cost of the transmission line).
- Utility capacity exists (or will exist) to supply the mine and process plant.

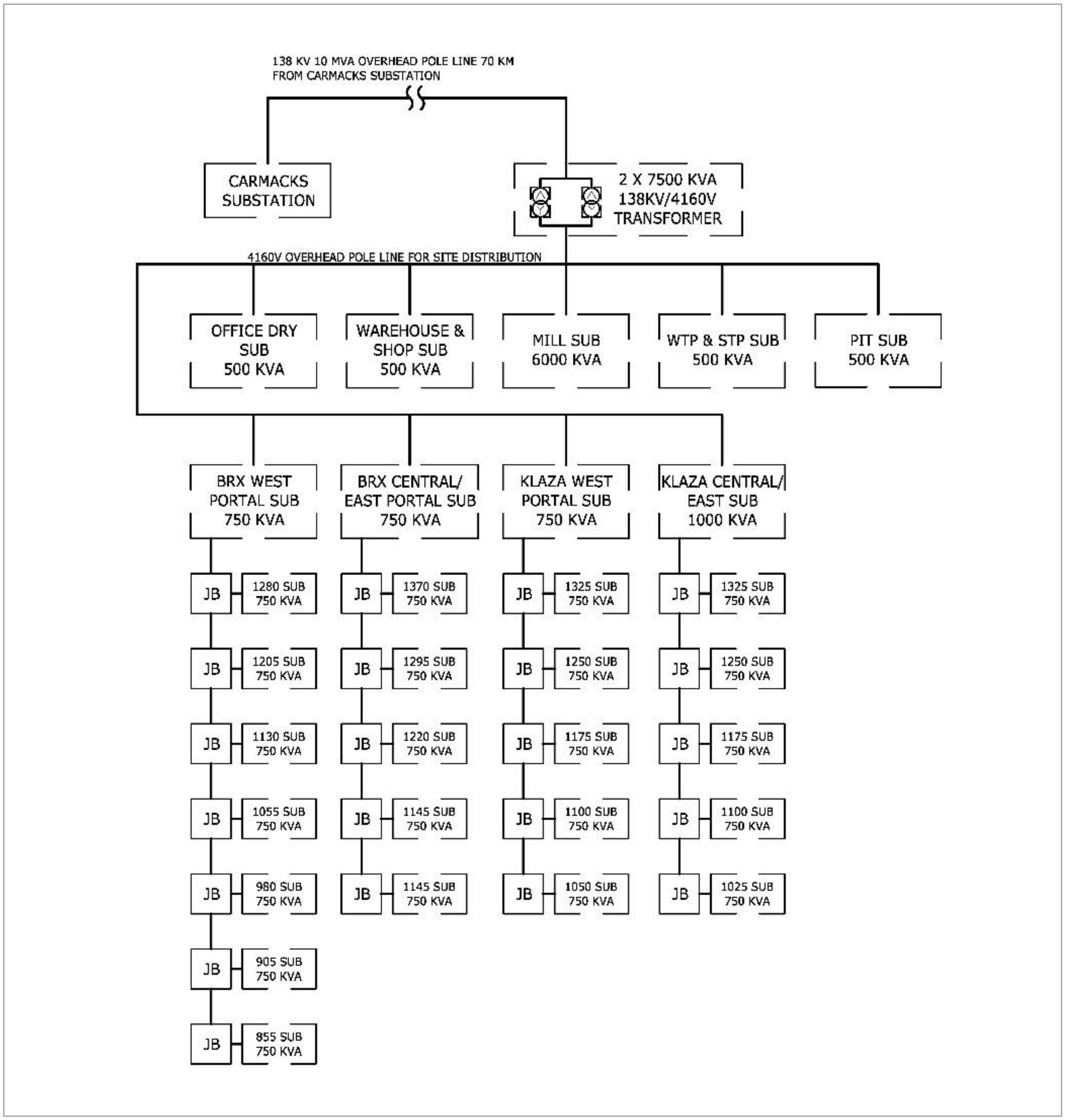
It was assumed that the transmission line would be provided by the utility and paid off over the LOM as a surcharge on the unit cost of power.

18.1.4.2 Distribution

The main electrical supply will consist of a 73 km, 138 kV 10 MVA transmission line from the territorial power grid at Carmacks. The main feeders will be brought to the main sub-station and primary disconnect near the process plant, as this is the location of the largest power load. The main substation will consist of two 7,500 kVA 138 kV / 4,160 V transformers.

4,160 V tapped from the sub-station will then be distributed via pole lines to the mine portals, oxygen plant, open pit mine complex and other site facilities. A block diagram of the distribution is shown in Figure 18.1.

Figure 18.1 Block diagram showing the electrical distribution



Note: JB = Junction box.
Source: AMC Mining Consultants (Canada) Ltd.

Each of the four underground mines will have local sub-stations for the mine supply. At every third level there will be a disconnect switch and junction box that will allow a sub-station to provide power on that level as well as the level immediately above and below. These sub-stations will provide ground fault interruption and voltage reduction to supply mobile equipment, lighting and stationary power loads such as mine dewatering pump stations or maintenance shops.

1 MW of diesel emergency generated power will be available on site to keep critical pumps and fans operating, as well as critical camp power in the event of a power outage.

18.1.5 Water supply

Water for the mine site and processing plant will be supplied from the Klaza river. Knight Piésold (KP) has estimated 685 m^3/d of make-up water is required. The water will be pumped using a 50 hp pump housed near the river via a heated and insulated 0.8 km pipeline to the processing plant for treatment. A modular filtration unit will be used to clean the water for industrial use. Potable water will be supplied via a potable water treatment plant.

18.1.6 Maintenance shop and warehousing

The underground mines will be supported by a centrally located maintenance facility near the offices, a heated warehouse and a cold storage warehouse. The maintenance shop will consist of a pre-engineered steel structure placed on a slab cast on grade. The building will have three maintenance bays and one wash bay. Fast-acting vehicle doors will help prevent the loss of heat from the building during the winter season.

The shop will be sufficient to handle larger, longer and more complex maintenance repairs that will be needed by the underground mining operation. Smaller repairs and routine maintenance will be handled underground.

The maintenance shop will provide administration space and will be attached to the heated warehouse. Both structures will be fitted with sprinklers and fire alarms. The fire water pumps will be installed in the wash bay mechanical room.

The maintenance shop will be equipped with carbon monoxide detection equipment that will activate the high flow ventilation requirement for the shop. Indirect fired radiant heaters will heat the shop floor and allow for the quick de-icing of equipment. Space for tools and the maintenance asset management system will also be provided. The waste oil storage facility will be placed near the shop.

The heated warehouse will provide enough inventory space for daily operations as well as critical maintenance spares. Stock levels for routine and minor maintenance will be set at a one week supply, which will provide enough buffer given the direct access to the mine site. Other major stock items for planned maintenance will be brought in via the Mount Nansen road.

18.1.7 Surface mobile equipment

The site will have the following equipment available:

- Ambulance
- Fire truck
- Grader (CAT 14G or equivalent)
- Loader (CAT 966 or equivalent) equipped with quick disconnect for GP bucket, snow bucket, forks, and boom
- Excavator (Komatsu PC-200 or equivalent)

- Boom Truck
- Telehandler
- 2.5 tonne forklift
- Water truck 4,000 USG
- Mechanics truck
- 2 Haul trucks

The mill and shops will be equipped with overhead bridge cranes and the boom truck can act as a small crane for most light loads. For maintenance and larger lifts, cranes, with operators, can be rented as required from Whitehorse.

18.1.8 Fire detection and suppression systems

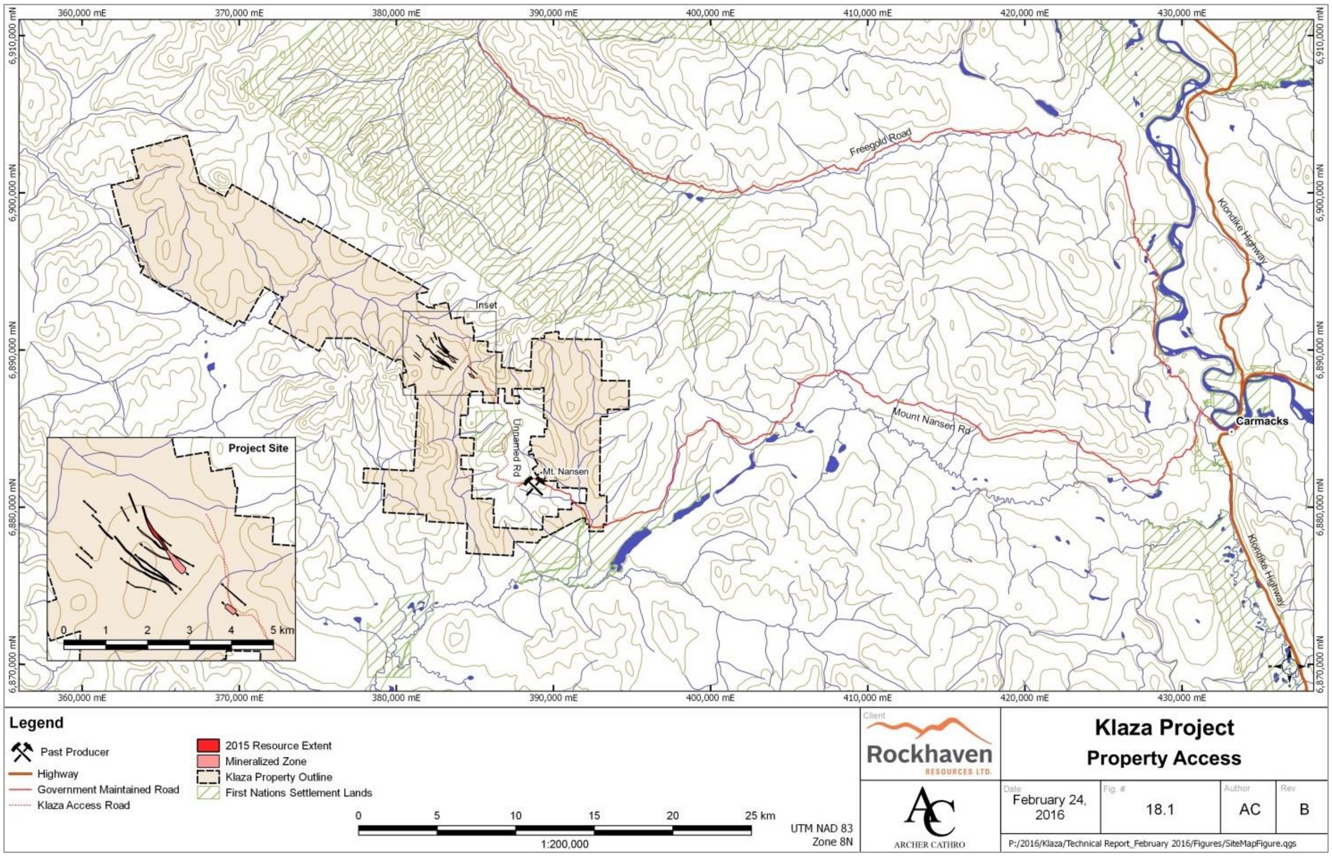
Each of the mine ventilation systems will be provided with an ethyl mercaptan (stench gas) system (activated manually or remotely) in order to warn underground personnel in the event of an emergency. Radio contact via the leaky feeder system provides an alternative method of communication. The supply, exhaust, and heater fans can be shut down or adjusted to assist with fire control systems in the mine. A fire-water supply tank, fire pumps, and distribution system will feed hydrants and Y-fittings in the surface buildings to provide fire-fighting capability to the process plant, mine offices, shops, and mine dry. Buildings (particularly warehouses and shops) will be fitted with alarms and equipped with sprinklers.

18.1.9 Roads

A network of light vehicle roads will be provided to keep personnel vehicles separate from the open pit and underground mine haul traffic. These roads will be constructed in accordance with applicable permafrost design requirements. Generally they will have a one metre thick crushed rock layer, a five metre wide crown, and sloping shoulders on a 3:1 gradient to control snow accumulation.

The final section of the public road to the mine site will be refurbished (approximately 13 km), however the public road to Carmacks will be maintained by the local rural municipality or territory. Figure 18.2 shows the Mount Nansen road from Carmacks to the Klaza project.

Figure 18.2 Klaza project access road



Source: Archer, Cathro & Associates (1981) Limited.

18.1.10 Waste water

Underground mine water from operations, surface water from the open pits, and grey water from the office and mine dry will be routed via dual wall heat-traced high-density polyethylene (HDPE) piping systems, partially or completely buried, to the plant for processing as part of the tailings system.

18.1.11 Explosives

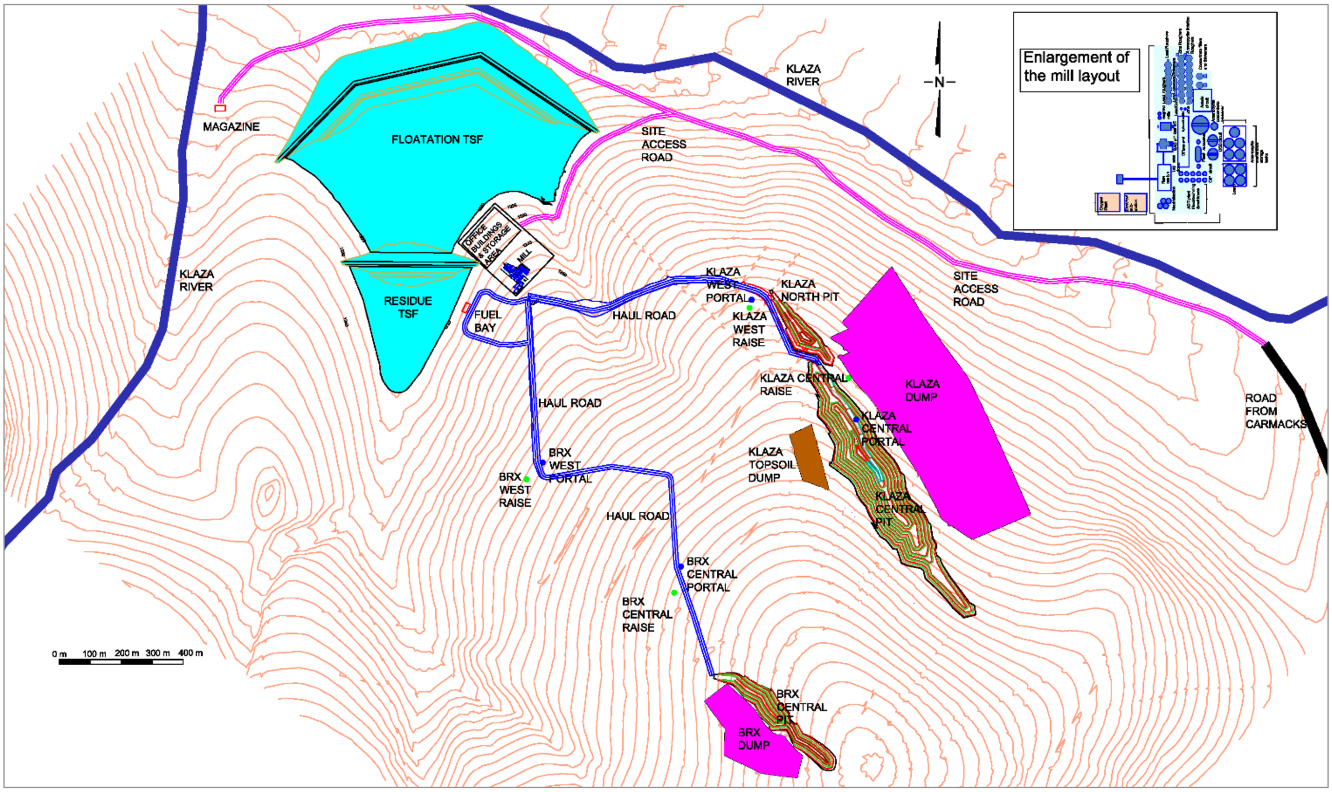
Explosives will be transported into the underground mines and two shifts of capacity stored in simple, secure underground facilities. The majority of explosives will be stored on surface as part of the open pit mine operations. The underground mines will continue to use the open pit explosives facilities when the open pit phase of the project is completed.

18.1.12 Open pit

The open pit facilities will be provided by the contractor and will consist of offices, maintenance shop, warehouse, and the explosives magazine. Part of these facilities could be used for the longer term underground operation and an allowance for this arrangement has been included in the capital cost estimate for surface infrastructure.

A general layout of the surface infrastructure, process plant, offices, roads, waste dumps and stockpiles, and the TSF is shown in Figure 18.3.

Figure 18.3 General layout of the surface infrastructure for Klaza



Source: AMC Mining Consultants (Canada) Ltd.

18.2 Underground infrastructure

Underground mine services will include lunchrooms, a small maintenance shop for minor and urgent repairs, fuel and lubricant storage, and small explosives and caps magazines.

18.2.1 Lunchrooms

The lunchrooms will provide a clean heated space with potable water, tables, and chairs. They will also be used as mine refuge areas. The mine rescue team will be able to use the space for training and to store equipment and supplies. A lunchroom will be provided in each of the four main mining areas.

18.2.2 Workshop

A six bay maintenance shop with an overhead crane will be constructed on surface to maintain the underground mine equipment as well as the light duty surface equipment. In addition, a three bay maintenance shop will be provided for open pit equipment maintenance.

Each mine will have a single underground service bay with a concrete floor and monorail for light duty servicing only. All major rebuilds and scheduled maintenance will be conducted in the surface shop.

18.2.3 Fuel and lube bay

A fuel and lubrication area will be provided underground in each mine. While most fueling will be conducted via tankers from surface, some storage via Satstats® will be provided underground.

Storage will also be provided for lubricants and waste oil. Satstats are self contained fueling systems complete with fire doors and a fire suppression system.

18.2.4 Explosives magazine

The explosives magazine located in each mine will be a few rounds deep and equipped with a lockable door and wooden benches covered with rubber matting. The space will be ventilated and kept cool. The intent is to provide a small stockpile of detonators, cord, and high velocity explosive for daily task specific activities. Explosives handling and delivery from surface will be accomplished using mobile loading equipment drawing from the surface magazine.

18.2.5 Compressed air

Compressed air will be supplied by two separate compressor plants, one for each of the two underground mining areas. Compressed air will be carried underground via DN150 HDPE pipe down the main ramps. Reticulation on the levels will be via DN100 HDPE pipe.

18.2.6 Service water

Mine water will be supplied via DN100 HDPE pipe down the declines that will be installed as the decline progresses. At required levels, pressure reducing and isolation valves will be installed to maintain the system at operating pressures. A two inch distribution system of DN50 HDPE pipe, together with hoses will be laid out on the operating levels.

18.2.7 Communications

Underground telecommunications will be provided by a conventional leaky feeder system strung down the decline and feeding the operating levels.

18.2.8 Dewatering

The mine dewatering system will consist of staged 50 hp submersible dirty water pumps at nominally 50 m vertical intervals. Each sump will have two pumps to provide continual redundancy. A DN100 HDPE line will stage water between each pump station. An insulated DN100 line will carry pumped water from each mine and direct it to the process water treatment facility.

Mine water is largely expected to be a product of the mining activities (drilling and washing the face and muck piles) with minimal ground water inflows. Dewatering facilities will be installed as the mine development proceeds.

18.2.9 Mine escape and rescue

Portable refuge stations will be located appropriately relative to operating levels. Lunchrooms near the maintenance area will also serve as refuge stations. Self-rescue storage will be provided in the lunchrooms as well as first aid kits at the refuge stations.

Main egress is provided by the declines and a second means of egress via the ladders in the ventilation raises.

18.3 Logistics

The Klaza mine location does not pose significant logistical challenges that may affect the movement of people to and from the site, supplies inbound, and concentrates outbound. There is an existing road to the site. This entire road except for the final 13 km to the site is maintained by the Yukon department of highways year-round. Apart from capital and emergency spares, processing reagents and fuel, most necessary materials and supplies can be brought to site as required during the year. A warehouse and cold storage will be constructed to accommodate any critical items.

18.4 Inbound freight

Freight will be shipped in from Carmacks via the main access road. Public all-weather road access to tidewater is available at Skagway (5 hours) and Prince Rupert (20 hours). Inland freight access is also available from Edmonton. No special or unusual freight handling requirements are expected.

Laydown areas will be provided on surface as well as cold storage and in the heated warehouse.

18.5 Outbound concentrate

An area for loading, unloading and servicing concentrate and other trucks will be established at the plant site. Concentrate will be loaded directly into steel boxes for shipment to markets via the main road access. Concentrate containers will be handled by forklift. Concentrate will not be stockpiled at site but will be transported as soon as a container is full.

18.6 Fuel storage

Fuel storage will consist of two tanks that will have the capacity to support two months consumption at peak production. The tank system will be enclosed by a lined berm of sufficient capacity to contain 110% of the contents of a full tank in the event of a major leak or spillage. Fuel will be trucked to site on a year-round basis.

18.7 Tailings disposal

Tailings disposal is discussed in Section 17.

18.8 Stockpiles

No large mineralized rock stockpiles are anticipated over the life of the mine. ROM stockpiles will be located in areas that will allow control of potential run-off. The stockpiles will be affected by freezing and, as such, these will be kept to a minimum. Material from the mine will be loaded from the ROM pad onto a conveyor and fed into the storage bin located at the process plant.

19 Market studies and contracts

19.1 Marketing

Initial metallurgical tests showed elevated levels of penalty elements in the forecast lead and zinc concentrates. In addition, the low levels of zinc in zinc concentrates and the high levels of gold in lead concentrates were seen to be potentially difficult in the marketing of both concentrates.

Rockhaven undertook an additional degree of research regarding the marketability of the concentrates in 2016 in order to assess the viability of the products produced and identify any potential detriments. The high percentage of precious metal reporting to the concentrates would potentially impact marketability and lead to inclusion of the Acacia leach process. In addition, some effort was made to reduce the potential deleterious elements present in the concentrates and, thereby, improve the marketability. Potential testing to further reduce the deleterious elements is warranted.

Prices shown in the following sections were taken from the 2016 PEA and have not been updated.

Test results from the additional research showed a significant lowering of concentrate impurities, an increase in the base metal grades of the concentrates and a shift of the recovered gold from the lead concentrate to doré.

While Rockhaven has investigated possible markets and potential terms, no detailed market study has been undertaken at this stage of the project.

It is envisaged that the gold-rich zinc concentrate would be sold primarily to smelters in Asia.

The lead concentrate could also potentially be sold to smelters in Asia or other offshore smelters.

If sold to a North American smelter, transportation costs may be reduced, but it is reasonable to anticipate that these savings, if any, might need to be shared with the smelter.

Both lead and zinc concentrates would be expected to incur some penalties for impurities.

19.2 Lead concentrate treatment terms

Treatment terms for the lead and zinc concentrates used to estimate the NSR values have been advised by the 2016 market research study and are shown in Table 19.1 and Table 19.2 respectively.

Table 19.1 Projected lead concentrate terms

Treatment terms	Value
Gold payment terms (% of contained metal in concentrate)	95%
Minimum deduction from gold grade	1.0 g
Silver payment terms (% of contained metal in concentrate)	95%
Minimum deduction from silver grade	50 g
Lead payment terms (% of contained metal in concentrate)	95%
Minimum deduction from lead concentrate grade	3 units
Lead Concentrate treatment charge	$205/dmt
Deduction for penalty elements (per tonne of concentrate)	$130.00/dmt
Price participation-threshold price per tonne of lead metal in concentrate	$1,700/mt
Price participation for each dollar the metal price is below threshold price	$0.065/dmt
Price participation for each dollar the metal price is above threshold price	$0.065/dmt
Gold refining charge (% of gold price) applied to payable gold metal	1.7%
Silver refining charge (% of silver price) applied to payable silver metal	9.5%

Source: H. M. Hamilton & Associates Inc.

19.3 Zinc concentrate treatment terms

Table 19.2 Projected zinc concentrate terms

Treatment terms	Value
Gold payment terms (% of contained metal in concentrate)	70%
Initial deduction from gold grade	1.0 g
Silver payment terms (% of contained metal in concentrate)	70%
Initial deduction from silver grade	3 oz
Zinc payment terms (% of contained metal in concentrate)	85%
Minimum deduction from zinc concentrate grade	8 units
Zinc Concentrate treatment charge	$170/dmt
Deduction for penalty elements (per tonne of concentrate)	$27.00/dmt
Price participation threshold price per tonne of zinc metal in concentrate	$1,000/mt
Price participation for each dollar the metal price is below threshold price	$0.10/dmt
Price participation for each dollar the metal price is above threshold price	$0.10/dmt

Source: H. M. Hamilton & Associates Inc.

19.4 Transportation

Deliveries would be in containers to Asian ports, most likely to Japan, South Korea, or Northern China. The containers would be trucked from the mine site to the port in Skagway, Alaska, barged from Skagway to Seattle, Washington, and then transferred to a container vessel for shipment to Asia. These deliveries are assumed to be delivered to the Asian port with the cost of the freight and insurance (CIF) for the shipper's account.

19.4.1 Total transportation costs

On the basis of a moisture content of 8% and using a C$:US$ exchange rate of 0.72:1, the total transportation cost is projected to be US$155 per dry metric ton.

20 Environmental studies, permitting, and social or community impact

The Section refers to the Study Area which is defined to include those lands within the mineral claim boundary and Quartz Mining Land Use approval LQ00434, and the alignment of the access road, commencing at the Mt. Nansen Mine Road. The Study Area overlies multiple Placer Claims held by other claim holders.

The following section summarizes the existing environmental and social studies broadly relevant to the Study Area or commissioned specifically for the Project, or as part of earlier exploration activities. This section also identifies any known and / or apparent environmental issues that could potentially influence Rockhaven's ability to extract the Mineral Resource.

20.1 Summary of available environmental and socio-economic information

Based on the information reviewed as part of this PEA, much of which is quite general and / or limited in scope and content, there are no known significant environmental issues or sensitive receptors / features that could influence project viability, nor affect the major design components for future mine development.

20.1.1 Government databases

A search was conducted of Yukon government databases to gather existing data on environmental conditions in the Study Area. Database searches were conducted for both the Klaza and Mt. Nansen mining area, due to the Project's proximity to the Mt. Nansen mine and in an effort to try and capture data that was specific to the Mt. Nansen mine project area (undergoing reclamation). The following Government databases were reviewed:

- Yukon Fisheries Information System (FISS) and Fish Sampling (FISS 2017).
- Yukon Lands Viewer including Species at Risk and Wildlife (GeoYukon 2020).
- Yukon Mining Map Viewer (GeoYukon 2020).
- Environment Yukon – Water Resources Branch (hydrometric stations).

20.1.2 Existing environmental studies and data

Table 20.1 provides a summary of existing environmental and social studies, either underway or completed. Studies were undertaken by consultants on behalf of Rockhaven, available through government databases or were completed for nearby projects in proximity to the Study Area. The depth of information contained in the existing studies and data varies between areas of environmental consideration. For example, there is a significant amount of 'raw' air quality / climate data available for the Study Area, but very little analysis and interpretation of this data. Conversely, for wildlife and species at risk there is significant analysis and interpretation of the (wildlife) raw data and it is presented in a format that can be easily drawn upon to guide and refine future studies. In addition to identifying what environmental considerations have available raw data, Table 20.1 also identifies which environmental consideration has undergone some degree of meaningful assessment / discussion / interpretation. Section 20.1.3 further examines the depth and adequacy of the meaningful assessment / discussion / interpretation for those identified environmental considerations.

Table 20.1 Summary of existing environmental studies and data

Environmental consideration	Study name and details	Data source	Additional information / discussion
Air quality / climate	Klaza Camp continuous meteorological monitoring. Parameters include: Temperature, precipitation, wind speed and direction; and solar.	J. Gibson, 2017	Site specific for all of 2013 through 2016.
	Carmacks historical climate data – Carmacks Airport	Environment Canada	Regional data
Terrain	Klaza Property - Terrain and Geohazard Assessment and Access Route Evaluation Report, including supporting figure.	EBA consultants, May 2013	See Section 20.1.3.1
Surface hydrology and water quality	Baseline Water Quality Assessment / Hydrology Survey (2012-2015), Ten monitoring stations across Study Area identified in raw data sets.	J. Gibson, 2012 - 2017	Site specific and adjoining waterways.
	Mt. Nansen 09CA-SC01 Yukon Snow Survey 1976-present; seasonal monitoring; Yukon Snow Survey network.	Environment Yukon, 1976-present	
	Hydrometric: Nisling River below Onion Creek; WSC 09CA006; streamflow discharge, surface water level; 1995-present; continuous.	Environment Canada, Water Survey of Canada	
	Nisling River at Klaza confluence. WH-DO-NI02; Placer Water Quality Objectives Monitoring; 2008-present; periodic / seasonal water chemistry.	Site specific data: 2008-present	
	Nisling River upstream of Nansen Creek WH-DO-NI04; Placer Water Quality Objectives Monitoring; 2008-present; periodic / seasonal.	Site specific data: 2008-present	
	Klaza River Station YPS-323 2008-present; infrequent; CABIN protocol for FW Quality monitoring (water chemistry; aquatic organisms; invertebrates; nutrients).	Site specific data: 2008-present	
	Nansen Creek YPS-321 2008-present; infrequent; CABIN – same parameters as Klaza Station	Yukon Water, 2008 - present	
Hydrogeology	Preliminary Hydrogeological Assessment, Klaza Property, Yukon	Tetra Tech EBA Inc. December 2015	See Section 20.1.3.2
Fisheries and aquatic resources	Baseline Aquatic Studies, Klaza Project, 2014	Laberge Environmental Services, February 2014	See Section 20.1.3.3
Wildlife and species at risk	Helicopter Wildlife Surveys – Moose and Caribou November 2012, February 2013, May 2013	Laberge Environmental Services, 2012 - 2013	See Section 20.1.3.4
	Class 3 Quartz Exploration- Klaza Property – 2015-0148, YESAB Mayo Designated Office Evaluation Report.	YESAB, November 2015	See Section 20.1.3.4
Access, land use, mineral tenure, and protected areas	**Access:** Klaza Property – Terrain And Geohazard Assessment and Access Route Evaluation Report.	Tetra Tech EBA Inc., May 2013	Multiple alignment options available.
	Land tenure and land use: Preliminary baseline information can be obtained through database interpretation.	GeoYukon	See Section 20.1.3.5
	Mineral tenure: • Quartz Claims – Archer Cathro / Rockhaven • Class III Mining Land Use Approval LQ00434, expiring 6 December 2020 • Placer Claims (creek claims)	GeoYukon	See Section 20.1.3.5

Environmental consideration	Study name and details	Data source	Additional information / discussion
	Protected areas: None identified within the Study Area. The nearest protected area is the Nordenskiold Habitat Protection Area, approximately 60 km to the south-east.	GeoYukon	
Heritage	Heritage Resources Overview Assessment for the Klaza Property.	Matrix Research Ltd, April 2011	See Section 20.1.3.6
	Heritage Resources Impact Assessment for the Klaza Property.	Matrix Research Ltd, June 2013	See Section 20.1.3.6
	Heritage Resource Overview Assessment: 2019 Klaza Property Class 3 Quartz Exploration Areas	Ecofor Consulting Ltd, Novemeber 2019	See Section 20.1.3.6
	Freegold and Mt. Nansen Road Assessment	G. Hare, 1995, 1997	See Section 20.1.3.6
Traditional land use	Some traditional land use and traditional knowledge information from Little Salmon Carmacks First Nation is available for the area (pers. comm. S. Wright).		
Socio-economic	Very limited data currently available: • Trapping Concession ID 148 • Outfitting Concession ID 13	YESAB, November 2015	

Source: Archer, Cathro & Associates (1981) Limited.

20.1.3 Discussion on available environmental information

As summarized in Table 20.1, it is evident that there has been a steady development of environmental, and to a lesser extent social, studies undertaken for the Project to support exploratory activities at the Klaza Property. There is also a limited amount of high-level data available from Government databases that is relevant to the Study Area. It is understood that this available data will need to be supplemented with detailed and targeted baseline data as the project progresses and regulatory requirements, including YESAA, are addressed.

The following subsections provide a synopsis of the data available for certain environmental considerations identified in Table 20.1.

20.1.3.1 Terrain and access

The terrain and access assessment by Tetra Tech EBA was a desktop study undertaken to scope terrain and geophysical characteristics of a 2 km-wide access corridor for the Project. The assessment included:

- A high level topographical, hydrological and geological summary of the (reports') study area.
- An assessment of the terrain including surficial geology, hazards and drainage.
- The existing access route to the mine site including alternative alignment options and sources for road building material (borrow pits).

The assessment was based solely on aerial photograph interpretation with no ground-truthing or verification done in the field.

20.1.3.2 Hydrogeology

The purpose of this preliminary hydrogeological assessment was to initiate the collection of hydrogeological information in the area of the currently understood main mineralized zone at the Klaza Property. The preliminary hydrogeological assessment involved the successful installation of a preliminary monitoring well network down-gradient of the main mineralized zone. This assessment identified the following:

- Permafrost appears to act as a confining layer for the deeper bedrock aquifer.

- The groundwater flow regime at the site is controlled by the steep terrain with groundwater flow from areas at higher elevations on the mountain slopes toward the valley bottoms and generally mimicking the local topography.
- Groundwater monitoring results show all groundwater samples are of a calcium and / or magnesium-dominant cation type, and bicarbonate and / or sulphate anion type and show a near neutral to slightly basic pH (between 7 and 8) reading.
- Groundwater chemistry results showed several natural exceedances of the FIG guidelines. Exceedances included sulphate, fluoride, and the dissolved metals aluminum, cadmium, copper, iron, lead, selenium, silver, and zinc.

The preliminary hydrogeological assessment states that the information collected as part of the assessment will be very useful for the design of a comprehensive hydrogeological baseline and effects assessment that will form part of the environmental assessment (EA). Additional data collection will be required to satisfy the requirements under YESAA and the *Yukon Waters Act* if the project moves ahead toward the EA and approval process. The preliminary hydrogeological assessment makes a number of recommendations, including but not limited to:

- A minimum of one year of baseline groundwater data will be required to support future approval / regulatory requirements.
- Collecting additional ground temperature and hydrogeological data from the existing observation and monitoring wells to update the preliminary conceptual hydrogeological model with an emphasis on permafrost-groundwater interaction.
- Additional monitoring wells should be installed in the areas up and down-gradient of proposed mine infrastructure.
- The groundwater and surface water baseline data collection should be integrated and both data sets be interpreted with respect to groundwater-surface water interaction.

20.1.3.3 Fisheries and aquatic resources

Rockhaven commissioned Laberge Environmental Services in September 2014 to conduct aquatic baseline surveys on sites on the upper Klaza River. Laberge Environmental Services assessed the water quality, stream sediment geochemistry, benthic invertebrate populations and fish assemblage. The results of this aquatic baseline survey include:

- Water Quality and Sediment load.
- The abundance, taxonomic richness and distribution of benthic invertebrates at the survey sites.
- Fish species, aquatic habitat, fish distribution and abundance, and metal contaminants in fish.

20.1.3.4 Wildlife and species at risk

Rockhaven commissioned Laberge Environmental Services for three helicopter wildlife surveys during November 2012, February 2013, and May 2013. The purpose of these wildlife surveys was to determine the winter distribution and abundance of large mammals, primarily moose and caribou, within the Study Area. These studies concluded that there were low densities of Moose and Caribou observed in the Study Area.

In addition to these helicopter wildlife surveys, the YESAB Designated Office Evaluation Report (2015) provides a more detailed assessment of the potential Wildlife and Species at Risk requirements in the Study Area. The YESAB report is the most comprehensive and complete 'picture' of environmental and approval requirements prepared for the Project to date. Key wildlife and Specifies at Risk findings from the YSEAB review identified:

- Terrestrial wildlife species of concern that may use the Project Area include, but are not limited to:
  - Little Brown Myotis (Myotis lucifugus) – SARA Schedule 1: Endangered.
  - Northern Mountain Population of Woodland Caribou (Rangifer tarandus caribou) - SARA Schedule 1: Special Concern.
  - Gypsy Cuckoo Bumble Bee (Bombus bohemicus) - COSEWIC: Endangered.
  - Western Bumble Bee (Bombus occidentalis mckayi) – COSEWIC: Special Concern.
  - Collared Pika (Ochotona collaris) – COSEWIC: Special Concern.
  - Wolverine (Gulo gulo) – COSEWIC: Special Concern, Western population.
  - Grizzly Bear (Ursus arctos) – COSEWIC: Special Concern.
- Migratory birds that are at risk and may be found in the Project Area include:
  - Common Nighthawk (Chordeiles minor) – SARA Schedule 1: Threatened.
  - Olive-sided Flycatcher (Contopus cooperi) – SARA Schedule 1: Threatened .
  - Bank Swallow (Riparia riparia) – COSEWIC: Threatened.
  - Barn Swallow (Hirundo rustica) – COSEWIC: Threatened.
- Rare plant species that may be found in the Project Area include:
  - Yukon Woodroot (Podistera yukonensis).
  - Ogilvie Mountain Spring Beauty (Claytonia ogilviensis).

20.1.3.5 Land use, mineral tenure, and protected areas

While there has not been any form of detailed land use baseline assessment done for the Study Area, preliminary findings can be deduced through reviewing and interpreting results that are available through existing government databases. The Yukon Lands Viewer and Yukon Mining Viewer show all of the existing tenure (surface and subsurface tenure) as well as any active land use permits in the Study Area. These databases also identify Current Mineral Tenure of the Project and over / underlying Mineral Tenure including Placer Claims held by Canaan Gold Resources and Kehong Wu. It is noted that Placer Claims follow the alignment of almost all surface water drainage features on and adjoining the Study Area.

Existing / Historical Mineral Tenure held by Rockhaven includes:

- Class III Quartz Exploration - Klaza Property – 2008-0086, YESAB Office Evaluation Report - not reviewed.
- Class III Quartz Mining Land Use Operating Plan Application 2011.
- Class III Quartz Mining Land Use Operating Plan Application 2011 – additional information request response.
- Class III Quartz Exploration - Klaza Property – 2011-0007, YESAB Office Evaluation Report.

20.1.3.6 Heritage impact assessment

Rockhaven commissioned Matrix to complete a Heritage Resource Overview Assessment (2011) and a Heritage Resources Impact Assessment (2013) as required by YESAB for the project to progress through the regulatory framework.

Typically, heritage resource overview assessments are conducted to determine the potential for locating heritage / archaeological sites within the Study Area. Where warranted, more detailed heritage resource impact assessments are then conducted to identify and evaluate the significance of heritage / archaeological sites within the Study Area, evaluate possible impacts to those sites from development and recommend appropriate impact management measures where necessary.

The Heritage Resource Overview Assessment found numerous areas associated with hydrological features or distinct landforms that have been classified as moderate to high heritage resource potential. The overview assessment recommended that further heritage resource investigations be conducted within these areas prior to any potentially ground disturbing development activities.

Subsequently the Heritage Resource Impact Assessment found that no heritage resources were identified within the Study Area. However, one newly identified precontact heritage site was located outside of the (Heritage Resource Impact Assessment) Study Area and recorded during the assessment. This assessment also included management recommendations to mitigate potential impacts to heritage resources and instructions to personnel and contractors should heritage resources unexpectedly be uncovered.

It must be noted that the Heritage Resource Impact Assessment was not intended to evaluate or comment on the traditional First Nation land use of the areas in which development is proposed.

In 2019 a Heritage Resource Overview Assessment was conucted in exploration areas on the Property, east of the Study Area. While this assessment identified multiple areas with elevated potential for surface / subsurface heritage resource sites, none are within the Study Area.

20.1.3.7 Available information summary

The level of information contained in the existing environmental and social data is sufficient to facilitate scoping of a comprehensive environmental baseline study for meeting future approval requirements. Any future baseline studies would require biophysical and socio-economic considerations and would build on the information gathered to date with an aim of filling the identified gaps and supplementing any existing information with more recent data.

Once the initial stages of mine planning have been completed and conceptual level detail is determined, it will be possible to identify and define a baseline assessment program that is relevant to the Project, the Study Area and the level of information currently available.

20.1.4 Identified (to date) environmental concerns

While no ‘showstoppers’ or ‘roadblocks’ to the future development of the Project have been identified as part of the PEA, recent exploratory activities undertaken by Rockhaven Resources have been assessed by the YESAB through an application for a Class III Quartz Exploration license. This YESAB assessment accorded to the preliminary findings of this PEA in that the following valued environmental and socio-economic components required specific attention and warrant further detailed investigation:

- Wildlife and Wildlife Habitat including Species at Risk, Moose, Caribou and Raptors.
- Environmental Quality including release of deleterious substances, introduction of invasive plants and the loss of rare plants.
- Heritage Resources.

The following environmental considerations have not been identified in the environmental studies undertaken to date:

- Traditional Land Use.
- Socio-Economic baseline information.

20.2 Approvals and permitting

20.2.1 Yukon Environmental and Socio-Economic Assessment Board (YESAB)

Before projects proceed to the licensing phase, they are first assessed through an EA. The YESAB administers EAs in the Yukon. The Project will be subject to an EA under the YESAA. YESAB is an independent Board established in 2005 to assess most projects in the Yukon for environmental and socio-economic effects. The Project will require an assessment under the YESAA because it will involve the construction of a mine and many of the proposed activities are considered assessable under the YESAA regulations.

The level of assessment for the Project will be at the Executive Committee Screening level, as activities proposed as part of the Project exceed the thresholds listed in *Schedule 3 of the Activities Regulations*. One of these thresholds is mine production rate, and it is understood that the proposed Klaza mine is envisaged to operate at a production rate of 1,900 tpd; the threshold in *Schedule 3* is 1,500 t/day.

To adequately identify the potential environmental and socio-economic effects of a project, a project proposal must be submitted to YESAB that contains sufficient information about the environment and the proposed project development. Baseline studies will play a key role in the Klaza project for developing the project proposal, as the review of existing data sources indicated that there is somewhat limited information available for the area. Rockhaven has already begun to collect site data (weather, aquatics, and wildlife observations) that will be useful to develop a longer term baseline of site specific information.

20.2.2 Existing approvals and permits

20.2.2.1 Current exploration activities

Exploration activities are subject to Mining Land Use Regulations of the Yukon Mining Quartz Act and are assessed under the YESAA. YESAB undertakes an assessment of exploration activities, and issues its recommendations and a Decision Document. Once a Decision Document is issued, a Proponent then obtains all necessary permits, including the Mining Land Use approval, before large-scale exploration is conducted.

Approval for the current exploration activities has been obtained by Rockhaven under:

- Class III Mining Land Use Approval LQ00434, expiring 6 December 2020.

20.2.2.2 Quartz mining license

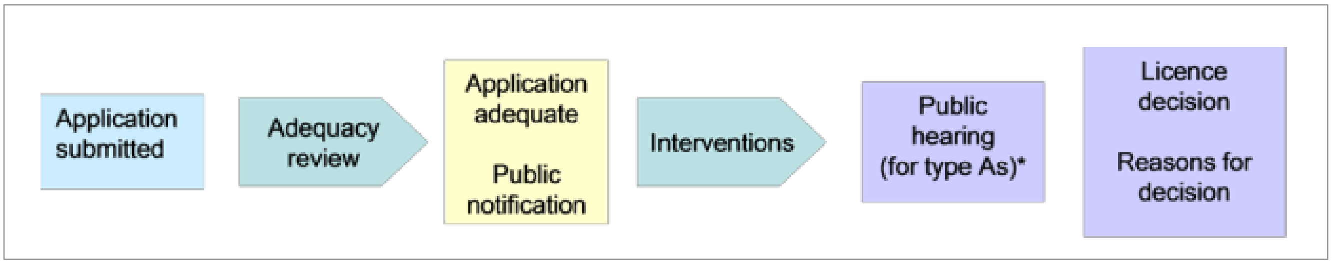
Under the *Quartz Mining Act* a Quartz Mine License is issued and administered by the Yukon Department of Energy, Mines and Resources, and enables the Government of Yukon to regulate mining developments. Any operator who wishes to construct a facility or do physical work in support of the commercial production of most minerals (other than placer gold and coal) will require a Quartz Mining License. This applies to all mines whether or not they have an existing Water License. A Quartz Mining License is required before development or production can begin. Although permits and licenses cannot be granted prior to the completion of the YESAB assessment, regulatory processes can be initiated while the assessment is in progress. The Quartz Mining License contains terms and conditions regarding reclamation of mining activities as well as financial security for reclamation and closure activities.

20.2.2.3 Water license

A Type A Water License under the *Waters Act* will be required for the mining project and is issued and administered by the Yukon Water Board. This Act regulates the use of, and deposit of waste in, water in the Yukon.

The Yukon Water Board reviews all applications for Water Licensing, but requires the YESAB Decision Document, as described in Section 20.2.1, in order to proceed with Licensing. Figure 20.1 depicts the process for a Water license application.

Figure 20.1 Water license application process



Source: Archer, Cathro & Associates (1981) Limited.

The license will include conditions related to water use and waste disposal, water control and diversion structures, the submission of studies and plans, monitoring and surveillance, and the modification and construction of water-related structures. The detailed information required for Type A Licenses for mining projects can be found on the Yukon Water Board's website: http://www.yukonwaterboard.ca/forms_info.html. It is expected that the licensing process would take up to one year, following submission of the completed license application.

20.3 First Nation consultation

The Project is located in the traditional territories of LSCFN and SFN. The following section summarizes the known First Nations and wider community consultation and engagement activities that Rockhaven has undertaken.

20.3.1 Little Salmon Carmacks First Nation

The Klaza Property is primarily located within the traditional territory of the LSCFN, although a small section of the mineral claim block overlaps with the SFN traditional territory. When Rockhaven acquired the Property in 2009, they contacted LCSFN to provide information about the extent of exploration work proposed. Rockhaven and LSCFN are in regular communication to ensure that information about exploration activities, proposed work, and the project is discussed. During a site visit held with Chief Eric Fairclough in August 2015, Rockhaven and LSCFN formalized their relationship through an EBA that includes employment, business and financial opportunities for the LSCFN.

There has been extensive involvement and interaction between Rockhaven and LSCFN over the course of several years regarding the Project, with ongoing discussions surrounding potential contract / employment opportunities for LSCFN. It is understood that both parties are working towards an Impact Benefits Agreement with further details about the scope and content of this Impact Benefits Agreement still being determined.

During the preparation of the 2016 PEA, Morrison Hershfield contacted LSCFN to explore what information exists (i.e. local indigenous knowledge, traditional land use) that might be accessible to the project team, or that could be generated.

It was learned that traditional land use information exists, and the First Nation is presently setting up a land data base for their traditional territory to better organize existing spatial information. It is anticipated that, provided appropriate agreements are made regarding confidentiality of information, this information could be made available to Rockhaven as the project advances through the baseline and project design phases.

20.3.2 Selkirk First Nation (SFN)

There has been limited contact with the SFN to date based on the understanding that no project activity is proposed to occur within or in proximity to their traditional territory. Rockhaven will recommence dialogue with SFN should any activity be planned in their traditional territory.

20.4 Closure and remediation

20.4.1 Environmental liabilities

Outstanding environmental liabilities relating to the Property are currently limited to progressive reclamation during seasonal exploration activities and final decommissioning required prior to expiration of the Class 3 Quartz Exploration Land Use Approval. Progressive reclamation generally entails backfilling or re-contouring disturbed sites and leaving them in a manner conducive to re-vegetation by native plant species. Back-hauling of scrap materials, excess fuel and other seasonal supplies is also done.

Final decommissioning requires that all vegetated areas disturbed by the exploration activities be left in a manner conducive to re-vegetation by native plant species, all petroleum products and hazardous substances be removed from the site, all scrap metal, debris and general waste be completely disposed of, structures be removed, and the site be restored to its previous level of utility.

Invasive species control is a concern for most disturbed areas in Yukon and it is expected that stringent management measures will be implemented to limit the spread of invasive plants (http://www.env.gov.yk.ca/animals-habitat/invasiveplants.php).

20.4.2 Reclamation and closure planning

According to the Reclamation and Closure Planning for Quartz Mining Projects guidance document published in 2013 by the Yukon Water Board and the Yukon Department of Energy, Mines and Resources:

A Reclamation and Closure Plan (RCP) describes how a quartz mine will be reclaimed and closed to return the site to an environmentally stable condition suitable for future land uses. The plan also provides the basis for estimating the financial liability associated with a mining project.

The RCP is intended to remain a living document and is required to be updated and revised as the project progresses. A standalone RCP will be developed that will address regulatory requirements and provide reclamation and closure activities based on the design and layout of mining infrastructure areas, the location of plant and equipment operating and laydown areas, offices and camp infrastructure, waste dumps and the chosen method of tailings management. The RCP will also ensure impacts to wildlife, wildlife habitat and other land uses in the Study Area will be minimized.

21 Capital and operating costs

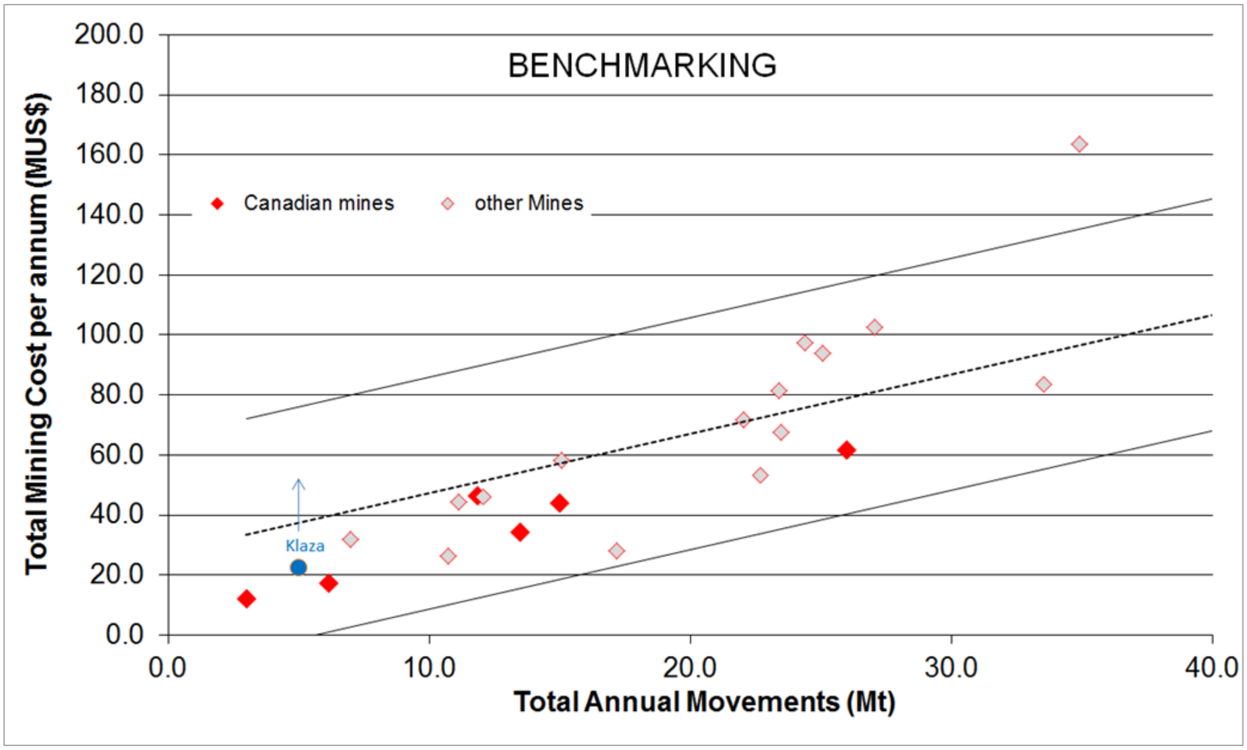
21.1 Operating cost estimate

The operating cost estimate allows for all labour, equipment, supplies, power, consumables, supervision, and technical services.

21.1.1 Open pit

AMC estimated open pit mining costs assuming a contractor mining operation. Estimated costs for the proposed fleet and labour were sourced from AMC's database and benchmarked against knowledge of similar sized, local operations as presented in Figure 21.1. The LOM average mining cost is approximately $4.5/t mined.

Figure 21.1 Open pit benchmarking costs



Source: AMC Mining Consultants (Canada) Ltd.

A summary of the benchmark cost split and AMC's estimate for the open pits is provided in Table 21.1.

Table 21.1 Summary of estimated open pit operating cost

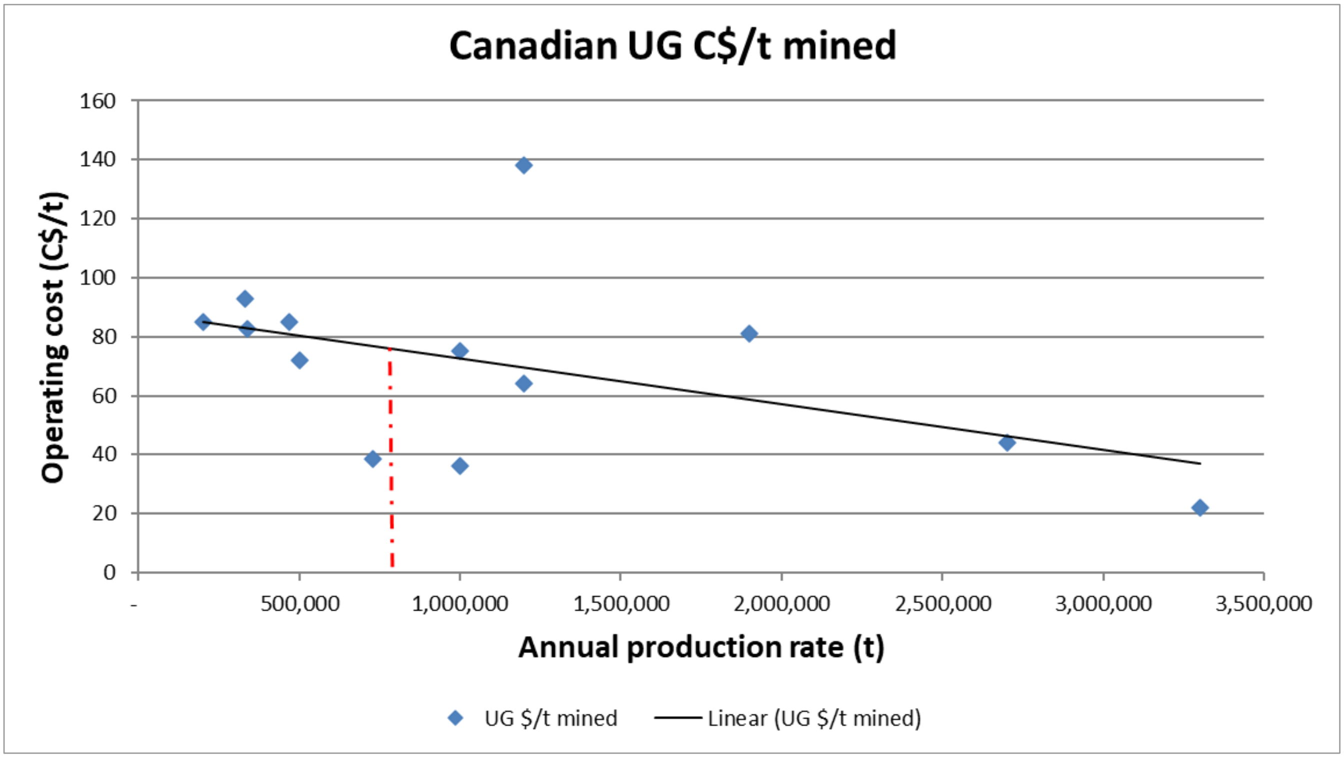
Category	%	Klaza estimate ($/t)
Drill and blast	24	1.08
Load	19	0.86
Haul	13	0.60
Ancillary	10	0.45
Labour	34	1.51
Total	**100**	**4.50**

Note: Totals may not add up exactly due to rounding.
Source: AMC Mining Consultants (Canada) Ltd.

21.1.2 Underground mine

Operating costs for mining are based on AMC's database of underground mine costs and knowledge of international operations. AMC has referenced the benchmark costs against a number of Canadian mining operations located either in the Yukon or the Northwest Territories. The benchmark costs for underground mines in the Yukon and Northwest Territories are shown in Figure 21.2.

Figure 21.2 Canadian underground benchmark mining costs ($/t)



Source: AMC Mining Consultants (Canada) Ltd.

AMC has used a benchmark mining cost of $52.5/t for mineralized material extracted by stoping and $100/t when extracted by development (tonnage ratio of 5:1 for projected LOM). Cost for waste development is estimated at $4,500/m, for an overall mining cost of $58.40/t. When considered against comparable local mining costs and accounting for variations in mining method and backfilling, the estimated mining cost for Klaza is well supported by the benchmark data. A summary of the benchmark cost split and AMC's estimate for Klaza is provided in Table 21.2.

Table 21.2 Summary of estimated underground operating cost

Category	Benchmark split ($/t)	%	Klaza estimate ($/t)	%
Labour	24.2	46	32.1	55
Power	7.9	15	5.8	10
Consumables	14.7	28	15.2	26
Services	3.7	7	3.5	6
Other	2.1	4	1.8	3
Total	**52.5**	**100**	**58.4**	**100**

Note: Totals may not add up exactly due to rounding.
Source: AMC Mining Consultants (Canada) Ltd.

21.1.3 Processing

This cost estimate was generated by BCM.

The operating costs summaries in Table 21.3 and Table 21.4 show projected average processing costs over the 12-year LOM, which are variable depending on the average grade of arsenic. Costs are separated out in two components. Firstly, the mineral processing sections of the Mill: comminution, flotation, flotation concentrate filtration and load-out; and secondly, the hydromet sections, which include POX, hot cure, CCD's, neutralization, cyanide leach - CIP, and gold recovery to doré.

Operating costs were determined from both first principles work-up and vendor quotes. They include all labour and supervision, consumables, reagents, and power. Vendor quoted delivered costs were obtained for the major consumables: power, flotation reagents, lime, limestone, cyanide, and oxygen. Maintenance costs were determined from similar projects and industry standards.

The mill operations labour complement was set per plant area, and as appropriate for a well automated, efficient plant. Labour schedules provided to BCM by AMC are based on actual schedules for similar remote operating mines.

Operating costs are broken down in two separate ways and presented below. Firstly, by labour, power, consumables and maintenance categories on a per tonne milled basis (Table 21.3). Secondly, by the fixed cost on both a daily (labour and maintenance) and per tonne (mineral processing consumables) basis and the POX feed rate driven costs, which is based on arsenopyrite concentrate mass pull as a function of arsenic grades, shown in Table 21.4.

Table 21.3 Summary of estimated mill operating cost

Category	Mineral processing ($/t)	Hydromet ($/t)	Total ($/t)
Labour	7.68	4.82	12.50
Power	4.48	0.82	5.30
Consumables and Reagents	7.28	12.69	19.97
Maintenance supplies	1.78	1.53	3.31
Total	**21.23**	**19.87**	**41.09**

Source: Blue Coast Metallurgy Ltd.

The POX driven processing operating costs have been developed from first principle algorithms, i.e. estimated based on primarily mill throughput, arsenic head grade and float mass pull of the arsenopyrite. The arsenopyrite concentrate production rate drives the tonnage throughput in the hydrometallurgical POX and CIP leach sections. Fixed costs are separated out for both the mineral processing and hydromet sections. For simplicity, a fixed cost per tonne is applied each year throughout the LOM.

Table 21.4 Estimated mill operating costs – fixed and POX driven

Category	Mineral processing ($/t)	Hydromet ($/t)	Total ($/t)
Fixed daily costs	9.47	6.35	15.82
Fixed costs per tonne	11.76	-	11.76
POX feed rate driven costs	-	13.51	13.51
Total	**21.23**	**19.87**	**41.09**

Source: Blue Coast Metallurgy Ltd.

21.1.4 Tailings storage

The operating cost estimate for the TSF was generated by Knight Piésold.

The operating costs summary in Table 21.5 show annual costs over the 12-year LOM.

Table 21.5 Tailings storage facility operating cost

Description	Unit	Value	Unit cost ($)	Total cost ($/year)
TSF Access Road Maintenance	km	2	21,600	43,200
Diversion ditch access road maintenance	km	2	10,800	21,600
General Maintenance	year	1	27,000	27,000
Manpower	year	1	108,000	108,000
Power – Reclaim pumping	Mwh	200	113	23,000
Power – Seepage pond pumping	Mwh	100	113	11,000
Environmental compliance	Year	1.00	54,000	54,000
Engineering support and reporting	Year	1.00	54,000	54,000
Total operating cost				**341,800**

Note: Totals may not add up exactly due to rounding.
Source: Knight Piésold Ltd.

21.1.5 General and administration (G&A)

G&A costs generally cover site administration and corporate costs. AMC benchmarked its estimate of $15/t against knowledge of the G&A cost ($13/t) for a similar operation in northern Canada. It is anticipated that, on a comparative basis, the Klaza estimate would be somewhat higher. Although it is relatively easy access to the site, the assumption that the workforce would be a FIFO with site camp operation and bussed to and from the mine.

21.2 Total operating cost estimate

The total operating cost estimate is summarized in Table 21.6. The mining cost is a combination of open pit and underground mining costs with an average of $55.14/t ore. The processing operating cost including tailings operating cost is $41.64/t.

Table 21.6 Total operating cost estimate

Description	LOM average cost ($/t)	Total LOM cost ($M)
Mining cost	55.14	412
Processing and tailings storage cost	41.64	311
G&A cost	15.00	112
Total operating cost	**111.78**	**834**

Note: Totals may not add up exactly due to rounding.
Source: AMC Mining Consultants (Canada) Ltd.

The total operating expenditure (opex) is summarized in Table 21.7.

Table 21.7 Total operating cost expenditure

Annual mean daily tonnes		1,704	1,664	1,885	1,884	1,879	1,884	1,884	1,884	1,840	1,848	1,849	1,250	698
LOM			1	2	3	4	5	6	7	8	9	10	11	12
Total mine production														
Mill feed	kt	7,464	607	688	688	686	688	688	688	672	675	675	456	255
Au	g/t	3.4	2.7	2.9	4.6	4.8	4.6	4.0	3.9	3.1	2.5	2.3	2.0	1.7
Ag	g/t	78.9	41.3	47.4	102.1	93.3	95.8	88.8	96.8	72.7	65.2	71.7	84.1	93.2
Pb	%	0.61	0.36	0.52	0.55	0.62	0.65	0.67	0.77	0.60	0.60	0.67	0.72	0.69
Zn	%	0.73	0.63	0.74	0.66	0.68	0.71	0.77	0.85	0.74	0.68	0.76	0.81	0.80
As	ppm	5,192	3,233	4,015	6,582	6,881	6,820	6,100	6,320	4,987	3,669	4,033	3,936	4,757
Total mining opex ($M)		412	32	23	37	40	40	40	40	39	39	39	27	15
Total process opex ($M)		311	25	29	29	29	29	29	29	28	28	28	19	11
Total G&A opex ($M)		112	9	10	10	10	10	10	10	10	10	10	7	4
Total opex ($M)		**834**	**67**	**61**	**76**	**79**	**79**	**79**	**79**	**77**	**78**	**78**	**53**	**30**

Source: AMC Mining Consultants (Canada) Ltd.

21.3 Capital cost estimate

The capital cost estimate is split into project capital over the first three years (one year pre-production and two years of prooduction) and sustaining capital (remainder of the mine life). Project capital includes the cost of the process plant, underground equipment and infrastructure, underground development and surface infrastructure.

21.3.1 Open pit

AMC has assumed that, due to the short life of the pits (three years), a contractor will be used to mine the open pits. Mark-ups on the operating costs have been assumed to cover the contractor's mining equipment and infrastructure capital costs and, therefore, no capital has been allowed for the open pits.

21.3.2 Underground mine

The underground capital cost is comprised of, primarily, underground development (lateral and vertical), underground mobile equipment and underground infrastructure. Capital costs for equipment are based on supplier quotes (2019). Equipment numbers were estimated to meet the production target of 688 ktpa. Infrastructure costs are based on estimated quantities and some supplier quotes. If no direct quotes were obtained, costs were derived from benchmark construction costs, and assumptions and quotes from recent projects undertaken by AMC.

21.3.2.1 Underground development

Cost for development is estimated at $4,500/m for lateral waste development and $8,500/m for vertical development (Alimak raise equiped with ladderways for secondary egress). The underground capital cost estimate for development is $114.9M and is summarized in Table 21.8.

Table 21.8 Underground development cost estimate

Capital development costs	Length (m)	Unit cost ($/m)	Project capital ($M)	Sustaining capital ($M)	Total cost ($M)
UG Lateral Development (waste)	22,589	4,500	32.5	69.1	101.7
UG Vertical Development	1,562	8,500	4.8	8.5	13.3
Total	**24,151**		**37.3**	**77.6**	**114.9**

Note: Totals may not add up exactly due to rounding.
Source: AMC Mining Consultants (Canada) Ltd.

21.3.2.2 Underground mobile equipment

The underground capital cost estimate for mobile equipment is $30.7M and is summarized in Table 21.9.

Table 21.9 Underground mobile equipment cost estimate

Description	Unit cost ($M)	Total cost ($M)
Longhole production drill (3)	0.9	2.7
2- boom development jumbo (4)	1.1	4.6
Scoops production (3)	1.1	3.3
Scoops development (3)	1.1	3.3
Haul truck (5)	1.2	6.2
Bolter (2)	0.9	1.8
Cable bolter (2)	0.9	1.8
Explosives loader (2)	0.5	1.8
Boom truck (2)	0.5	1.1
Lube / fuel truck (2)	0.4	0.8
Personnel carrier (4)	0.4	1.6
Scissor lift (4)	0.5	1.8
Utility vehicle (5)	0.1	0.3
Grader (1)	0.4	0.4
Total		30.7

Note: Totals may not add up exactly due to rounding.
Source: AMC Mining Consultants (Canada) Ltd.

21.3.2.3 Underground infrastructure

The underground infrastructure capital cost estimate is $20.6M and is summarized in Table 21.10. The costs are based upon supplier quotations, pricing in the public domain, and unit rates from previous experience. A portion of some costs (such as dewatering pumps in later years) is carried in the sustaining capital costs and is not included here. The underground infrastructure costs largely consist of electrical distribution, ventilation, and dewatering system costs.

Table 21.10 Underground infrastructure cost estimate

Description	Total cost ($M)
Electrical	
Underground electrical distribution and equipment	4.4
Communications system and hardware	1.0
Ventilation	
Primary exhaust fans	3.3
Portal heaters	1.8
Portal fans	0.6
Drop-board regulators	0.4
Refuge chambers	0.6
Stench system	0.1
Auxiliary fans - ramp development	0.3
Auxiliary fans - 75 hp	0.8
Dewatering	
Dewatering and reticulation	2.9
Other underground infrastructure	
UG shop / fuel / magazine / lighting	1.1
Underground air compressors	0.6
Portal infrastructure	2.8
Total	**20.6**

Source: AMC Mining Consultants (Canada) Ltd.

21.3.3 Process plant

The process plant capital cost estimate is $103M and is summarized in Table 21.11. This cost estimate was provided by BCM.

The capital costs were developed using vendor quotes, BCM internal files, other project benchmarks, industry factors, and literature.

These costs represent the direct capital costs, with delivery, construction and installation. Included are construction labour, civils, electrical, instrumentation, piping, pumps, and control systems. No EPCM, contingency, and other indirect costs are included.

Package quotes were received for the major equipment suppliers in crushing and grinding, flotation, autoclave, acacia leach, CIP, and carbon handling and refinery. Approximately 69% of the total equipment costs are based on external quotes and this provides a reasonable level of accuracy for the estimate. The balance of costs were taken from industry standards, and typical capital cost estimation factors.

The process plant building cost was estimated by AMC ($13.3M). This included the building shell, HVAC, electrical, and plumbing for the envelope only. The raw mineralized rock bin (structural steel and plate fabrication) was estimated as part of the process building cost.

Table 21.11 Mill area capital estimate

Description	Total cost ($M)
Crushing	3.5
Grinding	15.3
Flotation	11.8
Dewatering	4.0
Autoclave, CCD Neutralization	30.2
Acacia Leach	2.4
Leach and CIP	5.9
Carbon handling and refinery	3.1
Reagents	5.4
Ancillaries	8.3
Building	13.3
Total	**103.3**

Source: BCM.

21.3.4 Tailings storage facility (TSF)

As part of the updated PEA, AMC completed a trade-off study between the cost of building the TSF all on surface versus disposing tailings in the TSF for the first few years and then in-pit in the remaining years. The results showed minimal variance between value of the additional ore recovered from the crown pillar versus the additional capital cost of building the larger TSF on surface. However, other benefits including the ability to construct the portal at the bottom of the pit, reducing development capital cost as well as the increased recovery of the Mineral Resource, led to the decision to build the entire TSF on surface.

The capital cost estimated for the TSF is $17M and is summarized in Table 21.12. This cost estimate was provided by Knight Piésold.

Table 21.12 Tailings storage facility estimate

Year	Flotation stage	Flotation TSF cost ($M)	Residue stage	Residue TSF cost ($M)	Total TSF cost ($M)
0	1	4.4	1	2.9	7.2
1					
2	2	1.7			1.7
3					
4	3	1.7			1.7
5			2	2.4	2.4
6	4	1.7			1.7
7					
8	5	2.1			2.1
Total		**11.3**		**5.3**	**16.7**

Note: Totals may not add up exactly due to rounding.
Source: Knight Piésold Ltd.

21.3.5 Surface infrastructure

The capital cost estimate for the surface infrastructure is based upon supplier quotations, factored costs from previous projects, pricing in the public domain, factored published labour productivities, and experience regarding unit rates.

The capital cost estimate for surface infrastructure, including surface ancillary equipment, is $17.4M, and is summarized in Table 21.13. The major component of this cost estimate is site roads and refurbishment of access roads, mine office, camp, mine dry, and maintenance workshop. Water for the mine site and processing plant will be supplied from the Klaza river. The cost estimate allows for heated and insulated steel piping, pump and housing, modular filtration unit, and modular potable water treatment.

As part of the PEA, AMC completed a trade-off study between the cost of diesel generated and grid electrical power. Over the life of the mine the use of grid power was shown to be advantageous to the value of the project. The capital cost of the power line (provided by the utility) is included as a component of the operating costs for electrical power and is, therefore, not included here.

Table 21.13 Surface infrastructure cost estimate

Description	Total cost ($M)
Surface ancillary equipment	1.3
Access road upgrading	0.7
Site prep and excavations for foundations	0.3
Foundations	0.9
Camp / dry	4.5
Office	0.7
Mine rescue / first aid / safety / training	1.1
Shop / warehouse	2.7
Contractor shop / warehouse	1.0
Propane and fuel facilities	0.2
Magazines	0.1
First fills	0.5
Main distribution and power lines	2.2
Backup generations	1.0
Insulated & heat traced dewatering lines (from the mines to the mill)	0.2
Total	**17.4**

Source: AMC Mining Consultants (Canada) Ltd.

21.3.6 Closure costs

Closure costs are expected to be low and involve re-handle of top soil over waste dumps and seeding, monitoring of the TSF, removal of buildings, clearing of storage and laydown areas, plugging the portals and ventilation raises and reclamation costs. Closure of the Hydromet Residue and FTFs will involve a progressive capping of the facility with a waste rock and overburden blanket. Also, closure of the pits involves capping with a waste rock and overburden blanket. These costs are assumed to be covered by the sale of mobile equipment and processing and surface infrastructure at the end of the LOM.

21.3.7 Sustaining capital

Capital costs for ongoing underground development after the project period (first three years), are considered to be sustaining costs. These costs have been summarized above in Table 21.8.

Additional sustaining capital is based on 5% of total project capital expenditure to cover equipment rebuilds / replacement, and repairs to fixed equipment and infrastructure. The sustaining capital over the LOM is estimated to be $114M.

21.3.8 Indirect capital

Indirect capital (owners cost and EPCM) is assumed to be 5% of the project capital cost estimate. Indirect capital costs are estimated to be $10M.

21.3.9 Contingency

Contingency is applied to the project capital only (not sustaining capital) at 15% of the capital expenditure. The estimated contingency for the project is $32M.

21.3.10 Total capital cost estimate

The total capital cost is estimated to be $358M and is summarized in Table 21.14. The capital cost estimate is split into project and sustaining capital and detailed by year in Table 21.15.

Table 21.14 Total capital cost estimate

Description	Total cost ($M)
Underground lateral development	102
Underground vertical development	13
Floatation tailings storage & residue tailings storage	17
Underground mine infrastructure	21
Mobile equipment	32
Processing plant	103
Surface infrastructure	16
Capital indirects	10
Contingency	32
Additional 5% sustaining for equipment rebuilds	12
Total capital cost	**358**
Project capital	**244**
Sustaining capital	**114**

Note: Totals may not add up exactly due to rounding.
Source: AMC Mining Consultants (Canada) Ltd.

Table 21.15 Project and sustaining capital cost estimate

Capital cost estimate	Yr0	Yr1	Yr2	Yr3	Yr4	Yr5	Yr6	Yr7	Yr8	Yr9	Yr10	Yr11	Yr12	Total
Project capital ($M)	134	84	26	-	-	-	-	-	-	-	-	-	-	244
Sustaining capital ($M)	-	-	-	26	27	26	15	7	6	3	1	1	1	114
Total ($M)	**134**	**84**	**26**	**26**	**27**	**26**	**15**	**7**	**6**	**3**	**1**	**1**	**1**	**358**

Note: Totals may not add up exactly due to rounding.
Source: AMC Mining Consultants (Canada) Ltd.

22 Economic analysis

22.1 Assumptions

All currency is in Canadian dollars (C$) unless otherwise stated. Pricing in US dollars (US$) was converted to C$ using the exchange rate C$1:US$0.72. The cost estimate was prepared with a base date of Year 0 and does not include any escalation beyond this date. AMC have allowed a one-year construction period in Year 0, whereas mining would commence in Year 1. For Net Present Value (NPV) estimation, all costs and revenues are discounted at 5% from the base date. Metal prices were selected after discussion with Rockhaven and referencing current markets and forecasts in the public domain. A corporate tax rate of 27% is applied as the mining income will be earned in the Yukon. It is assumed that there are no royalties to be paid.

22.2 Economic analysis

AMC conducted a high level economic assessment of the conceptual operation of a combined Klaza open pit and underground mine. The combined open pit and underground mine is projected to generate approximately $529M pre-tax NPV and $378M post-tax NPV at 5% discount rate, pre-tax IRR of 45% and post-tax IRR of 37%. Project capital is estimated at $358M with a payback period of 4 years (discounted pre-tax cash flow from base date of Year 0). Key assumptions and results of the Klaza combined open pit and underground mine economics are provided in the Table 22.1 below. The LOM production schedule, average metal grades, recovered metal, and cash flow forecast is shown in Table 22.2.

The PEA is preliminary in nature. It includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves. There is no certainty that the PEA will be realized.

Table 22.1 Klaza combined open pit and underground mine - key economic assumptions and results

Klaza	Unit	Value
Total mineralized rock	kt	7,464
Total waste production	kt	10,130
Gold grade [1]	g/t	3.4
Silver grade [1]	g/t	79
Lead grade [1]	%	0.6
Zinc grade [1]	%	0.7
Gold recovery [1]	%	95
Silver recovery [1]	%	90
Lead recovery [1]	%	80
Zinc recovery [1]	%	80
Gold price	US$/oz	1,450
Silver price	US$/oz	17.00
Lead price	US$/lb	0.95
Zinc price	US$/lb	1.00
Exchange rate	US$1 : C$	0.72
Gold payable [2]	%	97
Silver payable [2]	%	81
Lead payable [2]	%	62
Zinc payable [2]	%	52
Payable gold metal	kg	23,373
Payable silver metal	kg	429,222
Payable lead metal	Tonnes	22,691
Payable zinc metal	Tonnes	22,706
Revenue split by commodity (gold)	%	77
Revenue split by commodity (silver)	%	16
Revenue split by commodity (lead)	%	3
Revenue split by commodity (zinc)	%	4
Total net revenue	C$M	1,975
Capital costs	C$M	358
Operating costs (total) [3]	C$M	834
Mine operating costs [4]	C$/t	55.1
Process and tails storage operating costs	C$/t	41.6
General and administrative costs	C$/t	15.0
Operating costs (total) [3]	C$/t	111.8
Operating cash cost (AuEq)	US$/oz AuEq	612.6
Total all in sustaining cost (AuEq)	US$/oz AuEq	875.3
Payback period [5]	Yrs	4
Cumulative net cash flow [6]	C$M	783
Pre-tax NPV [7]	C$M	529
Pre-tax IRR	%	45
Post-tax NPV [7]	C$M	378
Post-tax IRR	%	37

Notes:
[1] LOM average.
[2] Overall payable % includes treatment, transport, refining costs and selling costs.
[3] Includes mine operating costs, milling, and mine G&A.
[4] Includes open pit and underground operating costs.
[5] Values are pre-tax and discounted at 5%, from base date of Year 0.
[6] Pre-tax and undiscounted.
[7] At 5% discount rate.
Source: AMC Mining Consultants (Canada) Ltd.

Table 22.2 Klaza production and cash flow forecast

	Unit/yr	Total	0	1	2	3	4	5	6	7	8	9	10	11	12
Open pit - (Klaza and BRX)	kt	**1,181**	-	521	598	62	-	-	-	-	-	-	-	-	-
Underground - (Klaza and BRX)	kt	**6,283**	-	86	90	626	686	688	688	688	672	675	675	456	255
Total mined – Mineralized rock	kt	**7,464**	**-**	**607**	**688**	**688**	**686**	**688**	**688**	**688**	**672**	**675**	**675**	**456**	**255**
Open pit - waste (Klaza and BRX)	kt	**8,743**	-	5,501	3,242	-	-	-	-	-	-	-	-	-	-
Underground - waste (Klaza and BRX)	kt	**1,386**	-	248	212	170	240	245	143	75	28	26	-	-	-
Total mined - waste	kt	**10,130**	**-**	**5,749**	**3,454**	**170**	**240**	**245**	**143**	**75**	**28**	**26**	**-**	**-**	**-**
Total development - lateral	m	**45,949**	-	5,760	5,334	5,874	5,471	6,569	6,383	6,612	2,528	1,418	-	-	-
Total development - vertical	m	**1,562**	-	227	339	214	354	346	83	-	-	-	-	-	-
Stockpile (Klaza and BRX)	kt	**-**	-	-	-	-	-	-	-	-	-	-	-	-	-
Total mill feed	kt	**7,464**	**-**	**607**	**688**	**688**	**686**	**688**	**688**	**688**	**672**	**675**	**675**	**456**	**255**
Gold	g/t	**3.4**	-	2.7	2.9	4.6	4.8	4.6	4.0	3.9	3.1	2.5	2.3	2.0	1.7
Silver	g/t	**79**	-	41	47	102	93	96	89	97	73	65	72	84	93
Lead	%	**0.6%**	-	0.4%	0.5%	0.5%	0.6%	0.6%	0.7%	0.8%	0.6%	0.6%	0.7%	0.7%	0.7%
Zinc	%	**0.7%**	-	0.6%	0.7%	0.7%	0.7%	0.7%	0.8%	0.9%	0.7%	0.7%	0.8%	0.8%	0.8%
Overall gold recovery	%	**95%**	-	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%
Overall silver recovery	%	**90%**	-	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%
Overall lead recovery	%	**80%**	-	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%
Overall zinc recovery	%	**80%**	-	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%
Gold	kg	**23,373**	-	1,513	1,822	2,893	3,054	2,917	2,560	2,491	1,941	1,554	1,415	822	392
Silver	kg	**429,222**	-	18,297	23,798	51,179	46,638	48,028	44,493	48,530	35,620	32,086	35,266	27,986	17,302
Lead	t	**22,691**	-	1,083	1,777	1,861	2,102	2,207	2,291	2,634	1,992	2,006	2,227	1,638	874
Zinc	t	**22,706**	-	1,600	2,119	1,886	1,950	2,019	2,210	2,439	2,056	1,917	2,132	1,529	848
Overall gold payable	%	**97%**	-	97%	97%	97%	97%	97%	97%	97%	97%	97%	97%	97%	97%
Overall silver payable	%	**81%**	-	81%	81%	81%	81%	81%	81%	81%	81%	81%	81%	81%	81%
Overall lead payable	%	**62%**	-	62%	62%	62%	62%	62%	62%	62%	62%	62%	62%	62%	62%
Overall zinc payable	%	**52%**	-	52%	52%	52%	52%	52%	52%	52%	52%	52%	52%	52%	52%
Total net revenue	$M	**1,975**	**-**	**120**	**148**	**237**	**245**	**238**	**213**	**213**	**165**	**137**	**131**	**84**	**44**
Mining	$M	**412**	-	32	23	37	40	40	40	40	39	39	39	27	15
Processing and tailings storage	$M	**311**	-	25	29	29	29	29	29	29	28	28	28	19	11
G&A	$M	**112**	-	9	10	10	10	10	10	10	10	10	10	7	4
Total operating cost	$M	**834**	**-**	**67**	**61**	**76**	**79**	**79**	**79**	**79**	**77**	**78**	**78**	**53**	**30**

	Unit/yr	Total	0	1	2	3	4	5	6	7	8	9	10	11	12
Project capital	$M	**244**	134	84	26	-	-	-	-	-	-	-	-	-	-
Sustaining capital	$M	**114**	-	-	-	26	27	26	15	7	6	3	1	1	1
Total capital cost	$M	**358**	**134**	**84**	**26**	**26**	**27**	**26**	**15**	**7**	**6**	**3**	**1**	**1**	**1**
Undiscounted cash flows (pre-tax)	$M	**783**	(134)	(31)	60	136	140	133	119	127	82	56	53	30	13
Undiscounted cash flows (post-tax)	$M	**573**	(134)	(31)	57	103	105	99	91	97	64	44	41	24	11
Discounted cash flows (pre-tax)	$M	**529**	(127)	(28)	52	112	110	99	85	86	53	34	31	17	7
Discounted cash flows (post-tax)	$M	**378**	(127)	(28)	49	85	82	74	65	66	41	27	24	13	6

Note: Totals may not add up exactly due to rounding.
Source: AMC Mining Consultants (Canada) Ltd.

22.3 Sensitivity analysis

AMC has carried out a sensitivity analysis of the projection for combined open pit and underground mine economics. The sensitivity analysis examined the impact on pre-tax and post-tax NPV (at 5% discount rate) of a 20% positive or negative change in metal prices, operating costs, capital costs, corporate tax rate and exchange rate. The results of the pre-tax sensitivity analysis are summarized in Table 22.3 and Figure 22.1. The results of the post-tax sensitivity analysis are summarized in Table 22.4 and Figure 22.2.

The results show that the pre-tax NPV is robust and remains positive for the range of sensitivities evaluated. The post-tax NPV is more marginal, but also remains positive for the range of sensitivities evaluated.

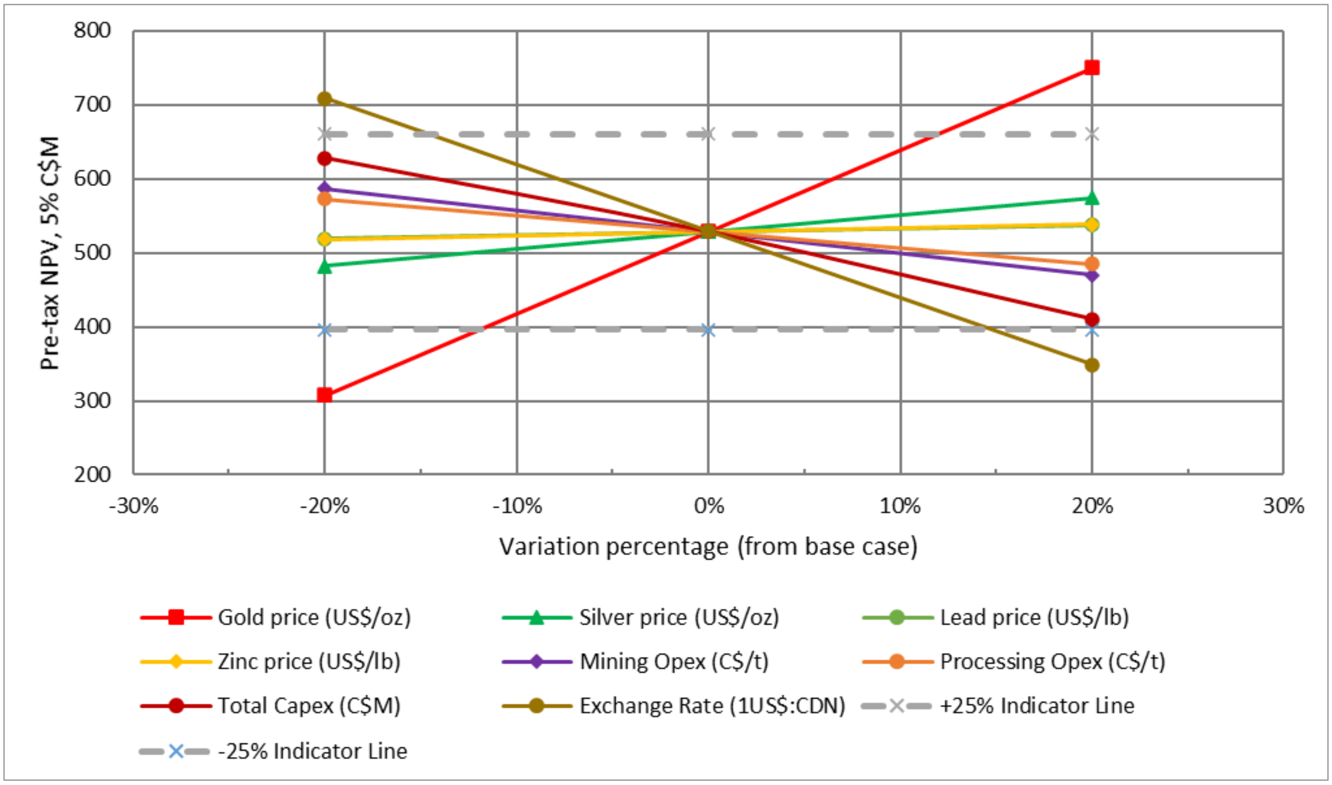
Pre-tax and post-tax NPV is most sensitive to changes in the gold price (as well as grade or recovery). It is also significantly sensitive to changes in total capital costs and exchange rate. The NPV is moderately sensitive to changes in mining operating costs, processing operating costs, silver price and corporate tax. Sensitivity to changes in the lead price and zinc price is minimal. Note in Figure 22.1 and Figure 22.2, lead price and zinc price follow the same line.

Table 22.3 Klaza economic sensitivity analysis (pre-tax)

Item	Value	Unit	Pre-tax NPV (C$M)	Pre-tax IRR %
Base case (NPV @ 5%)			**529**	**45**
Gold price - fall 20%	1,160	US$/oz	307	30
Gold price - increase 20%	1,740	US$/oz	750	59
Silver price - fall 20%	13.60	US$/oz	482	43
Silver price - increase 20%	20.40	US$/oz	575	48
Lead price - fall 20%	0.76	US$/lb	519	45
Lead price - increase 20%	1.14	US$/lb	538	46
Zinc price - fall 20%	0.80	US$/lb	519	45
Zinc price - increase 20%	1.20	US$/lb	538	46
Mining operating cost - decrease 20%	44.1	C$/t	587	49
Mining operating cost - increase 20%	66.2	C$/t	470	42
Processing operating cost - decrease 20%	33.3	C$/t	572	48
Processing operating cost - increase 20%	50.0	C$/t	485	42
Total Capex - decrease 20%	286	C$M	628	72
Total Capex - increase 20%	429	C$M	410	29
Exchange rate - decrease 20%	0.58	US$1:C$	709	67
Exchange rate - increase 20%	0.86	US$1:C$	348	30

Source: AMC Mining Consultants (Canada) Ltd.

Figure 22.1 Sensitivity analysis – pre-tax NPV at 5% discount rate



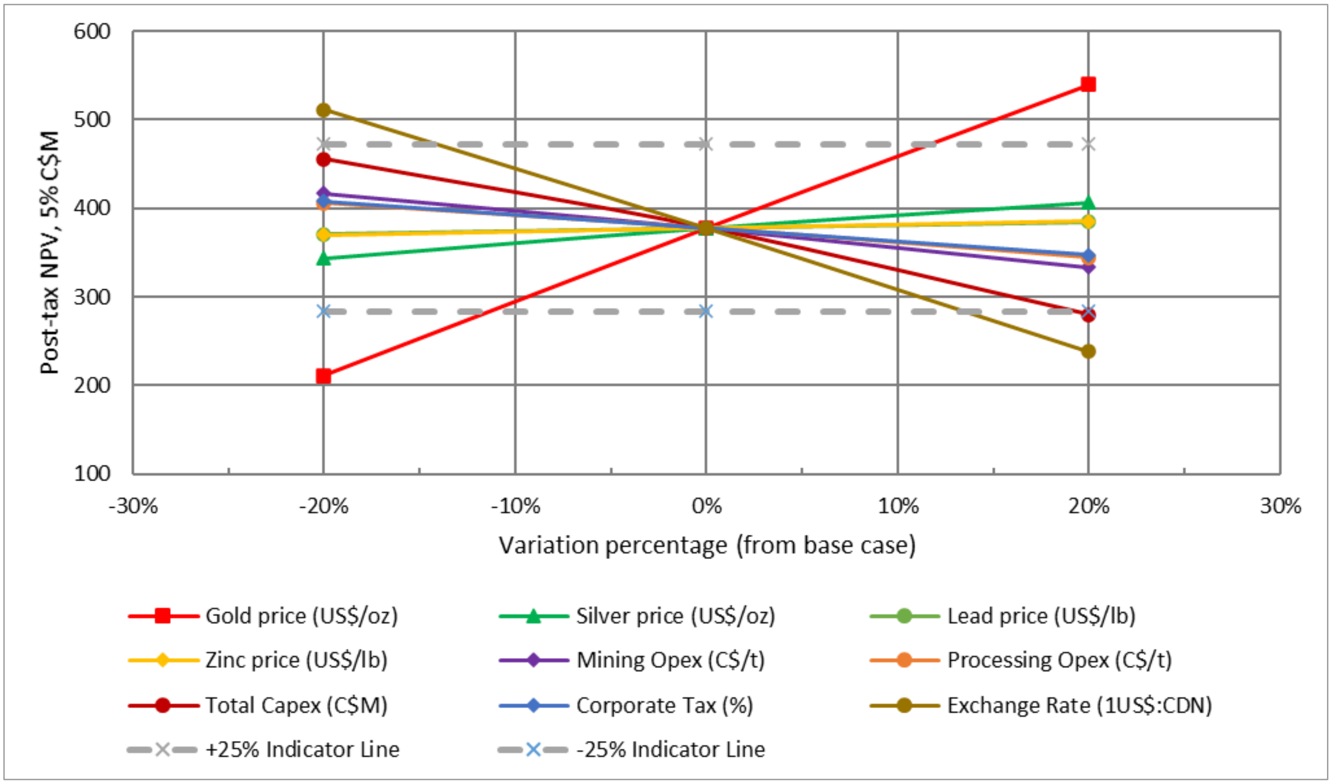
Source: AMC Mining Consultants (Canada) Ltd.

Table 22.4 Klaza economic sensitivity analysis (post-tax)

Item	Value	Unit	Post-tax NPV (C$M)	Post-tax IRR %
Base case (NPV @ 5%)			**378**	**37**
Gold price - fall 20%	1,160	US$/oz	211	24
Gold price - increase 20%	1,740	US$/oz	540	49
Silver price - fall 20%	13.60	US$/oz	343	35
Silver price - increase 20%	20.40	US$/oz	407	39
Lead price - fall 20%	0.76	US$/lb	371	37
Lead price - increase 20%	1.14	US$/lb	385	37
Zinc price - fall 20%	0.80	US$/lb	370	37
Zinc price - increase 20%	1.20	US$/lb	385	38
Mining operating cost - decrease 20%	44.1	C$/t	417	39
Mining operating cost - increase 20%	66.2	C$/t	334	34
Processing operating cost - decrease 20%	33.3	C$/t	406	39
Processing operating cost - increase 20%	50.0	C$/t	344	35
Total Capex - decrease 20%	286	C$M	456	60
Total Capex - increase 20%	429	C$M	280	23
Corporate tax rate - decrease 20%	21.6%	%	408	39
Corporate tax rate - increase 20%	32.4%	%	347	35
Exchange rate - decrease 20%	0.58	US$1:C$	511	55
Exchange rate - increase 20%	0.86	US$1:C$	239	23

Source: AMC Mining Consultants (Canada) Ltd.

Figure 22.2 Sensitivity analysis – post-tax NPV at 5% discount rate



Source: AMC Mining Consultants (Canada) Ltd.

23 Adjacent properties

There are two main types of adjacent properties near the Property – active placer gold operations and the Mount Nansen lode gold-silver deposit. The QP has not verified the information provided regarding adjacent properties. Mineralization on the adjacent properties is not considered indicative of the mineralization on the Property.

The most proximal property is an active placer gold operation located about 300 m along strike to the north-west of the BRX and Klaza zones and another operation lies immediately south-east of the Property (Van Loon and Bond 2014). Placer claims have been staked by other parties along all of the creeks draining and surrounding the known mineralized zones. None of the placer claims significantly overlap any of the Mineral Resources on the Property.

Production figures for placer gold operations based on royalty reporting are considered a minimum. Placer gold was discovered in 1899, but early production figures are not available. Total cumulative gold production from the Mount Nansen area, between 1980 to 2019, is reported to be 42,641 crude ounces (VanLoon, pers. comm. 2020). The available figures do not attribute ounces to specific operations, but rather state production by drainage.

Creeks and tributaries draining the north-west and northern portion of the ridge hosting the mineralized structural zones on the Property, including the Western BRX, have reported a total cumulative gold production of 3,022 crude ounces. To the south-east, the eastern zones drain into Summit creek, and portions of Nansen creek. These reported a total cumulative gold production of 1,781 crude ounces and 27,874 crude ounces respectively.

The Mount Nansen lode gold-silver Property is located approximately 9 km south-east of the deposit area. The Mount Nansen Property covers the former Mount Nansen mine site, including disused buildings, a tailings facility, and an open pit at the Brown-McDade Deposit and underground workings at the Huestis Deposit. The Mount Nansen Property is under care and maintenance.

The Mount Nansen Property hosts two gold-silver deposits and part of a third deposit (Deklerk and Burke 2008). Although mineralization found at the Mount Nansen Property and the adjoining property owned by 1011308 B.C. Ltd. is similar in tenor to mineralization found at the Property, the mineralogy and resources at those properties are not considered to be representative of mineralogy and resources on the Property.

Gold and silver mineralization occurs on the Mount Nansen Property in a series of anastomosing veins within north-westerly trending fault or shear zones. Mineralized structures consist of quartz-sulphide veins and associated clay-rich alteration zones.

Production from the Mount Nansen Property occurred over three periods: the first in 1967 and 1968; the second in 1975 and 1976; and, the last from 1996 to 1999. The latest operation continued intermittently until BYG Natural Resources Inc., the owner at the time, was placed into receivership. Published statistics state total production of 26,685 oz of gold and 214,897 oz of silver between 1967 and 1999. This total does not account for missing data from 1976.

On 6 May 2019 the Supreme Court of the Yukon issued an order approving the sale of the former Mount Nansen mine and Property to Alexco Environmental Group Inc. and JDS Energy & Mining Inc. Under the terms of the sale, the new property owners must assume care and maintenance, and reclaim the site within ten years.

There are a number of other gold-silver showings on properties owned by other parties within 5 km of the Property. Although encouraging drill and trench results have been returned from some of these showings, none of them has a Mineral Resource estimate.

24 Other relevant data and information

There is no other relevant data and information.

25 Interpretation and conclusions

This report discloses the results of an updated PEA that is based on the Mineral Resource estimate publicly reported in August 2018.

25.1 Summary

The results of this PEA suggest that the Project has good economic potential and warrants further study.

The PEA is preliminary in nature. It includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves. There is no certainty that the PEA will be realized.

25.2 Risk and opportunity management

Standard industry practices, equipment and processes were assumed for the PEA. The authors of the report are not aware of any unusual risks or uncertainties that could affect the reliability or confidence in the PEA results relative to the data and information available and the level of study.

Most mining projects are exposed to risks that may impact the economic outlook to varying degrees. External factors that are largely beyond the control of the project proponents can be difficult to anticipate and mitigate; although, in many instances, some reduction in risk may be achieved by regular reviews and interventions over the life of the project. Certain opportunities that can enhance project economics may also be identified during subsequent studies.

Table 25.1 and Table 25.2 summarize currently perceived project risks and opportunities for the Klaza Property, including potential impacts, and possible mitigations. A formal review of the likelihood and consequence ratings and pre and post-mitigation rankings was not conducted but is recommended for the next stage of project development.

Although the Property is remotely located in the Yukon, this does not pose significant logistical challenges that may affect the movement of people to and from the site, supplies inbound, and concentrates outbound. There is an existing road to the site which, except for the final 13 km, is maintained year-round by the Yukon Department of Highways and Public Works. Necessary materials and supplies can be brought to site as required throughout the year.

Extreme winter temperatures, possibly between -20° to -30°C, may impact personnel and equipment productivities during construction and operations.

General risks associated with open pit and underground mining related to aspects such as geotechnical conditions, equipment availability and productivity, and personnel productivity are anticipated to be similar to those experienced at other northern operations.

Table 25.1 Significant project risks

Area	Description	Aspect	Impact		Risk mitigation
			Description	**Risk**	
Geology and exploration	Geological model	Accuracy of geological model	Model does not represent the deposit location and grade distribution accurately.	Faults and mineralized zones might not be accurately identified; significant change in mine design required. Mineral Resources are overstated. Project economics reduced.	Infill drilling
Mineral processing and metallurgical testwork	Metallurgy	Gold recovery	Gold recovery process is not optimized for the mineralogy and variations within it.	Reduced recoveries reduce Project economics. Redesign of the gold recovery process plant might lead to additional capital expenditure.	Additional sampling and metallurgical testwork; further review of testwork done to date.
Open pit mine	Geotechnical considerations	Pit design and stability	Under estimation of waste in the mine design.	Under estimation of operating costs.	Ongoing geotechnical testwork and mapping to optimize pit wall design.
	Geotechnical considerations	Ground conditions	Under-estimation of the zone of weathering and geological structures.	Structure may impact on the pit design, mining assumptions, and economics of the pit.	Ongoing geotechnical testwork and drilling to better define zone of weathering.
Underground mine	Geotechnical considerations	Ground conditions	Under estimation of dilution and mining factors and inadequate crown pillar design.	Under-estimation of operating costs, over estimation of metal grades impact on economics.	Ongoing geotechnical monitoring and mapping; ongoing stope design and rock support review.
		Hydrology	Inadequate hydrological modelling of ground water inflow.	Under design of dewatering infrastructure and pit slopes.	Modelling of ground water inflow in the mineralized zones.
		Ground conditions	Reduced inter-level spacing.	Under estimation of ground conditions leading to reduced inter-level spacing, increased costs and lower project economics.	Ongoing geotechnical monitoring and mapping; ongoing review of open pit and underground scheduling.
	Cost estimation	Project economics	Over estimation of project economics.	Benchmark costs used may be too low for remote site, significant changes to mine design and economics.	Advance level of detail for cost estimation, first principles estimation.
Recovery methods	Crushing	Crushing throughput	Two-stage crushing inadequate.	Reduce throughput. Installation of third crusher and resulting increase in capital and operating costs.	Ongoing sampling and metallurgical testwork; further review of testwork done to date.
	Grinding	Mill throughput	Final mineralized rock size of 70 microns P_{80} not achieved or coarser grind size adequate.	Reduction in gold recovery.	Ongoing sampling and metallurgical testwork; further review of testwork done to date.
	Gold recovery	Gold recovery	Gold recovery lower than testwork indicates.	Reduced revenue and negative impact on Project economics.	Ongoing sampling and metallurgical testwork; further review of testwork done to date. Mineralized rock batching from specific deposits.

Area	Description	Aspect	Impact		Risk mitigation
			Description	Risk	
	Gold recovery	Hot cure	Validation of hot cure process stage.	Higher operating costs due to incorrect assumptions on hot cure process.	Additional batch and semi-continuous metallurgical testwork.
On-site infrastructure	Power supply	Inadequate supply locally available	Require alternate source of power.	Increased capital and operating cost.	Early engagement with local utility to determine future power expansion capacity.
	Human resources	Mine camp	240 person accommodation on site	May not be sufficient space during construction and during surges in production and / or maintenance shutdowns	Temporary accommodation of excess workers in Carmacks and provision for daily bus to site during peak HR loading
Off-site infrastructure	Access road	Failure to maintain	Poor road access due to improper maintenance.	Materials and equipment cannot be transported to site as planned or required. Results in loss of production and / or increase in cost of air freighting.	Confirmation of future intent from local rural municipality or territory.
Environmental studies, permitting, and social or community impact	Permitting	Regulatory process	Delay or failure to obtain necessary permits.	Delay in commencement of mining operations reduces Project economics and might impact Project financing.	Robust permitting process that addresses all Project requirements within the regulatory timelines.
Manpower	Manpower	Workforce complement and skills	Inability to attract or retain personnel with appropriate skills.	Increase in operating cost.	Explore the availability of workforce locally.
Financial	Financing	Securing reasonable capital	Failure to secure funding could slow or stop Project development.	Lower NPV.	Explore means to reduce capital.

Source: AMC Mining Consultants (Canada) Ltd.

Table 25.2 Significant project opportunities

Area	Description	Aspect	Impact		Opportunity steps
			Description	**Opportunity**	
Geology and exploration	Geological model	Accuracy of geological model	Model does not represent the deposit location and grade distribution accurately.	Mineral Resources are understated.	Infill drilling
Mineral processing and metallurgical testwork	Metallurgy	Gold recovery	Gold recovery process is not optimized for the mineralogy and variations within it.	Increased recoveries improve Project economics.	Additional sampling and metallurgical testwork; further review of testwork done to date.
Mineral Resource estimate	Mineral Resources	Resource expansion	Zones open at depth and laterally.	Inferred Mineral Resources may convert to Indicated Mineral Resources and improve confidence. Significant exploration potential within large land package with multiple greenfield targets.	Infill and exploration drilling.
Mining	Additional Resources	Extension	Additional resources to the east of the deposit to be further explored to include in the mine plan	Opportunity to increase the mineable inventory and extend mine life or capacity.	Increased value to project economics.
Process	On-site POX vs sale of concentrate	Capital and operating cost	Potential to sell arsenopyrite concentrate to a smelter.	Opportunity to reduce capital and operating costs.	Additional metallurgical testwork; trade-off study; market research.
Recovery method	Preconcentration POX versus bioleaching	Mill throughput, capital and operating cost	Potential to reduce mill throughput and increase crush size.	Visually, Klaza materials appear to be candidates for pre-concentration at a coarse crush size. POX tends to be more power-intensive than bioleaching.	Ongoing sampling and metallurgical testwork; further review of testwork done to date. Trade-off study
Financial	Capital Costs	Cost reductions	Reduce capital requirements through alternate recovery methods.	Refer to recovery method	Refer to recovery method
	Operating Costs	Cost reductions	Reduce operating costs through alternate recovery methods.	Refer to recovery method	Refer to recovery method

Source: AMC Mining Consultants (Canada) Ltd.

26 Recommendations

26.1 Geology and Mineral Resources

Work at the Klaza Property has defined significant, high-grade gold-silver-lead-zinc Mineral Resources. AMC recommends the following:

- Conduct further drilling of the BRX East and other Eastern Zones in order to sufficiently increase the drillhole density to enable completion of a Mineral Resource estimate.
- Infill drilling in the BRX and Klaza zones to upgrade Inferred Mineral Resources currently considered in the PEA to the Indicated category.
- Collect samples from previously unsampled drill core intervals within the Eastern Zones in order to complete the sample record.
- Conduct exploration diamond drilling beneath and along strike of prospective targets identified by trenching and diamond drilling.
- Update the Mineral Resource estimate on completion of the drill program and additional sampling within the Eastern Zones.
- Going forward, take duplicate samples only from mineralized material.

26.2 Hydrology

The following recommendations are made for further hydrogeological assessment at Klaza:

- Continue seasonal groundwater monitoring for the existing monitoring and observation wells.
- Survey all monitoring wells for their location and elevation of the top of the PVC casing with an accuracy of about ±1 cm or better.
- Collect additional ground temperature and hydrogeological data from the existing observation and monitoring wells, and drill additional wells as required. This data will allow updates to the preliminary conceptual hydrogeological model with an emphasis on permafrost-groundwater interaction.
- As mine planning progresses, install additional monitoring wells in the areas up and down gradient of proposed mine infrastructure.
- Integrate groundwater and surface water baseline data collection and interpret both datasets to assess groundwater-surface water interaction.

26.3 Geotechnical

Obtain a better understanding of the factors affecting open pit and stope stability and the proposed mining method from additional data collection, interpretation, and analysis, including the following:

- Develop a series of 3D models that includes lithology, alteration, and major structure.
- Using data from these models develop a 3D geotechnical model.
- Continue collecting geotechnical information during infill and exploration drilling. Preferably using oriented core whenever possible to increase confidence and understanding of structures.
- Implement a laboratory testing program on the various lithologies to assist in understanding rock properties. The following suite of rock property tests is recommended: UCS with Young's modulus (E) and Poisson's ratio (v), Confined compressive strength (triaxial), Indirect tensile strength (Brazilian test).
- As the mine is likely to be developed to depth > 300 m below ground level, in-situ stress testing will likely be needed. This could be carried out once mining has commenced.

26.4 Mining and infrastructure

AMC recommends the following work to be undertaken during the next phase of study:

- Re-evaluate open pit and underground mining opportunities for any updates to the Mineral Resource estimate.
- Reassess open pit-underground interface and specifics of crown pillar requirements.
- Further investigate underground stope sizing and confirm mining method.
- Further investigate the open pit mining method and bench height to evaluate means of reducing dilution.
- Develop development and production schedules.
- Project groundwater inflow to the proposed pits and underground mines from updated hydrogeological modelling.
- Should the hydrogeological modelling and study of the ground water regime indicate potential for large quantities of inflow into the mine, investigate a non-contact dewatering system. Water captured prior to entering the mining floor can reduce the cost of water treatment later.
- Undertake cost estimation and obtain contractor quotes for operating costs.
- Increase the level of detail for infrastructure engineering to better define capital costs.
- Undertake further work to support the assumptions that:
  - Sufficient local grid power is available.
  - The capacity and / or need for site emergency generation.
  - Investigate the potential to supply mine water from wells closer to site rather than from the river.

26.5 Processing and metallurgical testwork

BCM recommends the following for the Klaza project, ahead of preparation of a revised PEA study:

- Eastern BRX process development: For the flotation-leach process developed using whole ore material, demonstrate its application to EBRX pre-concentrate together with a projection of metallurgical recoveries.
- Pre-concentration: Conduct sufficient testing to establish metallurgical projections and process design for the use of pre-concentration on all zones except Western Klaza.
- Refractory gold concentrate development: Continue testing aimed at maximizing the gold grade / recovery relationship for arsenopyrite-hosted refractory gold.
- Conduct a marketing study on the refractory gold concentrate.

26.6 Tailings Storage Facility

- Complete rheology testing and geotechnical testing of the tailings streams.
- Complete geotechnical investigations to evaluate foundation conditions and construction material sources.

26.7 Environmental

Continue ongoing collection and evaluation of baseline data.

26.8 Proposed budget for recommendations

An approximate budget for the recommended work described above is presented in Table 26.1.

Table 26.1 Estimated cost to complete recommended work

Parameter	**Cost (C$000's)**
BRX and Klaza zones infill drilling (23,000 m @ $220/m)	5,060
Hydrological monitoring	50
Geotechnical testwork, modelling, and interpretation	50
Metallurgical studies	60
Marketing study	20
Updated Resource and next level of study	500
Environmental baseline studies to stage ready for EA	600
Contingency @ 15%	950
Total (excluding taxes)	**7,290**

Source: Archer, Cathro & Associates (1981) Limited.

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Section 23

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28 QP Certificates

CERTIFICATE OF AUTHOR

I, Adrienne A Ross, P.Geo., P.Geol., of Vancouver, British Columbia, do hereby certify that:

1 I am currently employed as a Principal Geologist and Geology Manager with AMC Mining Consultants (Canada) Ltd., with an office at Suite 202, 200 Granville Street, Vancouver British Columbia, V6C 1S4.

2 This certificate applies to the technical report titled "Technical Report and Preliminary Economic Assessment Update for the Klaza Property, Yukon, Canada" with an effective date of 10 July 2020 (the "Technical Report") prepared for Rockhaven Resources Limited ("the Issuer").

3 I am a graduate of the University of Alberta in Edmonton, Canada (Bachelors of Science (Hons) in Geology in 1991). I am a graduate of the University of Western Australia in Perth, Australia (Ph.D. in Geology). I am a registered member in good standing with Engineers and Geoscientists British Columbia (License #37418) and the Association of Professional Engineers and Geoscientists of Alberta (Reg. #52751). I have practiced my profession for a total of 26 years since my graduation and have relevant experience in precious and base metal deposits.

I have read National Instrument 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

4 I have visited the Property from 18 to 19 August 2015 for 2 days.

5 I am responsible for Sections 11 and 12, as well as part of Sections 1, 14, 25, 26, and 27 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 prior involvement with the Property that is the subject of the Technical Report. I co-authored the Technical Report entitled "NI 43-101 Technical Report describing Updated Diamond Drilling, Metallurgical Testing and Mineral Resources on the Klaza Property Yukon, Canada", dated 9 December 2015, "Technical Report and PEA for the Klaza Au-Ag Deposit, Yukon, Canada for Rockhaven Resources Ltd.", dated 26 February 2016, and "Technical Report Describing Updated Diamond Drilling, Metallurgical Testing and Mineral Resources on the Klaza Property, Yukon, Canada", dated 5 June 2018.

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: 10 July 2020
Signing Date: 15 July 2020

Original signed and sealed by

Adrienne Ross, P.Geo., P.Geol.

CERTIFICATE OF AUTHOR

I, Nicholas Ingvar Kirchner, FAusIMM, M.A.I.G., of Perth, Western Australia, do hereby certify that:

1 I am currently employed as Principal Geologist and Geology and Corporate Manager (Perth) with AMC Consultants Pty Ltd, with an office at Level 1, 1100 Hay Street, West Perth, Western Australia, 6005.

2 This certificate applies to the technical report titled "Technical Report and Preliminary Economic Assessment Update for the Klaza Property, Yukon, Canada" with an effective date of 10 July 2020 (the "Technical Report") prepared for Rockhaven Resources Limited ("the Issuer").

3 I am a graduate of Monash University in Melbourne, Victoria, Australia (Bachelor of Science (Hons) in Geology in 1988). I am a practicing geologist registered as a Fellow of the Australasian Institute of Mining and Metallurgy (Membership Number 108770) and as a Member of the Australian Institute of Geoscientists (Member ID 4727). I have practiced my profession for a total of 32 years and have relevant experience in precious and base metal deposits and mineral resource estimation.

I have read National Instrument 43-101 – Standards of Disclosure for Mineral Projects ("NI 43 101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

4 I have not visited the Property, subject of this Technical Report.

5 I am responsible for part of Sections 1 and 14 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 prior involvement with the Property that is the subject of the Technical Report. I co-authored the Technical Report entitled "Technical Report Describing Updated Diamond Drilling, Metallurgical Testing and Mineral Resources on the Klaza Property, Yukon, Canada", dated 5 June 2018.

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: 10 July 2020
Signing Date: 15 July 2020

Original signed and sealed by

N. Ingvar Kirchner, FAusIMM, M.A.I.G.

CERTIFICATE OF AUTHOR

I, Christopher J Martin, C.Eng., of Parksville, British Columbia, do hereby certify that:

1 I am currently employed as President and Principal Metallurgist with Blue Coast Metallurgy Ltd., with an office at Unit 2 - 1020 Herring Gull Way, Parksville British Columbia, V9P 1R2.

2 This certificate applies to the technical report titled "Technical Report and Preliminary Economic Assessment Update for the Klaza Property, Yukon, Canada" with an effective date of 10 July 2020 (the "Technical Report") prepared for Rockhaven Resources Limited ("the Issuer").

3 I am a graduate of Camborne School of Mines in Cornwall, UK (BSc(Hons).ACSM), in Mineral Processing Technology, 1984, and McGill University, Montreal, Canada (M.Eng in Metallurgical Engineering, 1988). I have been a Chartered Engineer and a member in good standing of the Institution of Materials, Minerals and Mining since 1990 (License #46116). I have practiced my profession for 34 years. I have experience in mineral processing operations management, and plant support and flowsheet development from roughly 400 projects located worldwide.

I have read National Instrument 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

4 I have not visited the Property, subject of this Technical Report.

5 I am responsible for Sections 13 and 19 and parts of Sections 1, 17, 25, 26, and 27 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 prior involvement with the Property that is the subject of the Technical Report. I co-authored Technical Reports entitled "Geology, Mineralization, Geochemical Surveys, Geophysical Surveys, Diamond and Percussion Drilling, Metallurgical Testing and Mineral Resources on the Klaza Property, Yukon, Canada" dated 11 March 2015 and amended 19 June 2015, "NI 43-101 Technical Report describing Updated Diamond Drilling, Metallurgical Testing and Mineral Resources on the Klaza Property Yukon, Canada", dated 9 December 2015, "Technical Report and PEA for the Klaza Au-Ag Deposit, Yukon, Canada for Rockhaven Resources Ltd.", dated 26 February 2016, and "Technical Report Describing Updated Diamond Drilling, Metallurgical Testing and Mineral Resources on the Klaza Property, Yukon, Canada", dated 5 June 2018.

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: 10 July 2020
Signing Date: 15 July 2020

Original signed and sealed by

____

Christopher Martin, C.Eng.

CERTIFICATE OF AUTHOR

I, Matthew R. Dumala, P.Eng., of Vancouver, British Columbia, do hereby certify that:

1 I am currently employed as a Senior Engineer and Partner with Archer, Cathro & Associates (1981) Limited, with an office at 1016-510 West Hastings Street, Vancouver British Columbia, V6B 1L8.

2 This certificate applies to the technical report titled "Technical Report and Preliminary Economic Assessment Update for the Klaza Property, Yukon, Canada" with an effective date of 10 July 2020 (the "Technical Report") prepared for Rockhaven Resources Limited ("the Issuer").

3 I am a graduate of the University of British Columbia in Vancouver, Canada (Bachelor of Science in Geological Engineering in 2002). I am a member in good standing of the Engineers and Geoscientists British Columbia (Reg. #32783). I have practiced my profession continuously since 2003 and have relevant experience in precious and base metals deposits.

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

I have visited the Klaza Property on an ongoing basis since 2013, most recently on 14 October 2017.

5 I am responsible for Sections 4 through 10 inclusively, 23, and 24, and parts of Sections 1, 3, 25, 26, and 27 of the Technical Report.

6 I am not independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101.

7 I have had prior involvement with the property that is the subject of the Technical Report. I have been directly involved with the property since 2013. I co-authored the Technical Report entitled "Technical Report Describing Updated Diamond Drilling, Metallurgical Testing and Mineral Resources on the Klaza Property, Yukon, Canada", dated 5 June 2018.

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: 10 July 2020
Signing Date: 15 July 2020

Original signed and sealed by

____

Matthew Dumala, P.Eng.

CERTIFICATE OF AUTHOR

I, Gary Methven, P.Eng., of Vancouver, British Columbia, do hereby certify that:

1 I am currently employed as a Principal Mining Engineer and Underground Manager with AMC Mining Consultants (Canada) Ltd. with an office located at Suite 202, 200 Granville Street, Vancouver, British Columbia, V6C 1S4.

2 This certificate applies to the technical report titled "Technical Report and Preliminary Economic Assessment Update for the Klaza Property, Yukon, Canada" with an effective date of 10 July 2020 (the "Technical Report") prepared for Rockhaven Resources Limited ("the Issuer").

3 I graduated from the University of Witwatersrand in Johannesburg, South Africa with a Bachelor of Science degree in Mining Engineering in 1993. I am a registered member in good standing with Engineers and Geoscientists British Columbia (License #180019), a member of Registered Professional Engineers of Queensland (License #06839), and a member of the Australian Institute of Mining and Metallurgy (CP). I have experience in narrow-vein gold deposits, flat and steeply dipping, bulk and selective mining methods for base metals, mine infrastructure, design and planning, mine production and financial evaluation, reserve estimation, technical reviews, feasibility and pre-feasibility studies, project and construction management, contracts management and cost estimation.

I have read National Instrument 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

4 I have visited the Property from 18 to 19 August 2015 for 2 days.

5 I am responsible for Sections 2, 15, 20, 22 and parts of Sections 1, 3, 16, 21, 25, 26, and 27 of the Technical Report.

6 I am independent of the Issuer and related companies as described in Section 1.5 of the NI 43-101.

7 I have had previous involvement with the property that is the subject of the Technical Report. I completed a Scoping Study on the Property. I also co-authored the Technical Report entitled "Technical Report and PEA for the Klaza Au-Ag Deposit, Yukon, Canada for Rockhaven Resources Ltd.", dated 26 February 2016, and "Technical Report Describing Updated Diamond Drilling, Metallurgical Testing and Mineral Resources on the Klaza Property, Yukon, Canada", dated 5 June 2018.

8 I have read NI 43-101 and the section of the Technical Report for which I am responsible has been prepared in compliance with NI 43-101.

9 As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the section of the Technical Report for which I am responsible contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: 10 July 2020
Signing Date: 15 July 2020

Original signed and sealed by

____

Gary Methven, P.Eng.

CERTIFICATE OF AUTHOR

I, Mo Molavi, P.Eng., of Vancouver, British Columbia, do hereby certify that:

1 I am currently employed as a Director, Principal Mining Engineer, and Mining Services Manager with AMC Mining Consultants (Canada) Ltd., with an office located at Suite 202, 200 Granville Street, Vancouver, British Columbia, V6C 1S4.

2 This certificate applies to the technical report titled "Technical Report and Preliminary Economic Assessment Update for the Klaza Property, Yukon, Canada" with an effective date of 10 July 2020 (the "Technical Report") prepared for Rockhaven Resources Limited ("the Issuer").

3 I graduated with a B Eng in Mining Engineering from the Laurentian University in Sudbury Ontario in 1979, and an M Eng in Mining Engineering specializing in Rock Mechanics and mining methods from the McGill University of Montreal in 1987. I am a registered member in good standing of the Association of Professional Engineers and Geoscientists of Saskatchewan and Engineers and Geoscientists British Columbia, and a Member of the Canadian Institute of Mining, Metallurgy and Petroleum.

I have worked as a Mining Engineer for a total of 40 years since my graduation from university and have relevant experience in project management, feasibility studies, and technical report preparations for mining projects.

I have read National Instrument 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

4 I have not visited the Property, subject of this Technical Report.

5 I am responsible for Section 18 and parts of Sections 1, 25, and 26 of the Technical Report.

6 I am independent of the Issuer and related companies as described in Section 1.5 of the NI 43-101.

7 I have had prior involvement with the Property that is the subject of the Technical Report. I co-authored the Technical Report entitled "Technical Report Describing Updated Diamond Drilling, Metallurgical Testing and Mineral Resources on the Klaza Property, Yukon, Canada", dated 5 June 2018.

8 I have read NI 43-101 and the section of the Technical Report for which I am responsible has been prepared in compliance with NI 43-101.

9 As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the section of the Technical Report for which I am responsible contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: 10 July 2020
Signing Date: 15 July 2020

Original signed and sealed by

Mo Molavi, P.Eng.

CERTIFICATE OF AUTHOR

I, David Warren, P.Eng., of Vancouver, British Columbia, do hereby certify that:

10 I am currently employed as a Principal Mining Engineer with AMC Mining Consultants (Canada) Ltd. with an office located at Suite 202, 200 Granville Street, Vancouver, British Columbia, V6C 1S4.

11 This certificate applies to the technical report titled "Technical Report and Preliminary Economic Assessment Update for the Klaza Property, Yukon, Canada" with an effective date of 10 July 2020 (the "Technical Report") prepared for Rockhaven Resources Limited ("the Issuer").

12 I am a graduate of Helsinki University of Technology in Helsinki, Finland (MA Science in Materials Science and Rock Engineer in 1978). I am a registered member in good standing with Engineers and Geoscientists British Columbia (License #15053) and a member of the Canadian Institute of Mining, Metallurgy, and Petroleum (CIM). I have practiced my profession continuously since 1978, and have been involved in open pit mine operations engineering, mine optimization, design and planning, due diligence and technical reviews, feasibility studies, operational reviews and improvement, project management, and mine financial analysis.

I have read National Instrument 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

13 I have not visited the Property, subject of this Technical Report.

14 I am responsible for parts of Sections 1, 16, 21, 25, 26, and 27 of the Technical Report.

15 I am independent of the Issuer and related companies as described in Section 1.5 of the NI 43-101.

16 I have not had prior involvement with the property that is the subject of the Technical Report.

17 I have read NI 43-101 and the section of the Technical Report for which I am responsible has been prepared in compliance with NI 43-101.

18 As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the section of the Technical Report for which I am responsible contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: 10 July 2020
Signing Date: 16 July 2020

Original signed and sealed by

____

David Warren, P.Eng.

CERTIFICATE OF AUTHOR

I, Bruno Borntraeger, P.Eng., of Vancouver, British Columbia, do hereby certify that:

1 I am currently employed as a Specialist Geotechnical Engineer 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 "Technical Report and Preliminary Economic Assessment Update for the Klaza Property, Yukon, Canada" with an effective date of 10 July 2020 (the "Technical Report") prepared for Rockhaven Resources Limited ("the Issuer").

3 I am a graduate of the University of British Columbia in Vancouver, Canada (Bachelor of Applied Science in Geological Engineering, 1990). I am a member in good standing of the Engineers and Geoscientists British Columbia (License #20926), and a member of the the Association of Professional Engineers Yukon. I have practiced my profession continuously for 30 years. I have been directly involved in geotechnical engineering, mine waste and water management, mine development with practical experience from feasibility studies, detailed engineering, construction, operations, and closure.

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 Property, subject of this Technical Report.

5 I am responsible for parts of Sections 1, 17, and 26 in 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 prior involvement with the Property that is the subject of the Technical Report. I co-authored the Technical Report entitled "Technical Report and PEA for the Klaza Au-Ag Deposit, Yukon, Canada for Rockhaven Resources Ltd.", dated 26 February 2016, and "Technical Report Describing Updated Diamond Drilling, Metallurgical Testing and Mineral Resources on the Klaza Property, Yukon, Canada", dated 5 June 2018.

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: 10 July 2020
Signing Date: 15 July 2020

Original signed and sealed by

____

Bruno Borntraeger, P.Eng.

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