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Report

Technical Report and PEA for the Klaza Au-Ag Deposit, Yukon, Canada for Rockhaven Resources Ltd.

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 a Preliminary Economic Assessment (PEA) and corresponding NI 43-101 Technical Report (Technical Report) for the Klaza property (Property) in Yukon, Canada.

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, which is located in the southern part of the more regionally extensive Dawson Range Gold Belt, in southwestern Yukon.

The Property comprises 1,317 mineral claims that are 100% owned by Rockhaven. A total of 213 claims are subject to a 1.5% Net Smelter Return royalty, but the other 1,104 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 25,400 hectares and is located approximately 50 km due west of the town of Carmacks in southwestern Yukon. Access is via the Mount Nansen Road, which extends from the Klondike Highway at the town of Carmacks to the former Mount Nansen Mine site and from there along 13 kilometres of unnamed placer access roads to the Property.

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 southeast corner of the Property and is thought to be the main heat source for hydrothermal cells that deposited mineralization along a series of northwesterly trending, structural conduits.

A swarm of northwesterly trending, Late Cretaceous feldspar porphyry dykes emanate from the stock in the southeastern part of the Property and cut the granodiorite in the main areas of interest. These porphyry dykes are up to 30 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 nine main zones, which individually range from 1 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 zones. 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, Central and Eastern 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 Eastern BRX zone comprises a series of sub-parallel veins. Mineralization in these veins differs from elsewhere along the BRX zone in that arsenopyrite is minimal, and chalcopyrite is more prevalent. Pyrite and chalcopyrite are the dominant sulphide minerals.

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.

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-molybdenum porphyry centre. Rockhaven purchased claims in the core of the Property in 2009, and since then has greatly expanded its claim holdings. Most recently, Rockhaven purchased claims through four separate agreements that tripled the size of its landholding.

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.

Exploration and drilling

Between 2010 and 2015, Rockhaven conducted systematic exploration that better defined northwesterly 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, 22,366 m of mechanized trenching and 70,100 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 2015.

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. Western BRX is open at depth. 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 northeast of the corresponding BRX subzones. They have been tested by both diamond drillholes and excavator trenches. Mineralization within these subzones, which is included in the Mineral Resource estimation, extends along a 1,200 m strike length and from surface to a maximum depth of 325 m down-dip.

Between 2010 and 2015, a total of 122 holes have been drilled at the BRX zone, while 127 holes have been drilled at the Western Klaza zones. During that period, additional holes have been drilled to test the Eastern Klaza and other mineralized zones on the Property, which are not part of the current Mineral Resources. The 2010 to 2015 programs on the Property were all managed by Archer, Cathro & Associates (1981) Limited (Archer Cathro) on behalf of Rockhaven.

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 conducted at Blue Coast Research in 2015 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. Carbonates are not abundant and there was no evidence of the presence of preg-robbing carbonaceous matter. Gold occurs both as discrete grains and solid solution in both arsenopyrite and pyrite. The distribution of gold project-wide is approximately 55% refractory (solid solution) gold contained in arsenopyrite, 8% within pyrite and 37% discrete gold.

Comminution test work 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 test work focused on 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 program to develop the flowsheet featured 65 batch flotation tests and two locked-cycle tests along with four bottle roll tests. 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 ES1 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**

The lead concentrate is relatively high grade and should be quite attractive to many smelters, however arsenic, antimony and mercury levels may incur penalties, although this should not affect marketability. High gold grades may also limit the number of smelters which can accept the lead concentrate. Zinc concentrates are relatively low-grade but saleable at 48% zinc.

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

Pressure oxidation (POX) was assumed as the pre-oxidation process of choice for the refractory gold bearing arsenopyrite concentrate. This assumption will need to be tested as the project advances and other processes such as bio-oxidation and Albion may warrant investigation. A single pressure oxidation test was run at AuTec Innovative Extractive Solutions Ltd., located in Vancouver, BC, on the arsenopyrite sulphide flotation concentrate from the flotation circuit. The extraction of gold by carbon-in-leach after pressure oxidation was 98%.

Although no hot cure test work was conducted on the POX residue, the hot cure process is very likely to be used in practice and has been included into the process flowsheet. Hot cure is expected to have a substantial effect on lime consumption, in part by enabling the use of cheaper limestone in place of lime. Hot cure is likely to be effective, but needs to be tested.

Intensive leach testing was conducted on the lead concentrate, consisting of shaker tests followed by a single bottle roll test. The testwork explored if the gold contained within the lead concentrate could be converted to doré on-site, and if associated reagent consumptions would be economic. The leach achieved 84% gold extraction within four hours. Reagent consumption was low indicating that the process would likely be economic. Therefore, intensive cyanidation has been built into the processing flowsheet, using the Acacia – Consep leach technology, as a packaged plant.

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 6%. 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 is provided in Table ES2.

Table ES2 Summary of metal recovery to doré and concentrates from testwork on the project-wide composite

	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%

Mineral Resource estimates

The Mineral Resource estimate was completed using 248 drillholes on the Property totalling 58,955 m and 23,890 assays. Seventy-two (72) mineralized domains were constructed by Archer Cathro to constrain the estimate. As much as possible, high-grade solids were built to capture only vein mineralization. The large number of mineralization domains reflects a strategy of subdividing the veins on either side of the porphyry unit and minimizing splays in a domain that can hinder the estimation process. The number of mineralization domains varied between the five zones.

AMC selected a compositing interval of 1 m, which is the most common sample length in the database. To allow for similar sample support, residual compositing intervals < 0.4 m in length were discarded.

Composited assay data for gold, silver, lead, zinc, copper, arsenic and iron were examined on probability plots for each of the 72 domains, and outliers examined. Capping was employed were required and varied by domain.

The estimations were carried out using Datamine software, with Ordinary Kriging (OK) employed as the interpolation method. A 3D block model with sub-celling was used. The search parameters chosen for the estimations were based on the drill spacing and variography. Data density allowed for only Inferred Resources to be classified.

Contained metals are reported below. Iron and copper were estimated for the purpose of the regression equations to derive density values. Examination of correlation coefficients demonstrates a strong relationship between measured density and a sum of the base metal grades. The mineralized portions of the block model were assigned a density based on the combined estimated grades of lead, zinc, copper and iron and the regression equations for the BRX and Klaza zones.

Arsenic was estimated for metallurgical purposes.

The pit-constrained Mineral Resources are reported within a base of overburden surface and a conceptual pit shell based on a US$1,300/ounce gold price. The cut-off applied for reporting the open pit Mineral Resources is 1.3 g/t gold equivalency. 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 2.75 g/t gold equivalency. 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.

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 estimate.

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 ES3 below, and expanded in Table ES4.

Table ES3 Summary of Inferred Mineral Resources as of 9 December 2015

	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)
Pit-Constrained	2,366	5.12	95	0.93	1.18	6.71	389	7,190	48,258	61,475	510
Underground	7,054	4.27	87	0.69	0.88	5.65	969	19,772	107,159	136,416	1,282
Total	**9,421**	**4.48**	**89**	**0.75**	**0.95**	**5.92**	**1,358**	**26,962**	**155,417**	**197,891**	**1,793**

CIM definition standards were used for the Mineral Resource.
Using drilling results to 30 September 2015.
Near surface Mineral Resources are constrained by an optimized pit shell at a gold price of US$1300 oz.
Cut-off grades applied to the pit-constrained and underground Resources are 1.3 g/t Au EQ and 2.75 g/t Au EQ respectively.
Gold equivalent values were calculated using the following formula: Au EQ=Au+Ag/85+Pb/3.74+Zn/5.04 and assuming: US$1300 oz Au, US$20 oz Ag, US$0.90 lb Pb and US$0.90 lb Zn with recoveries for each metal of Au: 96%, Ag: 91%, Pb: 85% and Zn: 85%.
Numbers may not add due to rounding.
All metal prices are quoted in US$ at an exchange rate of $0.80 US to $1.00 Canadian.
Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

Table ES4 Inferred Mineral Resources as of 9 December 2015 by area

Zone	PC/UG	Tonnes (kt)	Au (g/t)	Ag (g/t)	Pb (%)	Zn (%)	AuEQ (g/t)	Au (koz)	Ag (koz)	Pb (klb)	Zn (klb)	AuEQ (koz)
Western BRX	Pit-Constrained	554	8.21	110	1.03	1.03	9.99	146	1,960	12,608	12,557	178
	Underground	814	7.87	147	1.49	1.68	10.34	206	3,853	26,764	30,194	271
	Total	**1,368**	**8.01**	**132**	**1.31**	**1.42**	**10.20**	**352**	**5,813**	**39,372**	**42,750**	**448**
Central BRX	Pit-Constrained	283	3.67	192	1.34	1.39	6.57	33	1,752	8,361	8,693	60
	Underground	1,027	2.65	152	1.26	1.39	5.05	87	5,019	28,561	31,506	167
	Total	**1,311**	**2.87**	**161**	**1.28**	**1.39**	**5.38**	**121**	**6,771**	**36,922**	**40,198**	**227**
Eastern BRX	Pit-Constrained	193	4.42	90	0.21	0.42	5.62	27	559	908	1,798	35
	Underground	2,213	4.07	50	0.21	0.29	4.77	289	3,568	10,296	14,230	340
	Total	**2,406**	**4.10**	**53**	**0.21**	**0.30**	**4.84**	**317**	**4,127**	**11,203**	**16,028**	**374**
Western Klaza	Pit-Constrained	81	6.86	288	1.01	0.98	10.72	18	752	1,803	1,748	28
	Underground	461	5.41	182	0.58	0.87	7.88	80	2,703	5,879	8,820	117
	Total	**542**	**5.62**	**198**	**0.64**	**0.88**	**8.30**	**98**	**3,455**	**7,682**	**10,567**	**145**
Central Klaza	Pit-Constrained	1,255	4.07	54	0.89	1.33	5.20	164	2,168	24,578	36,680	210
	Underground	2,539	3.74	57	0.64	0.92	4.76	305	4,628	35,661	51,668	389
	Total	**3,794**	**3.85**	**56**	**0.72**	**1.06**	**4.91**	**470**	**6,796**	**60,239**	**88,347**	**599**

CIM definition standards were used for the Mineral Resource.
Using drilling results to 30 September 2015.
Near surface Mineral Resources are constrained by an optimized pit shell at a gold price of US$1300 oz.
Cut-off grades applied to the pit-constrained and underground Resources are 1.3 g/t Au EQ and 2.75 g/t Au EQ respectively.
Gold equivalent values were calculated using the following formula: Au EQ=Au+Ag/85+Pb/3.74+Zn/5.04 and assuming: US$1300 oz Au, US$20 oz Ag, US$0.90 lb Pb and US$0.90 lb Zn with recoveries for each metal of Au: 96%, Ag: 91%, Pb: 85% and Zn: 85%.
Numbers may not add due to rounding.
All metal prices are quoted in US$ at an exchange rate of $0.80 US to $1.00 Canadian.
Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

Mineral Reserve estimates

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

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 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 zone is accessed by an independent decline system and for mining purposes is termed Eastern Klaza. The Eastern BRX zone is only considered as upside potential that warrants further study.

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. The open pit is planned to be mined using conventional drill-and-blast and utilizing truck and excavator mining equipment by 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 production rate of 550 ktpa. This production rate is well supported by the detailed production scheduling.

The Klaza and BRX veins are approximately parallel and 800 m apart. Separate declines were designed for Klaza, Eastern Klaza and BRX. Access to the Klaza zone underground mine is via three 5 m by 5 m declines and crosscuts on each level. Levels are spaced at a vertical distance of 30 m floor to floor. Development (4 m by 4 m) was designed to follow the vein along strike from a central access crosscut. The Western and Central Klaza decline commences from the portal on surface and splits into two declines to access the Western and Central zones. The Eastern Klaza stopes are accessed by an independent decline from surface. Main access to the BRX zone has a similar design, with two declines and crosscuts as necessary. 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 30 m, at a 65° dip, stope lengths of 18 m for unsupported walls and 44 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.

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 total of approximately 290 m^3/s is planned for ventilation of both the Klaza and BRX underground mines.

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 six inch 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 four inch pipeline for mine service water and a six inch 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). RAR’s 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 enable placement of process tailings in the depleted pits.

The proposed combined life of mine production schedule for the open pit and underground is summarized in Table ES5.

Table ES5 Combined OP and UG life of mine production schedule

Production	**YR0**	**YR1**	**YR2**	**YR3**	**YR4**	**YR5**	**YR6**	**YR7**
Waste (kt)	4,433	4,398	4,128	4,048	1,176	165	155	45
Mineralization (kt)	144	337	596	664	552	391	551	550
AuEQ (g/t)	3.3	4.8	4.1	4.7	6.2	4.5	5.1	5.1
UG Production	**YR8**	**YR9**	**YR10**	**YR11**	**YR12**	**YR13**	**YR14**	**Total**
Waste (kt)	8	104	23	1				18,686
Mineralization (kt)	549	510	524	470	310	203	93	6,444
AuEQ (g/t)	4.6	3.7	3.4	3.5	3.3	2.9	2.9	4.4

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 85% capacity (470 ktpa) during YR1 and 100% capacity in YR2. The process plant has been designed for a maximum throughput of 1,650 tpd. The proposed processing schedule is summarized in Table ES6.

Table ES6 Processing schedule

Total Production	**YR0**	**YR1**	**YR2**	**YR3**	**YR4**	**YR5**	**YR6**	**YR7**
Mill feed (kt)		470	550	550	550	550	550	550
Aueq (g/t)		4.3	4.3	5.2	6.2	3.7	5.2	5.1
Au (g/t)		3.7	3.6	4.3	4.8	2.9	4.0	3.7
Ag (g/t)		49	49	72	100	57	95	102
Pb (%)		0.6	0.8	0.7	1.0	0.6	0.6	0.7
Zn (%)		1.1	1.0	1.0	1.0	0.7	0.7	0.8
As (%)		0.5	0.5	0.6	0.6	0.4	0.5	0.5
Total Production	**YR8**	**YR9**	**YR10**	**YR11**	**YR12**	**YR13**	**YR14**	**Total**
Mill feed (kt)	550	523	524	470	310	203	93	6,444
Aueq (g/t)	4.7	3.8	3.4	3.5	3.3	2.9	2.9	4.4
Au (g/t)	3.4	2.5	2.2	2.3	2.6	2.2	2.5	3.3
Ag (g/t)	95	95	82	86	44	50	24	77
Pb (%)	0.6	0.7	0.8	0.9	0.6	0.3	0.2	0.7
Zn (%)	0.8	0.8	0.9	0.8	0.5	0.6	0.7	0.8
As (%)	0.6	0.4	0.4	0.4	0.3	0.3	0.1	0.5

Recovery methods

The processing facility will feature standard crushing, SAG and ball mill grinding followed by sequential lead, zinc and arsenopyrite flotation circuits. The arsenopyrite concentrate will then be treated by pressure oxidation and cyanide leaching to recover gold.

Crushing will take place using a jaw crusher ahead of stockpiling with 24 hour live capacity (1,650 t). The crushed material will feed into a SAG mill followed by a ball mill operated in closed circuit with hydrocyclones, to grind the mineralized rock to 80% passing 70 microns. The cyclone overflow would then report to the flotation circuit.

The flotation circuit begins with lead roughers, with the lead rougher concentrate stream then subjected to regrinding in a ball mill down to a size of 80% passing 28 microns. The regrind product is then cleaned in three stages to produce a final lead concentrate. The lead concentrate is then thickened and fed to an Acacia intensive leach reactor to recover gold, ahead of filtration and washing, and sale to a lead smelter.

Lead circuit tailings are conditioned ahead of zinc rougher flotation, with zinc rougher concentrate subjected to regrinding to 80% passing 32 microns. The zinc regrind product is then cleaned in two stages to produce a final zinc concentrate, which is thickened and filtered ahead of sale to a zinc smelter.

Zinc circuit tailings are conditioned ahead of arsenopyrite rougher flotation with the arsenopyrite rougher concentrate thickened ahead of hydrometallurgical processing.

Thickened arsenopyrite concentrate continuously feeds storage tanks providing five days of surge capacity ahead of the pressure oxidation (POX) circuit, to allow for autoclave downtime while the mill-flotation plant continues to operate. The POX autoclave circuit is followed by hot cure, neutralization, gold leaching and CIP recovery circuits. The POX autoclave and downstream processes have been sized to operate at an effective 85% availability.

The hot slurry from the autoclave flash vessel is treated through a hot cure stage to cure the basic iron sulphates (BIS), and break down the BIS into ferric sulphate and iron hydroxide into solution.

After hot curing, two-stage counter-current decantation (CCD) is used to separate acid liquor from the solid residue. Overflow solution from the first CCD is directed to a neutralization process using cheaper limestone to reduce cost. This neutralized liquor is directed to a lined Hydromet Residue Storage pond. Overflow from this pond will flow into the main flotation tailings storage facility (TSF).

The second CCD underflow slurry is neutralized with lime to pH 10 and transferred to a series of cyanide leach tanks, followed by a set of carbon-in-pulp (CIP) pump cell tanks. The total leach and CIP residence time is 36 hours. The CIP circuit is operated counter-current with gold-loaded carbon removed batch-wise from each pump-cell. The CIP tails are pumped to cyanide destruction, which uses the INCO Air/SO_2 process before pumping to the Hydromet Residue Storage pond. Sodium metabisulphite (SMBS) is used to replace SO_2, to reduce transportation and handling hazards.

Loaded carbon from the CIP plant circuit is acid washed, neutralized and then stripped of gold in two elution columns. Eluate is circulated through electrowinning cells where gold is deposited on the cathodes, removed and the gold and silver precipitate (sludge) filtered and placed in a calcining oven with a retort system to remove any mercury before smelting in an induction furnace. The furnace is tapped to produce gold and silver doré bars which are cleaned and weighed prior to storage in the gold room safe, ready for dispatch.

Carbon is periodically regenerated in an electric kiln at 650°C to remove contaminants and reactivate the carbon, which is returned to the CIP circuit.

Thickened lead concentrate is leached to recover gold and silver using an Acacia intensive cyanide leach process. The process includes concentrate storage, leaching, and slimes removal with electrowinning of gold and silver. High gold recoveries from the lead concentrate are expected, at or above 85%. Electrowon gold and silver (sludge) washed from the cathodes is bagged and transported to the mill gold room for drying and smelting. The leached lead concentrate residue is pressure filtered and washed, with filtrate solutions sent to cyanide detoxification. The clean, filtered lead concentrate cake (7% moisture) is sampled, weighed and bagged for dispatch to a lead smelter.

Tailings 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:

- Hydromet residue tailings
- Flotation tailings

The hydromet tailings will be managed in a separate facility sized to contain the production schedule volume. The flotation tailings will be managed using a surface facility with a containment structure until approximately Year 6, when the open pits will be available to receive tailings. The tailings facility designs minimize the project footprint, prevent surface effluent discharge during operations, and allow for simple and effective water management at closure.

A preliminary site water balance has indicated a net water input requirement to provide sufficient process water. It is proposed 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 Flotation Tailings Facility and recycled to the mill for use during processing.

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 placer 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 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 center, meeting rooms, lunch rooms, and training rooms. The mine dry will also be an assembly of construction industry grade, portable trailers. Space for 200 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. The workforce will be encouraged to live in Carmacks and a daily bus service will be provided to drive the workforce 73 km to the mine and back. The total mining labour and supervision requirements were estimated to be 180 employees; this excludes contractor labour for the open pit. A further 75 employees are required for processing with a total workforce of 255 employees. There is no allowance for a mine camp.

Mine water will be supplied via a four inch 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 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 hp submersible dirty water pumps at 60 m levels. Each sump will have two pumps to provide continual redundancy. A six inch 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 mobile electric compressors. These will be situated on active levels and at the maintenance bay. The compressors will be relocated to active mining levels as needed.

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.

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.

Environmental Studies and Permitting

A preliminary environmental assessment 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 environmental assessment, 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.

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 located either in the Yukon or North-West Territory. Actual costs for an operating mine near Klaza 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 – 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.

G&A costs generally cover site administration and corporate costs. AMC benchmarked its estimate of $12/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 ES7.

Table ES7 Total operating cost

Description	Cost (C$/t)	Cost (C$M)
Mining cost	59.7	384
Processing cost	43.4	279
General and Administration cost	12.0	77
Total operating cost	**115.1**	**741**

Totals may not add up exactly due to rounding.

The capital cost estimate is split into project capital (first four 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 (five 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. 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 550 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 ES8.

Table ES8 Total capital cost

Description	Total cost (C$M)
Underground lateral development	125
Underground vertical development	11
Floatation tailings storage & residue tailings storage	10
Underground mine infrastructure	17
Mobile equipment	32
Ancillary equipment	1
Processing plant	91
Surface infrastructure	14
Capital indirects	11
Contingency	34
Additional 5% sustaining for equipment rebuilds	13
Total capital cost	**358**
Project capital	**262**
Sustaining capital	**96**

Economic Analysis

All currency is in CAD dollars (C$) unless otherwise stated. Foreign exchange rates were applied as required. Pricing submitted in US dollars (US$) were converted to C$ dollars using the exchange rate 1C$: 0.75US$. 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) 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 30% 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$150M pre-tax NPV and C$86M post-tax NPV at a 5% discount rate, pre-tax IRR of 20% and post-tax IRR of 14%. Project capital is estimated at C$262M with a payback period of 7 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 ES9.

Table ES9 Klaza combined open pit and underground mine economics

Klaza	Unit	Value
Total Mineralized rock	kt	6,444
Total Waste Production	kt	18,686
Gold Grade	g/t	3.3
Silver Grade	g/t	77
Lead Grade	%	0.7%
Zinc Grade	%	0.8%
Gold price	US$/oz	$1,200
Silver price	US$/oz	$16.00
Lead price	US$/lb	$0.80
Zinc price	US$/lb	$0.85
Payable Gold metal	kg	19,606
Payable Silver metal	kg	353,457
Payable Lead metal	t	23,233
Payable Zinc metal	t	23,792
Revenue by Commodity (Gold)	%	74%
Revenue by Commodity (Silver)	%	18%
Revenue by Commodity (Lead)	%	4%
Revenue by Commodity (Zinc)	%	4%
Total Net Revenue	C$M	1,365
Capital Costs	C$M	357
Operating Costs (Total)	C$M	741
Mine Operating Costs	C$/t mineralized rock	59.7
Process and Tails Storage Operating Costs	C$/t mineralized rock	43.4
Operating Costs (Total)	C$/t mineralized rock	115
Operating Cash Cost (AuEQ)	US$/oz AuEQ	651.5
Total All In Sustaining Cost (AuEQ)	US$/oz AuEQ	965.9
Payback Period	Yrs	7
Cumulative Cash flows (pre-tax)	C$M	266
Pre-tax NPV (1)	C$M	150
Pre-tax IRR	%	20%
Post-tax NPV (1)	C$M	86
Post-tax IRR	%	14%

1 - Discount rate of 5%

Interpretations and Conclusions

The results of this 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 Mineral Resource for the Klaza Property is entirely classified as an Inferred Mineral Resource.

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.

Recommendations

Geology

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

- Conduct an infill drilling programme over the summer (2016) with the primary aim of converting Inferred Mineral Resources to Indicated Mineral Resources.
- Update the Mineral Resource estimate on completion of the drill programme.
- Undertake a Pre-Feasibility Study (PFS) based on the new Mineral Resource estimate.
- Conduct additional drilling to expand the current Mineral Resource to depth and along strike.
- Conduct additional drilling to assess whether areas immediately east along strike of the current Mineral Resources (Eastern BRX) are well enough mineralized to warrant further geological and engineering work.
- A Certified Reference Material reflecting the average grade of the Mineral Resource estimate also be included in the QA/QC program.
- Going forward, duplicate samples are to be taken only from mineralized material.

Hydrology

AMC concurs with the Tetra Tech EBA recommendations for further hydrogeological assessment at Klaza:

- Seasonal groundwater monitoring should be continued for the existing monitoring and observation wells. A future Type A Water Licence application will require a minimum of two consecutive years of baseline data.
- All monitoring wells should be surveyed for their location and elevation of the top of the PVC casing in the summer of 2016 with an accuracy of about ±1 cm or better.
- Collecting additional ground temperature and hydrogeological data from the existing observation and monitoring wells is recommended as well as additional wells if required. The data should be used to update the preliminary conceptual hydrogeological model with an emphasis on permafrost-groundwater interaction.
- As the mine planning progresses, additional monitoring wells should be installed in the areas up and down gradient of proposed mine infrastructure, including any proposed infrastructure that may affect groundwater quality and/or quantity.
- The groundwater and surface water baseline data collection should be integrated and both data sets be interpreted with respect to groundwater-surface water interaction.

Geotechnical

A better understanding of the factors affecting open pit and stope stability and the proposed mining method could be gained 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.
- Hydrogeological characterization of the site as per recommendations above.
- 2D-3D modelling with updated parameters to assess stope, open pit, and crown pillar stability (between open pit and underground).

- A geotechnical diamond drilling program with oriented core should be carried out to assist in increasing the geotechnical understanding of the test area.
- A laboratory testing program should be performed on the various lithologies to assist in understanding the 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).
- A Ground Control Management Plan (GCMP) should be developed, with ongoing geotechnical data collection and interpretation, and updated annually.
- As the mine is likely to be developed to depth >300 m below ground level, in-situ stress testing will likely be needed.

Mining and infrastructure

AMC recommends the following work to be undertaken during the PFS:

- Re-evaluate open pit and underground mining opportunities for the updated Mineral Resource estimate based on an Indicated Mineral Resource.
- Re-evaluate planned production rate based on a new Mineral Resource estimate.
- Undertake geotechnical drilling within the proposed crown pillars to better define pillar sizes, particularly for the proposed Klaza pit where tailings would be stored.
- Undertake hydrological modelling of the proposed pits and underground mines to establish expected ground water inflow.
- Undertake first principals 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:
  - Mine workforce would be based in Carmacks and bussed to and from site on a daily basis.
  - Sufficient local grid power is available.

Processing and metallurgical testwork

BCM recommends the following for the Klaza project:

- Eastern BRX process development: A process needs to be developed to produce separate copper, lead and zinc concentrates, mainly to allow for payment of the precious metals that would be associated with each of these products. Selective flotation of arsenopyrite should also be explored as, although the arsenic grades are low in Eastern BRX, the arsenopyrite could still contain a significant amount of refractory gold.
- Pre-concentration development: Visually, Klaza materials appear to be candidates for pre-concentration at a coarse crush size. Pre-concentration should be explored further through testwork.
- Demonstration of workability of flowsheet on pre-concentrate: If pre-concentration is proven, then the effect of pre-concentration on downstream metallurgy needs to be established.
- Grindability testing: Grindability data will be needed on the individual zones to ensure the mill as designed can handle the different materials.
- Upgrade testing on lead and zinc concentrates: Presently the lead and zinc concentrates leave opportunity for improvement by (a) lowering the arsenic grade in the lead concentrate and (b) raising the zinc grade in the zinc concentrate.
- Evaluate POX vs BIOX: pressure oxidation should be evaluated versus bioleaching. It is recommended that this trade-off is conducted initially as a paper exercise so as to identify if bioleaching offers any potential advantage over pressure oxidation, before testwork is initiated to validate the workability of bioleaching.
- Evaluate the Albion process: If the concentrates are amenable to Albion, it is possible that Albion would offer both capital and operating cost savings over POX.

- Validate hot cure: The hot cure process has been assumed for the PEA but has not been tested. This needs to be proven in the laboratory, and the impact on lime and limestone consumption needs to be determined.
- Prove up use of limestone for neutralization: A study is needed to characterize (assay) the candidate limestone sources and to test their effectiveness in neutralization of the hot cure solutions.
- Testing of the chemical stability of the residues to account for the regulatory requirements from a permitting perspective.
- More variability flotation testing: A better picture of the spatially-driven variability in metallurgy needs to be established, this would allow a better prediction of mine life metallurgy to be developed.
- Development of a process water management strategy: This complex process potentially has several water recirculation systems. These need to be balanced.

Tailings storage facility

- Evaluate additional tailings storage potential within the Klaza open pits KL1 and 2 by constructing containment dams across the low point of the pits and increasing storage capacity.
- Complete rheology testing and geotechnical testing of the tailings streams.
- Complete geotechnical investigations to evaluate foundation conditions and construction material sources.
- Complete geotechnical investigations at the Klaza KL1 and 2 open pits to determine suitability for tailings storage.

Environmental

- Continue ongoing and constructive dialogue with Little Salmon/Carmacks First Nation and Selkirk First Nation to keep them informed of project progress.

Proposed budget for recommendations

An approximate budget for the work described above is presented in Table ES10.

Table ES10 Proposed budget for recommendations

Parameter	Cost (C$000's)
Diamond drilling – 30,000 m (including consumables and mobilization)	3,200
Labour for exploration drilling	750
Exploration camp, field gear, rentals, food and consumables	500
Assay and analytical	520
Excavator and fuel	110
Office and Senior supervision	350
Hydrological monitoring and modelling	200
Geotechnical drilling,testwork and modelling and interpretation	200
Metallurgical and Mineralogical studies	360
PFS (Geology and mining components)	800
Logistics, airfares, ground transportation and shipping	75
Expediting and safety	75
Environmental baseline studies to stage ready for environmental asessment	600
Aerial photos and other studies	170
Consultants management fee	390
Contingency @ 15%	1,245
Total (excluding Taxes)	**9,545**

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1 e-copy to Rockhaven Resources Limited
1 e-copy to AMC Vancouver office

2 Introduction

AMC Mining Consultants (Canada) Ltd. (AMC) was commissioned by Rockhaven Resources Ltd. (Rockhaven) to prepare a Preliminary Economic Assessment (PEA) and corresponding NI 43-101 Technical Report (Technical Report) for the Klaza property (Property) in Yukon, Canada.

The Property is located in the southwestern Yukon Territory of Canada approximately 50 km due west of the town of Carmacks. The Property is 100% owned by Rockhaven. Rockhaven acquired the original Property from ATAC Resources Ltd. in 2009 and subsequently added more claims through both acquisition and additional staking. Of a total of 1,317 claims, 213 claims are subject to a 1.5% Net Smelter Return (NSR) royalty. The other 1,104 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 mineable by open pit and underground methods for the BRX and 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), Knight Piesold (KP), Morrison Hershfield (MH) and H. Hamilton & Associates Inc. (HMH).

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**
Mr. G Methven	Principal Mining Engineer	AMC Mining Consultants (Canada) Ltd.	Yes	18-19 August 2015	P.Eng (BC)	1 (part), 2, 3, 15, 16 (part), 20, 21 (part), 22, 24, 25 (part), 26 (part), 27
Dr. A Ross	Principal Geologist	AMC Mining Consultants (Canada) Ltd.	Yes	18-19 August 2015	P.Geo (BC)	1 (part), 4-12,14, 23, 25 (part), 26 (part)
Mr. C Martin	Principal Metallurgist	Blue Coast Metallurgy Ltd.	Yes	No visit	C.Eng	1 (part), 13,17, 19, 25 (part), 26 (part)
Mr. W Hughes	Principal Mechanical Engineer	AMC Mining Consultants (Canada) Ltd.	Yes	No visit	P.Eng (BC)	1 (part), 5 (part), 18, 25 (part), 26 (part)
Mr. P Lebleu	Principal Mining Engineer	AMC Mining Consultants (Canada) Ltd.	Yes	No visit	P.Eng (BC)	1 (part), 16 (part), 21 (part), 26 (part)
Mr. B Borntraeger	Engineer	Knight Piesold	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. Glenn R Yeadon	Secretary & Director	Tupper, Jonsson & Yeadon	No	No	4	
Ms. Jennifer Turner	Senior Environmental Planner/Department Manager	Morrison Hershfield Limited	Yes	No	20	
Mr. M Dumala	Senior Engineer and Partner	Archer, Cathro & Associates (1981) Limited	No	Ongoing visits	Overall assistance	
Mr. A Carne	Project Engineer	Archer, Cathro & Associates (1981) Limited	No	June 4, 2012	Overall assistance	
Mr. H Hamilton	President	Hugh Hamilton & Associates	Yes	No	19	

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 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 26 February 2016.

3 Reliance on other experts

The Qualified Persons have relied, in respect of legal aspects, upon the work of the Expert listed below. To the extent permitted under NI 43-101, the Qualified Persons 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 Qualified Person(s).

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

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

The following disclosure is made in respect of this Expert:

Jennifer Turner, 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 Qualified Person(s).

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

4 Property description and location

The Property is located in southwestern Yukon at latitude 62°15´11" north and longitude 137°7´23" west on NTS 115I/3 (Figure 4.1). It comprises 1,317 contiguous mineral claims (totalling approximately 25,400 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, J. Bill#, Jon-Wedge, Rat, Wedge, Ox and Bull) are subject to a 1.5% NSR royalty payable to Janet Dickson of Whitehorse. The six Desk claims have a 1.5% NSR royalty payable to R. Hulstein and R. Stroshein Estate. The other 1,104 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	April 15, 2023
	97-152	YD58527-YD58582	April 15, 2023
	153-172	YD62853-YD62872	April 15, 2023
	173-255	YD113413-YD113495	April 15, 2023
	256-384	YE60326-YE60454	April 15, 2023
Dic	1-7(1)	YA93470-YA93476	January 11, 2033
	101-106(1)	YB35470-YB35475	January 11, 2034
Eagle	1-12(1)	YB35415-YB35426	January 11, 2034
Etzel	1-12(1)	YA86336-YA86347	December 1, 2040
	13-17(1)	YA86348-YA86352	December 1, 2039
	18-20(1)	YA86353-YA86355	December 1, 2040
	21-28(1)	YA86356-YA86363	December 1, 2039
	29-32(1)	YA86364-YA86367	December 1, 2040
	33(1)	YS86368	December 1, 2036
	34(1)	YA86369	December 1, 2040
	35-44(1)	YA86370-YA86379	December 1, 2037
	45-50(1)	YA86380-YA86385	December 1, 2039
JCS	1-3	YC25916-YC25918	December 01, 2028
Klaza	1-17*	YC37984-YC38000	January 11, 2036
	18-24*	YC39051-YC39057	January 11, 2036
	25-40	YD09205-YD09220	January 7, 2036
	43-64	YD09223-YD09244	January 7, 2036
	65F-66F	YC99541-YC99542	January 11, 2036
	68-129	YD07149-YD07210	January 11, 2036
	133-166	YD07214-YD07247	January 11, 2036
	167-308	YD119737-YD119878	January 11, 2032
	309	YD110502	January 11, 2032
	310-311	YC97706-YC97707	January 11, 2033
	314-316	YC97722-YC97724	January 11, 2033
	317-319	YC99801-YC99803	January 11, 2027
	320-357	YE66241-YE66278	January 11, 2024
VG	1-4 (1)	YA86406-YA86409	December 01, 2037
	5-8 (1)	YA86410-YA86413	December 01, 2029
VIC	2 (1)	YA86309	December 01, 2039
	75 (1)	YC19429	December 01, 2037

Claim Name		Grant Number	Expiry Date**
	76-78 (1)	YC19430-YC19432	December 01, 2040
Wedge	11-14 (1)	YA82177-YA82180	December 01, 2029
Dade	1-16	YD07685-YD07700	March 23, 2027
	17-54	YD108507-YD108544	March 23, 2024
	77-90	YD108567-YD108580	March 23, 2020
	91-96	YC97716-YC97721	March 23, 2025
	97-106	YD07248-YD07257	March 23, 2021
Krast	1-32	YD74101-YD74070	January 11, 2020
Queen	1-121	YE60731-YE60851	April 24, 2020
Val	1-9	YC25903-YC25911	February 24, 2026
	10-15*	YE85801-YE85806	February 24, 2026
Nor	1-74	YE60651-YE60724	April 24, 2020
Bull	1-2 (1)	YA81420-YA81421	December 1, 2036
	12 (1)	YA86291	February 29, 2028
	14 (1)	YA86293	February 29, 2028
	16-20 (1)	YA86295-YA86299	February 28, 2027
	21-28 (1)	YA86300-YA86307	February 28, 2022
D	1-2	YB57373-YB57374	January 20, 2022
	3-4	YB57375-YB57376	January 20, 2030
Desk	1-6 (2)	YC47461-YC47466	March 23, 2020
J. Bill	1-2 (1)	YA78049-YA78050	February 28, 2022
	3-4 (1)	YA78051-YA78052	February 28, 2023
	5-12 (1)	YA78053-YA78060	February 28, 2022
	13 (1)	YA78061	February 2, 2030
	14 (1)	YA78062	February 2, 2034
	15-16 (1)	YA78063-YA78064	February 2, 2030
	17-28(1)	YA78065-YA8076	February 28, 2022
	29-30 (1)	YA78077-YA78078	February 28, 2030
	31-32 (1)	YA78079-YA78080	February 28, 2034
JBF	6	YB36958	December 01, 2028
	10	YB54543	December 05, 2029
Jon-Wedge	1 (1)	YB35895	December 01, 2030
	2 (1)	YB35896	December 01, 2028
	3 (1)	YB35897	December 01, 2020
	4 (1)	YB35898	December 01, 2021
	5-6 (1)	YB35899-YB35900	December 01, 2020
Ox	1-20 (1)	YA86386-YA86405	December 20, 2020
Rat	1-8 (1)	YA81428-YA81435	February 28, 2022
	9-24 (1)	YA81436-YA81451	February 28, 2023
	25-40 (1)	YA81452-YA81467	February 28, 2022
Sked	1-30	YD07655-YA07684	March 23, 2022
	31-36	YC99722-YC99727	March 23, 2022

Notes: (1) A total of 207 mineral claims (the Dic, the Eagle, the Etzel, the VG, the VIC, the Wedge, the Bull, the J.Bill, the Jon-Wedge, the Ox, and the Rat) 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 2015 work which has been filed for assessment credit but not yet accepted.

Figure 4.1 Property location - Klaza Property

Figure 4.2 Claims location – Western part of Property

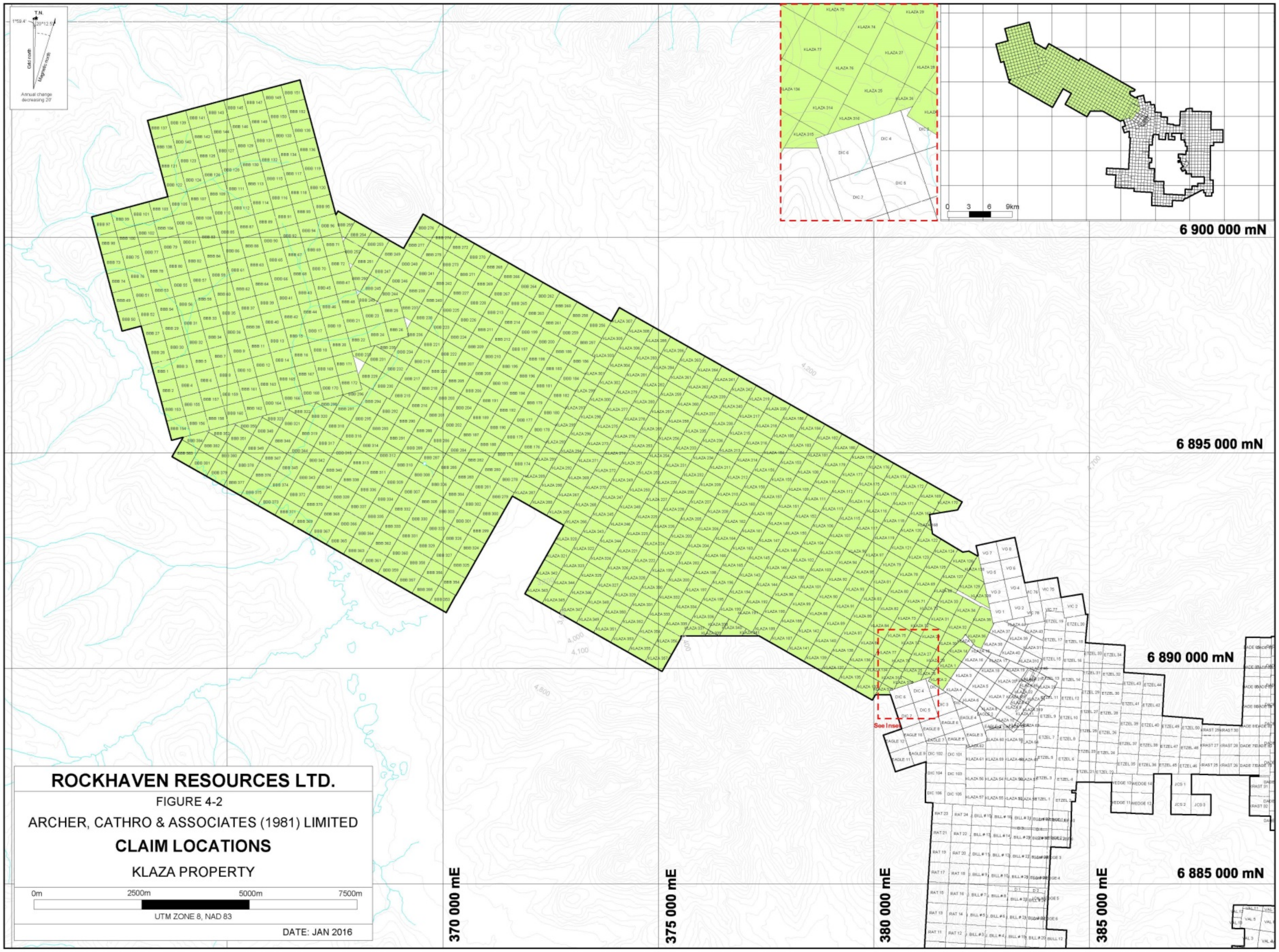
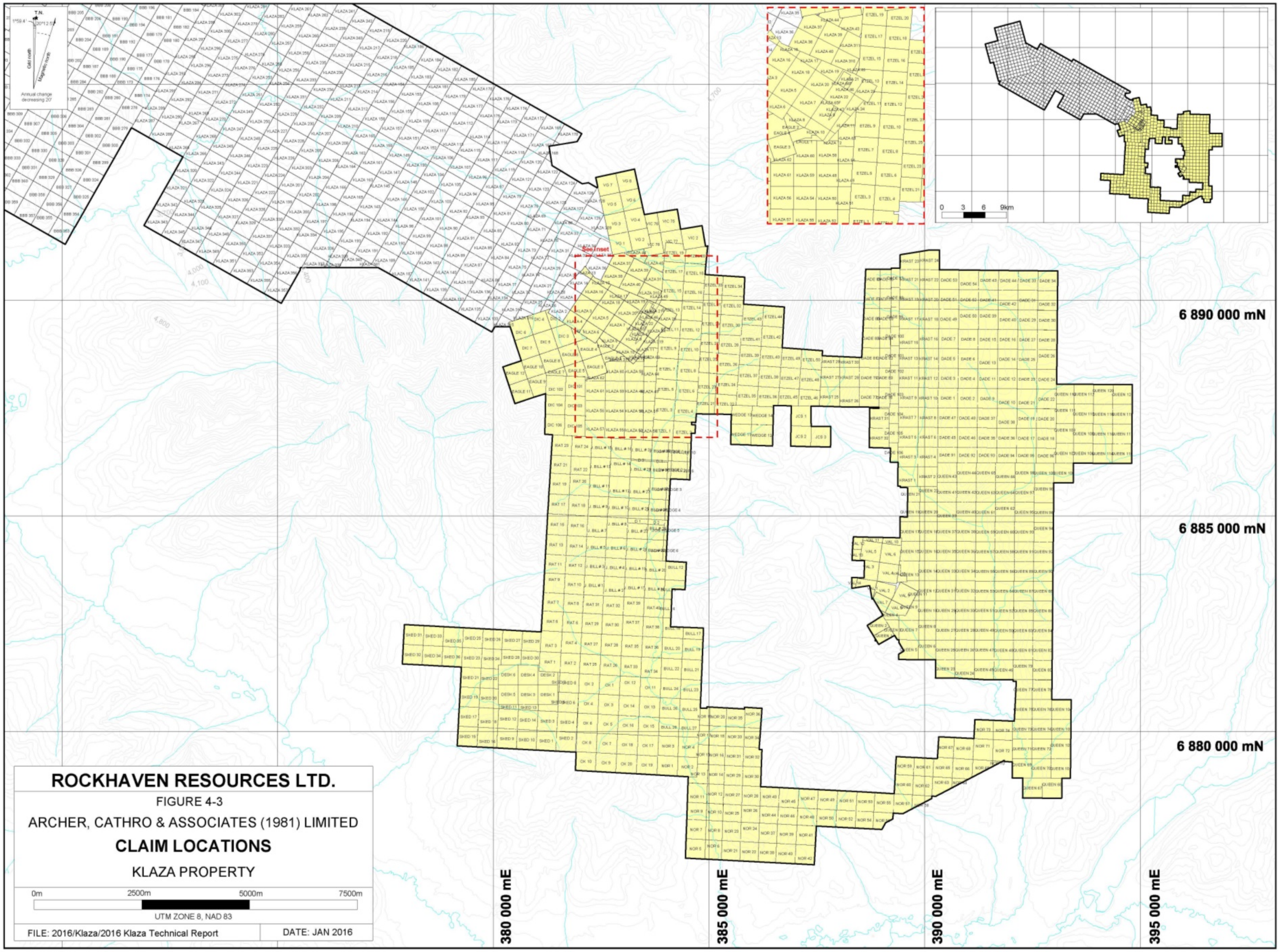


Figure 4.3 Claims location – Eastern part of Property



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

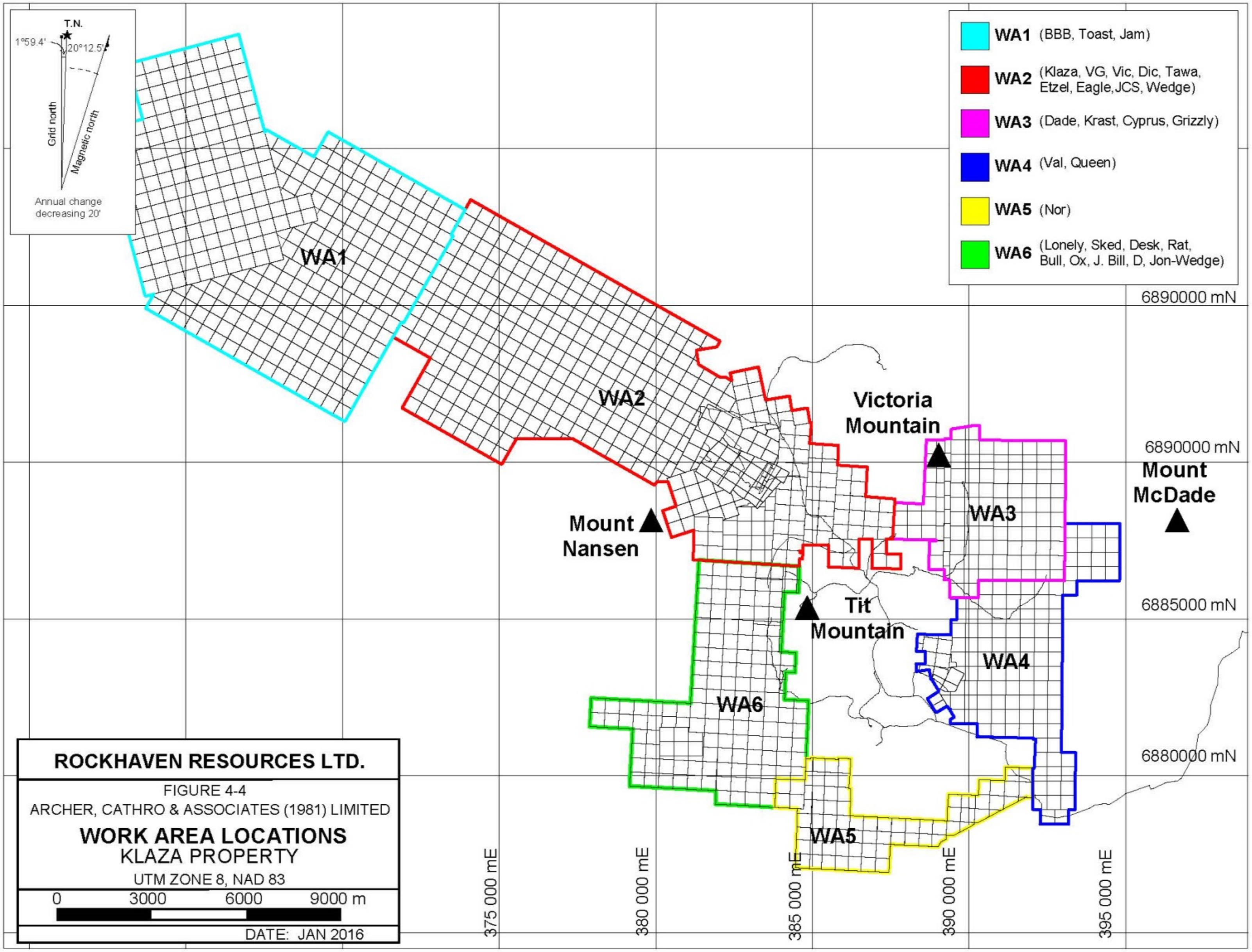


Table 4.2 Work Area claims

Work Area	Claim Groups	Area (km^2)
1	BBB	78.5
2	Dic, Eagle, Etzel, JCS, Klaza, Tawa, VG, Vic, and Wedge	84.7
3	Dade, Krast	21.0
4	Queen, Val	25.2
5	Nor	13.2
6	Bull, D, Desk, J. Bill, JBF, Jon-Wedge, Ox, Rat, and Sked.	31.5

The Klaza Property is the primary focus of this report. No mineral resources or 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. 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 by Rockhaven under Class III Mining Land Use Approval LQ00434, which expires 6 December 2020.

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 northwestern corner overlaps with the traditional territory of the Selkirk First Nation. 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 town 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 420 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 gently sloping area. Mineralized rock stockpiles will be avoided during operations by careful scheduling of production. Tailings will be stored in nearby containment areas and the open pits.

Personnel will reside in Carmacks and commute to the mine site daily. The existing road will be extended to the mine-site for this purpose. 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 an 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.

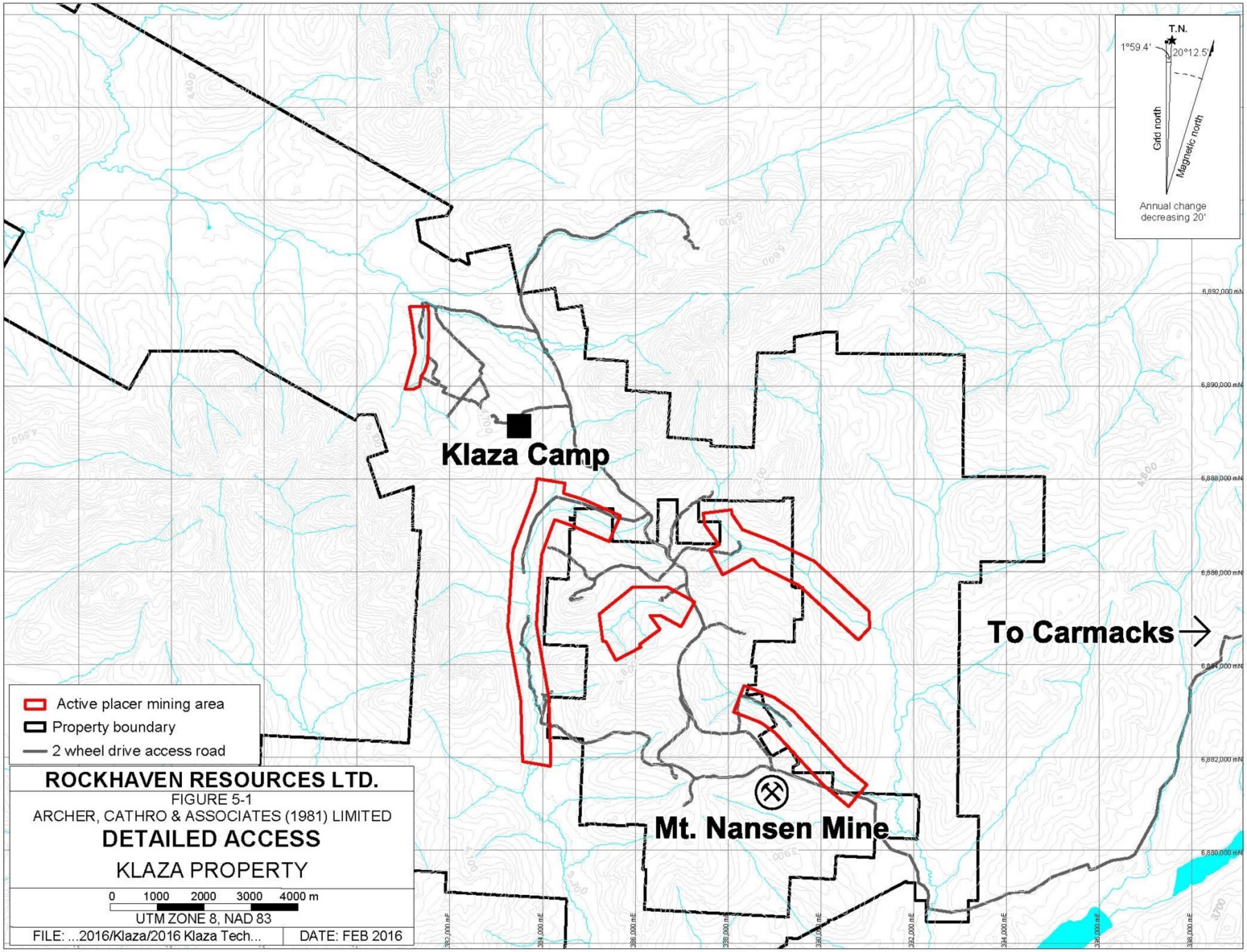
All work programs to date have been conducted from a tent frame camp on the Property. Drilling and excavator trenching sites have been accessed using All Terrain Vehicles, four-by-four trucks or heavy equipment. In 2013, EBA Engineering Consultants Ltd. of Whitehorse was retained to prepare a Terrain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

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.

Figure 5.1 Detailed access - Klaza Property



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 bottom, flanking the Klaza River. The main areas of interest lie along a northwesterly elongated ridge. Elevations on the Property range between 1,200 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 spruces 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 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 18ºC during the day and 5ºC at night. Winter temperatures average -12ºC 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 rarely reached bedrock, often because of permafrost, and small diameter drillholes 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 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 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.

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 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 northwesterly 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, northwesterly trending anomalies.
1984 (McClintock, 1986)	G. Dickson	Wedge	Staked the Wedge claims	n/a
1985 (McClintock, 1986)	G. Dickson	Wedge	Limited trenching	n/a
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	Very low frequency electromagnetic, magnetic (VLF-EM) and geochemical surveys.	Northwesterly trending VLF-EM and magnetic anomalies and soil geochemical values up to 1,825 ppb gold and 1,049 ppm copper.
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.

Year of Work (Report)	Owner/ Operator	Claim Group/Target	Work Performed	Results
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 graded16.01 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.
2011 (Tarswell and Turner, 2012)	Rockhaven Resources Ltd.	Klaza	Geophysical surveying, diamond and reverse circulation drilling, excavator trenching, soil geochemical sampling, and air photo surveys.	Best drill intercept returned 5.43 g/t gold and 50 g/t silver over 14.80 m. Peak soil geochemical values were 320 ppb gold, 61.3 ppm silver, 606 ppm lead and 213 ppm arsenic.
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 (Tarswell and Turner, 2013)	Rockhaven Resources Ltd.	Klaza, Dic and Eagle	Diamond drilling, excavator trenching, and soil geochemical sampling.	Best drill intercept returned 5.78 g/t gold and 111 g/t silver over 15.62 m. Peak soil geochemical values were 920 ppb gold, 20.9 ppm silver, 722 ppm lead and 1,750 ppm arsenic.
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
2013 (Tarswell and Turner, 2014)	Rockhaven Resources Ltd.	Klaza	Excavator trenching	Best trench result was 5.61 g/t gold and 300 g/t silver over 18.79 m
2014 (Dumala, Tarswell and Turner, 2014)	Rockhaven Resources Ltd.	Klaza	Geophysical surveys, excavator trenching, diamond drilling, and metallurgical testing.	Best drill intercept returned 16.29 g/t gold and 1,435 g/t silver over 1.37 m.

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 March 11, 2015 and amended June 19, 2015 (Wengzynowski et al., 2015).

In 2005, ATAC Resources Ltd. (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.

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
2013	Rockhaven Resources Ltd.	Val	Ran a two trench program.	Collected 82 samples over 330 m of trenching, locating mineralized veins below soil anomalies. Trenches returned 3.09 g/t Au with 5.93 g/t Ag over 1.00 m and 1.91 g/t Au with 131 g/t Ag over 1.30 m.

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.

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
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
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.
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
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

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 northwest 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. All but the Sked claims are subject to a 1.5% NSR payable to previous owners.

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 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). The following discussion is primarily based on maps prepared by Gordey and Makepeace and the Yukon Geological Survey (YGS) website.

The Property lies within the Yukon-Tanana Terrane (YTT) approximately 100 km southwest of the Tintina Fault and 100 km northeast 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 southeastern Yukon from the main body of YTT, which lies southwest 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 gneisses (Pelly Gneiss), metasedimentary and metavolcanic rocks (Nisling) and enigmatic ultramafic and mafic units (Amphibolite). 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, Casino Suite, Prospector Mountain Suite and Carmacks).

Figure 7.1 Regional geology and tectonic setting Klaza Property

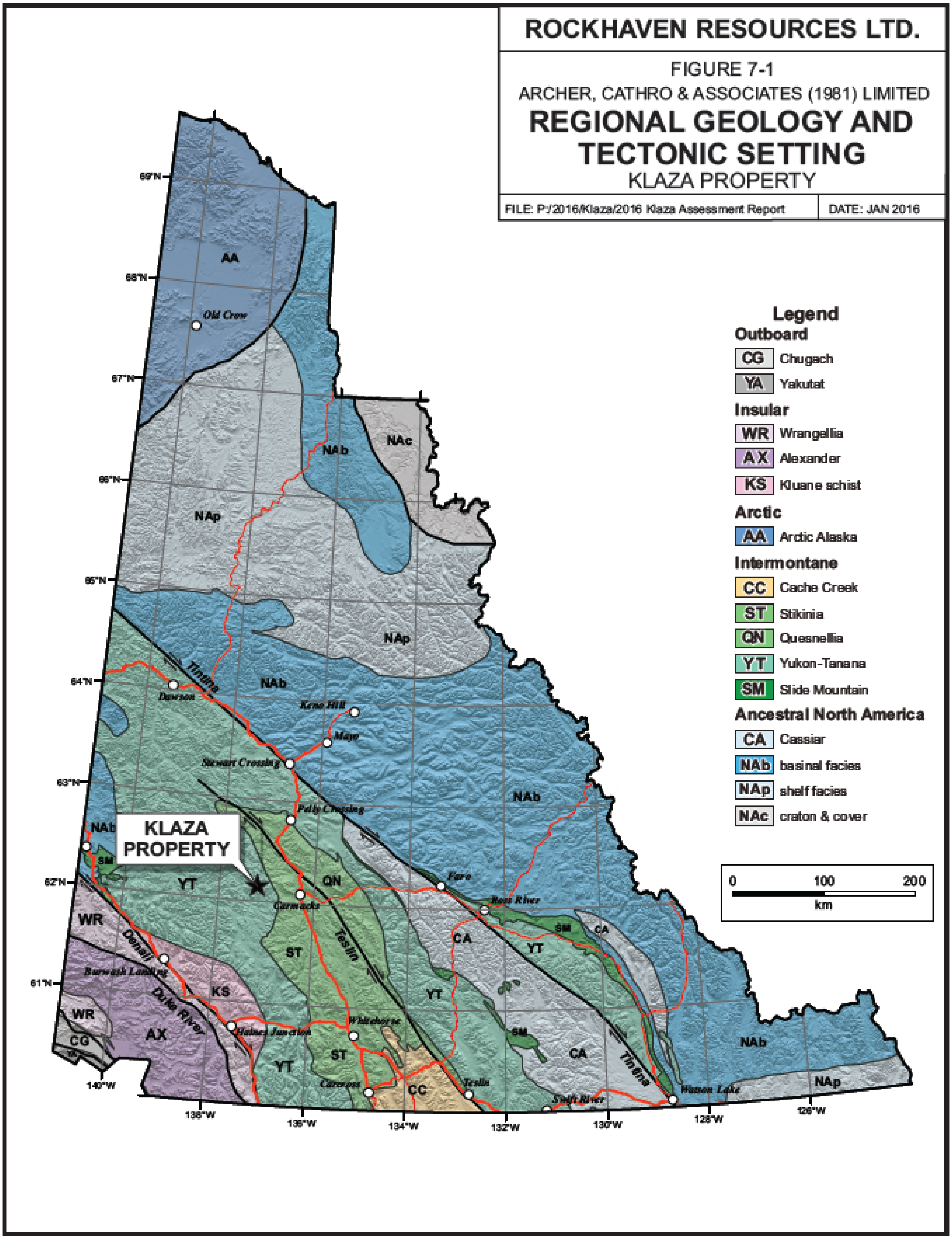


Table 7.1 Regional lithologies

UPPER CRETACEOUS	
uKC	**uKC: CARMACKS** 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.)**
uKcs2	**uKcs2: CASINO SUITE** Porphyry: quartz monzonite to dacite; fine to medium grained; alkali feldspar-plagioclase phyric, biotite and quartz porphyritic; tuff, breccias, and breccia pipes, with some breccias cemented by tuffaceous matrix; equigranular granodiorite and fine quartz diorite.
LATE CRETACEOUS TO TERTIARY	
LKP	**LKP: 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
MID-CRETACEOUS	
mKN	**mKN: MOUNT NANSEN** 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)**

PROTEROZOIC AND PALEOZOIC	
PPa	**PPa: AMPHIBOLITE** metamorphosed mafic rocks including amphibolite (1) and ultramafic rocks (2) of unknown association; i.e. may belong in part or entirely to Nisling, Nasina, and Slide Mountain assemblages and (3), mafic-ultramafic intrusions within Nasina assemblage
LATE DEVONIAN TO MISSISSIPPIAN	
DMPW	**DMPW: PELLY GNEISS SUITE** variably deformed granitic rocks of predominantly felsic (q) to intermediate composition (g) southwest of Tintina Fault: q. foliated equigranular medium grained muscovite quartz monzonite; moderately to strongly foliated K-feldspar augen bearing quartz monzonitic to granitic gneiss **(S. Fiftymile Batholith, Mount Burnham Orthogneiss)** g. foliated medium grained, homogeneous biotite granite gneiss to biotite or hornblende granodiorite gneiss; massive to strongly foliated dioritic to granodioritic gneiss; includes interfoliated amphibolite, quartz-mica schist and phyllite **(Selwyn Gneiss, Pelly Gneiss, N. Fiftymile Batholith, Moose Creek Orthogneiss)**
LATE PROTEROZOIC AND PALEOZOIC	
PPN	**PPN: NISLING ASSEMBLAGE** assemblage characterized by mica quartz feldspar schist (1) and abundant locally thick limestone (2) members; includes possibly equivalent strata northeast of Tintina Fault(3): 1. dark grey to brown, biotite-muscovite-quartz-feldspar schist, quartzite and micaceous quartzite, garnetiferous; felsic chlorite-biotite orthogneiss; rare amphibolite; minor (?) two-mica gneiss and hornblende diorite gneiss; may include Nasina assem. **(Nisling assemblage)** 2. bleached white weathering, white to grey, coarsely crystalline, flow banded, fetid marble; graphite, chert, metabasite and calcsilicate lamina are common **(Nisling assemblage)** 3. calcareous quartz psammite, marble, calcareous chlorite-biotite schist and calcsilicate; calcareous garnet-biotite-muscovite schist, rare amphibolite; biotite-quartz-muscovite schist and lesser quartz-feldspar- muscovite augen schist **(assignment uncertain, could belong to Nasina assemblage)**

Modified after Gordey and Makepeace (2000)

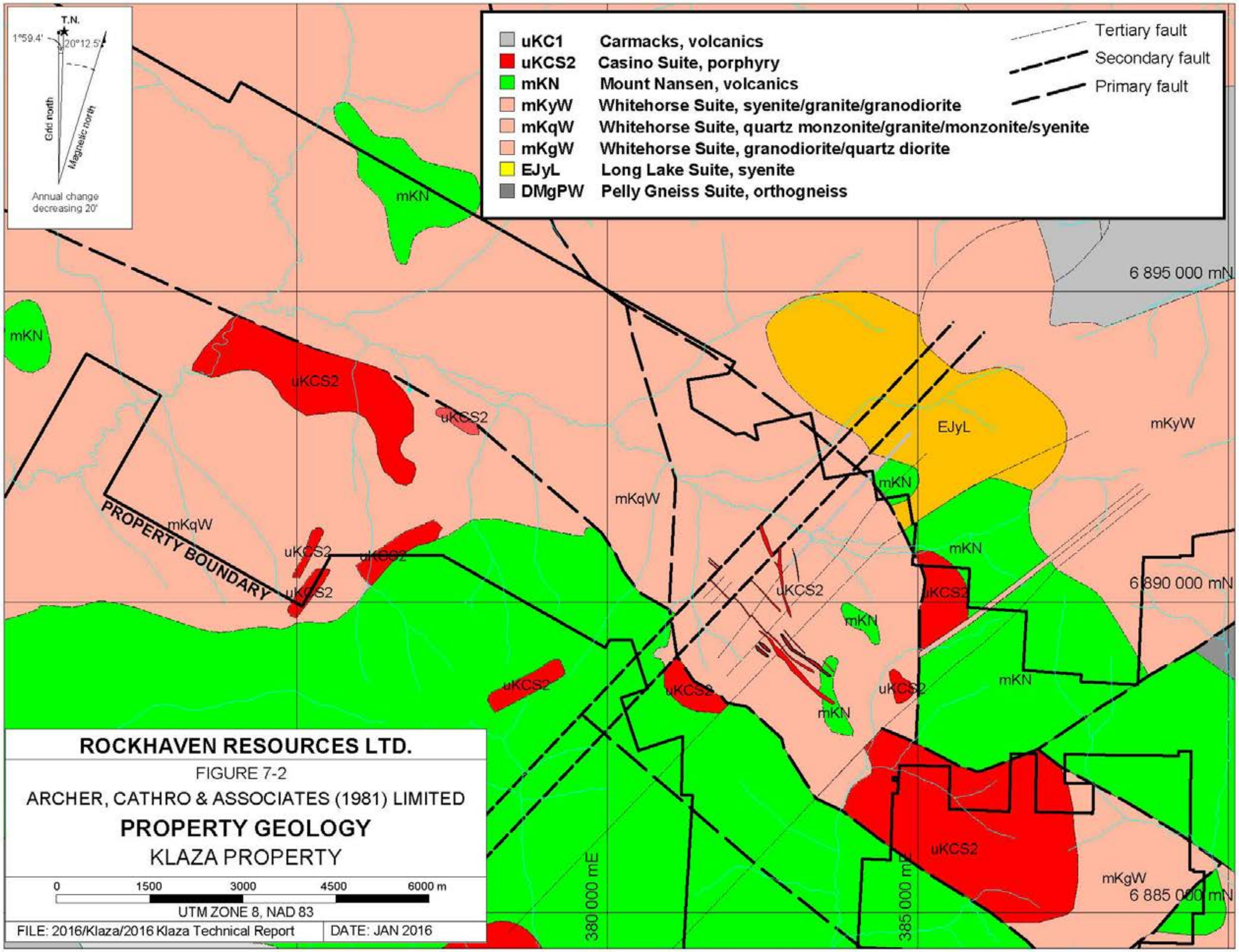
7.2 Property geology

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 magnetic data, mostly collected over the past five exploration seasons on the core of the Property. This corresponds to Work Area 2 as discussed in Section 6.

No 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 is a pluton of the Early Jurassic Long Lake Suite, which outcrops in the northeast corner on the map. Most of the map area is underlain by Mid-Cretaceous Whitehorse Suite granodiorite. This granodiorite contains 30% hornblende and biotite. It is coarse-grained and non-foliated.

Figure 7.2 Property geology –Klaza Property



A moderate size quartz-rich granite to quartz monzonite stock (LKq) intrudes granodiorite in the southeast corner of the map area and is thought to be the main heat source for hydrothermal cells responsible for mineralization. This pluton, and feldspar porphyry dykes (LKfp) related to it, are now assigned by YGS to the recently recognized Casino Suite (uKcs2), which is younger than the Mount Nansen suite but older than Prospector Mountain Suite (Mortenson et al. in press). The Casino Suite is associated with most porphyry-style mineralization within the Dawson Range.

A series of northwesterly trending feldspar porphyry dykes (LKfp), emanating from the stock in the southeastern 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 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.

Sub-aerial volcanic and volcaniclastic rocks belonging to the Mount Nansen and Carmacks volcanics are found on the periphery 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 the Mid and Late Cretaceous intrusions, respectively.

Two main fault trends (NW and NE) are present in the map area. The first set strikes northwesterly and dips between 60 and 80° to the southwest. 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 northeasterly, 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 northwesterly trending structures is still uncertain, but they appear to have been in part coeval and may have played an important role in ground preparation.

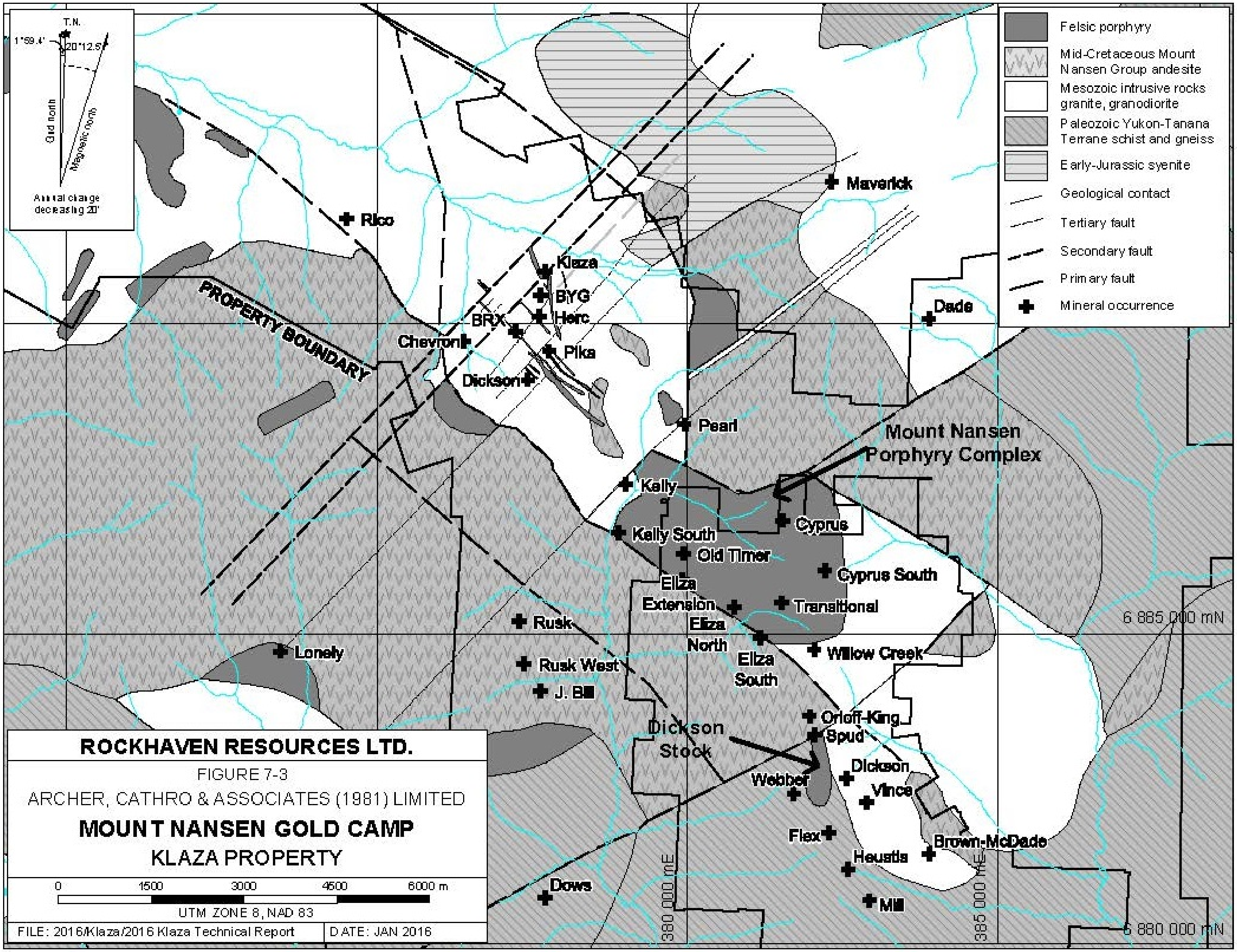
A third set of structures is slightly oblique to the main mineralized faults, striking more westerly. They are less continuous and are considered to be Riedel shears. High-grade mineralization is sometimes localized at junctions between these shears and the northwesterly trending structures.

7.3 Mineralization

The Property lies within the northern part of the Mount Nansen Gold Camp (MNGC), a northwest 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



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 southeast 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.

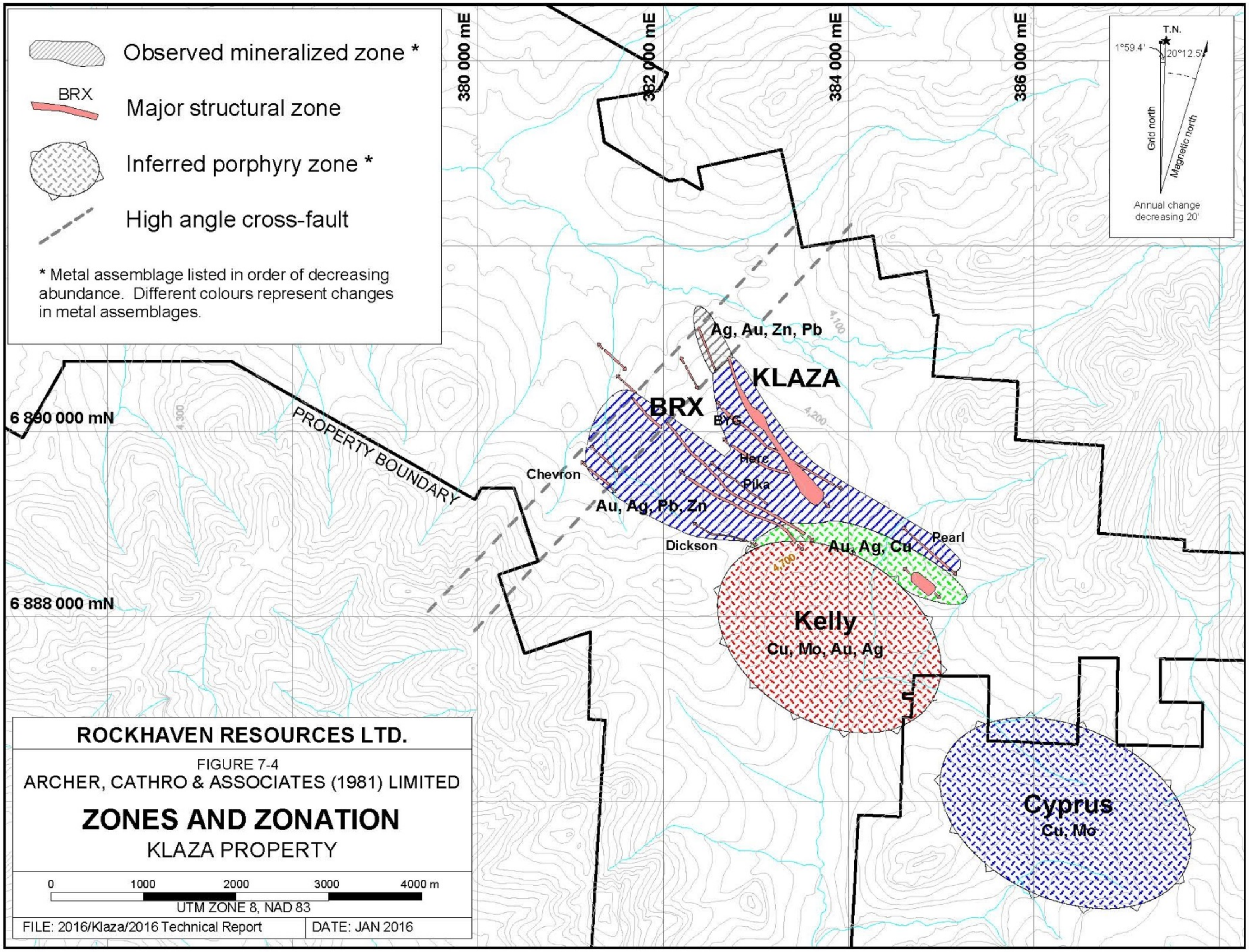
The western porphyry centre (Kelly zone) is located on the Property and was explored as early as 1973. The Kelly zone is defined by coincident geochemical and geophysical anomalies, including: 1) strongly elevated gold, copper and molybdenum soil geochemical response; 2) high chargeabilities with moderate 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 nine main mineralized structural zones in Work Area 2 that are developed northwest of the porphyry targets. The structural zones collectively form a 2 km wide corridor that cuts northwesterly 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 Chevron, Dickson, AEX, BRX, Pika, Herc, BYG, Klaza and Pearl. 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 and Central Klaza zones. The current Mineral Resource estimate contains mineralization from parts of these five areas.

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



Depth of surface oxidation ranges from 5 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 southwest. 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 southeast.

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 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 northwesterly 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, northeast trending faults have been observed to cut across the northwestern 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 northeast 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: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 March 11, 2015 and amended June 19, 2015 (Wengzynowski et al., 2015).

7.3.2 Mineralization – other work areas

Historic work on Rockhaven's recently acquired claims identified various mineralized trends; however, they require further investigation. The most noteworthy of these veins are found in Work Areas 3, 4, and 6. Rockhaven has not performed any work on the recently acquired claims.

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°-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 to 20 m along a 175 m strike length and V2 over widths of 2 to 12 m along a 125 m strike length (Burrell, 2013). The veins comprise white to grey quartz with boxwork limonite and locally 1-3% 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 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 northwest trending structural corridor, which is interpreted to have focused fluid flow away from weak porphyry centres related to a Late Cretaceous stock in the southeastern 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 sulfidation 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 (New Guinea), Buritica (Colombia) and Kelian (Indonesia).

The CBM class of deposits has yet to be identified elsewhere 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 develop 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 (nine) mineralized structures with long strike lengths (>2,400 m)
Bonanza grade gold mineralization	Some bonanza grade intercepts

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 2015 within the main area of interest, Work Area 2, are described below, except for drilling which is discussed in detail in Section 10.0. Work Areas 1 and 3-6 are on ground acquired by Rockhaven in the summer of 2015 and have no exploration work conducted 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 by 3,000 m) area of moderately to strongly anomalous copper-in-soil response, which partially defines the Kelly porphyry target in the southeastern corner of Work Area 2.

Grid soil sampling was performed by Rockhaven from 2010 to 2012, Rockhaven expanded grid sample coverage to the west and north of the earlier grids, and collected samples on a few contour - controlled lines in the northwestern part of Work Area 2. 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

The structural corridor hosting the nine 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 southwest and northeast of the structural corridor, where no mineralized zones have been discovered to date. Northwest 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 southeastern 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. Zone locations are shown in Figure 9.6.

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

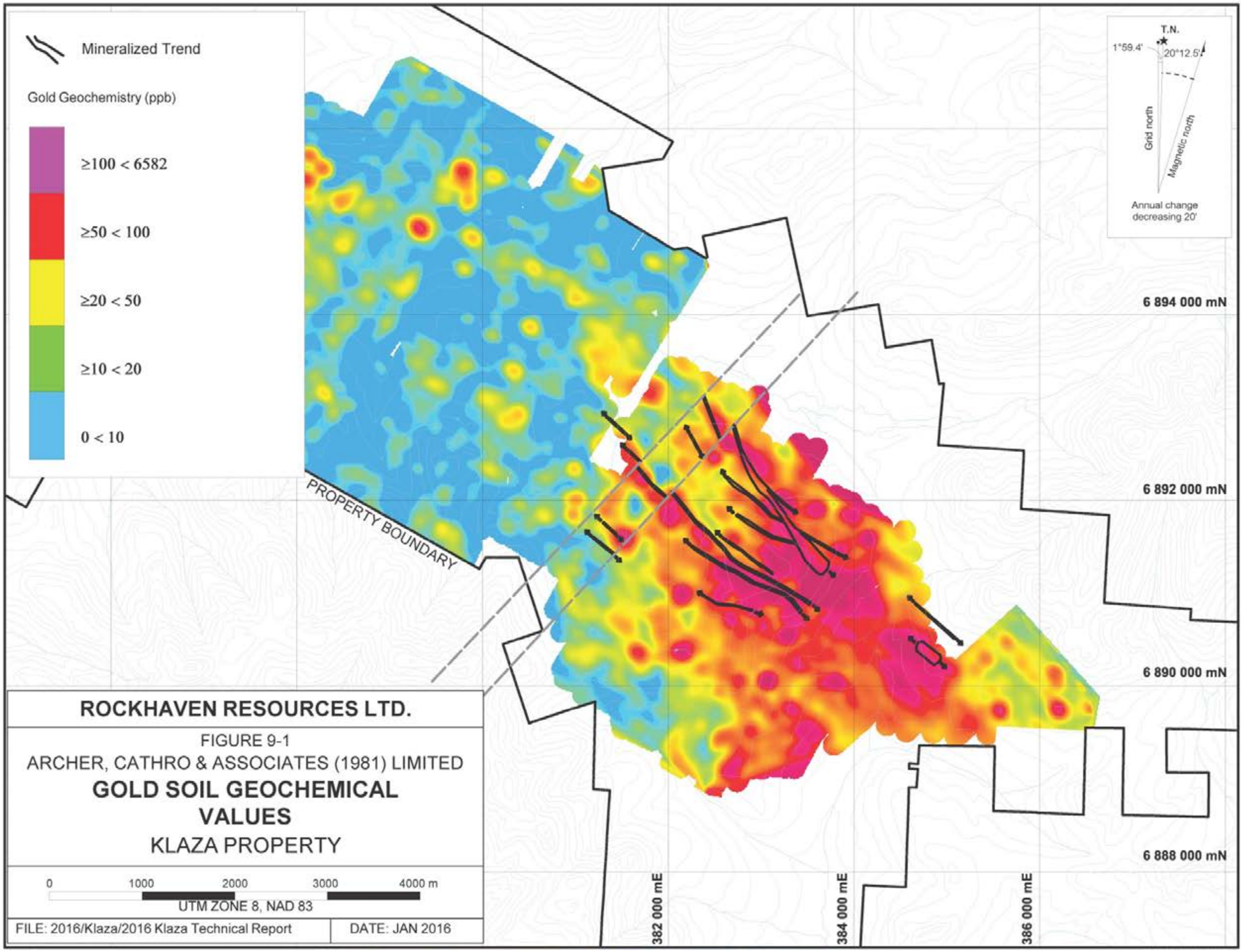


Figure 9.2 Silver soil geochemical values – Work Area 2 Klaza Property

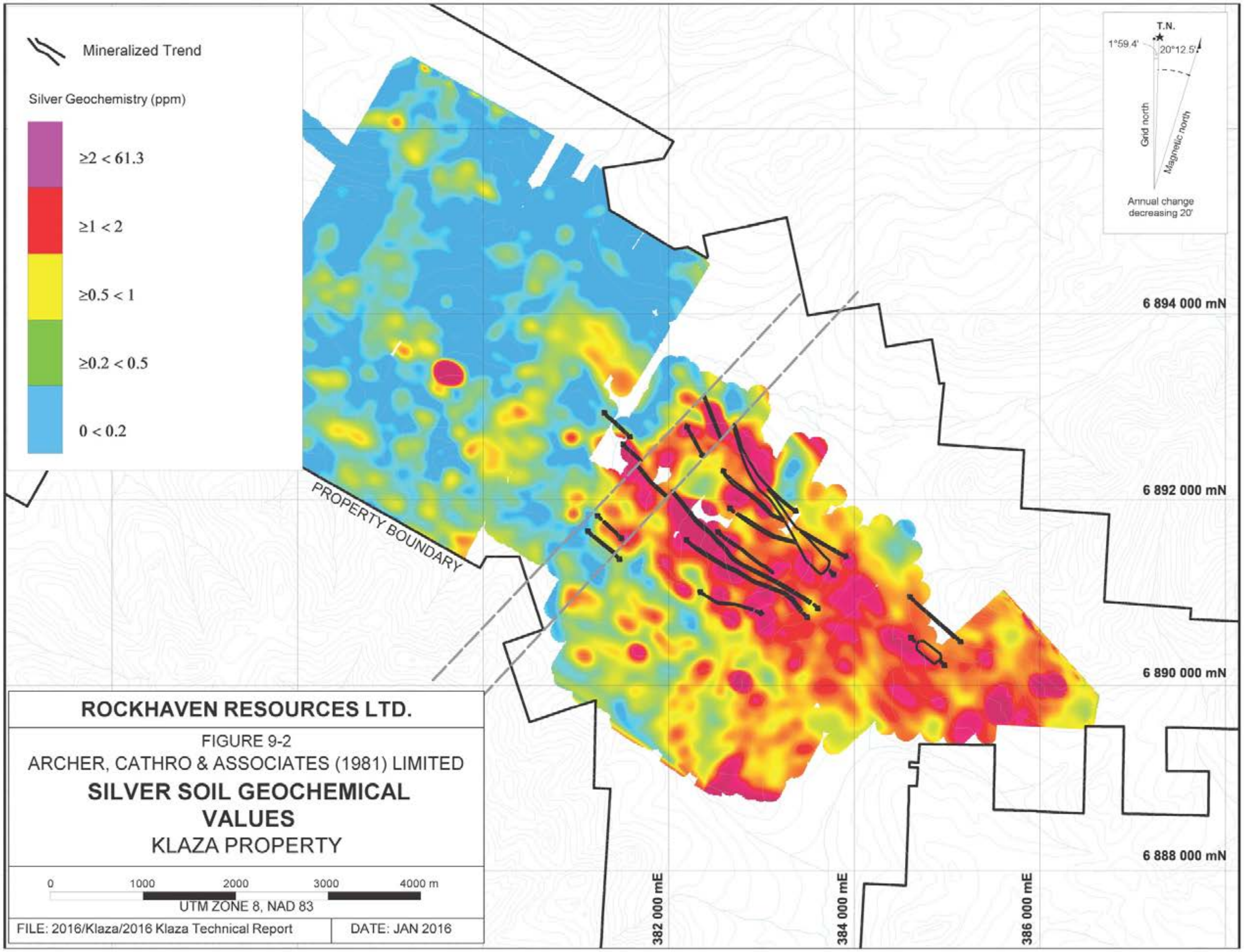


Figure 9.3 Lead soil geochemical values – Work Area 2 Klaza Property

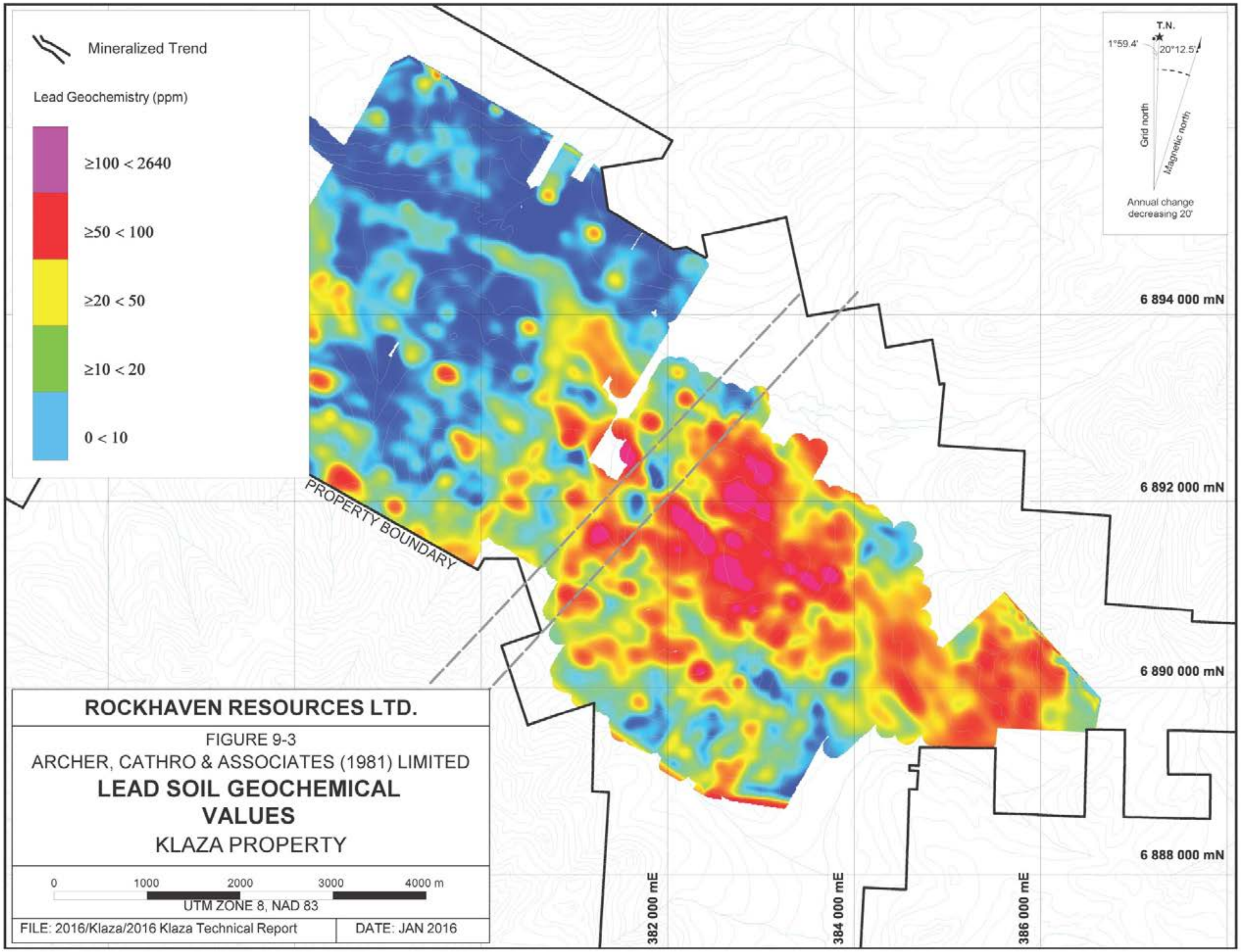


Figure 9.4 Arsenic soil geochemical values – Work Area 2 Klaza Property

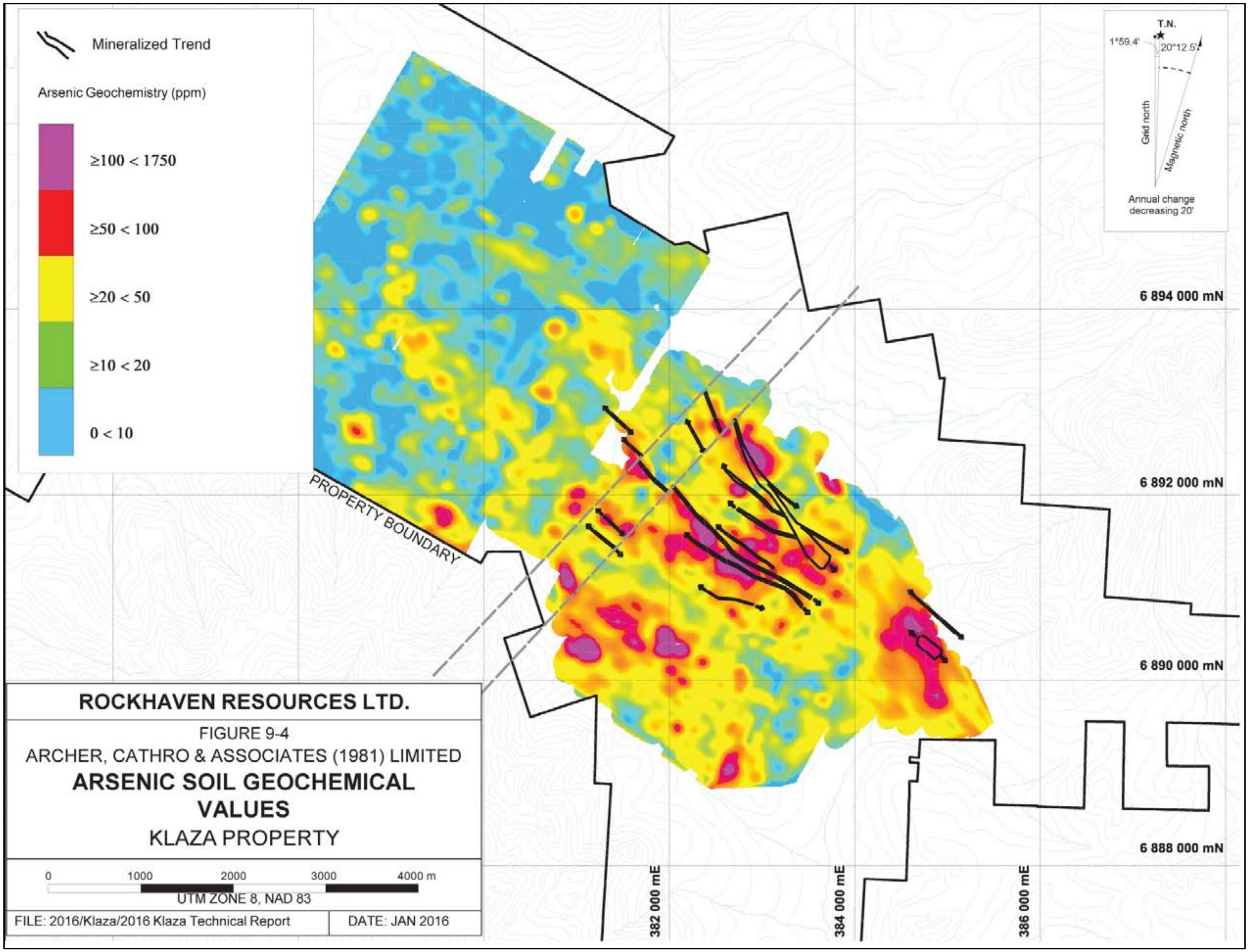
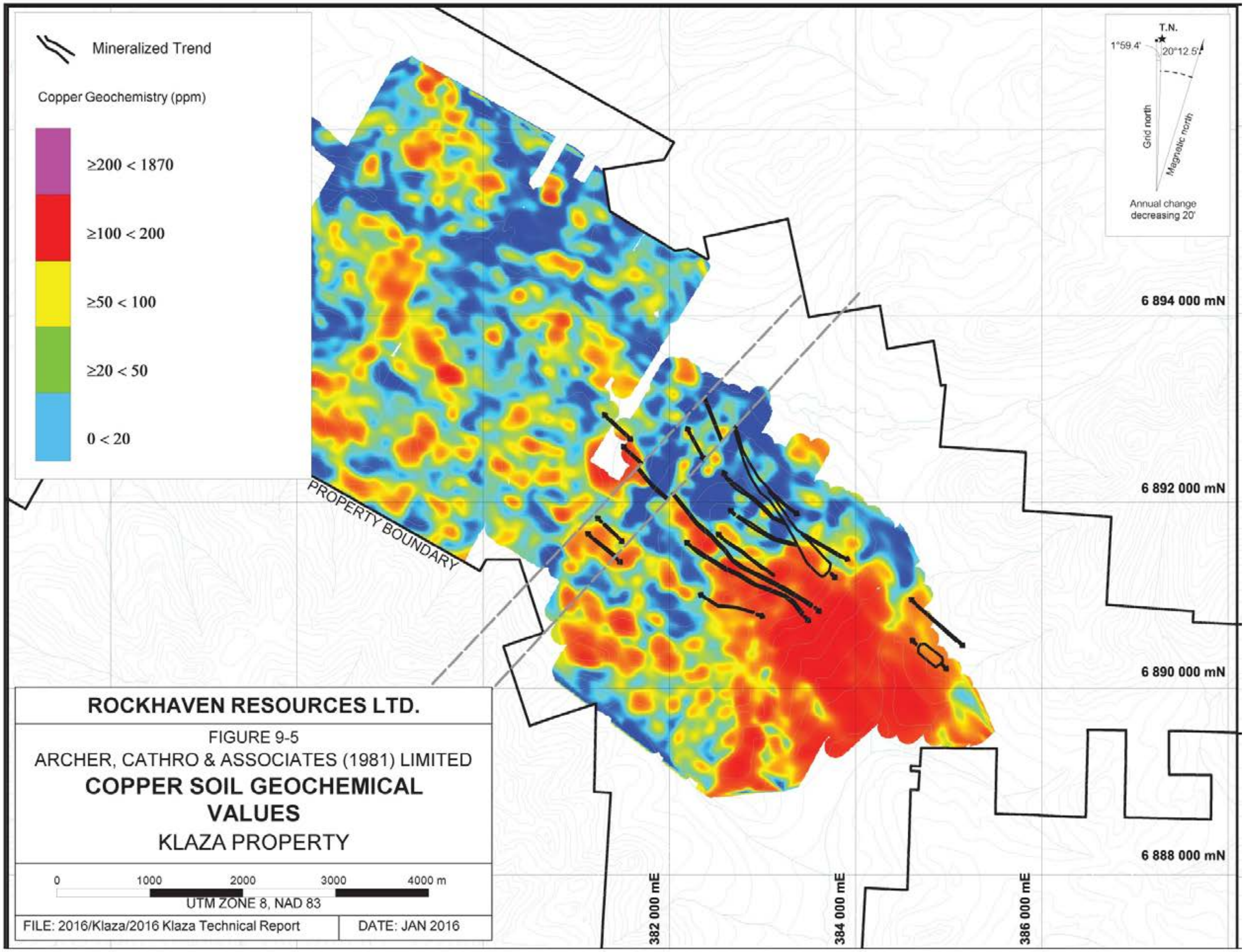


Figure 9.5 Copper soil geochemical values - Work Area 2 Klaza Property



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 22,366 m of trenching in 89 trenches between 2010 and 2015. Table 9.2 lists the total number and combined lengths of trenches completed by Rockhaven each year during that period.

Table 9.2 2010 to 2015 Excavator trenching summary

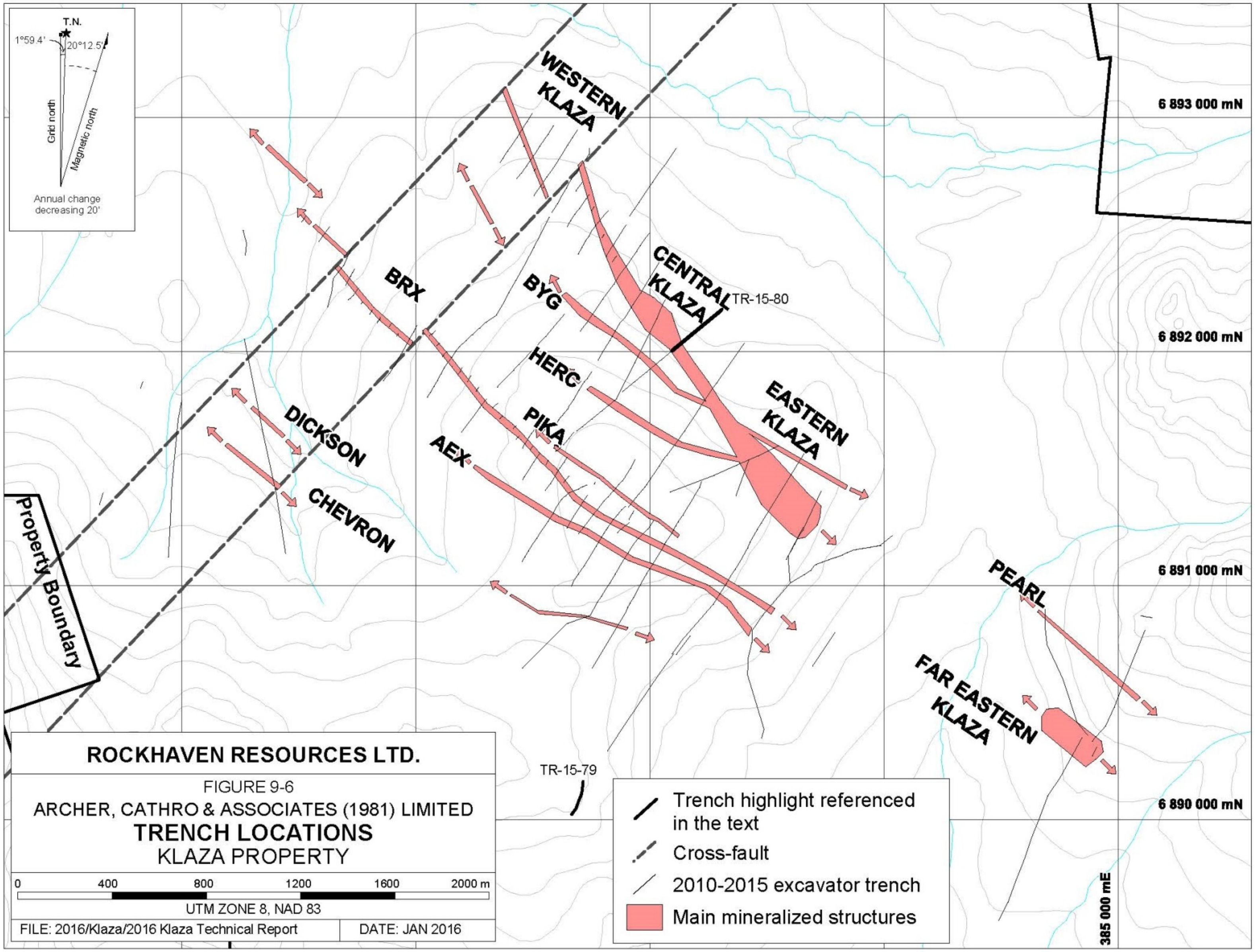
Year	Number of Trenches	Total Length (m)
2010	21	8,000
2011	12	4,050
2012	11	4,000
2013	38	5,000
2014	5	880
2015	2	436
TOTAL	**89**	**22,366**

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 Sections 11.

Individual zones and key trench results obtained prior to 2015 are discussed 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 2015, only two trenches were dug. Trench 80 extended a high-grade splay belonging to the Central Klaza zone in a previously untested area. The best intersection in this trench graded 36.60 g/t gold and 374 g/t silver over 2.00 m. Trench 81 did not reach bedrock.

Figure 9.6 Trench locations - Klaza Property



Note: The solid black trenches refer to 2015 trenches and are discussed in the text.

9.4 Geophysical surveys

To date, four types of geophysical surveys have been completed on Work Area 2:

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).
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 (Klaza 1 to 319 claims). 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 northwesterly trending lows coincide with mineralized structural zones, while northeasterly 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 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.

The gradient array and pole-dipole IP survey conducted by Aurora Geosciences covered a 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 southeastern 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 remains open to the south and east. 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. To date, only two trenches and one drillhole have tested the northern edge of the anomaly with the best result coming from hole KL-12-134, which included an interval that averaged 0.15% copper, 0.14 g/t gold, 2.70 g/t silver and 0.010% molybdenum over 93.15 m.

The second IP anomaly includes three westerly trending chargeability features of weak to moderate intensity. These chargeability features are 710 to 1,200 m long and are offset 30 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 Airborne magnetics first vertical derivative - Work Area 2 Klaza Property

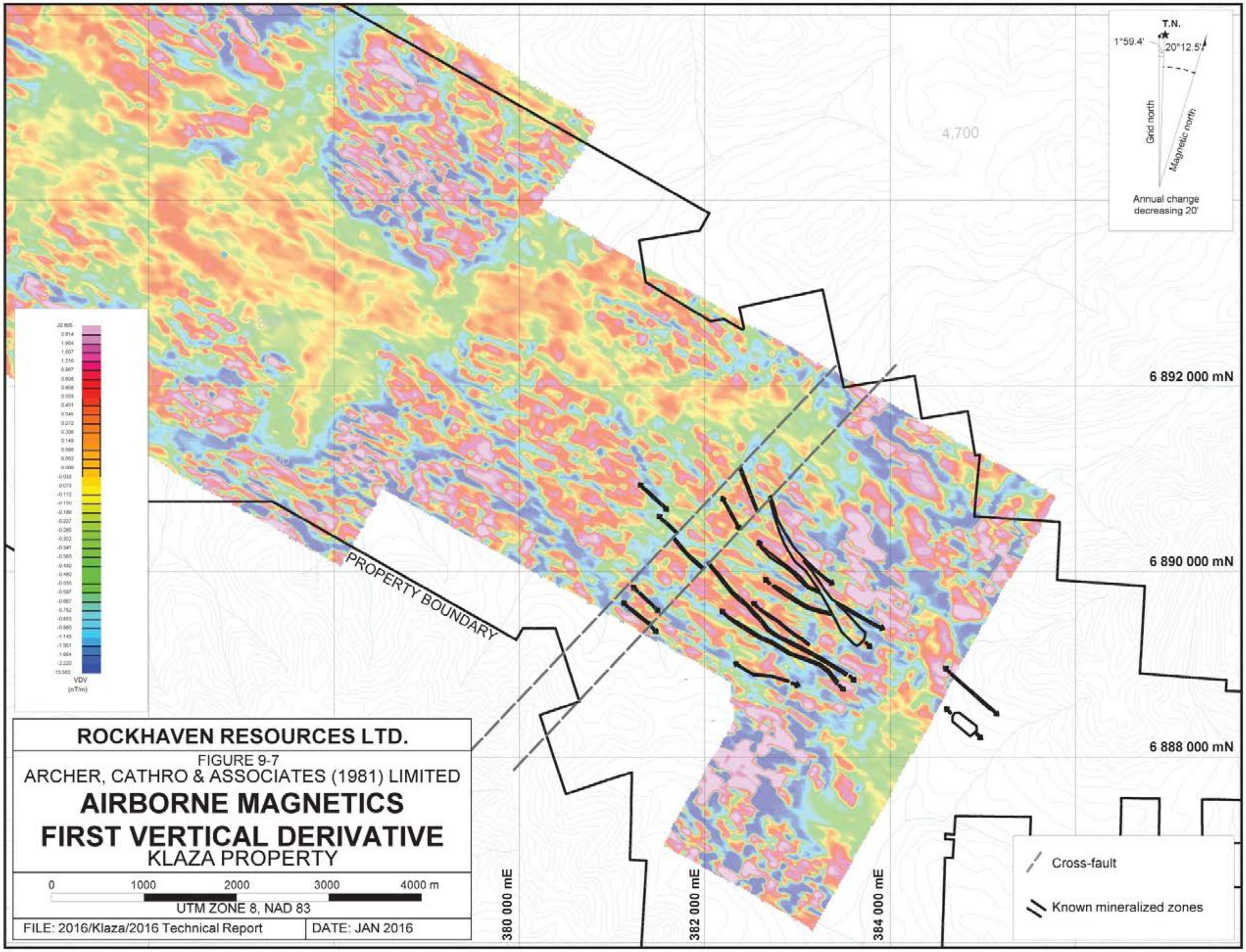
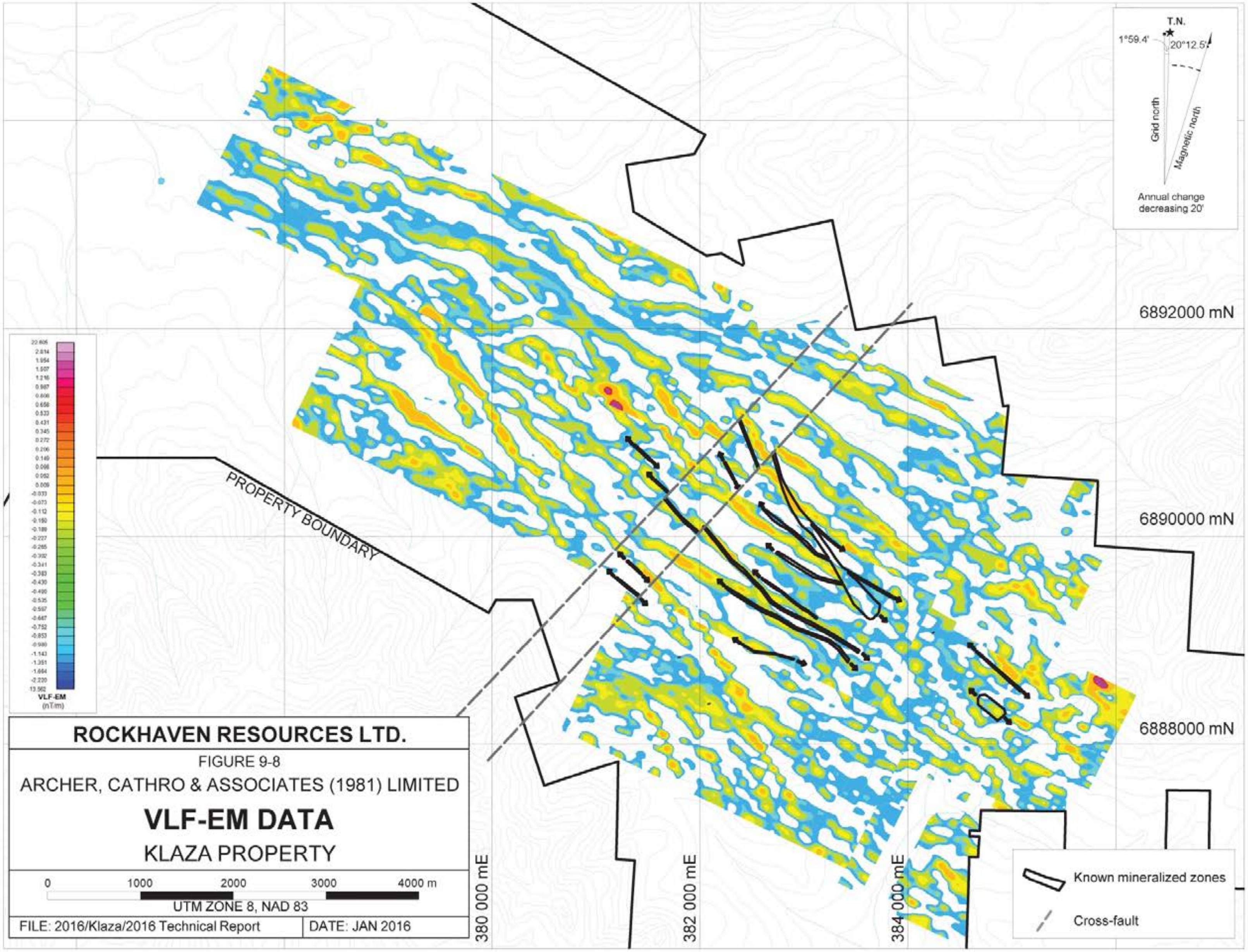


Figure 9.8 VLF-EM data - Work Area 2 Klaza Property



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 2015. It does not include any of Rockhaven's percussion drill results or any historical drill data. It also does not include assays from trenching.

10.1 Diamond drilling summary

Between 2010 and 2015, a total of 70,100 m of exploration and definition drilling was done by Rockhaven in 295 diamond drillholes on the Property. Figure 10.1 shows the location of all 295 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.

During the 2010 to 2012 programs, core recovery was good, averaging 95%, excluding the near surface portions of the holes where core recovery was poor. In 2015, core recovery averaged 96%. 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 and 2015. No drilling was conducted in 2013.

Final drillhole depths within the Klaza zone averaged 251.49 m, which included a maximum drillhole depth of 550.77 m. At the BRX zone, final drillhole depths averaged 229.25 m and reached a maximum of 559.92 m. The number of holes and total metres drilled on the Property each year between 2010 and 2015 are listed by zone in Table 10.1.

Table 10.1 2010 to 2015 Diamond drilling summary

Target – Year	Holes Drilled	Total Drilled (m)
Klaza zone – 2010	7	1,035.10
BRX zone – 2010	4	606.98
Klaza zone – 2011	39	11,211.85
BRX zone – 2011	9	1,717.25
Klaza zone – 2012	27	8,269.10
BRX zone – 2012	31	9,652.55
Klaza zone – 2014	33	6,488.33
BRX zone – 2014	57	9,882.12
Klaza zone – 2015	21	5,639.42
BRX zone – 2015	21	6,420.01
Nearby Exploration – 2011-2015	46	9,176.64

Note: nearby exploration includes all holes drilled outside the BRX and Klaza zones

Figure 10.1 Diamond drillhole locations - Klaza Property

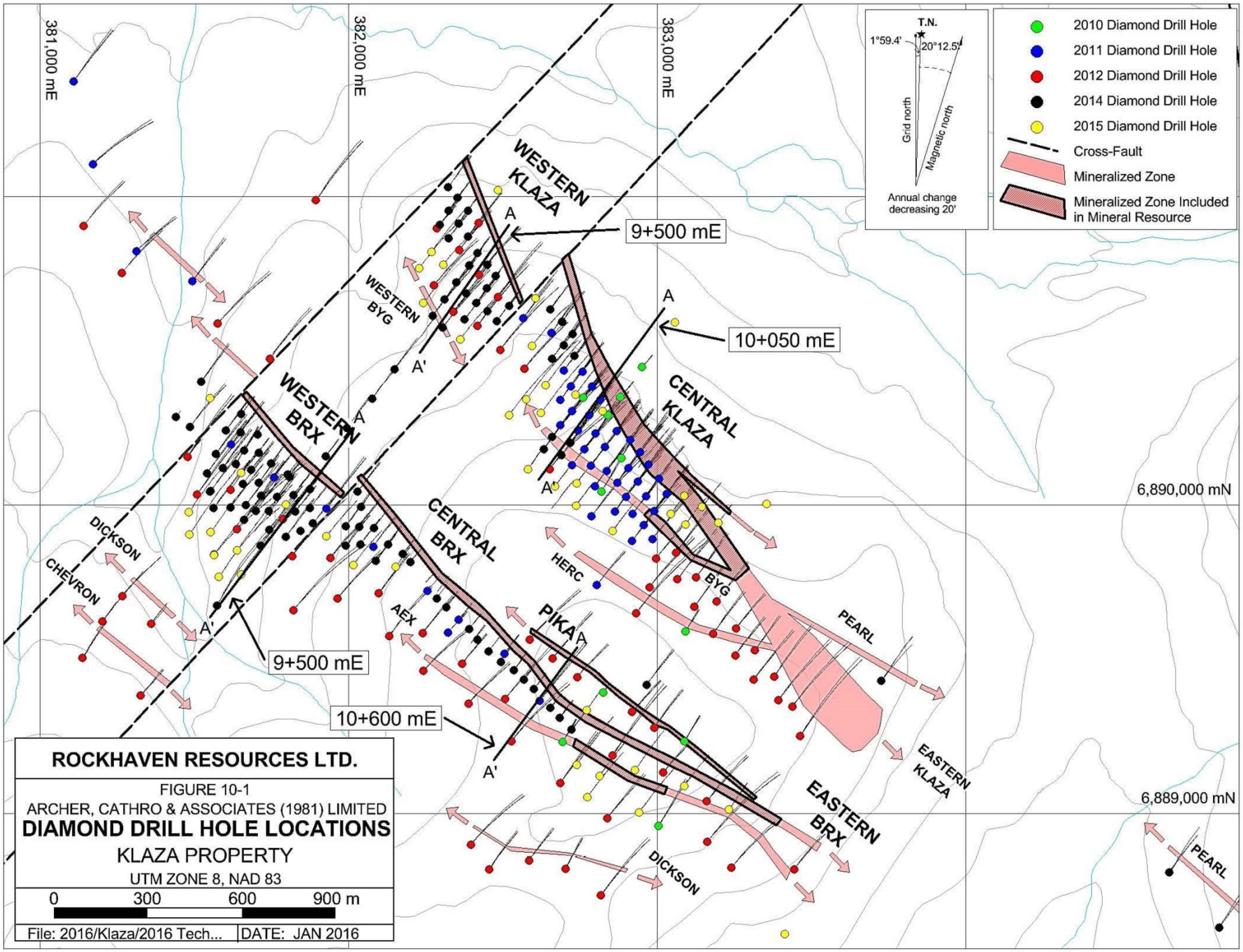


Table 10.2 shows the drill confirmed strike length of each of the main zones and the maximum down-dip intersection depth in each zone.

Table 10.2 Data for main mineralized zones

Zone	Mineralized Strike Length (m)	Maximum Down-dip Drill Intersection (m)
Western Klaza	400	250
Central Klaza	800	325
Eastern Klaza	1,100	180
Western BRX	500	520
Central BRX	950	400
Eastern BRX	950	275
Pika	740	250
AEX	1,650	310
BYG	650	150
Dickson	450	100
HERC	460	310
Chevron	250	90
Pearl	450	100

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

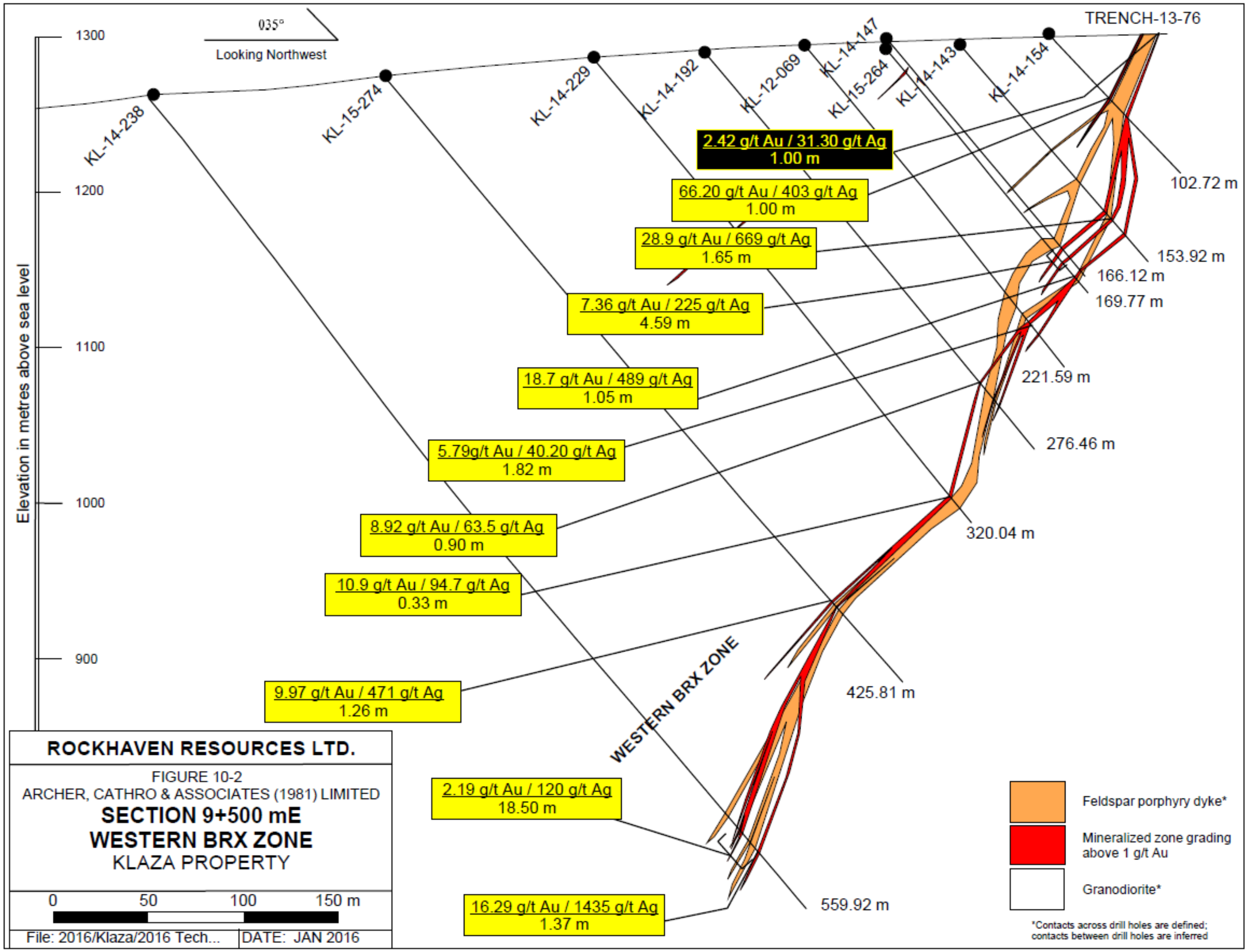
Although significant drill intersections have been obtained from all of the nine main mineralized zones, the focus of the most recent exploration has been the BRX and Klaza zones. For the purposes of deposit modelling and Mineral Resource estimation, the zones have been subdivided as follows:

- **BRX zone** – Central BRX, Western BRX and Eastern BRX zones.
- **Klaza zone** – Central Klaza and Western Klaza zones (drill density within Eastern Klaza does not support modelling at this time).

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 northwest striking and moderately to steeply southwest 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



Note: Black labelling refers to trench results. These results were not used in the Mineral Resource estimate.

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 Eastern BRX zone 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 BRX zones, sulphide mineralization in the Eastern BRX zone contains little arsenopyrite, galena and sphalerite.

The Klaza zone is located about 800 m northeast 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. Only the former two subzones are described below because the distance between drillholes in the Eastern Klaza zone is too great for the data to be included in the Mineral Resource estimate. The Western and Central Klaza zones are off-set by the same cross-fault that separates the corresponding sections of the BRX zone.

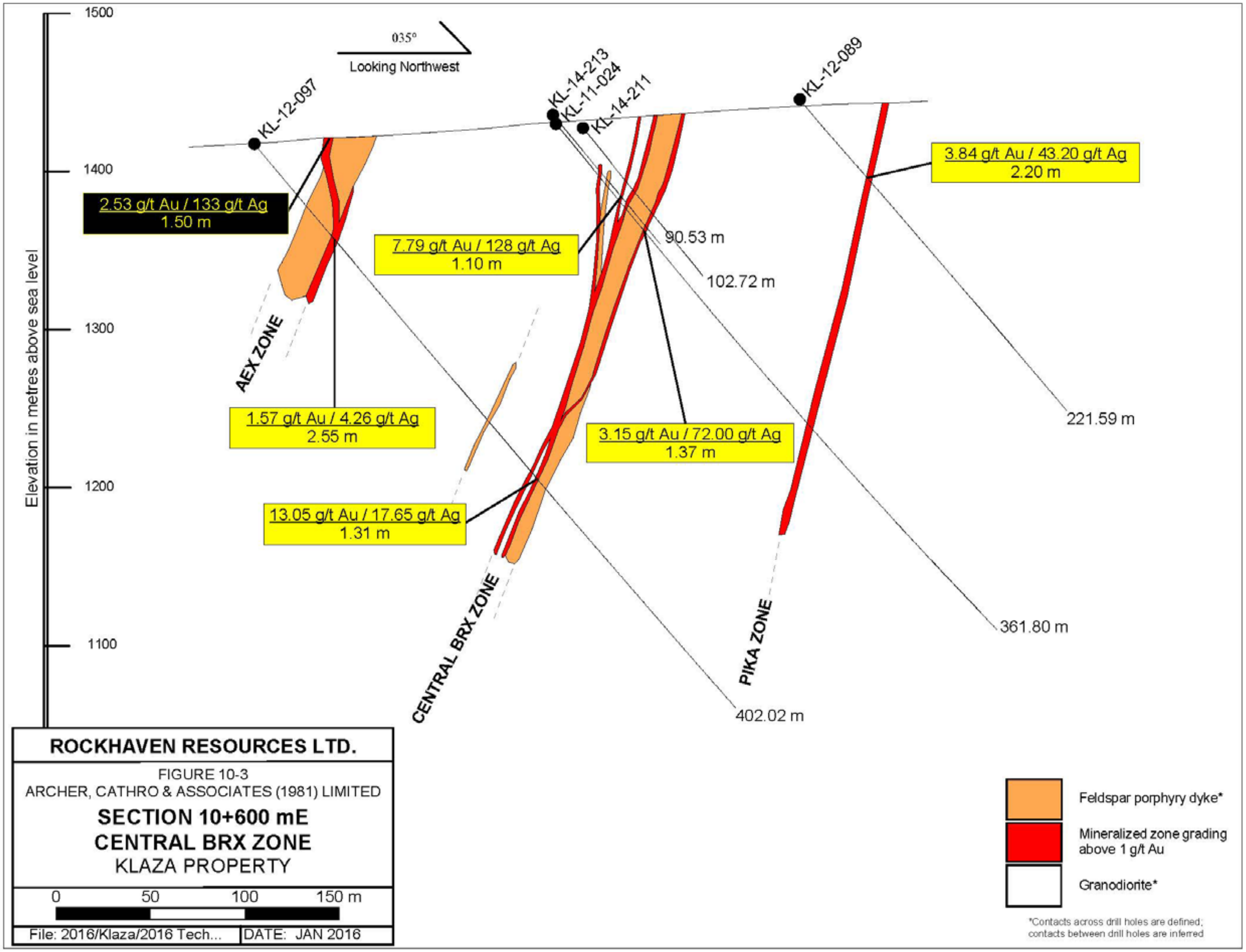
The Western Klaza zone is defined by two narrow high-grade silver-gold veins (extending west from section KL 9+700). Unlike other zones, these veins are not emplaced alongside a feldspar porphyry dyke and they are not flanked by the type of sheeted veining seen elsewhere in the Klaza zone. The mineral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

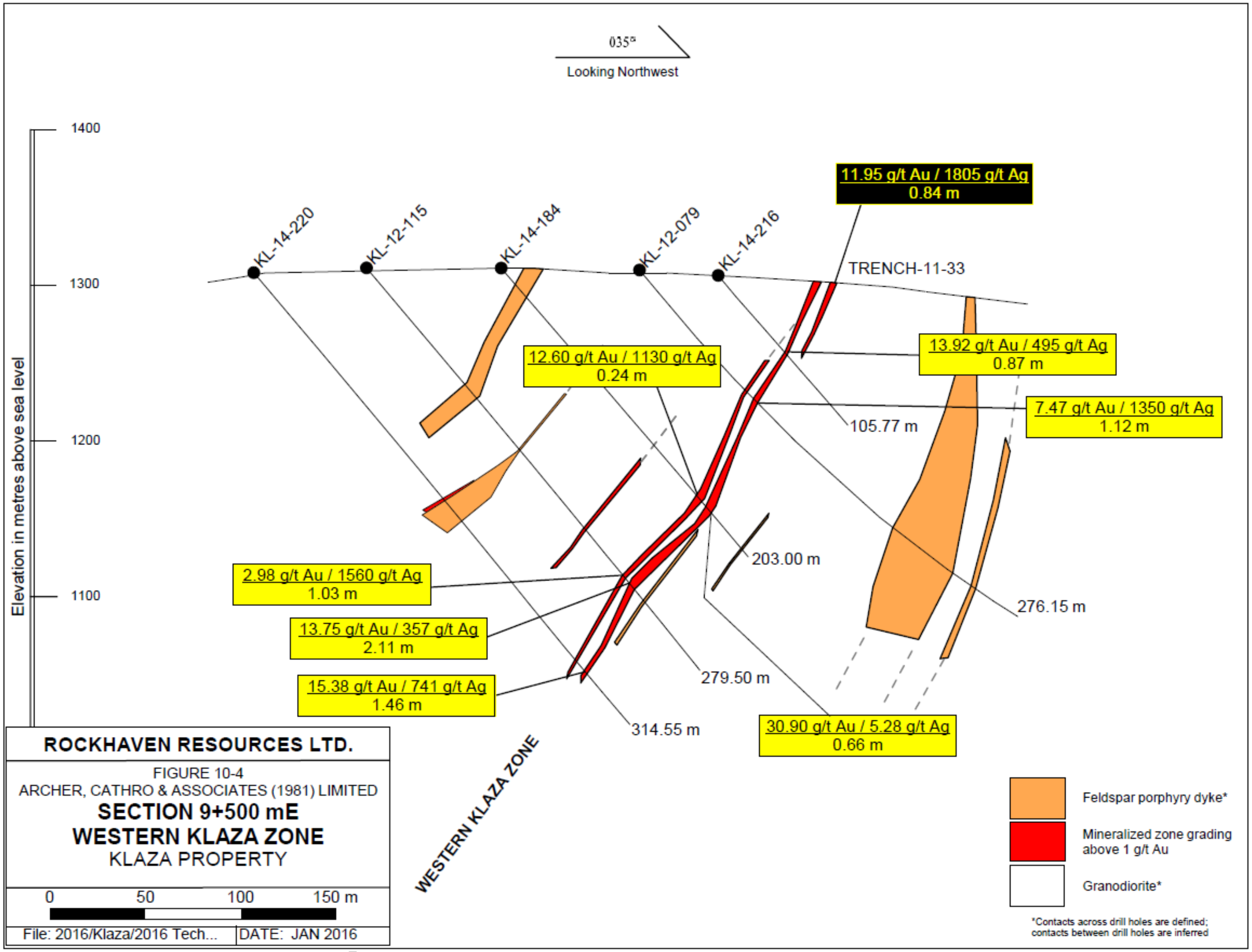
The Qualified Person does not know of any drilling, sampling or recovery factors that could materially impact the accuracy and reliability of the 2010 to 2015 results.

Figure 10.3 10+600 mE Central BRX zone - Klaza Property



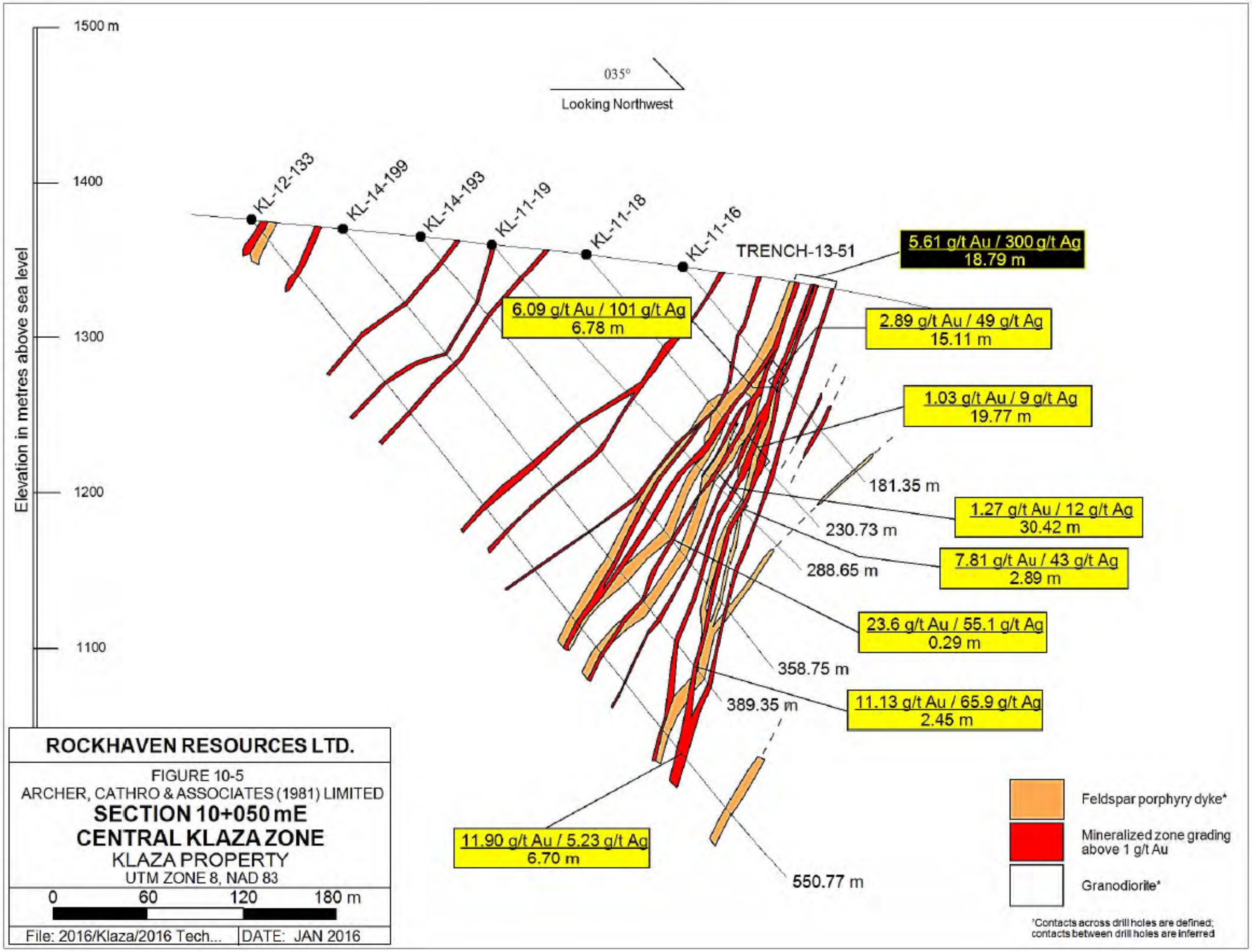
Note: Black labelling refers to trench results. These results were not used in the Mineral Resource estimate

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



Note: Black labelling refers to trench results. These results were not used in the Mineral Resource estimate.

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



Note: Black labelling refers to trench results. These results were not used in the Mineral Resource estimate.

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 and 2015, 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.

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). 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 (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 and 2015 programs were routinely surveyed every 50 feet (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

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, QA/QC and data validation, followed during the 2010 to 2015 exploration programs. The programs were supervised by Archer Cathro on behalf of Rockhaven. It is the data from these years on which the Mineral Resource estimate is based.

11.2 Sampling methods

11.2.1 Soil sampling methods

In 2010, 2011 and 2012, 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 aluminium 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 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 2015. 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 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. In 2011 to 2015, one blank and one standard samples 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 2015 drill programs. Prior to 2015, all logging data were recorded as a hardcopy during the day and transcribed to digital format during the evenings. 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. In 2014 and 2015 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. In 2012, 2014 and 2015 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.

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. In 2015, 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 2,198 density measurements were taken over the course of four drill programs 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:

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

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:

Density = weight in air ÷ (Final Volume – Initial Volume)

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

Specific Gravity = weight in air ÷ (weight in air – 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 rock that 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 in 2014 and 2015. 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 2015, drill core was logged and sawn or split at a processing facility on the Property. Chip samples taken between 2010 and 2015 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 2015, 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 2015, 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 fibreglass 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 are individually certified to standards within ISO 9001:2008.

All 2010 to 2012 soil samples were dried and screened to -180 microns. All 2010 to 2014 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. In 2014 and 2015, 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 then 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).

In 2012, 2014 and 2015, 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 in 2011 and 2012, 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 Certified Reference Materials (CRMs), blanks and duplicates into each batch. In the 2010 and 2011 drill programs, commercially available assay standard samples for gold and silver were purchased. Six project specific assay standards were prepared from coarse reject material from the 2011 and 2012 core samples for use during the 2012 through 2015 programs. These assay standards were prepared, homogenized and packaged by CDN Resource Laboratories Ltd. of Delta, British Columbia. All assay standards were certified by Smee & Associates Consulting Ltd. of North Vancouver, British Columbia.

In 2015, batches comprised 30 samples. Two standards and blanks were inserted into the sample sequence in each batch. Standards 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. One sample in each batch was selected at random and a duplicate coarse reject sample was created from the original coarse reject material and analyzed at the same time as the rest of the batch. Prior to 2012, coarse reject duplicates were not included in sample batches.

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	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
Total	**34,447**	**28,891**	**1,924**	**1,954**	**927**	**751**

Note: No drilling was conducted in 2013. Coarse reject duplicates, as defined in this report, are a second pulp prepared from the same coarse reject material as the main sample.

Results from the QA/QC program are reviewed immediately upon receipt. Over time as data were accumulated, results are reviewed to identify potential biases and other issues.

Samples from visually well-mineralized intervals and the adjacent samples were placed in different batches than those from weakly or un- mineralized intervals. A second coarse reject duplicate was inserted into the sample sequence within these batches in place of the quarter-core duplicate.

Rockhaven's 2015 quality assurance/quality control (QA/QC) program comprised 4,998 samples, including 277 CRMs, 277 blanks, 132 quarter-core duplicates and 146 coarse reject duplicates.

In 2015, 140 pulp samples were randomly selected from the recent exploration program and submitted to SGS Minerals Services (SGS) for re-analysis. These samples were selected using a random number generator in Microsoft Excel.

Below are examples of QA/QC results from the 2015 field program.

11.5.1 Assay Results of Certified Reference Materials

A total of four different CRMs were used during Rockhaven's 2015 program. The CRMs were prepared by CDN Laboratories from coarse reject material taken from Rockhaven's 2012 drill program. One higher-grade CRM was obtained from CDN laboratories. The CRMs contain standard amounts 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, most of the following tables and figures focus on the gold assays.

Table 11.2 shows the recommended values for the CRMs.

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
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

It is good industry practice to have a CRM at the average grade of the deposit. This is absent from the CRMs being used. The current CRMs reflect a time when the deposit was being considered in a low-grade, bulk-tonnage scenario and were appropriate for this objective at the time. AMC recommends that a CRM reflecting the average grade of the Mineral Resource estimate also be included in the QA/QC program.

Table 11.3 shows a summary of the CRM assay results from the 2015 program.

Table 11.3 2015 Assay results of Certified Reference Materials for gold

CRM	Expected Au Value (g/t)	No. of Assays	Warnings	Fails	Sample Switch	True Fails
KL4	0.355	118	14	3	1	2
KL5	0.88	117	13	3	0	3
KL6	1.72	35	2	0	0	0
CDN-ME-1402	13.9	7	0	0	0	0
TOTAL	**-**	**277**	**29**	**6**	**1**	**5**

Note: A sample switch is when the geologist has inserted a CRM into the sample bag, different from what he recorded on the sample sheet.

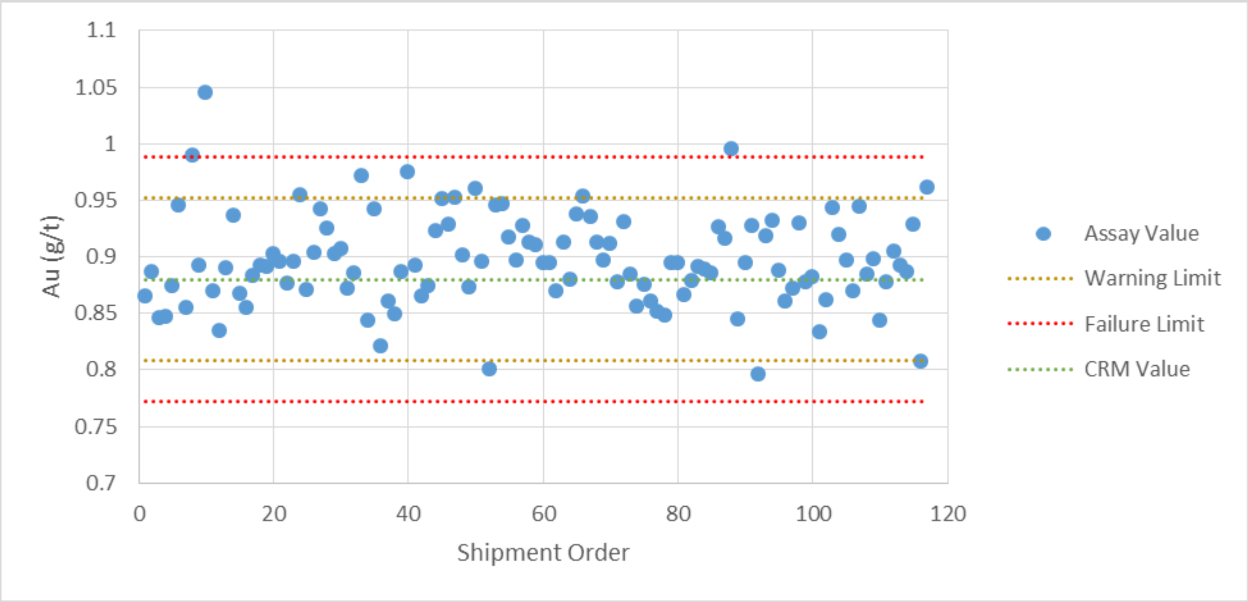
Figures 11.1 and 11.2 show the results for selected CRMs.

A CRM fails when the assay value is outside three standard deviations of the mean. When a CRM assays fails, or there are two consecutive warnings, the assay batch is re-run.

Figure 11.1 Control chart for 2015 KL4

Note: Warning limit is outside two standard deviations. Failure limit is outside three standard deviations.

Figure 11.2 Control chart for 2015 KL5



Note: Warning limit is outside two standard deviations. Failure limit is outside three standard deviations.

11.5.2 Assay results of blank samples

Coarse blanks test for contamination during both the sample preparation and assay process. Two blanks are inserted 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 277 blank samples were inserted into the sample sequence during Rockhaven's 2015 program.

A warning was generated if a blank returned greater than 10 ppb Au, two times the detection limit (5 ppb Au). All warnings were investigated. New pulps were prepared from coarse rejects for any that were deemed to be failures. A total of five warnings were generated in 2015, two of which were deemed to be failures. When a blank fails, new pulps are prepared for the whole batch and assayed.

As 98.2% 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

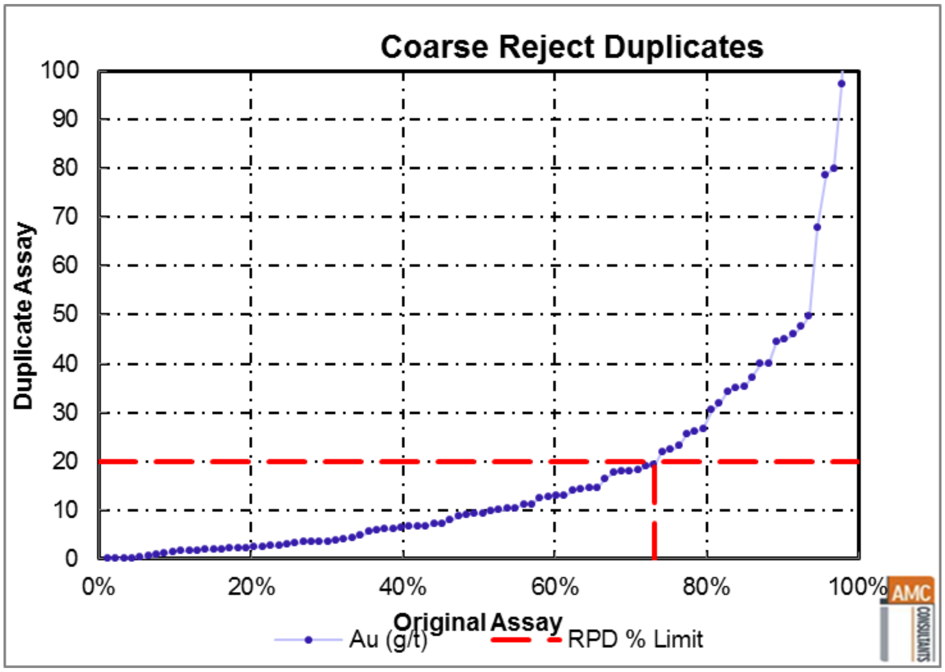
In AMC's opinion, duplicates should constitute around 5% of the samples submitted to the laboratory. Unmineralized samples should not be sent as duplicates because assays near the detection limit are commonly inaccurate. Duplicate data can be viewed on a scatterplot but should also be compared using the relative paired difference (RPD) plot. This method measures the absolute difference between a sample and it's 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 2012 to 2015, Rockhaven selected 751 coarse reject duplicates and from 2010 to 2015, 927 quarter-core (field) duplicates to test for repeatability. These samples were not based solely on gold assay results. As a result, only 93 coarse reject duplicates and 38 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:

- 73% 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; and
- Given the limited number of samples (<100) that were > 15 x detection limit for the coarse duplicates, no conclusions can be drawn from the data.

Figure 11.3 Relative paired difference plot for coarse reject duplicates

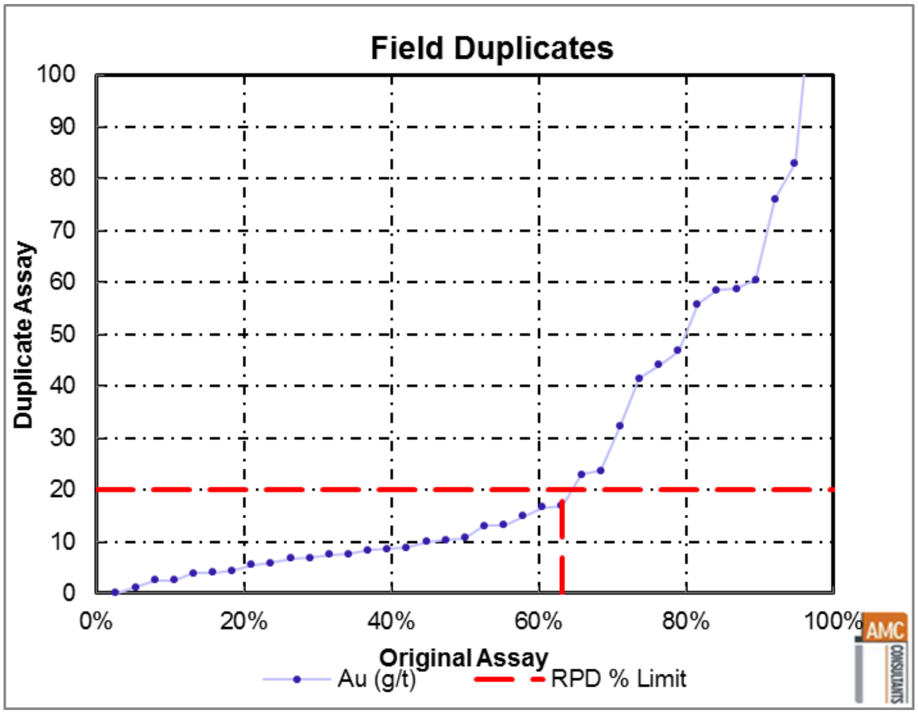


RPD plots are presented in Figure 11.4 for the field duplicate datasets.

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

- 63% 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; and
- 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



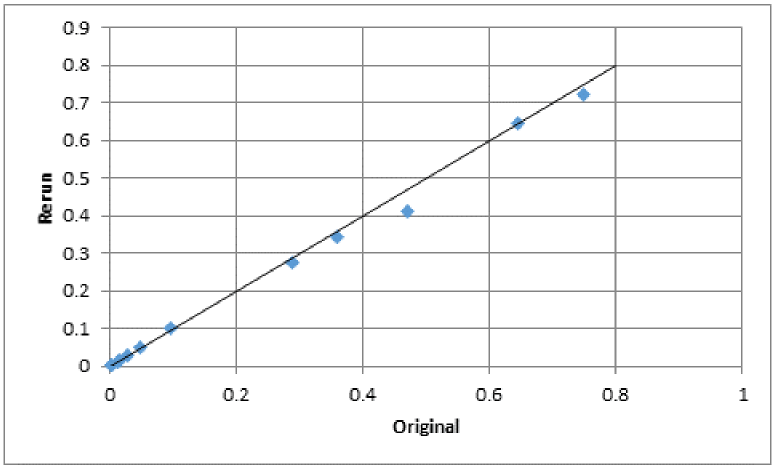
AMC recommends that going forward, duplicate samples be taken only from mineralized material.

11.5.4 Results of external check 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



11.6 Data validation

M.R. Dumala, P.Eng. of Archer Cathro, has supervised the exploration programs at the Property from 2013 through 2015. 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 him 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.

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). In 2015, drill logs were entered directly into the Database. All of the pre-2014 data were 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.

Visual comparisons of hardcopy data and digital data were conducted on selected holes to ensure accuracy. Any discrepancies identified by this process were investigated, by examining the core stored on the Property, and corrected.

11.6.2 Collar location verification

Early drillhole collars were all re-surveyed in 2012 using a Trimble RTK GPS system and, where necessary, survey data collected in previous years was corrected. Differences between the 2012 surveys and the earlier surveys are explained by the lesser accuracy of the hand held GPS devices used in previous years. The RTK GPS system was also used to survey the collars for all 2014 and 2015 drillholes.

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 2015 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 completed between 2010 and 2015, 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 are rigorous and meet or exceed 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, the 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

Dr. Ross reviewed the processes used in the data collection and handling in 2015.

Under the supervision of Dr. Ross, Mary Alejo, P.Eng of AMC undertook random cross-checks of over 5% of the assay results in the database with original assay results on the assay certificates returned from ALS Laboratories. Of the 140 batches of samples sent to ALS during the 2015 field season, 10 batches were randomly selected. This verification consisted of comparing 360 of the 4,998 assays for the 2015 drilling on the Property (7%). The original ALS certificates as well as an export from the Archer Cathro database were provided by Archer Cathro. AMC recognizes the small risk in having an export of the database sent to AMC by Archer Cathro but believes the risk is minimal based on what was observed on site.

No errors were detected in the comparison study. 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

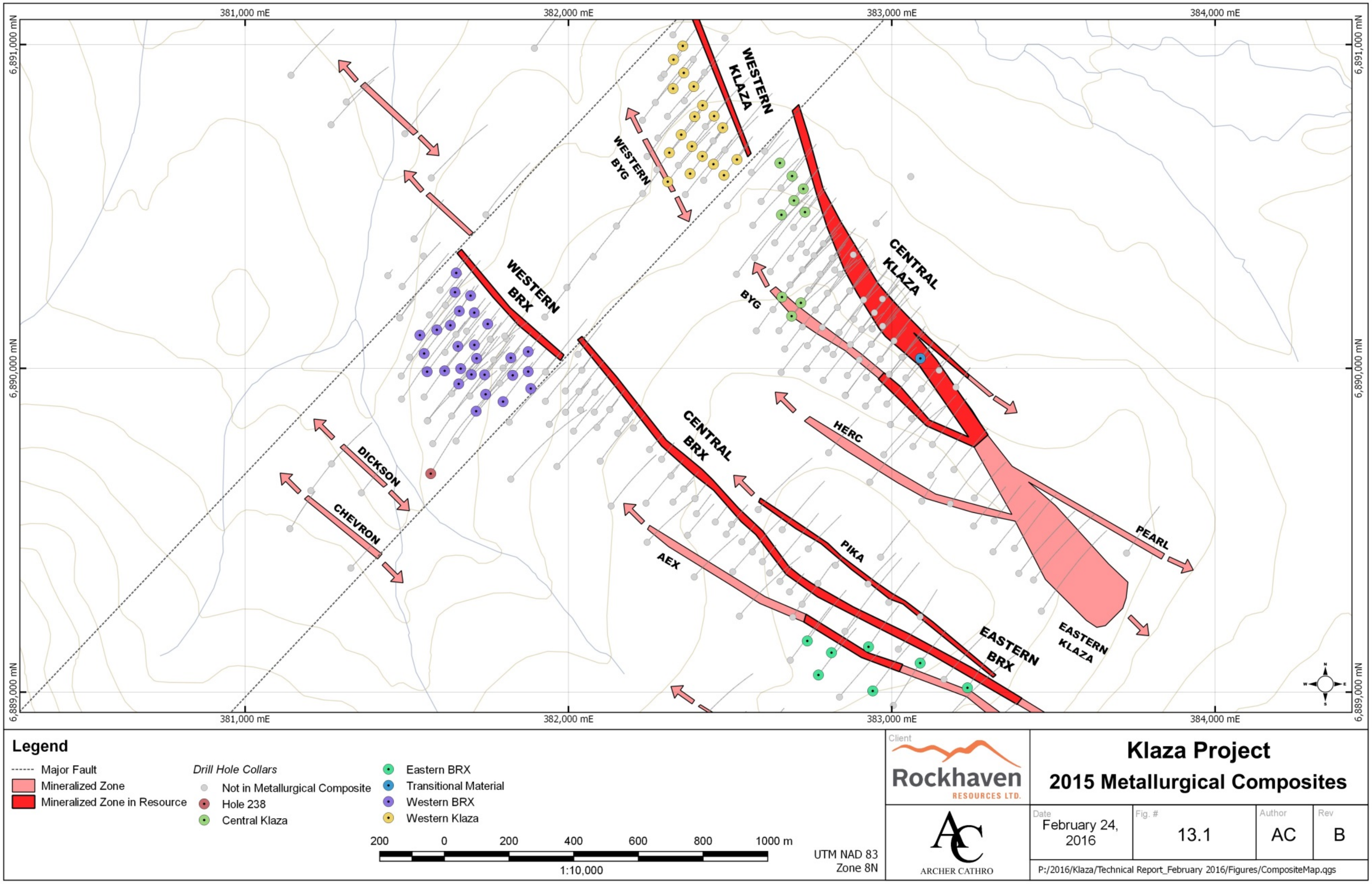
In 2014, Rockhaven contracted SGS to conduct basic scoping flotation and leaching testwork on four composites: two from the Klaza zone and two from BRX.

Subsequently, in 2015, Rockhaven contracted BCM to conduct a more in-depth metallurgical testwork program on representative samples selected by Rockhaven from a number of zones within the Klaza property. The 2015 work will be the focus of the following discussion.

13.1 Sample selection and head grades

Samples from Eastern and Western BRX, and from Central and Western Klaza zones were the subject of the majority of the testing in 2015.

Figure 13.1 Overview of the Klaza Property showing sources of samples for 2015 metallurgical testing



The mean head assays of the tested composites 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 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
Eastern BRX	0.20	0.21	4.00	51	0.09
Project-wide	0.79	1.28	5.35	111	1.00

13.2 Mineralogical analysis

Subsamples of the Western and Central Klaza and Western BRX Variability Composites were submitted to Process Mineralogical Consultants (PMC) in Maple Ridge, British Columbia. Each sample was ground to a p80 of ~100 µm and sized to produce a +53 µm and -53 µm fraction. The samples were analysed via quantitative scanning electron microscopy (TIMA) to determine mineral abundance, liberation and grain size.

The modal abundance of the three samples is summarized below.

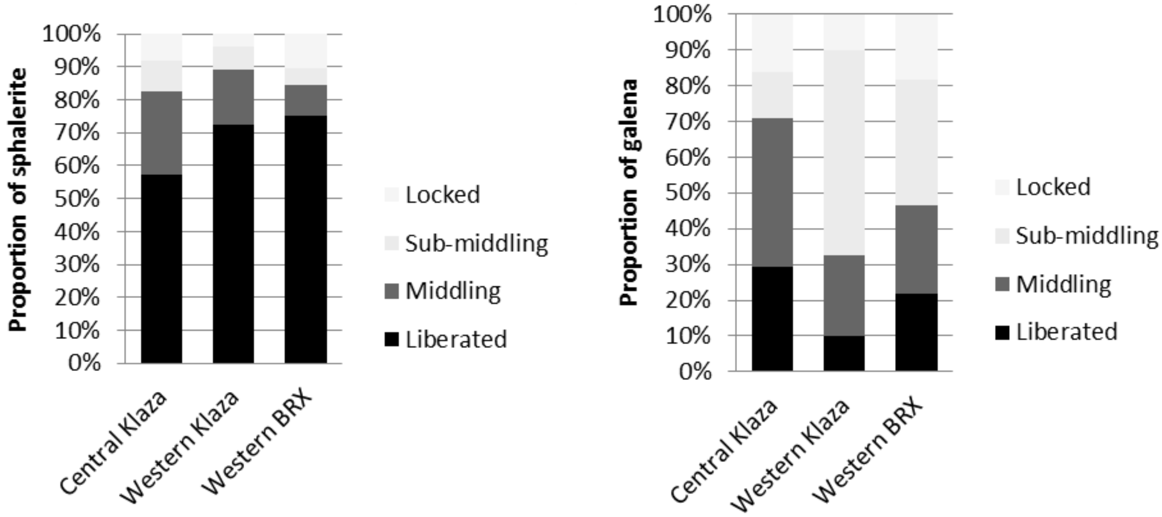
Table 13.2 Summary of modal abundance for Central Klaza, Western Klaza and Western BRX

	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, and there is no evidence of the presence of preg-robbing carbonaceous matter.

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

Figure 13.2 Sphalerite and Galena Liberation characteristics for Central Klaza, Western Klaza and Western BRX

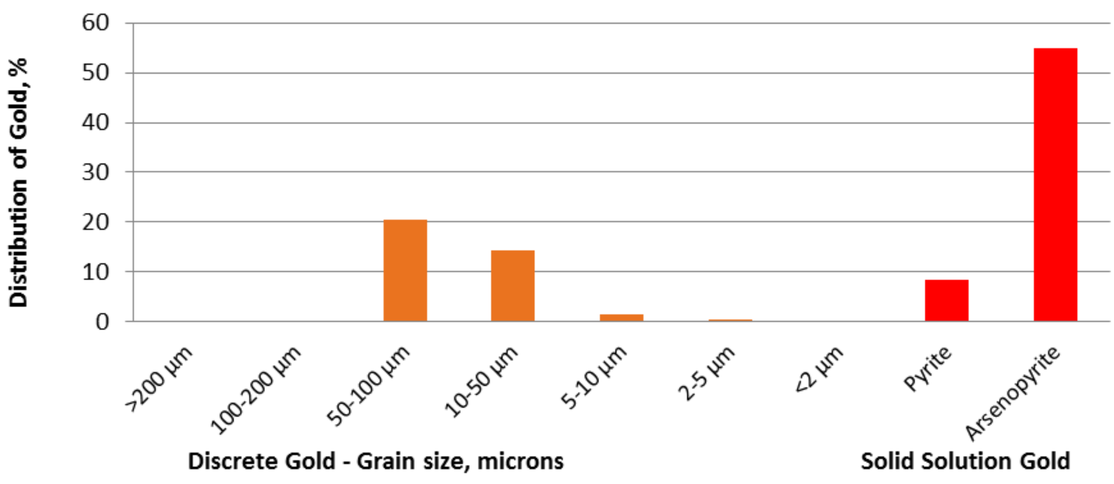


Note: Liberated: >80% free, Middling: 50-80% free, Sub-middling: 20-50% free, Locked: <20% free

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 13.3 Gold occurrence in the Project Wide Composite (orange: discrete; red: solid solution)



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

13.3 Comminution testwork

A single "grindability" composite comprised of material from the Klaza and BRX zones, yielded the following indices. The material has moderate resistance to grinding either by SAG or ball milling:

- Bond Ball Mill Work Index (BWi): 16.4 kWh/tonne
- Bond Rod Mill Work Index (RWi): 15.3 kWh/tonne
- SAG Mill Comminution (SMC) Test: A: 68.1, b: 0.71, Axb: 48.4, t_a: 0.45

13.4 Flotation flowsheet development

The deposit contains potentially economic quantities of gold, silver, lead and zinc, with roughly 72% of the in-situ value contained in gold and 16% in silver[1]. 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, flowsheet development at BCM produced saleable products containing all the economic metals as these not only facilitated the recovery of value contained in the lead and zinc (roughly 12% of the total contained value in the deposit), but also a means of generating strong revenue streams from most of the precious metals. A flowsheet comprising sequential lead, zinc, and arsenopyrite flotation was developed, with the aim of creating saleable lead and zinc concentrates, and a gold-bearing arsenopyrite concentrate that may be processed economically on site or sold.

The flowsheet was developed through a program comprising 65 batch flotation tests and four bottle roll leaches. The developed flowsheet, while moderately complex, uses exclusively conventional and widely used processes. The metallurgical program itself was conventional in nature for such projects, using testwork to identify and tune (at a PEA level) 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, as expected given the relatively straightforward mineralogy.

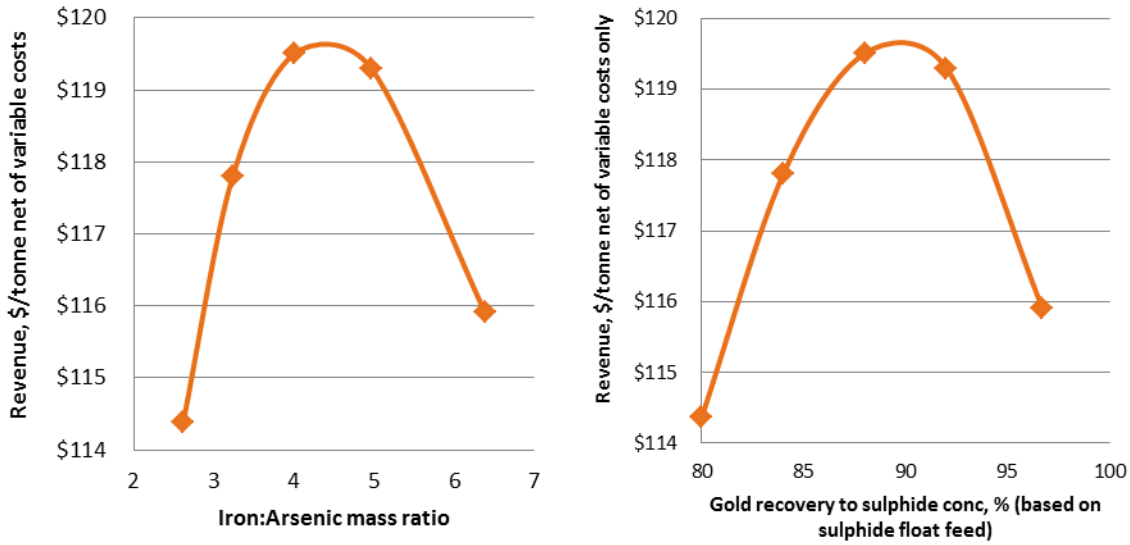
Much of the latter part of the program focused on developing a process to produce a refractory gold-bearing sulphide concentrate that would maximize the financial return from either sale to a third party or on-site processing.

In parallel with the early lead-zinc flotation work, a greater understanding of the deportment of the gold was developed through mineralogical studies, as described in an earlier sub-section of this report. The recognition that the vast majority of the gold was tied up in arsenopyrite was a key finding that has been exploited in process development. Arsenopyrite flotation behaves somewhat similarly to sphalerite at high pH levels. It is responsive to flotation following copper activation, however typically much more copper sulphate is needed to float arsenopyrite than sphalerite (which allows the prior selective flotation of the zinc sulphide). In this project, arsenopyrite has been floated selectively from pyrite at pH levels of 11.4 or higher, with high doses of copper sulphate (~500 g/t) and starvation doses of collector. The 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. The variable cost component of pre-oxidation and cyanidation (per tonne of mill feed) was modelled as a function of the recovery of arsenopyrite and pyrite. Complete flotation of both minerals yielded the highest gold recoveries, but directed large quantities of sulphur to downstream processing and this significantly increased costs. Floating very little pyrite led to a lower capital and operating cost, but adversely affected gold recoveries.

Accordingly, the economics of the process form a bell-curve, shown below (based on preliminary cost information developed at the time of the test program), as a function of gold recovery and the iron:arsenic mass ratio. The curve peaks at an iron to arsenic mass ratio close to 4.0 and a gold recovery (based on arsenopyrite flotation feed) of 91%.

[1] In-situ values based on resource average grades and metal prices as used in the PEA study

Figure 13.4 Economics of Arsenopyrite: Pyrite flotation selectivity



13.5 Locked cycle testing

The process as developed in the flowsheet development program, was confirmed through testing in locked cycle mode on the Project-wide Composite. Two tests were run. The first test was conducted mid-way through the flowsheet development exercise. The second was conducted at the end of the program, and has been used as a basis to both predict overall mine life[2] flotation metallurgy and assign the process for the PEA study and is the focus of this section of the report.

In this test, the lead flotation process employed 570 g/t zinc sulphate and 105 g/t sodium cyanide as zinc, pyrite and arsenopyrite depressants, distributed roughly 80%:20% between the primary mill and the lead regrind mill. The primary grind was 80% passing 70 microns, while the lead concentrate regrind size was 80% passing 28 microns. The pH was maintained at 8.5 throughout the lead flotation circuit (roughers and cleaners). Collectors included Cytec's dithiophosphinate Aero 3418A and the phosphine collector Aero 241, and were dosed at 17.5 g/t and 34 g/t respectively. MIBC was used as the frother throughout.

Zinc flotation was achieved using 82 g/t copper sulphate as an activator, the thionocarbamate Aero 5100 (14.5 g/t) and the mid-chain length sodium isopropyl xanthate (SIPX) (19 g/t) as collectors, and at a pH of 11 in the zinc roughers and 11.5 in the zinc cleaners, with lime used for pH adjustment. Once again, MIBC was used as the frother.

Arsenopyrite flotation employed copper sulphate added at 500 g/t to the zinc tails, at pH 11.6, while 10 g/t of SIPX was used to float the arsenopyrite with some of the pyrite. The conditions are shown in Table 13.3.

[2] The project-wide composite used for locked cycle testing excluded material from Eastern BRX.

Table 13.3 Conditions used in locked cycle testing the project-wide composite

Lead Circuit	**Reagents (g/tonne)**						**Time, minutes**			**pH**		**ORP (mV)**
Stage	**Lime**	**NaCN**	**$ZnSO_4$**	**3418A**	**Aero 241**	**MIBC**	**Grind**	**Cond.**	**Froth**	**Start**	**End**	**End**
Primary Grind	125	75	450				29.0				8.0	-161
Conditioning								3		8.0	8.6	-180
Lead Rougher				12.5	30	10		1	12	8.6	8.4	-60
Lead Regrind (Ceramic)	15	30	120				4.0				7.7	37
Lead Cleaner 1	5			3	2	5		2	6	8.6	8.2	50
Lead Cleaner 2	5			1	1	5		1	3	8.7	8.1	55
Lead Cleaner 3	5			1	1	5		1	3	8.7	8.2	49
Lead Cleaner 4	5					5		1	2	8.7	8.1	39
Zinc Circuit	**Reagents (g/tonne)**						**Time, minutes**			**pH**		**ORP (mV)**
Stage	**Lime**	**$CuSO_4$**	**Aero 5100**	**SIPX**		**MIBC**	**Grind**	**Cond.**	**Froth**	**Start**	**End**	**End**
Conditioning 1								3		8.2	11.0	-63
Conditioning 2		75						5		11.0	10.9	-59
Zinc Rougher			7.5	15		3		1	12	11.0	9.9	5
Zinc Regrind (Ceramic)	300	7.5					4.0				11.4	-19
Zinc Cleaner 1	140		5	2		3		1	5	11.8	11.9	-61
Zinc Cleaner 2	78		2	2		3		1	3	11.9	12.0	-64
Arsenopyrite Circuit	**Reagents (g/tonne)**						**Time, minutes**			**pH**		**ORP (mV)**
Stage	**Lime**	**$CuSO_4$**		**SIPX**		**MIBC**	**Grind**	**Cond.**	**Froth**	**Start**	**End**	**End**
AsPy Conditioner 1	903							3		10.3	11.6	15
AsPy Conditioner 2	0	500						5		11.6	11.6	-25
AsPy Rougher 1-1	193			4		3		1	5	11.6	11.5	-13
AsPy Rougher 1-2	163			3				1	5	11.6	11.5	-10
AsPy Rougher 1-3	185			3				1	5	11.6	11.5	-13

The metallurgical performance achieved under stable conditions is shown below.

Table 13.4 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**	**5957**	**129.9**	**3.6**	**19.4**
Zinc Cleaner 2 Conc.	89	2.2	**2.0**	**48.0**	**9.0**	**1318**	**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**

The quality of the lead and zinc concentrates is shown below. The lead concentrate is relatively high grade and should be quite attractive to many smelters. The arsenic content, however, is quite high and may be an issue with some buyers. Furthermore the grade of gold, though attractive, may limit the choice of smelters that can provide good returns on the precious metals. However, it should be noted that the inclusion of an intensive leach stage on the lead concentrate reduces the grade of gold in the saleable product to roughly 21 g/t. Antimony is also quite high and will likely trigger a penalty payment, but should not significantly affect marketability. The mercury content may also trigger a penalty, but again should not affect marketability.

The zinc concentrate is relatively low-grade, but still saleable at 48% zinc. This is partially due to the high iron content in the sphalerite. The iron and cadmium grades are both moderately high.

The base case assumption for the PEA is for the arsenopyrite concentrate to be treated on-site and as such its marketability is not an issue.

Table 13.5 Quality of concentrates

Analysis Method	Au ppm FA	Ag ppm AR-MS	Pb % 4A-AAS	Zn % 4A-AAS	As % 4A-AAS	S % LECO	Fe % 4A-AAS	Cu ppm AR-MS	Hg ppb 1G	Cd ppm AR-MS	Sb ppm AR-MS	Bi ppm AR-MS
AsPy Conc.	30.7	73	0.3	1.0	6.7	33.4	35.0	838	1,250	116	241	22
Zinc Conc.	13.5	1,318	2.0	48.0	1.0	30.7	9.0	9,380	25,100	6,000	3,160	82.5
Lead Conc.	129.5*	5,957*	59.8	3.1	3.6	19.4	9.3	7,420	19,800	417	9,800	1,140

Analysis Method	SiO2 % FUS-ICP	Al2O3 % FUS-ICP	Fe2O3(T) % FUS-ICP	MnO % FUS-ICP	MgO % FUS-ICP	CaO % FUS-ICP	Na2O % FUS-ICP	K2O % FUS-ICP	TiO2 % FUS-ICP	P2O5 % FUS-ICP	LOI % FUS-ICP	Total % FUS-ICP
AsPy Conc.	12.15	2.7	43.35	0.262	0.3	0.75	0.02	0.68	0.145	0.02	27.76	88.14
Zinc Conc.	1.24	0.38	5.09	0.153	0.07	0.2	< 0.01	0.02	0.009	< 0.01	14.83	22
Lead Conc.	1	0.08	12.4	0.153	0.08	0.17	< 0.01	0.01	0.013	0.01	12.16	26.08

Analysis Method	Ba ppm FUS-ICP	Cl % INAA	F % FUS-ISE	Sc ppm FUS-ICP	Zr ppm FUS-ICP	Be ppm FUS-ICP	V ppm FUS-ICP	Te ppm AR-MS	W ppm AR-MS	Ti % AR-MS	B ppm AR-MS	U ppm AR-MS
AsPy Conc.	557	0.03	0.01	3	18	< 1	23	1.3	0.9	0.001	< 1	5.2
Zinc Conc.	379	0.02	< 0.01	< 1	3	< 1	< 5	1.51	0.9	< 0.001	< 1	1.6
Lead Conc.	58	< 0.01	< 0.01	< 1	3	< 1	< 5	11.4	1.1	< 0.001	< 1	1.1

Analysis Method	Br ppm INAA	Ca % AR-MS	Ce ppm AR-MS	Cr ppm AR-MS	Co ppm AR-MS	Cs ppm AR-MS	Dy ppm AR-MS	Er ppm AR-MS	Eu ppm AR-MS	Ga ppm AR-MS	Gd ppm AR-MS	Ge Ppm AR-MS
AsPy Conc.	< 0.5	0.52	5.81	14	30.4	2.34	0.4	0.3	0.1	0.79	0.5	0.1
Zinc Conc.	< 0.5	0.28	2.21	5	3.6	0.96	0.2	< 0.1	< 0.1	17	0.2	< 0.1
Lead Conc.	< 0.5	0.1	1.44	< 1	8.9	0.06	< 0.1	< 0.1	< 0.1	0.95	0.1	0.2

Analysis Method	Hf ppm AR-MS	Ho ppm AR-MS	In ppm AR-MS	K % AR-MS	La ppm AR-MS	Li ppm AR-MS	Lu ppm AR-MS	Mg % AR-MS	Mo ppm AR-MS	Mn ppm AR-MS	Na % AR-MS	Nb Ppm AR-MS
AsPy Conc.	< 0.1	< 0.1	1.54	0.06	2.5	0.5	< 0.1	0.13	30.4	2,000	0.013	< 0.1
Zinc Conc.	< 0.1	< 0.1	52.5	0.03	< 0.5	0.2	< 0.1	0.07	19.3	2,620	0.013	< 0.1
Lead Conc.	< 0.1	< 0.1	3.36	< 0.01	< 0.5	< 0.1	< 0.1	0.04	16.8	1,310	0.043	< 0.1

Analysis Method	Nd ppm AR-MS	Ni ppm AR-MS	P % AR-MS	Pr ppm AR-MS	Re ppm AR-MS	Rb ppm AR-MS	Sc ppm AR-MS	Se ppm AR-MS	Sr ppm AR-MS	Sm ppm AR-MS	Sn ppm AR-MS
AsPy Conc.	2.14	11.4	0.01	0.6	0.003	4.1	0.7	0.6	15.1	0.5	6.25
Zinc Conc.	0.57	3.5	0.002	0.2	0.002	1.9	0.3	27.4	6.2	0.2	145
Lead Conc.	0.3	2.4	0.002	< 0.1	0.004	0.2	< 0.1	4.6	1.4	< 0.1	18.4

* In practice, inclusion of an intensive leach on the lead concentrate will facilitate the conversion of much of the contained gold and silver to doré while dropping the gold grade in the concentrate ultimately sold to smelters

For the sake of the PEA, it was assumed that pressure oxidation was the pre-oxidation process of choice for the project. This assumption will need to be tested as the project advances and other processes such as bio-oxidation and Albion may warrant investigation in due course. However, with roasting eliminated for environmental and permitting reasons, pressure oxidation was chosen for the PEA study over bio-oxidation due mainly to the greater level of familiarity and confidence within North America in autoclaving. A single pressure oxidation test was run at AuTec Innovative Extractive Solutions Ltd., located in Vancouver, BC, on arsenopyrite concentrate from the first locked cycle test. The conditions and results are summarized below:

Table 13.6 Summarized pressure oxidation conditions and metallurgy

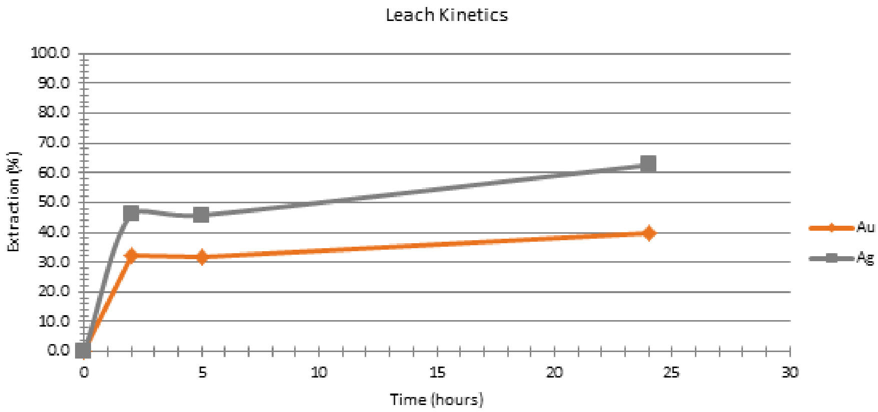
Head grades		**Conditions**		**Metallurgy**	
Ag, ppm	59.3	POX temperature, °C	220	S oxidation, %	99.8%
Au, ppm	27.3	POX % solids	9.5	CIL extraction, gold, %	98%
As, %	7.2	POX residence time, minutes	60	CIL extraction, silver, %	1.2%
Fe, %	32.4	Cyanidation % solids	30	Carbon loading, g/t	429
S_{total} %	31.9	Cyanide concentration, g/L	3		
		Leach time, hours	24		

Reagent consumption, basis POX residue			
	POX	Neutralization	CIL
Lime, kg/t	0	272	10
Cyanide, kg/t	0	10	4.2

The extraction of gold by carbon-in-leach after pre-oxidation was 98%. Lime consumption for both neutralization and leaching was 282 kg per tonne of POX feed, while cyanide consumption was 4.2 kg/tonne of POX feed. Note that no hot cure test work was conducted on the POX residue, although it is very likely that it would be used in practice (and has been built into the design of the process for this report). Hot cure can be expected to have a substantial effect on lime consumption, in part by enabling the use of potentially much cheaper limestone in place of lime. While there is no reason to believe that a hot cure will not be effective for this project, it will need to be tested in the future.

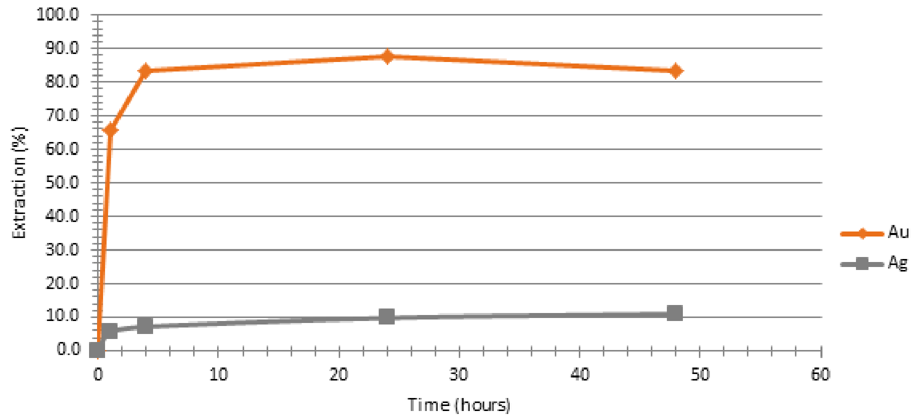
A single cyanide leach test was conducted on the sulphide flotation tails from the locked cycle test. This extracted 40% of the gold and 62% of the silver from the tails stream. Cyanide consumption was 0.8 kg/tonne of feed (arsenopyrite flotation tails) and lime requirement was 0.34 kg/tonne.

Figure 13.5 Arsenopyrite flotation tails leach kinetics



Intensive leach testing was conducted on the lead concentrate. This consisted of shaker tests followed by a single bottle roll test. The objective of the tests was to explore if the gold contained within the lead concentrate could be converted to doré on-site, and if the associated reagent consumptions would be economic. The leach achieved 84% gold extraction within four hours. Further extraction beyond this point was limited. Cyanide consumption was 4.7 kg/tonne of lead concentrate (or 1.7 kg per ounce of gold) and lime consumption was essentially zero, indicating that the process would indeed likely be economic, considering the higher payable values for gold in doré.

Figure 13.6 Extraction of gold and silver from the lead concentrate



13.6 Integrated metallurgical response to testing the project-wide composite

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 87% and 11%, respectively, accordingly the sale of doré will account for the vast majority for of the revenue from the operation.

Table 13.7 Summarized metal recoveries to doré and concentrates from testwork on the project-wide composite

	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%

13.6.1 Variability testing

The summarized results from four variability tests conducted on samples of material from Western Klaza, Central Klaza, Western BRX and Eastern BRX are shown below. These tests were conducted to examine if the flowsheet was sufficiently robust as to be able to process each of the materials in isolation.

For the most part, the samples responded well. All but Eastern BRX yielded lead cleaner concentrates assaying over 60% lead (Eastern BRX was copper-rich). The zinc grades were over 50% for all but Eastern BRX (again copper diluted the zinc concentrate). Both gold and silver recoveries were typically in the 90% range, though the split of gold to the arsenopyrite concentrate is higher for Klaza and lower for BRX.

Most of these tests were one-off tests, and were not optimized. The purpose of the tests was to ensure that each zone responded reasonably to the flowsheet, and indeed if the results achieved were comparable with those from the blended Project-wide Composite. This was indeed the case (except for the Eastern BRX zone), suggesting that the process as defined should yield good metallurgy largely irrespective of the blend of material fed into the mill.

A single sample of near-surface oxide/transition material was also tested. This also behaved quite well, and it was concluded that with further work this would be shown to respond adequately well to the flowsheet. Finally, a sample of material from Hole 238, located deep in the Western BRX zone, was tested; this high grade zone showed promise to yield good lead and zinc concentrates. Overall gold and silver recoveries were very high at 98% and 99% respectively, though the reader should be cautioned that these were only to rougher concentrates.

Table 13.8 Summarized variability test results

Batch cleaner flotation tests												
Western Klaza	Concentrate grade, %/g/t						Distribution, %					
F-65	Pb, %	Cu, %	Zn, %	Au, g/t	Ag, g/t	As, %	Pb	Cu	Zn	Au	Ag	As
Lead Cleaner Conc.	61.7	-	2.4	141	19,545	3.9	69	-	2	16	52	3
Zinc Cleaner Conc.	2.4	-	50.9	12.1	5,640	1	5	-	74	3	30	1
Arsenopyrite Conc.	0.5	-	0.5	37	167	6.7	10	-	6	72	8	85
Central Klaza	Concentrate grade, %/g/t						Distribution, %					
F-64	Pb, %	Cu, %	Zn, %	Au, g/t	Ag, g/t	As, %	Pb	Cu	Zn	Au	Ag	As
Lead Cleaner Conc.	61.5	-	3	85	3,071	3.6	81	-	2	17	47	4
Zinc Cleaner Conc.	1.3	-	50.5	10.5	739	1.6	5	-	89	6	31	4
Arsenopyrite Conc.	0.3	-	0.2	27	56	6.5	5	-	2	68	11	81
Western BRX	Concentrate grade, %/g/t						Distribution, %					
F-60	Pb, %	Cu, %	Zn, %	Au, g/t	Ag, g/t	As, %	Pb	Cu	Zn	Au	Ag	As
Lead Cleaner Conc.	66.3	-	2.2	191	4,635	1.9	82	-	2	33	60	2
Zinc Cleaner Conc.	1.9	-	51.9	22.4	1,062	0.6	3	-	76	6	20	1
Arsenopyrite Conc.	0.34	-	1	29.5	68	8.1	4	-	10	49	8	86
Eastern BRX	Concentrate grade, %/g/t						Distribution, %					
F-47	Pb, %	Cu, %	Zn, %	Au, g/t	Ag, g/t	As, %	Pb	Cu	Zn	Au	Ag	As
Lead Cleaner Conc.	8.7	19.3	3.3	271	3,443	2.1	60	41	6	51	59	19
Zinc Cleaner Conc.	0.7	9.5	27.7	26.5	486	0.6	8	34	84	8	14	9
Arsenopyrite Conc.	0.1	0.1	0	5.9	29	0.2	13	5	2	29	13	51
Rougher flotation tests												
Central Klaza (Oxide Transition)	Concentrate grade, %/g/t						Distribution, %					
F-56	Pb, %	Cu, %	Zn, %	Au, g/t	Ag, g/t	As, %	Pb	Cu	Zn	Au	Ag	As
Lead Rougher Conc.	9.3	3.7	18.7	175	947	-	53	41	14	50	51	-
Zinc Rougher Conc.	0.6	1	35	14	159	-	10	32	81	12	26	-
Arsenopyrite Conc.	0.3	0.1	0.3	23	43	-	6	4	1	27	9	-
Hole 238	Concentrate grade, %/g/t						Distribution, %					
F-10	Pb, %	Cu, %	Zn, %	Au, g/t	Ag, g/t	As, %	Pb	Cu	Zn	Au	Ag	As
Lead Rougher Conc.	28.9	0.9	6.5	35	5,270	-	95	52	22	38	72	-
Zinc Rougher Conc.	0.8	0.9	23.5	43	1,827	-	3	43	77	46	24	-
Arsenopyrite Conc.	0.4	0.1	0.3	12	172	-	2	4	1	15	3	-

14 Mineral Resource estimates

14.1 Introduction

This section was taken from the NI 43-101 technical report titled "NI 43-101 Technical Report describing Updated Diamond Drilling, Metallurgical Testing and Mineral Resources on the Klaza Property Yukon, Canada", dated 9 December 2015. This section remains predominantly unchanged and is included for completeness and continuity.

The Mineral Resource for the Klaza deposit has been estimated by Dr. A. Ross. P.Geo. Principal Geologist of AMC Mining Consultants (Canada) Ltd. (AMC), who takes responsibility for the estimate.

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.

This estimate is dated 9 December 2015 and supersedes the previous estimate outlined in the "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). The details on the previous estimate are shown in Table 14.1.

Table 14.1 Previous Mineral Resource Estimate for the Klaza Property

	Mineral Resource Effective Date	QP	Company	Cut-off Date of Data
Klaza Deposit	25 November 2014	G. Giroux	Giroux Consultants Ltd	25 November 2014

Source: AMC Mining Consultants (Canada) Ltd

The data used in the 9 December 2015 estimate (AMC 2015 estimate) include results of all drilling carried out on the Property to 30 September 2015. The estimation work was carried out in Datamine™ software. Interpolation was carried out using ordinary kriging (OK) for all the domains

The results of the AMC 2015 estimate are summarized in Table 14.2 and expanded in Table 14.3.

Table 14.2 Summary of Inferred Mineral Resources as of 9 December 2015

	Tonnes (kt)	Grade					Contained Metal				
		Au (g/t)	Ag (g/t)	Pb (%)	Zn (%)	Au EQ (g/t)	Au (koz)	Ag (koz)	Pb (klb)	Zn (klb)	Au EQ (koz)
Pit-Constrained	2,366	5.12	95	0.93	1.18	6.71	389	7,190	48,258	61,475	510
Underground	7,054	4.27	87	0.69	0.88	5.65	969	19,772	107,159	136,416	1,282
Total	**9,421**	**4.48**	**89**	**0.75**	**0.95**	**5.92**	**1,358**	**26,962**	**155,417**	**197,891**	**1,793**

CIM definition standards were used for the Mineral Resource.
Using drilling results to 30 September 2015.
Near surface mineral resources are constrained by an optimized pit shell at a gold price of US$1300 oz.
Cut-off grades applied to the pit-constrained and underground resources are 1.3 g/t AuEQ and 2.75 g/t AuEQ respectively.
Gold equivalent values were calculated using the following formula: AuEQ=Au+Ag/85+Pb/3.74+Zn/5.04 and assuming: US$1300 oz Au, US$20 oz Ag, US$0.90 lb Pb and US$0.90 lb Zn with recoveries for each metal of Au: 96%, Ag: 91%, Pb: 85% and Zn: 85%.
Numbers may not add due to rounding.
All metal prices are quoted in US$ at an exchange rate of $0.80 US to $1.00 Canadian.
Mineral resources that are not mineral reserves do not have demonstrated economic viability.
Source: AMC Mining Consultants (Canada) Ltd.

Table 14.3 Inferred Mineral Resources as of 9 December 2015 by area

Zone	PC/UG	Tonnes (kt)	Au (g/t)	Ag (g/t)	Pb (%)	Zn (%)	AuEQ(g/t)	Au (koz)	Ag (koz)	Pb (klb)	Zn (klb)	AuEQ (koz)
Western BRX	Pit-Constrained	554	8.21	110	1.03	1.03	9.99	146	1,960	12,608	12,557	178
	Underground	814	7.87	147	1.49	1.68	10.34	206	3,853	26,764	30,194	271
	Total	**1,368**	**8.01**	**132**	**1.31**	**1.42**	**10.20**	**352**	**5,813**	**39,372**	**42,750**	**448**
Central BRX	Pit-Constrained	283	3.67	192	1.34	1.39	6.57	33	1,752	8,361	8,693	60
	Underground	1,027	2.65	152	1.26	1.39	5.05	87	5,019	28,561	31,506	167
	Total	**1,311**	**2.87**	**161**	**1.28**	**1.39**	**5.38**	**121**	**6,771**	**36,922**	**40,198**	**227**
Eastern BRX	Pit-Constrained	193	4.42	90	0.21	0.42	5.62	27	559	908	1,798	35
	Underground	2,213	4.07	50	0.21	0.29	4.77	289	3,568	10,296	14,230	340
	Total	**2,406**	**4.10**	**53**	**0.21**	**0.30**	**4.84**	**317**	**4,127**	**11,203**	**16,028**	**374**
Western Klaza	Pit-Constrained	81	6.86	288	1.01	0.98	10.72	18	752	1,803	1,748	28
	Underground	461	5.41	182	0.58	0.87	7.88	80	2,703	5,879	8,820	117
	Total	**542**	**5.62**	**198**	**0.64**	**0.88**	**8.30**	**98**	**3,455**	**7,682**	**10,567**	**145**
Central Klaza	Pit-Constrained	1,255	4.07	54	0.89	1.33	5.20	164	2,168	24,578	36,680	210
	Underground	2,539	3.74	57	0.64	0.92	4.76	305	4,628	35,661	51,668	389
	Total	**3,794**	**3.85**	**56**	**0.72**	**1.06**	**4.91**	**470**	**6,796**	**60,239**	**88,347**	**599**

CIM definition standards were used for the Mineral Resource.
Using drilling results to 30 September 2015.
Near surface mineral resources are constrained by an optimized pit shell at a gold price of US$1300 oz.
Cut-off grades applied to the pit-constrained and underground resources are 1.3 g/t Au EQ and 2.75 g/t Au EQ respectively.
Gold equivalent values were calculated using the following formula: Au EQ=Au+Ag/85+Pb/3.74+Zn/5.04 and assuming: US$1300 oz Au, US$20 oz Ag, US$0.90 lb Pb and US$0.90 lb Zn with recoveries for each metal of Au: 96%, Ag: 91%, Pb: 85% and Zn: 85%.
Numbers may not add due to rounding.
All metal prices are quoted in US$ at an exchange rate of $0.80 US to $1.00 Canadian.
Mineral resources that are not mineral reserves do not have demonstrated economic viability.
Source: AMC Mining Consultants (Canada) Ltd.

14.2 Data used

14.2.1 Drillhole database

The data used in the AMC 2015 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 14.4.

Table 14.4 Drillhole data used in the Mineral Resource Estimate

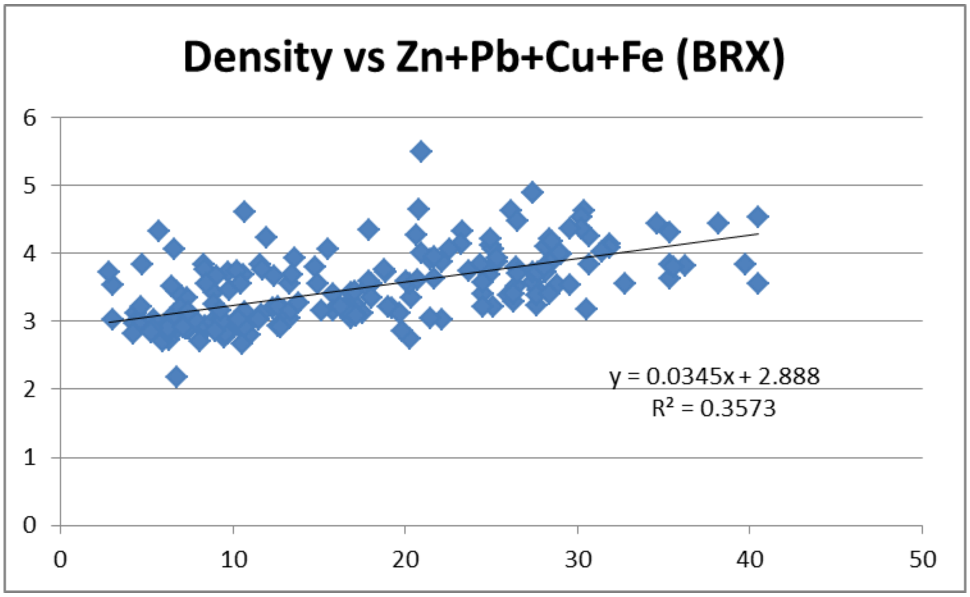
Year	No. of Drillholes	No. of Assays	Metres Drilled (m)
2010	11	746	1,642
2011	47	5,928	12,442
2012	47	7,517	14,828
2014	94	5,607	17,139
2015	49	4,092	12,904
Total	248	23,890	58,955

Note: All drillholes are surface diamond drillholes.
Source: Archer, Cathro & Associates.

14.2.2 Bulk density

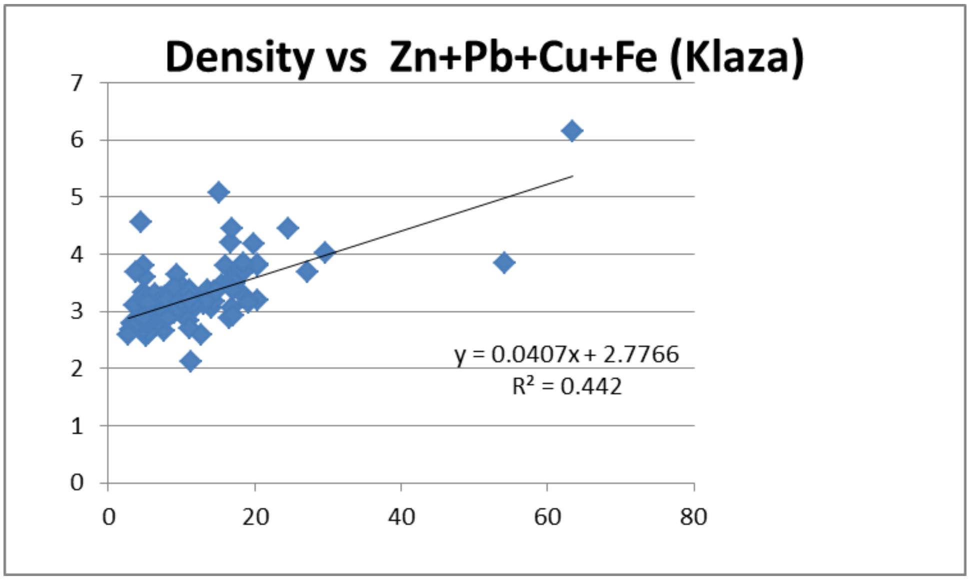
The collection of density measurements is described in Section 11.2.3. Mineralized veins usually contain significant concentrations of sulphide minerals, including pyrite and galena which have much higher specific gravities than normal rock forming minerals. Examination of correlation coefficients demonstrates a strong relationship between measured density and a sum of the base metal grades. Scatterplots of density versus combined lead, zinc, copper and iron assay values were produced for the BRX and Klaza zones as shown in Figure 14.1 and Figure 14.2.

Figure 14.1 Scatter plot of density versus percentage lead-zinc-copper-iron in the BRX zone



Source: AMC Mining Consultants (Canada) Ltd.

Figure 14.2 Scatter plot of density versus percentage lead-zinc-copper-iron in the Klaza zone



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 and the regression equations shown above. Barren rock, for the purpose of delineating a pit-constrained resource, was assigned a valued of 2.80 which is 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.

14.3 Domain modelling

The Klaza deposit consists of two main zones, the BRX and Klaza zones. These two zones are further subdivided into the Western BRX, Central BRX, Eastern BRX, Western Klaza and Central Klaza subzones (zones).

Matthew Dumala, P.Eng., of Archer, Cathro & Associates (1981) Limited, built 72 three dimensional solids to constrain the mineralization within the BRX (Western, Central, Eastern) zones and Klaza (Western and Central). Mineralized solids define the better known and higher grade continuous structures/veins. As much as possible, high-grade solids were built to capture only vein mineralization and often consist of only one or two samples in a drillhole. Dyke intersections were used as a marker to help constrain the orientation and position of mineralized structures.

The number of mineralization domains varied between the subzones. There were 17 domains at Western BRX, 13 domains at Central BRX and 9 domains at the newly modelled Eastern BRX. In the Klaza zone, Western Klaza has one domain and Central Klaza had 32 domains including 16 domains outlining the mineralized splay in central Klaza that were previously not modelled. The higher number of mineralization domains compared to the previous technical report reflects a strategy of subdividing the veins on either side of the porphyry unit and minimizing splays in a domain that can hinder the estimation process. This is summarized in Table 14.5.

Table 14.5 Domain nomenclature

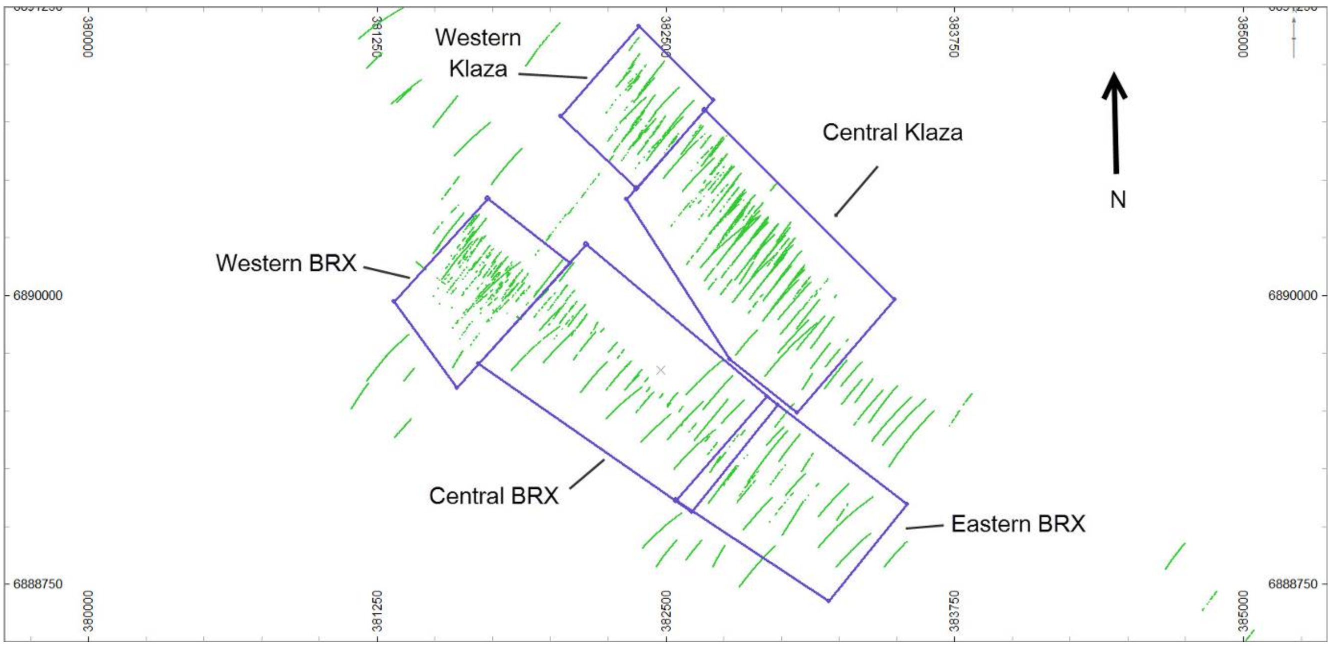
Zone	Subzones	No. of Domains
BRX	Western BRX	17
	Central BRX	13
	Eastern BRX	9
Klaza	Western Klaza	1
	Central Klaza	32

Source: AMC Mining Consultants (Canada) Ltd.

On completion of the domain modelling, visual checks were carried out to ensure that the constraining wireframes respected the raw data.

Figure 14.3 shows the location of the areas in which the domains were modelled.

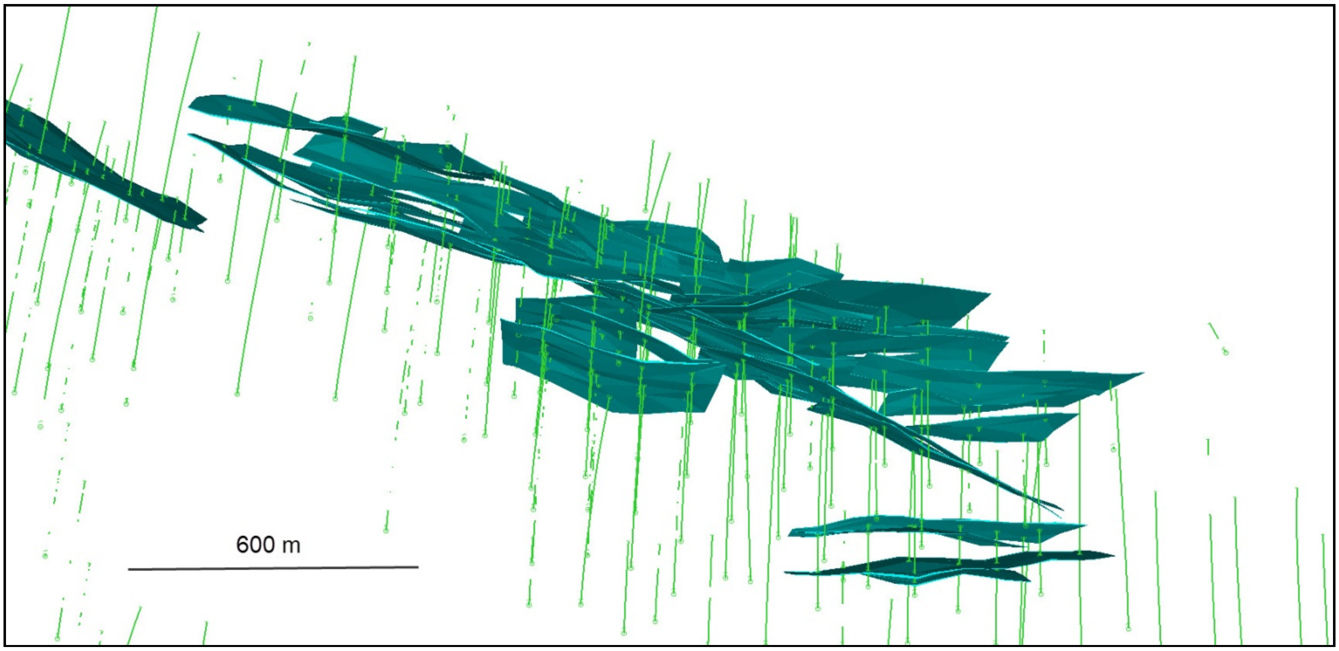
Figure 14.3 Subzone locations with drillholes



Source: AMC Mining Consultants (Canada) Ltd.

Figure 14.4 shows an isometric view of the mineralized domains in Western Klaza. It highlights the local complexity of the Klaza deposit.

Figure 14.4 Isometric view of the Central Klaza subzone looking north



Source: AMC Mining Consultants (Canada) Ltd.

14.4 Statistics and compositing

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. This length also gave the appropriate selectivity for the narrow-vein style of this mineralization. To allow for similar sample support, residual compositing intervals < 0.4 m in length were discarded.

Composited assay data for gold, silver, lead, zinc, copper, arsenic and iron were then examined on probability plots for each of the 72 domains, and outliers examined. This resulted in top cuts as shown in Table 14.6. 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 14.6 Top cutting ranges by element

Element	No. of Domains with Top Cut	% of Domains with Top Cut	Top Cut Range
Gold	26	36%	2 - 30 g/t
Silver	34	47%	12 - 1000 g/t
Lead	33	45%	0.02 - 10%
Zinc	29	40%	0.6 – 6%
Copper	30	41%	0.02 - 4%
Iron	25	34%	6 - 20%
Arsenic	28	38%	2,000 - 90,000 ppm

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

Table 14.7 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	28	28	28	28	28	28	28
Minimum	0.01	0.57	0.00	0.01	78	0.00	2.62
Maximum	68.20	725.00	11.25	8.99	98,300	1.61	26.10
Mean	**15.52**	**254.00**	**1.82**	**2.22**	**11,824**	**0.38**	**11.85**
Std Dev	17.59	251.62	2.94	2.22	19,710	0.46	7.62
Coeff Var	1.13	0.99	1.62	1.00	1.67	1.20	0.64
Composited							
Element	**Au (g/t)**	**Ag (g/t)**	**Pb (%)**	**Zn (%)**	**As (ppm)**	**Cu (%**	**Fe (%)**
Number	24	24	24	24	24	24	24
Minimum	0.291	10.596	0.047	0.075	366	0.017	3.92
Maximum	42.00	660.00	11.25	8.99	68,816	1.29	26.10
Mean	**15.17**	**242.64**	**1.89**	**2.18**	**12,943**	**0.35**	**11.50**
Std Dev	13.79	207.47	2.61	2.03	17,443.76	0.37	6.35
Coeff Var	0.91	0.86	1.38	0.93	1.35	1.08	0.55
Capped							
Element	**Au (g/t)**	**Ag (g/t)**	**Pb (%)**	**Zn (%)**	**As (ppm)**	**Cu (%**	**Fe (%)**
Number	24	24	24	24	24	24	24
Minimum	0.291	10.596	0.047	0.075	366	0.017	3.92
Maximum	42	660	11.25	5	60,000	1.285	26.1
Mean	**15.17**	**242.64**	**1.89**	**2.02**	**12,575**	**0.35**	**11.50**
Std Dev	13.79	207.47	2.61	1.56	16,269.38	0.37	6.35
Coeff Var	0.91	0.86	1.38	0.77	1.29	1.08	0.55

Notes: Std Dev = Standard Deviation; Coeff Var = Coefficient of Variation. Raw statistics are length-weighted.
Source: AMC Mining Consultants (Canada) Ltd.

Table 14.8 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	44	44	44	44	44	44	44
Minimum	0.03	1.22	0.02	0.02	82.00	0.00	2.61
Maximum	18.55	978.00	12.20	13.80	49,900.00	3.11	38.20
Mean	**2.75**	**181.39**	**1.57**	**1.63**	**6,800.66**	**0.24**	**11.17**
Std Dev	3.30	162.16	2.02	2.25	9,000.34	0.37	7.25
Coeff Var	1.20	0.89	1.29	1.38	1.32	1.56	0.65
Composited							
Element	**Au (g/t)**	**Ag (g/t)**	**Pb (%)**	**Zn (%)**	**As (ppm)**	**Cu (%**	**Fe (%)**
Number	47	47	47	47	47	47	47
Minimum	0.17	19.5	0.042	0.047	423	0.01	3.05
Maximum	13.2	978	10.1	12.323	44,629.19	1.59	29.94
Mean	**2.94**	**182.55**	**1.52**	**1.66**	**7,295.78**	**0.25**	**11.10**
Std Dev	3.03	152.30	1.67	2.14	8230.53	0.33	6.54
Coeff Var	1.03	0.83	1.10	1.29	1.13	1.36	0.59
Capped							
Element	**Au (g/t)**	**Ag (g/t)**	**Pb (%)**	**Zn (%)**	**As (ppm)**	**Cu (%**	**Fe (%)**
Number	47	47	47	47	47	47	47
Minimum	0.17	19.5	0.04	0.05	423	0.01	3.05
Maximum	9	400	4	12.32	20,000.00	1.59	29.94
Mean	**2.77**	**170.25**	**1.39**	**1.66**	**6,516.44**	**0.25**	**11.10**
Std Dev	2.53	101.56	1.14	2.14	5,529.14	0.33	6.539
Coeff Var	0.92	0.60	0.82	1.287	0.848	1.36	0.589

Notes: Std Dev = Standard Deviation; Coeff Var = Coefficient of Variation. Raw statistics are length-weighted.
Source: AMC Mining Consultants (Canada) Ltd.

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

Raw							
Element	**Au (g/t)**	**Ag (g/t)**	**Pb (%)**	**Zn (%)**	**As (ppm)**	**Cu (%**	**Fe (%)**
Number	21	21	21	21	21	21	21
Minimum	0.67	0.71	0.00	0.01	30.60	0.00	4.50
Maximum	11.55	396.00	1.77	3.28	28,900.00	2.76	20.30
Mean	**3.25**	**41.80**	**0.12**	**0.26**	**1,391.21**	**0.26**	**9.76**
Std Dev	3.13	79.54	0.33	0.63	3,875.15	0.58	5.02
Coeff Var	0.96	1.90	2.77	2.46	2.79	2.26	0.51
Composited							
Element	**Au (g/t)**	**Ag (g/t)**	**Pb (%)**	**Zn (%)**	**As (ppm)**	**Cu (%**	**Fe (%)**
Number	25	25	25	25	25	25	25
Minimum	0.67	0.71	0.00	0.01	30.6	0.00	4.5
Maximum	11.55	396	1.765	3.28	28,900.00	2.76	20.3
Mean	**3.78**	**50.65**	**0.13**	**0.27**	**2,102.83**	**0.31**	**10.37**
Std Dev	3.49	85.91	0.35	0.66	5701.97	0.63	5.21
Coeff Var	0.92	1.696	2.66	2.47	2.71	2.05	0.50
Capped							
Element	**Au (g/t)**	**Ag (g/t)**	**Pb (%)**	**Zn (%)**	**As (ppm)**	**Cu (%**	**Fe (%)**
Number	25	25	25	25	25	25	25
Minimum	0.673	0.71	0.00	0.01	30.6	0.00	4.5
Maximum	11.55	200	0.3	0.6	5,000.00	1	20.3
Mean	**3.78**	**42.81**	**0.07**	**0.16**	**1,142.03**	**0.21**	**10.37**
Std Dev	3.49	57.24	0.08	0.20	1396.67	0.31	5.21
Coeff Var	0.92	1.337	1.13	1.3	1.223	1.50	0.50

Notes: Std Dev = Standard Deviation; Coeff Var = Coefficient of Variation. Raw statistics are length-weighted.
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	80	80	80	80	80	80	80
Minimum	0.03	0.95	0.00	0.01	91.80	0.00	1.69
Maximum	49.50	1,890.00	6.06	9.16	64,900.00	0.42	13.9
Mean	**5.41**	**239.31**	**0.70**	**0.94**	**7,966.53**	**0.04**	**4.95**
Std Dev	7.36	389.16	1.14	1.41	12,271.92	0.07	2.13
Coeff Var	1.36	1.63	1.62	1.50	1.54	1.93	0.43
Composited							
Element	**Au (g/t)**	**Ag (g/t)**	**Pb (%)**	**Zn (%)**	**As (ppm)**	**Cu (%**	**Fe (%)**
Number	66	66	66	66	66	66	66
Minimum	0.20	1.74	0.00	0.01	159.26	0.00	2.56
Maximum	30.9	1560	5.55	4.76	55,400	0.42	13.9
Mean	**5.47**	**216.14**	**0.64**	**0.91**	**7,962.44**	**0.04**	**5.00**
Std Dev	6.15	304.26	0.96	1.03	10,006.12	0.07	2.06
Coeff Var	1.13	1.41	1.51	1.13	1.257	1.77	0.41
Capped							
Element	**Au (g/t)**	**Ag (g/t)**	**Pb (%)**	**Zn (%)**	**As (ppm)**	**Cu (%**	**Fe (%)**
Number	66	66	66	66	66	66	66
Minimum	0.20	1.74	0.00	0.01	159.26	0.00	2.56
Maximum	30.9	1000	5.55	4.76	30,000	0.42	8
Mean	**5.47**	**204.03**	**0.64**	**0.91**	**7,410.33**	**0.04**	**4.82**
Std Dev	6.15	259.70	0.96	1.03	7,956.4	0.07	1.51
Coeff Var	1.13	1.27	1.51	1.13	1.074	1.77	0.31

Notes: Std Dev = Standard Deviation; Coeff Var = Coefficient of Variation. Raw statistics are length-weighted.
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	72	72	72	72	72	72	72
Minimum	0.02	0.55	0.03	0.02	43.2	0.00	2.98
Maximum	46.8	981	15.85	33.81	41500	0.969	17.25
Mean	**4.44**	**66.21**	**0.80**	**1.53**	**4908.02**	**0.10**	**6.09**
Std Dev	6.05	115.07	1.54	3.65	6435.72	0.16	2.89
Coeff Var	1.36	1.74	1.94	2.39	1.31	1.66	0.47
Composited							
Element	**Au (g/t)**	**Ag (g/t)**	**Pb (%)**	**Zn (%)**	**As (ppm)**	**Cu (%**	**Fe (%)**
Number	81	81	81	81	81	81	81
Minimum	0.49	2.03	0.03	0.03	90.8	0.00	3.06
Maximum	31.8	510	7.59	13.52	40256.30	0.58	14.45
Mean	**4.4**	**62.51**	**0.77**	**1.34**	**4920.54**	**0.09**	**6.07**
Std Dev	5.51	94.35	1.36	2.51	6276.70	0.14	2.64
Coeff Var	1.25	1.51	1.77	1.87	1.28	1.52	0.44
Capped							
Element	**Au (g/t)**	**Ag (g/t)**	**Pb (%)**	**Zn (%)**	**As (ppm)**	**Cu (%**	**Fe (%)**
Number	81	81	81	81	81	81	81
Minimum	0.49	2.03	0.03	0.03	90.8	0.00	3.06
Maximum	12	300	4	13.52	40256.30	0.58	14.45
Mean	**3.82**	**57.56**	**0.68**	**1.34**	**4920.54**	**0.09**	**6.07**
Std Dev	3.72	74.39	1.01	2.51	6276.70	0.14	2.64
Coeff Var	0.97	1.29	1.48	1.87	1.28	1.52	0.44

Notes: Std Dev = Standard Deviation; Coeff Var = Coefficient of Variation. Raw statistics are length-weighted.
Source: AMC Mining Consultants (Canada) Ltd.

14.5 Block model

14.5.1 Block model parameters

The parent block size for the model was 10 m by 10 m by 10 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 dimensions are shown in Table 14.12.

Table 14.12 Block model parameters

Parameter	**X**	**Y**	**Z**
Origin (m)	381,530	6,888,850	795
Rotation Angle (deg.)	0	0	0
No. of Blocks	188	224	68

Source: AMC Mining Consultants (Canada) Ltd.

14.5.2 Variography and grade estimation

Variography was carried out on the entire data set 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. To account for over estimation of gold in the main Central Klaza domain (Domain 7101.11), a more restricted search radius was used.

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 = 3 x search distance

The search distances of the first pass are shown in Table 14.13 along with the minimum and maximum number of samples used for each pass. The orientation of the search ellipse was 40° counter clockwise in the Z direction and 30° counter clockwise in the Y direction. There was no rotation of the search ellipse along the X axis.

Table 14.13 Minimum and maximum sample parameters

Domain	Element	X (m)	Y (m)	Z (m)	Minimum No. of Samples	Maximum No. of Samples	Minimum No. of Drillholes
All Domains (Excluding 7101.11)	Au	60	100	80	4	10	2
	Ag	60	100	60	4	10	2
	Pb	60	100	80	4	10	2
	Zn	60	60	120	4	10	2
	As	60	100	100	4	10	2
	Cu	60	100	100	4	10	2
	Fe	60	120	60	4	10	2
Central Klaza Domain 7101.11	Au	60	100	50	4	10	2
	Ag	60	100	30	4	10	2
	Pb	60	100	50	4	10	2
	Zn	60	60	80	4	10	2
	As	60	100	60	4	10	2
	Cu	60	100	60	4	10	2
	Fe	60	120	40	4	10	2

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 resource tables. Copper and iron were estimated to provide a regression line for the density correlation. Arsenic was estimated for metallurgical assessment.

14.6 Mineral Resource classification

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. While local areas had sufficient drill density to be classified as Indicated these areas did not form significant contiguous bodies. As a result, the entire Mineral Resource is classified as Inferred at this time.

14.7 Block model validation

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 with satisfactory results.

14.8 Mineral Resource estimate

The Mineral Resource estimate consists of pit-constrained and underground Mineral Resources for all five zones at the Klaza Deposit. Pit-constrained Mineral Resources are reported between a base-of-overburden surface and a conceptual pit shell based on a US$1,300/oz gold price. 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 14.14. A cut-off of 1.3 g/t gold equivalent was applied for reporting the pit-constrained Mineral Resources.

Table 14.14 Conceptual pit shell parameters

Item	Pit Optimization Parameters	Unit
Gold Price	1,300	US$/oz
Silver Price	20	US$/oz
Lead Price	0.90	US$/lb
Zinc Price	0.90	US$/lb
Exchange Rate	0.80	C$ to US$
Refining Charge Au	8.00	US$/oz
Refining Charge Ag	2.00	US$/oz
Refining Charge Lead & Zinc	0.10	US$/lb
% Payable Gold	99%	
% Payable Silver	88%	
% Payable Lead	95%	
% Payable Zinc	83%	
Concentrate Costs[1]	261.44	US$/dmt of con
Royalties	0.00%	
Processing Costs	50.39	C$/tonne
G & A	15.00	C$/tonne
Base Mining Costs [2]	4.90	C$/tonne
Preliminary Overall Slope Angles	44 – 48	degrees
Dilution	25%	
Plant Rate	550	ktpa
AuEQ Formula	Au EQ=Au+Ag/85+Pb/3.74+Zn/5.04	
Metallurgical Recovery	96% gold, 91% silver, 85% lead, 85% zinc	

Notes: 1. Includes estimated transport, port handling, ocean freight and treatment costs 2. Includes incremental mining costs
Source: AMC Mining Consultants (Canada) Ltd.

The results of the Klaza deposit conceptual pit Mineral Resource estimates are shown by zones in Table 14.15 to Table 14.19 at a range of cut-offs, with the selected cut-off shown in bold. The estimate for the entire Klaza deposit is shown in Table 14.20 at a range of cut-offs.

Table 14.15 Western BRX pit-constrained Inferred Mineral Resource estimate

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)
0.00	595	7.70	103	0.97	0.98	9.36	147	1,973	12,687	12,836	179
0.50	595	7.70	103	0.97	0.98	9.36	147	1,973	12,687	12,836	179
0.75	595	7.70	103	0.97	0.98	9.36	147	1,973	12,687	12,836	179
1.00	585	7.82	105	0.98	0.99	9.51	147	1,971	12,671	12,780	179
1.30	**554**	**8.21**	**110**	**1.03**	**1.03**	**9.99**	**146**	**1,960**	**12,608**	**12,557**	**178**
1.50	546	8.31	111	1.05	1.04	10.11	146	1,955	12,583	12,512	177
1.75	537	8.43	113	1.06	1.05	10.25	145	1,948	12,539	12,440	177
2.00	520	8.65	116	1.09	1.08	10.52	145	1,940	12,468	12,336	176
2.50	490	9.07	122	1.14	1.12	11.03	143	1,920	12,265	12,068	174
2.75	484	9.16	123	1.15	1.12	11.13	142	1,916	12,236	11,992	173
3.00	472	9.32	126	1.17	1.14	11.34	142	1,907	12,193	11,881	172
3.50	454	9.58	129	1.21	1.16	11.66	140	1,888	12,106	11,660	170
4.00	434	9.89	134	1.25	1.19	12.03	138	1,865	12,001	11,403	168
5.00	412	10.20	139	1.30	1.22	12.42	135	1,837	11,857	11,119	165

CIM definition standards were used for the Mineral Resource.
Using drilling results to 30 September 2015.
Near surface mineral resources are constrained by an optimized pit shell at a gold price of US$1300 oz.
Gold equivalent values were calculated using the following formula: Au EQ=Au+Ag/85+Pb/3.74+Zn/5.04 and assuming: US$1300 oz Au, US$20 oz Ag, US$0.90 lb Pb and US$0.90 lb Zn with recoveries for each metal of Au: 96%, Ag: 91%, Pb: 85% and Zn: 85%.
Numbers may not add due to rounding.
All metal prices are quoted in US$ at an exchange rate of $0.80 US to $1.00 Canadian.
Mineral resources that are not mineral reserves do not have demonstrated economic viability.
Source: AMC Mining Consultants (Canada) Ltd.

Table 14.16 Central BRX pit-constrained Inferred Mineral Resource estimate

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)
0.00	283	3.67	192	1.34	1.39	6.57	33	1,752	8,361	8,693	60
0.50	283	3.67	192	1.34	1.39	6.57	33	1,752	8,361	8,693	60
0.75	283	3.67	192	1.34	1.39	6.57	33	1,752	8,361	8,693	60
1.00	283	3.67	192	1.34	1.39	6.57	33	1,752	8,361	8,693	60
1.30	**283**	**3.67**	**192**	**1.34**	**1.39**	**6.57**	**33**	**1,752**	**8,361**	**8,693**	**60**
1.50	283	3.67	192	1.34	1.39	6.57	33	1,752	8,361	8,693	60
1.75	276	3.75	196	1.36	1.40	6.69	33	1,738	8,301	8,541	59
2.00	276	3.75	196	1.36	1.40	6.69	33	1,738	8,301	8,541	59
2.50	274	3.77	197	1.37	1.41	6.73	33	1,734	8,275	8,490	59
2.75	274	3.77	197	1.37	1.41	6.74	33	1,733	8,271	8,476	59
3.00	274	3.77	197	1.37	1.41	6.74	33	1,733	8,271	8,476	59
3.50	269	3.81	199	1.39	1.42	6.80	33	1,718	8,222	8,380	59
4.00	249	3.94	206	1.45	1.46	7.04	32	1,647	7,964	7,992	56
5.00	199	4.29	226	1.59	1.56	7.68	27	1,445	6,976	6,866	49

Notes: see footnotes in Table 14.14
Source: AMC Mining Consultants (Canada) Ltd.

Table 14.17 Eastern BRX pit-constrained Inferred Mineral Resource estimate

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)
0.00	194	4.39	90	0.21	0.42	5.59	27	559	908	1,798	35
0.50	193	4.42	90	0.21	0.42	5.62	27	559	908	1,798	35
0.75	193	4.42	90	0.21	0.42	5.62	27	559	908	1,798	35
1.00	193	4.42	90	0.21	0.42	5.62	27	559	908	1,798	35
1.30	**193**	**4.42**	**90**	**0.21**	**0.42**	**5.62**	**27**	**559**	**908**	**1,798**	**35**
1.50	193	4.42	90	0.21	0.42	5.62	27	559	908	1,798	35
1.75	192	4.44	90	0.21	0.42	5.64	27	558	898	1,779	35
2.00	192	4.44	90	0.21	0.42	5.64	27	558	898	1,778	35
2.50	178	4.66	93	0.22	0.43	5.90	27	530	872	1,691	34
2.75	170	4.81	93	0.23	0.44	6.06	26	510	851	1,633	33
3.00	166	4.88	94	0.23	0.43	6.13	26	500	828	1,576	33
3.50	158	5.01	96	0.21	0.40	6.27	25	486	747	1,386	32
4.00	138	5.34	101	0.18	0.29	6.64	24	449	546	870	29
5.00	96	6.22	117	0.07	0.05	7.63	19	360	143	95	23

Notes: see footnotes in Table 14.14
Source: AMC Mining Consultants (Canada) Ltd.

Table 14.18 Western Klaza pit-constrained Inferred Mineral Resource estimate

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)
0.00	81	6.86	288	1.01	0.98	10.72	18	752	1,803	1,748	28
0.50	81	6.86	288	1.01	0.98	10.72	18	752	1,803	1,748	28
0.75	81	6.86	288	1.01	0.98	10.72	18	752	1,803	1,748	28
1.00	81	6.86	288	1.01	0.98	10.72	18	752	1,803	1,748	28
1.30	**81**	**6.86**	**288**	**1.01**	**0.98**	**10.72**	**18**	**752**	**1,803**	**1,748**	**28**
1.50	81	6.86	288	1.01	0.98	10.72	18	752	1,803	1,748	28
1.75	81	6.86	288	1.01	0.98	10.72	18	752	1,803	1,748	28
2.00	81	6.86	288	1.01	0.98	10.72	18	752	1,803	1,748	28
2.50	81	6.86	288	1.01	0.98	10.72	18	752	1,803	1,748	28
2.75	81	6.86	288	1.01	0.98	10.72	18	752	1,803	1,748	28
3.00	81	6.86	288	1.01	0.98	10.72	18	752	1,803	1,748	28
3.50	81	6.86	288	1.01	0.98	10.72	18	752	1,803	1,748	28
4.00	81	6.86	288	1.01	0.98	10.72	18	752	1,803	1,748	28
5.00	81	6.86	288	1.01	0.98	10.72	18	752	1,803	1,748	28

Notes: see footnotes in Table 14.14
Source: AMC Mining Consultants (Canada) Ltd.

Table 14.19 Central Klaza pit-constrained Inferred Mineral Resource estimate

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)
0.00	1,278	4.01	53	0.87	1.31	5.13	165	2,176	24,646	36,809	211
0.50	1,275	4.02	53	0.88	1.31	5.14	165	2,175	24,640	36,803	211
0.75	1,274	4.02	53	0.88	1.31	5.14	165	2,175	24,638	36,800	211
1.00	1,271	4.03	53	0.88	1.31	5.15	165	2,174	24,630	36,788	211
1.30	**1,255**	**4.07**	**54**	**0.89**	**1.33**	**5.20**	**164**	**2,168**	**24,578**	**36,680**	**210**
1.50	1,226	4.14	55	0.90	1.35	5.30	163	2,154	24,455	36,476	209
1.75	1,206	4.19	55	0.92	1.37	5.36	162	2,141	24,331	36,322	208
2.00	1,181	4.25	56	0.93	1.39	5.43	161	2,119	24,130	36,117	206
2.50	1,102	4.43	58	0.96	1.46	5.66	157	2,040	23,394	35,495	200
2.75	1,063	4.52	58	0.98	1.50	5.77	155	1,995	22,972	35,086	197
3.00	1,030	4.60	59	1.00	1.53	5.86	152	1,964	22,653	34,666	194
3.50	942	4.78	62	1.06	1.60	6.11	145	1,875	21,908	33,209	185
4.00	827	5.02	66	1.14	1.69	6.43	133	1,745	20,828	30,814	171
5.00	626	5.50	71	1.30	1.88	7.05	111	1,429	17,962	25,884	142

Notes: see footnotes in Table 14.14
Source: AMC Mining Consultants (Canada) Ltd.

Table 14.20 Klaza Property pit-constrained Inferred Mineral Resource estimate

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)
0.00	2,432	5.00	92	0.90	1.15	6.56	391	7,212	48,405	61,883	513
0.50	2,427	5.01	92	0.90	1.16	6.57	391	7,211	48,399	61,878	513
0.75	2,427	5.01	92	0.90	1.16	6.57	391	7,210	48,398	61,875	513
1.00	2,413	5.03	93	0.91	1.16	6.60	391	7,207	48,374	61,806	512
1.30	**2,366**	**5.12**	**95**	**0.93**	**1.18**	**6.71**	**389**	**7,190**	**48,258**	**61,475**	**510**
1.50	2,329	5.18	96	0.94	1.19	6.79	388	7,172	48,110	61,227	509
1.75	2,292	5.25	97	0.95	1.20	6.88	387	7,136	47,873	60,829	507
2.00	2,250	5.32	98	0.96	1.22	6.97	385	7,106	47,600	60,520	504
2.50	2,125	5.53	102	0.99	1.27	7.25	378	6,977	46,608	59,491	495
2.75	2,071	5.62	104	1.01	1.29	7.37	374	6,906	46,133	58,935	491
3.00	2,023	5.70	105	1.03	1.31	7.48	371	6,855	45,748	58,347	486
3.50	1,904	5.90	110	1.07	1.34	7.74	361	6,719	44,787	56,383	474
4.00	1,729	6.20	116	1.13	1.39	8.14	344	6,457	43,142	52,826	453
5.00	1,414	6.83	128	1.24	1.47	8.96	310	5,823	38,742	45,712	407

Notes: see footnotes in Table 14.14
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 2.75 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 Klaza deposit underground Mineral Resource estimates are shown in Table 14.21 and Table 14.25 at a range of cut-offs, with the selected cut-offs shown in bold. The estimate for the entire Klaza deposit is shown in Table 14.26 at a range of cut-offs.

Table 14.21 Western BRX underground Inferred Mineral Resource estimate

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)
1.00	939	7.06	131	1.32	1.51	9.26	213	3,961	27,364	31,177	279
1.50	934	7.09	132	1.33	1.51	9.30	213	3,954	27,346	31,151	279
2.00	904	7.28	135	1.37	1.56	9.55	212	3,938	27,251	31,003	277
2.50	849	7.63	142	1.44	1.63	10.02	208	3,887	27,006	30,516	274
2.75	**814**	**7.87**	**147**	**1.49**	**1.68**	**10.34**	**206**	**3,853**	**26,764**	**30,194**	**271**
3.00	799	7.98	150	1.52	1.71	10.48	205	3,843	26,682	30,036	269
3.50	757	8.27	156	1.58	1.77	10.88	201	3,799	26,423	29,592	265
4.00	723	8.51	162	1.65	1.83	11.22	198	3,768	26,204	29,137	261
5.00	655	9.00	175	1.78	1.94	11.91	189	3,680	25,659	28,062	251
6.00	573	9.64	191	1.95	2.09	12.83	178	3,522	24,665	26,411	236
7.00	511	10.20	204	2.09	2.22	13.59	168	3,344	23,578	24,981	223

CIM definition standards were used for the Mineral Resource.
Using drilling results to 30 September 2015.
Underground mineral resources are constrained outside an optimized pit shell at a gold price of US$1300 oz.
Gold equivalent values were calculated using the following formula: Au EQ=Au+Ag/85+Pb/3.74+Zn/5.04 and assuming: US$1300 oz Au, US$20 oz Ag, US$0.90 lb Pb and US$0.90 lb Zn with recoveries for each metal of Au: 96%, Ag: 91%, Pb: 85% and Zn: 85%.
Numbers may not add due to rounding.
All metal prices are quoted in US$ at an exchange rate of $0.80 US to $1.00 Canadian.
Mineral resources that are not mineral reserves do not have demonstrated economic viability.
Source: AMC Mining Consultants (Canada) Ltd.

Table 14.22 Central BRX underground Inferred Mineral Resource estimate

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)
1.00	1,311	2.36	129	1.07	1.21	4.41	99	5,455	30,830	35,106	186
1.50	1,303	2.36	130	1.07	1.22	4.42	99	5,449	30,788	35,007	185
2.00	1,182	2.50	138	1.14	1.29	4.69	95	5,257	29,707	33,571	178
2.50	1,058	2.63	149	1.23	1.37	4.98	89	5,067	28,804	31,898	169
2.75	**1,027**	**2.65**	**152**	**1.26**	**1.39**	**5.05**	**87**	**5,019**	**28,561**	**31,506**	**167**
3.00	991	2.67	155	1.29	1.42	5.13	85	4,952	28,226	31,020	163
3.50	872	2.75	167	1.39	1.51	5.38	77	4,690	26,780	28,979	151
4.00	743	2.79	183	1.52	1.62	5.67	67	4,361	24,877	26,553	135
5.00	488	2.97	211	1.74	1.81	6.28	47	3,306	18,666	19,455	98
6.00	253	3.37	234	1.84	1.90	7.00	27	1,901	10,271	10,599	57
7.00	108	4.00	243	1.93	1.89	7.75	14	847	4,609	4,518	27

Notes: see footnotes in Table 14.21
Source: AMC Mining Consultants (Canada) Ltd.

Table 14.23 Eastern BRX underground Inferred Mineral Resource estimate

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)
1.00	3,376	3.21	43	0.21	0.28	3.83	348	4,710	15,752	21,101	416
1.50	3,246	3.30	45	0.21	0.29	3.93	344	4,652	15,359	20,546	411
2.00	2,870	3.55	47	0.22	0.29	4.21	327	4,326	13,708	18,371	389
2.50	2,396	3.91	49	0.22	0.29	4.61	301	3,795	11,436	15,573	355
2.75	**2,213**	**4.07**	**50**	**0.21**	**0.29**	**4.77**	**289**	**3,568**	**10,296**	**14,230**	**340**
3.00	1,979	4.28	51	0.20	0.28	5.00	272	3,266	8,891	12,322	318
3.50	1,593	4.68	55	0.18	0.26	5.42	240	2,794	6,396	9,237	278
4.00	1,196	5.17	60	0.18	0.25	5.97	199	2,297	4,843	6,678	230
5.00	871	5.71	65	0.17	0.24	6.57	160	1,829	3,299	4,554	184
6.00	523	6.39	70	0.15	0.18	7.28	107	1,174	1,680	2,088	123
7.00	315	6.95	66	0.12	0.14	7.79	71	670	830	991	79

Notes: see footnotes in Table 14.21
Source: AMC Mining Consultants (Canada) Ltd.

Table 14.24 Western Klaza underground Inferred Mineral Resource estimate

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)
1.00	468	5.35	180	0.57	0.86	7.80	81	2,715	5,893	8,863	117
1.50	468	5.35	180	0.57	0.86	7.80	81	2,715	5,893	8,863	117
2.00	466	5.37	181	0.57	0.86	7.83	80	2,713	5,886	8,850	117
2.50	464	5.38	182	0.58	0.86	7.85	80	2,710	5,884	8,839	117
2.75	**461**	**5.41**	**182**	**0.58**	**0.87**	**7.88**	**80**	**2,703**	**5,879**	**8,820**	**117**
3.00	457	5.44	183	0.58	0.87	7.93	80	2,692	5,868	8,765	116
3.50	435	5.59	190	0.61	0.88	8.17	78	2,659	5,806	8,449	114
4.00	410	5.75	198	0.63	0.90	8.44	76	2,615	5,701	8,119	111
5.00	354	6.11	219	0.70	0.95	9.07	70	2,489	5,464	7,424	103
6.00	310	6.41	234	0.76	0.99	9.56	64	2,328	5,215	6,781	95
7.00	254	6.81	255	0.86	1.04	10.24	56	2,078	4,820	5,828	84

Notes: see footnotes in Table 14.21
Source: AMC Mining Consultants (Canada) Ltd.

Table 14.25 Central Klaza underground Inferred Mineral Resource estimate

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)
1.00	4,261	2.85	46	0.50	0.76	3.68	391	6,351	46,680	70,964	504
1.50	4,086	2.94	47	0.51	0.77	3.78	386	6,210	45,856	69,796	497
2.00	3,607	3.15	50	0.54	0.82	4.05	365	5,806	43,282	65,432	469
2.50	2,844	3.54	55	0.61	0.90	4.53	324	5,017	38,415	56,602	414
2.75	**2,539**	**3.74**	**57**	**0.64**	**0.92**	**4.76**	**305**	**4,628**	**35,661**	**51,668**	**389**
3.00	2,245	3.95	59	0.66	0.95	5.01	285	4,229	32,717	46,976	362
3.50	1,739	4.43	60	0.70	1.00	5.53	248	3,355	26,976	38,395	309
4.00	1,418	4.81	61	0.75	1.05	5.93	219	2,776	23,449	32,903	271
5.00	929	5.46	67	0.90	1.13	6.71	163	1,998	18,416	23,126	201
6.00	556	6.08	78	1.11	1.23	7.55	109	1,403	13,570	15,128	135
7.00	314	6.62	95	1.41	1.44	8.40	67	961	9,752	9,999	85

Notes: see footnotes in Table 14.21
Source: AMC Mining Consultants (Canada) Ltd.

Table 14.26 Klaza Property underground Inferred Mineral Resource estimate

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)
1.00	10,354	3.40	69.66	0.55	0.73	4.51	1,132	23,191	126,518	167,211	1,503
1.50	10,036	3.48	71	0.57	0.75	4.62	1,122	22,981	125,240	165,363	1,489
2.00	9,030	3.72	76	0.60	0.79	4.93	1,080	22,041	119,835	157,228	1,431
2.50	7,612	4.10	84	0.66	0.85	5.43	1,004	20,476	111,544	143,428	1,330
2.75	**7,054**	**4.27**	**87**	**0.69**	**0.88**	**5.65**	**969**	**19,772**	**107,159**	**136,416**	**1,282**
3.00	6,470	4.46	91	0.72	0.91	5.91	928	18,982	102,383	129,119	1,228
3.50	5,396	4.86	100	0.78	0.96	6.44	844	17,296	92,381	114,652	1,117
4.00	4,489	5.25	110	0.86	1.04	6.98	758	15,817	85,073	103,390	1,007
5.00	3,297	5.93	126	0.98	1.14	7.90	629	13,301	71,504	82,622	837
6.00	2,214	6.81	145	1.13	1.25	9.07	485	10,328	55,401	61,007	646
7.00	1,503	7.75	163	1.32	1.40	10.30	375	7,901	43,589	46,317	498

Notes: see footnotes in Table 14.21
Source: AMC Mining Consultants (Canada) Ltd.

Representative cross sections through the Klaza deposit are show in Section 10 in Figures 10.2 to Figure 10.5.

14.9 Comparison with previous Mineral Resource estimate

The previous Mineral Resource estimate on the Property was published in the “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).

Changes to the Mineral Resource estimate in this report are due predominately to:

- Additional drilling information
- Modelling of subsidiary structures in the Central Klaza zone
- Inclusion of the Eastern BRX zone into the modelled area

The Mineral Resource has also been pit-constrained with different cut-off grades for the near surface and underground mineralization.

A less significant change to the Mineral Resource estimate resulted from refinement of specific gravity regression lines.

Table 14.27 shows a comparison between the Giroux and AMC estimates. To make a more meaningful comparison, the estimates are both reported at a cut-off grade of 1.5 g/t Au.

Table 14.27 Comparison of Inferred Mineral Resource estimates

Author	Zone	Tonnes (kt)	Au (g/t)	Ag (g/t)	Pb(%)	Zn(%)	Cu (%)	Au (Koz)
Giroux 25 November 2014	All zones	7,040	4.19	96.23	0.78	0.93	0.09	948
AMC 9 December 2015	All zones	10,899	4.14	79.04	0.66	0.87	0.12	1,451
Giroux 25 November 2014	Western BRX	1,380	7.91	107.37	0.89	1.07	0.14	351
AMC 9 December 2015	Western BRX	1,430	7.76	127.20	1.25	1.37	0.15	357
Giroux 25 November 2014	Central BRX	2,140	2.58	115.52	0.92	0.94	0.14	178
AMC 9 December 2015	Central BRX	1,299	2.92	155.09	1.22	1.34	0.21	122
Giroux 25 November 2014	Eastern BRX	na	na	na	na	na	na	na
AMC 9 December 2015	Eastern BRX	3,011	3.66	47.20	0.20	0.28	0.18	354
Giroux 25 November 2014	Western Klaza	515	6.00	247.59	0.75	0.90	0.03	99
AMC 9 December 2015	Western Klaza	546	5.60	197.10	0.64	0.88	0.04	98
Giroux 25 November 2014	Central Klaza	3,010	3.32	51.59	0.65	0.86	0.05	321
AMC 9 December 2015	Central Klaza	4,612	3.51	49.50	0.62	0.96	0.06	520

CIM definition standards were used for the Mineral Resource estimates.
Drilling results to 30 September 2015 for AMC estimate. Drilling results to 25 November 2014 for Giroux Consultants Ltd estimate.
Results are reported for comparison purposes only at a 1.5 g/t Au cut-off.
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 and Wengzynowski et al., 2015.

In the Inferred Mineral Resource category, there has been an increase of 503,000 ounces of gold.

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. In the Central Klaza zone potential exists in the eastern extremity for underground mining. The eastern extent is naturally separated from the remainder of the Central Klaza zone by low grade mineralization. The eastern extent of the Central Klaza zone is accessed by an independent decline system and for mining purposes is termed Eastern Klaza. The Eastern BRX zone is only considered as upside potential that warrants further study.

16.1 Hydrological parameters

In anticipation of a future application under the Yukon Environmental and Socioeconomic Assessment Act (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:

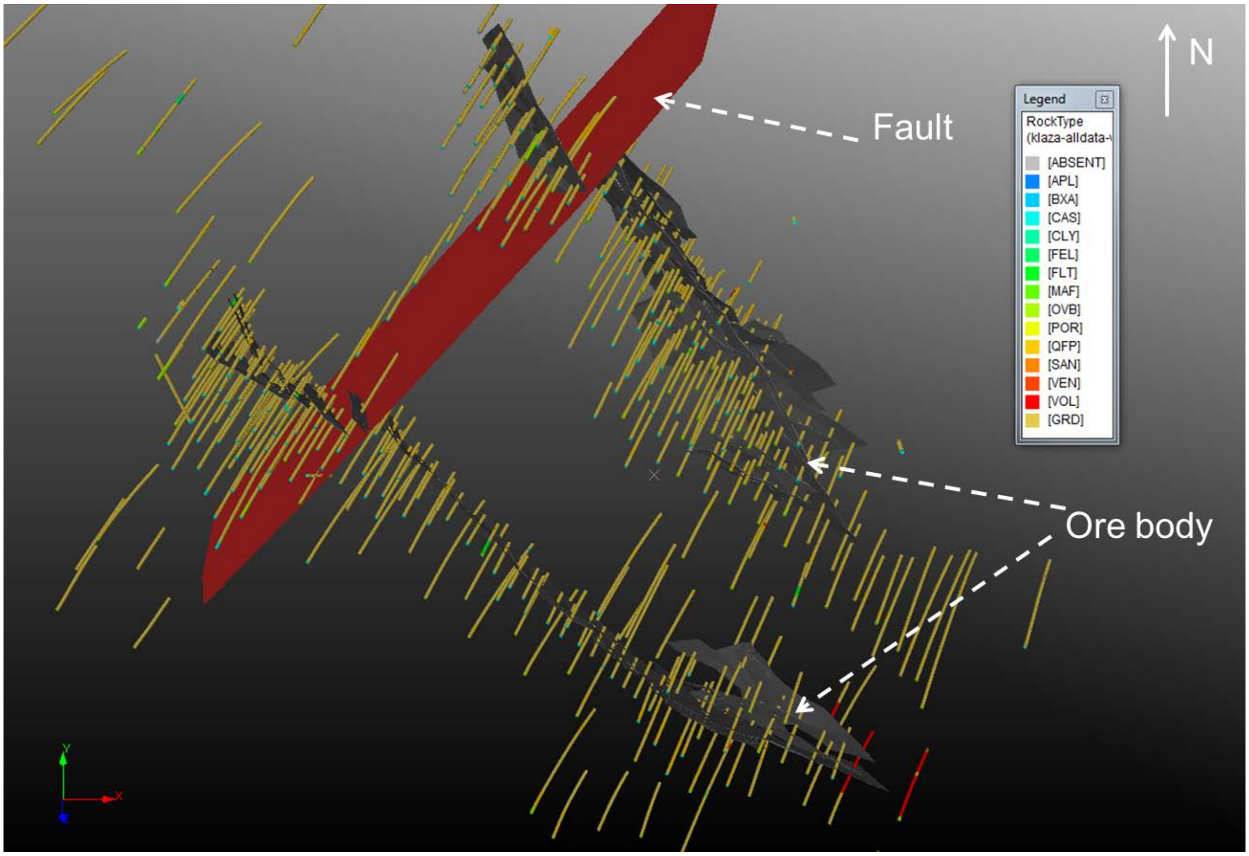
- Permafrost appears to act as a confining layer for the deeper bedrock aquifer. However, some uncertainty remains with respect to the permafrost temperature, extent, and interaction with groundwater that requires further data collection.
- The groundwater flow regime at the site is controlled by the 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 six inch 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 2010, 2011, 2012, 2014 and 2015 drilling information for the Property. Figure 16.1 shows drilling locations with lithology information. Granodiorite (GRD) represents 90% of the total drilled length.

Figure 16.1 Drillhole locations and lithological units: Klaza and BRX zones (2010-2015)



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 (90% of the total drill metres). The other lithologies make up the remaining 10% (Table 16.1).

Table 16.1 Total drillhole length within lithological units

Rock type		Total drillhole length (m)	% Freq.
Aplite	APL	26	0.0
Breccia	BXA	157	0.2
Casing	CAS	644	0.9
Felsic Rock	FEL	33	0.0
Fault	FLT	153	0.2
Granodiorite	GRD	63,096	90.1
Mafic Rock	MAF	47	0.1
Overburden	OVB	812	1.2
Porphyry	POR	232	0.3
Quartz Feldspar Porphyry	QFP	4,035	5.8
Sand	SAN	3	0.0
Siliceous Dyke	CLY	1	0.0
Vein	VEN	170	0.2
Volcanic	VOL	652	0.9
Total		**70,061**	**100.0**

16.2.2 Rock Quality Designation (RQD)

Rock quality designation (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 (Deere, 1964)

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

In general, the RQD assessment indicates that most lithologies identified at Klaza have a wide range of RQD values, typically from Poor to Good.

16.2.3 Rock mass characterization

Rock mass characterization was carried out using two main classification systems: RMR (Bieniawski, 1976) and Q-index (Barton et al., 1974). RMR is typically used for slope stability analysis. Average RMR and Q' values determined for the main lithology are shown in Figure 16.2 and Figure 16.3. All domains are classified as Fair to Good.

Figure 16.2 RMR_{76} by lithological domain

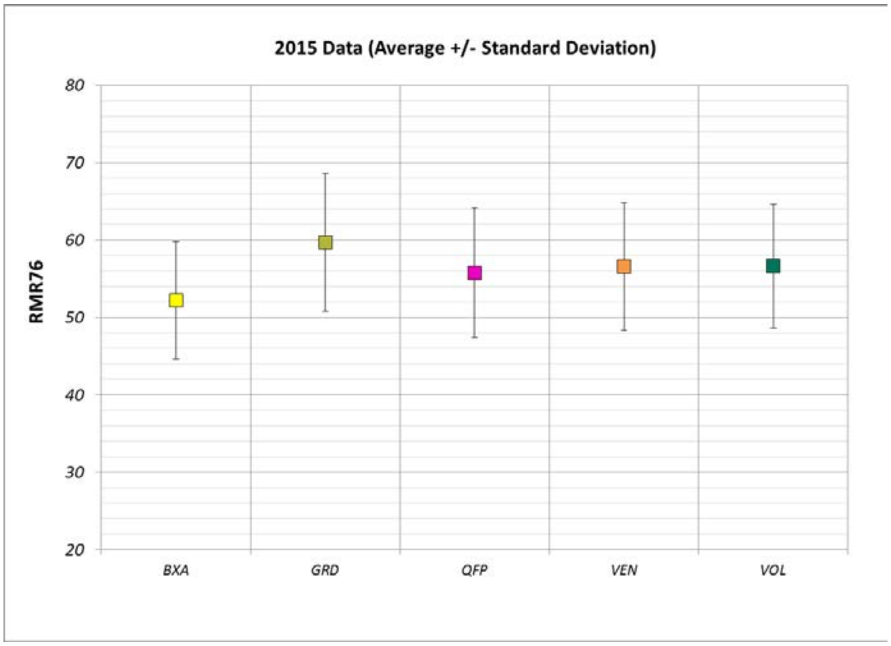
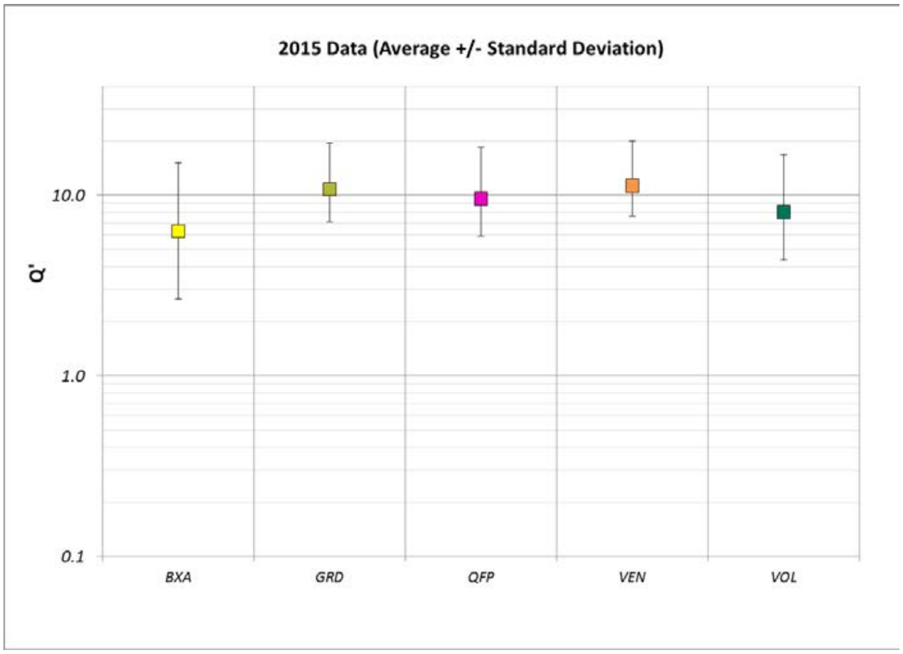


Figure 16.3 Q' by lithological domain



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. Typical values of FOS used in pit design are indicated in Table 16.3.

Table 16.3 Typical FOS and POF acceptance criteria values (Read & Stacey, 2009)

Slope Scale	Consequence of Failure[b]	Acceptance Criteria[a]		
		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%

[a] Needs to meet all acceptance criteria
[b] Semi-quantitatively evaluated

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	RMR_{av}	Weathering	Blasting	Joints	Water	Stress	$MRMR_{av}$	Slope height, m
GRD	60	0.96	0.94	0.8	0.9	1	41	100
QFP	56	0.96	0.94	0.8	0.9	1	39	100

The summary values can be plotted on the Haines-Terbrugge chart and pit slope angles determined (Table 16.5).

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

Domain	Haines-Terbrugge (1990)	
	FOS=1.5	FOS=1.2
GRD	45	55
QFP	44	54

16.2.4.3 Limit equilibrium analysis

A series of limit equilibrium (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 factor of safety (FOS) is calculated for each failure surface as follows:

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

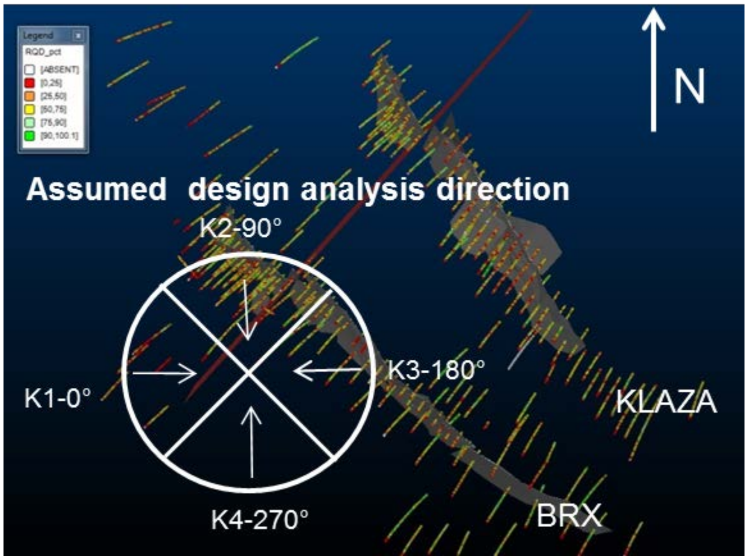
Where the resisting forces are related to the rock mass strength components resisting failure, and driving forces are related to the weight of the slices and resultant forces promoting failure.

The input parameters for LE modelling, UCS – 35 MPa and GSI – 55, were assumed based on the existing geotechnical data. The results show FOS for slope angles 45°- 55° is 3.4 - 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 16.4).

Figure 16.4 Design sectors of open pit for kinematic analysis



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 ϕ = 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 probability of failure (POF) can be related to the cumulative frequency (CF) in the kinematic assessments, whereby POF=50% would correspond to CF=50% (since 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 (for BRX) and Table 16.7 (for Klaza).

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

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

16.2.5 Pit wall design parameters

Pit slope design recommendations are shown in Table 16.8 (for BRX) and Table 16.9 (for Klaza). 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 projected. 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

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

16.2.6 Underground geotechnical considerations

16.2.6.1 Stable stope spans

AMC conducted the stope stability assessment using the empirical Stability Graph Method (SGM) proposed by Mathews et al. (1981) and Potvin (1988). This approach is widely practiced in Australia and North America to obtain a first-pass estimate of possible stope spans when little or no local stoping experience is available. Stability of each stope wall is assessed separately.

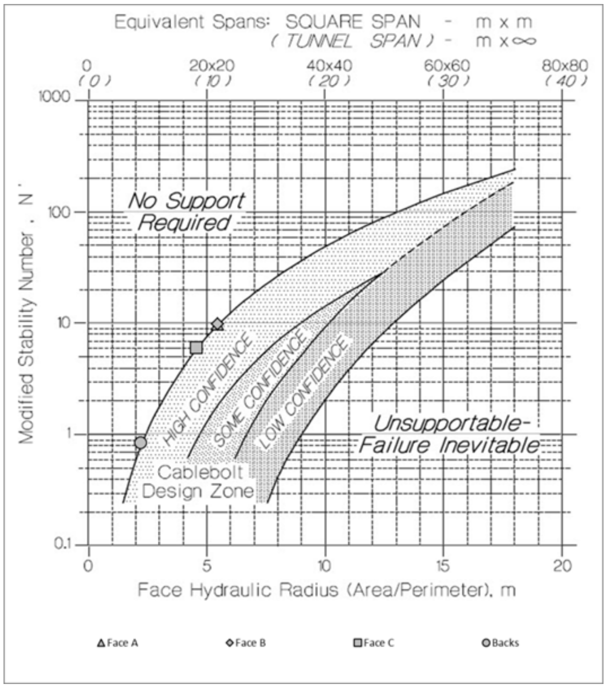
Under the SGM, the stability number N' is calculated using the following expression:

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

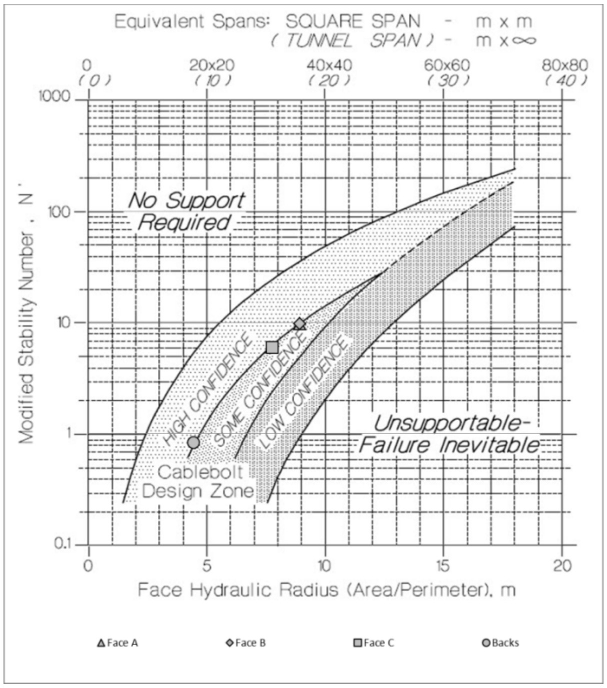
Where A is the rock stress factor, B is the joint orientation factor, and C is the gravity adjustment factor.

The derivation of the adjustment factors A, B, and C is based on interpretation of the induced stress around excavations, the dominant joint set orientation in relation to the face of interest and the dip of the face. A low to moderate in situ stress regime is assumed. Factor C was calculated based on an average range of the vein dip values (45° to 65°). The hydraulic radii of stable spans, with and without support, can be determined from the stability graph (Figure 16.5).

Figure 16.5 Stability graph results for the stope walls –(a) unsupported and (b) supported (Vein dip-55°)



(a)

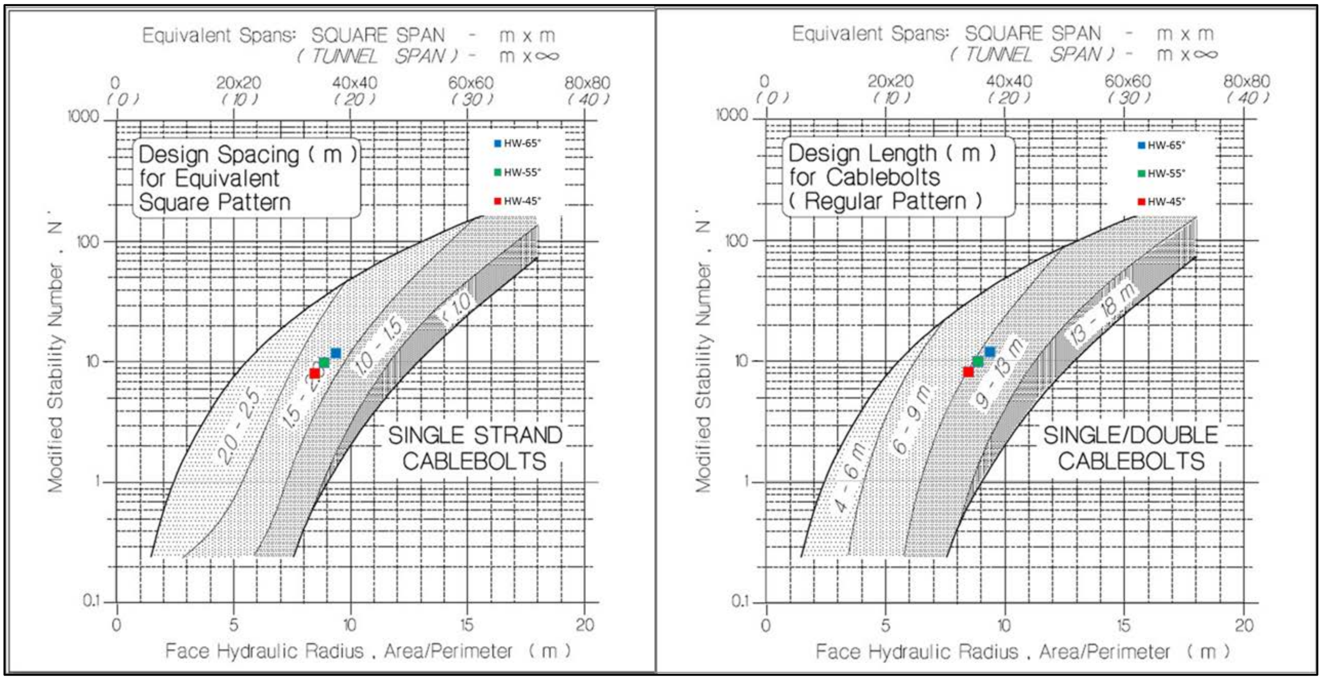


(b)

Both unsupported and supported cases have been assessed. The results are for the average HR threshold that AMC considers is applicable for this level of assessment. The assessment suggests that stope backs at typical envisaged width (3 m) will be stable at projected stope heights (30 m). The results indicate that hangingwall stability is strongly influenced by vein dip. Based on a floor to back height of 30 m, and 65° dip, stope lengths of 18 m for unsupported walls and 44 m for supported walls are projected to be stable. At 45° dip, stope lengths of 13 m for unsupported walls and 28 m for supported walls are projected to be stable at envisaged stope length.

Recommended cable bolt spacing and length can also be derived from empirical data (Hutchinson 1996). Figure 16.6 shows recommended spacing and minimum lengths for cable bolts. The suggested range of cablebolt length is 9 m to 13 m, and the spacing should be a minimum of 1.5 - 2.0 m.

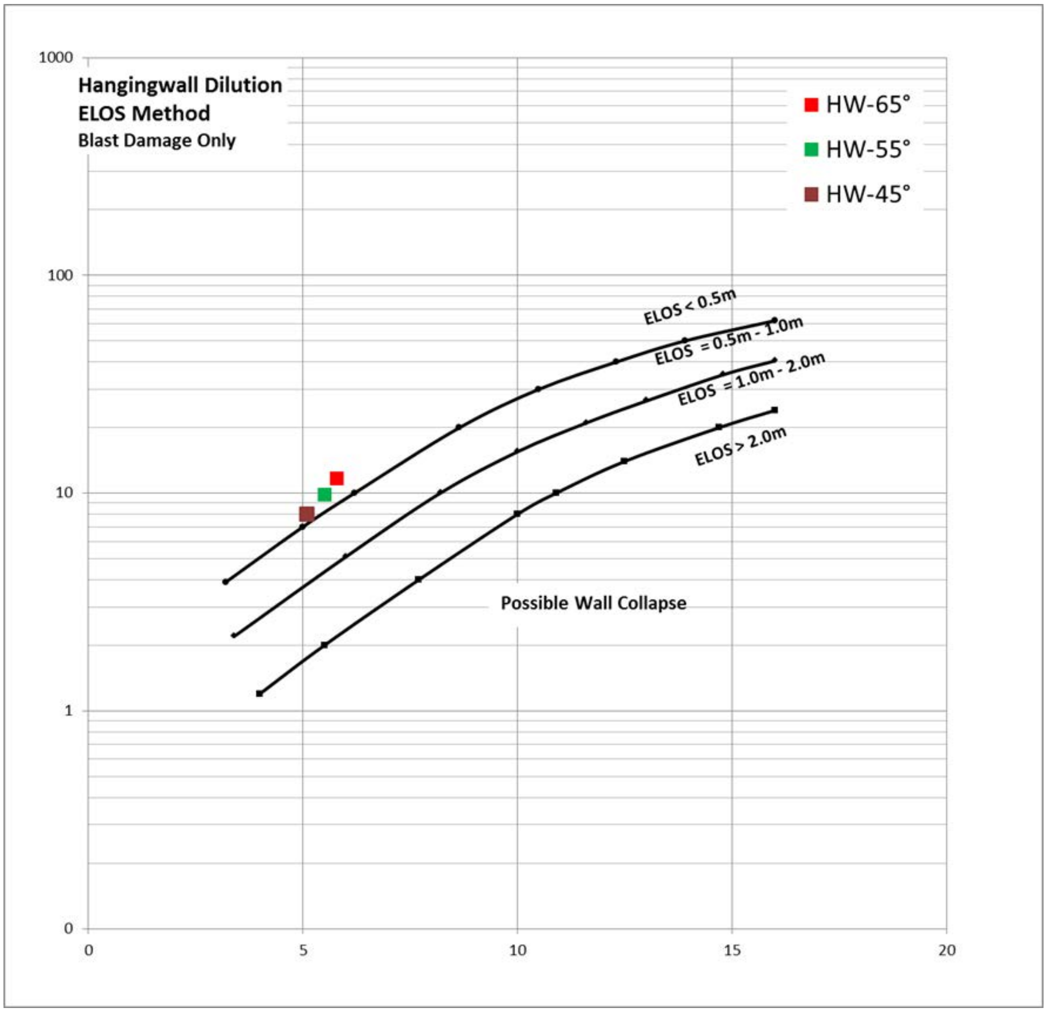
Figure 16.6 Recommended spacing and minimum length for cable bolts.



16.2.6.2 Stope dilution estimation

A preliminary estimate of dilution for Long Hole Open Stopes (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). The HW dilution estimation is presented in Figure 16.7; which shows dilution of less than ~ 0.5 m from hangingwall. It should be noted that the stability graph method is approximate only and early stoping should be carefully monitored, and designs adjusted in response to actual performance.

Figure 16.7 Stope dilution estimated for the HW



16.2.6.3 Preliminary ground support estimates

Preliminary ground support requirements have been estimated using the Q-system updated rock reinforcement design chart (Grimstad and Barton, 1993), which relates rock quality, excavation span, and service life to support requirements (Figure 16.8). The method converts the width (span) of the excavation to an equivalent dimension (D_e), which takes into account the function of the excavation.

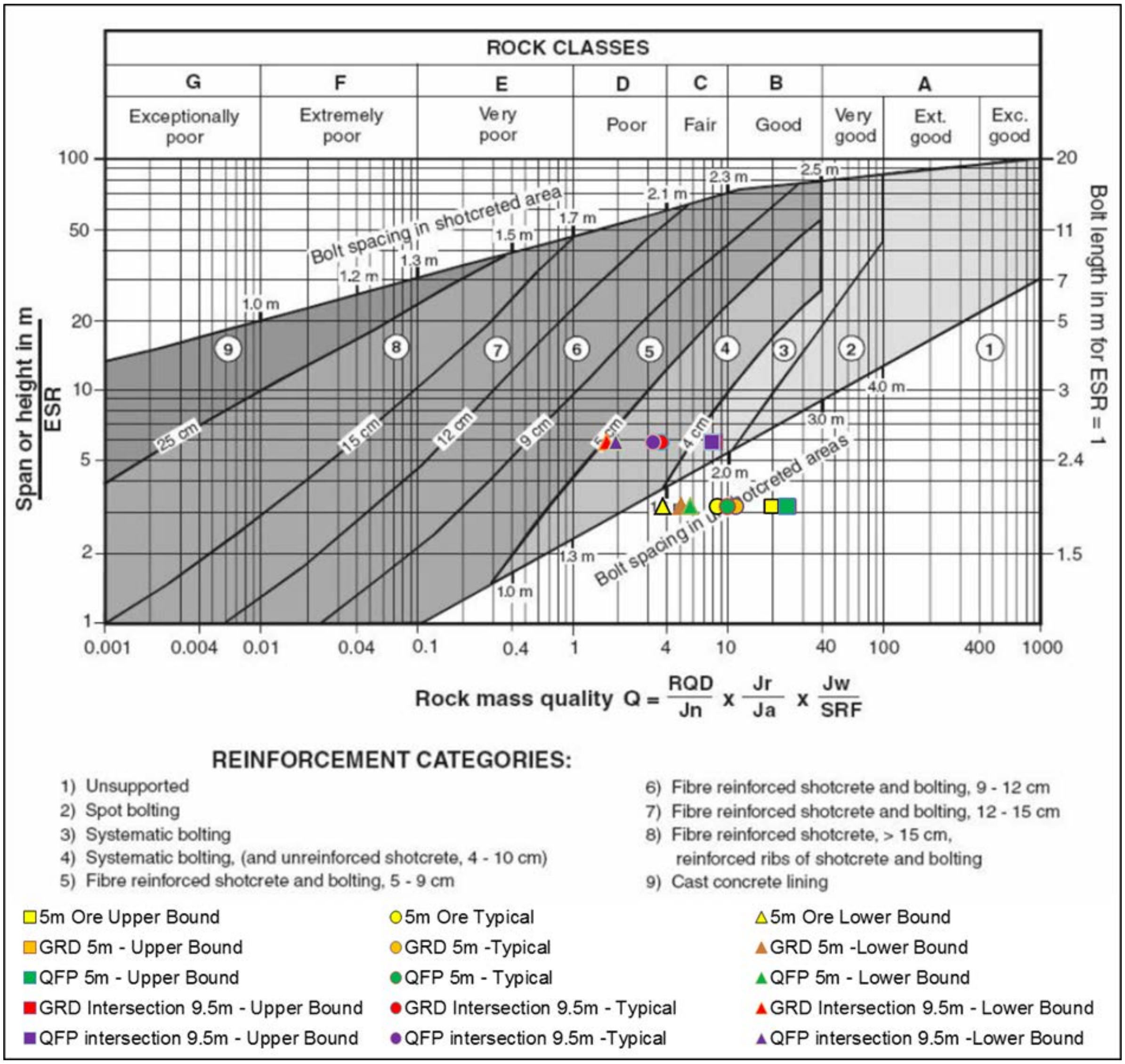
Equivalent Dimension (De)

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

and ESR = Excavation Support Ratio

Preliminary empirical ground support results are presented in Figure 16.8 It can be seen that all excavations are located in the ‘unsupported’ category except intersection area. Based on modern industry practices, such ground conditions will require moderate ground support installations.

Figure 16.8 Depth from 0 to 500 m below surface - Support from Q System (Grimstad and Barton, 1993)



Underground excavations classed as 'permanent mine openings' (for example, decline, magazines, substations, and pump cuddies) have a representative ESR value of 1.6, which has been assumed for this study.

Nominal development widths of up to 5.0 m have been assumed by AMC for the Klaza project. The equivalent dimension (D_e) is therefore 3.1 (i.e. 5.0/1.6).

Preliminary ground support standards are outlined in Table 16.10 to Table 16.12. All excavations are projected to require weld-wire screen as surface support covering the back and haunches to within 1.5 m of the floor. For the shorter-life requirements of temporary development, the option exists to use split-sets or Swellex.

It should be noted that additional secondary ground support in the form of cables are projected to be required in wide spans such as intersections. For permanent mine development, #6 (6/8 inch) or #7 (7/8 inch) threaded rebar should be used. Poor ground conditions would require systematic bolting as well as shotcrete support.

AMC notes that the ground-support guidance provided here is preliminary in nature and ultimate requirements must be determined on-site and in consideration of actual ground conditions encountered.

Table 16.10 Recommended ground support for Mineralized rock, GRD and QFP rocks

Ground Conditions	Support System
Upper Bound (good)	2.4 m bolts spaced 1.5 m x 1.5 m
Typical and Lower Bound (fair)	2.4 m bolts spaced 1.2 m x 1.2 m

Table 16.11 Recommended ground support for GRD and QFP rocks (Intersections)

Ground Conditions	Support System
Upper Bound (Fair)	2.4 m bolts spaced 1.2 m x 1.2 m with shotcrete (4 - 5 cm), minimum cable length 6 m with 2.0 m spacing
Typical and Lower Bound (Poor)	2.4 m bolts spaced 1.2 m x 1.2 m with shotcrete (4 - 5 cm), minimum cable length 6 m with 2.0 m spacing

Table 16.12 Recommended ground support for Fault zones

Ground Conditions	Support System
Very poor	2.4 m bolts spaced 1.0 m x 1.0 m with fibre-reinforced shotcrete (>12 cm), minimum cable length 6 m with 2.0 m spacing

16.3 Mineral Resource model for mining

AMC used the Mineral Resource estimate block models for the Klaza and BRX zones (reference: rxh_brx_grd and rxh_brxe_grd) for evaluation of mining potential. The data used in the evaluation includes results of all drilling carried out on the Property to 30 September 2015. The evaluation work was carried out in Datamine™ software.

16.3.1 Gold equivalent

For reporting purposes an AuEQ (gold equivalent) field was calculated in the block model; the formula used for BRX and Klaza was:

*AuEQ = 1 * Au + Ag/106.5 + Pb/7.83 + Zn/14.45*

16.4 Cut-off value

The cut-off value is based on NSR, which accounts for all downstream processing costs. A net payable recovery for each metal was determined by 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 16.13.

A variable processing cost was determined by BCM and validated by AMC based on the arsenic feed grade for the BRX and Klaza zones. The cost is determined as follows:

*Processing cost = 12.76*1650/1500+17.16+(12.3*((100-12.3/As%0.6)*As%/(As%*4.44+3.37))/12.28*12.1/10.7),*

Where throughput driven cost is C$12.76 * throughput ratio, fixed cost is C$17.16 and variable cost is a function of the As% * C$12.30.

Due to the variable processing cost used for the BRX and Klaza zones, the cut-off value is based on an adjusted NSR where:

Adjusted NSR Cut-off = NSR - Process Cost (Variable) = Process Cost (Fixed) + Mining Cost + G&A

This is purely for reporting purposes to allow a fixed value to be reported against, in practice a variable total cost is used for determining mineralization above cut-off value.

Table 16.13 NSR cut-off value assumptions

Mining Factors		Open pit	Underground
Mining Dilution Klaza	%	33	20
Mining Dilution BRX	%	37	20
Mining Recovery	%	95	95
Operating Costs			
Weathered waste mining cost	C$/tonne	2.91	
Fresh waste mining cost	C$/tonne	3.96	82.75
Ore mining cost	C$/tonne	4.87	58.4
Incremental haulage cost per 5 m bench	C$/tonne	0.03	
Fixed & throughput processing cost (BRX and Klaza)	C$/tonne	31.20	31.20
Variable processing cost based on As	C$/tonne	$12.3*((100-12.3/As\%^{0.6})*As\%/(As\%*4.44+3.37)$	$12.3*((100-12.3/As\%^{0.6})*As\%/(As\%*4.44+3.37)$
BRX and Klaza G&A	C$/tonne	12	12
Processing Recovery			
Gold	%	94%	94%
Silver	%	90%	90%
Lead	%	85%	85%
Zinc	%	85%	85%
Lead Concentrate Grade	%	60%	60%
Zinc Concentrate Grade	%	48%	48%
Refining Costs			
Refining Charge Au	US$/oz Au	6.00	6.00
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$/US$	1.33	1.33
Gold Price	US$/oz	1,200.00	1,200.00
Silver Price	US$/oz	16.00	16.00
Lead Price	US$/lb	0.80	0.80
Zinc Price	US$/lb	0.85	0.85
BRX and Klaza Adjusted NSR cut-off value	**C$/t**	**50.7 (weathered) 49.7 (Fresh)**	**105**

* Payable gold, silver, lead and zinc take into account all downstream costs for transport, port handling, ocean freight and treatment charges

16.5 Open pit

16.5.1 Dilution and mining recovery factors

The thickness of the mineralized veins generally varies between about 0.6 m and 2.4 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 main veins within the proposed open pits and assuming a 30 cm dilution skin, representative of the selectivity of a 30 t excavator. The analysis resulted in an overall estimated dilution of 37% for BRX and 33% 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 16.14 Open pit mining dilution and recovery

	Mining dilution (%)	Mining recovery (%)
BRX	37	95
Klaza	33	95

16.5.2 Mining method

AMC proposes to mine the open pits using a conventional truck and excavator mining method. At this stage, AMC has assumed that a 5 m bench height would be adopted and mineralized material mined in two 2.5 m flitches to increase mining selectivity.

Three 30 t excavators would be required to mine waste and mineralized material. The productivity of one 30 t excavator has been estimated at 396 t/hr in mineralized material, 439 t/hr in oxide waste and 354 t/hr in fresh waste. The hydraulic excavators are planned to load 30 t articulated dump trucks. At peak production in Year 1, five trucks are projected to be required.

The majority of the material will require blasting. Only a marginal portion of the material mined may be free-dug by excavators. Proposed drilling parameters by material type are presented in Table 16.15. AMC estimates that three top hammer drills would be required for production and contour drilling along the side of the hills.

Table 16.15 Open pit drilling parameters

		Mineralized material	Waste fresh	Waste weathered
Bench height	m	5	5	5
Drillhole diameter	mm	75	75	75
Burden	m	2.25	2.4	3
Spacing	m	2.6	2.8	3.5
Sub-drill	m	0.75	0.75	0.75
Charge length	m	4.25	4.25	4.25
Stemming	m	1.5	1.5	1.5
Powder factor	kg / cum	0.6	0.5	0.3

Due to the projected short life of the open pit mines and the shallow mining depth, AMC has assumed that no pre-split blasting would be required.

16.5.2.1 Stockpile rehandling

AMC assumed that one front-end loader would be required to rehandle material from the Run of Mine pad (ROM pad) to blend stockpiles and limit grade variability to the mill. The front end loader would also be used in the pit to provide support to the hydraulic excavators when required.

16.5.2.2 Ancillary equipment

AMC has determined ancillary equipment numbers as a relation to the total production fleet numbers and based on its experience. Activities considered when estimating the auxiliary equipment fleet include:

- Pioneering road establishment
- Clean up of digging and drilling areas
- Waste dump maintenance
- Dust suppression and maintenance of haul roads
- Servicing of the production fleet
- Topsoil clearing

The proposed support fleet consists of two 300 hp dozers for clearing topsoil, forming waste dumps and stockpiles, and creating surface roads to the waste dumps and open pits at the start of the operation. During the production phase, the dozers will provide support in the pits and at the waste dumps. One grader and one water truck are planned for road maintenance, re-contouring of rehabilitated areas and dust prevention.

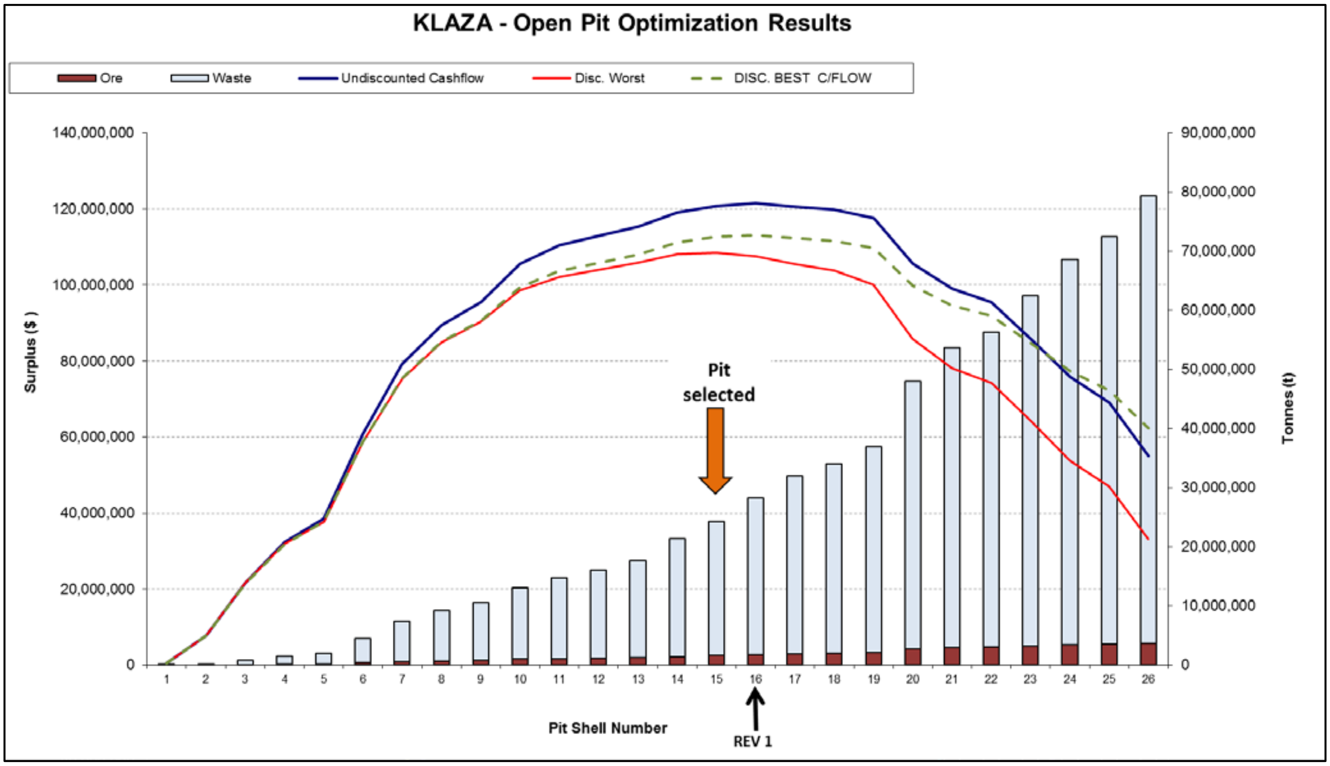
16.5.3 Pit design and selection

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.

16.5.3.1 Klaza ultimate pit selection

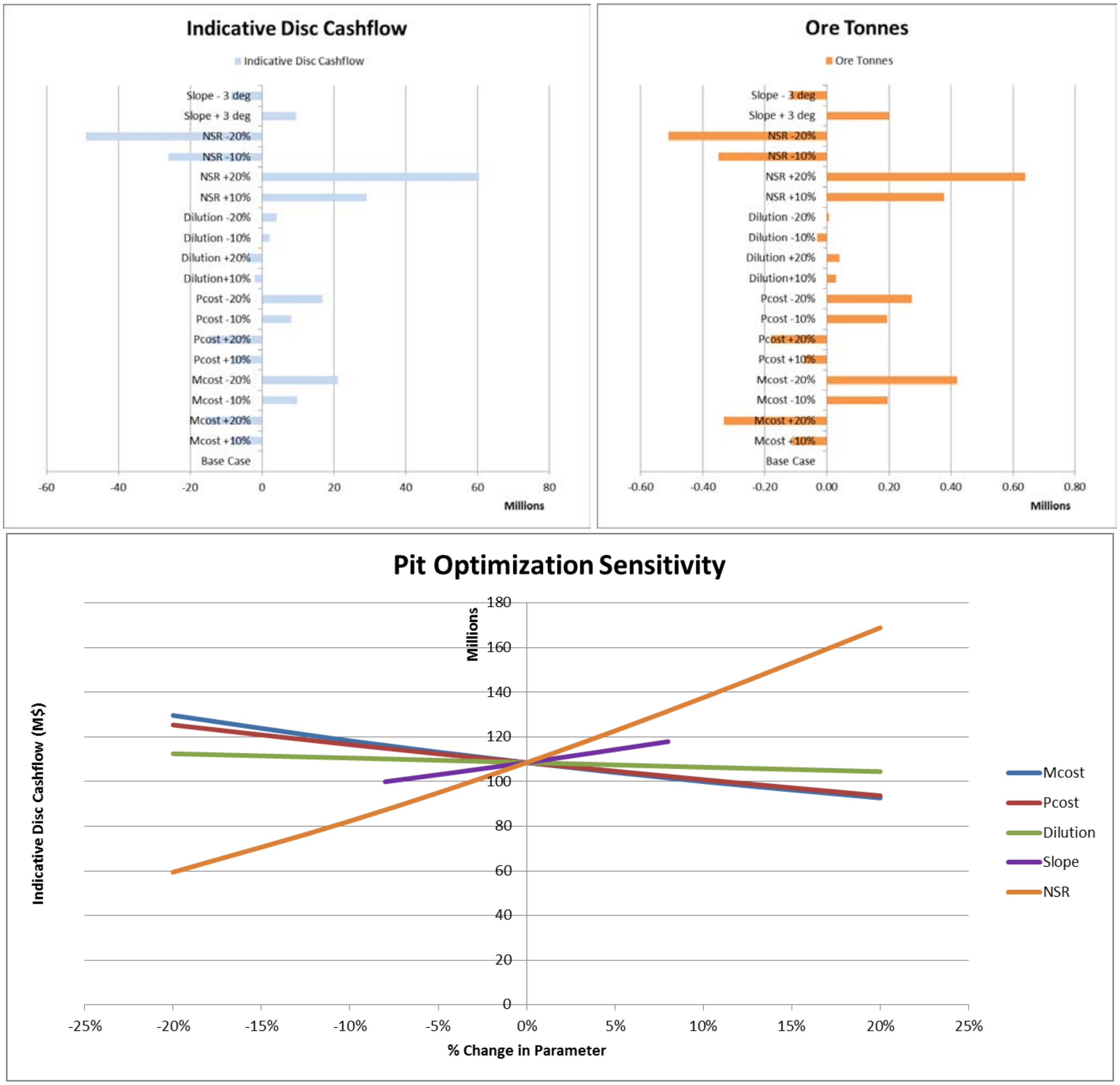
The optimization results for the Klaza zone are shown in Figure 16.9. The pit shell corresponding to the 100% revenue factor is pit shell 16. Pit shell 15 was selected as the basis for the mining schedule based on the results of the open pit to underground interface analysis.

Figure 16.9 Optimization Results - Klaza



A sensitivity analysis of the selected open pit optimization was conducted at +/-10%, +/-20% for processing costs, mining costs, dilution and NSR. Overall slope angle sensitivity of +/-3° was also evaluated; the results are shown in Figure 16.10. The pit size, tonnes of mineralized material and discounted (5%) cash flow are most sensitive to NSR, followed by mining and processing costs.

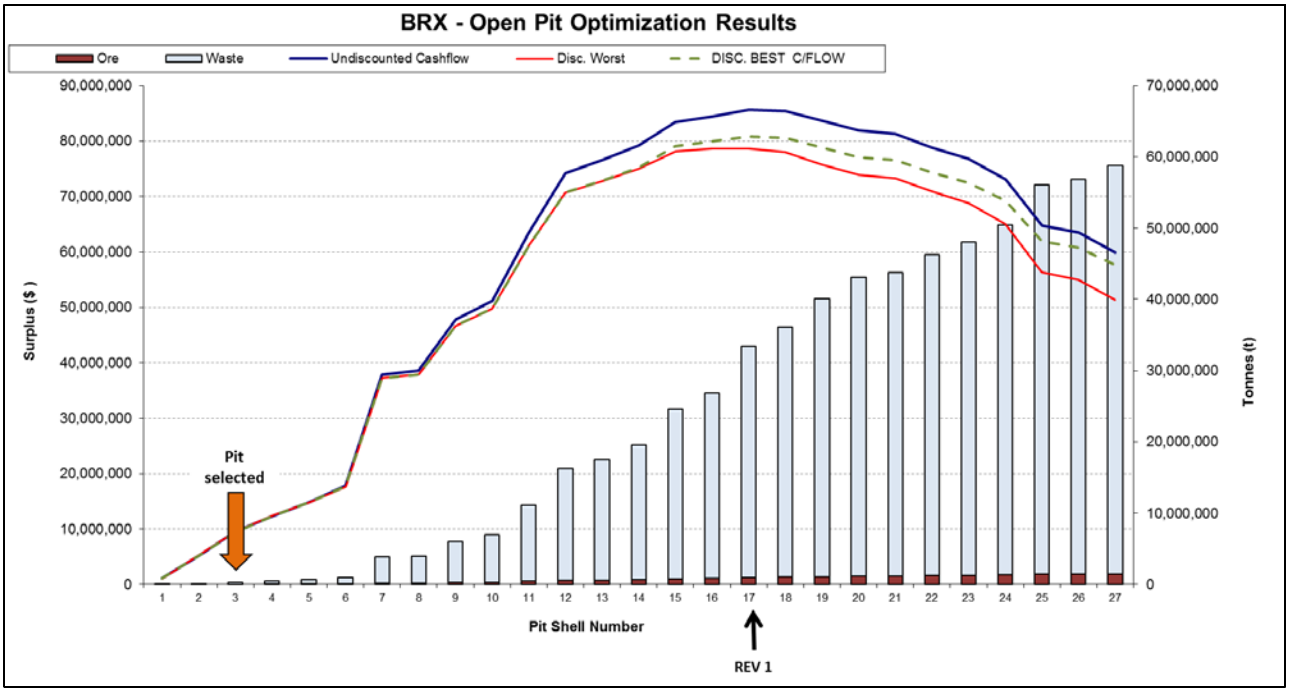
Figure 16.10 Pit optimization sensitivity analysis – discounted at 5%



16.5.3.2 Western BRX ultimate pit selection

The optimization results for the Western BRX zone are shown in Figure 16.11. The pit shell corresponding to the 100% revenue factor is pit shell 17. However, based on the open pit to underground interface analysis, described in Section 16.6, more value is generated by mining the fresh mineralization using an underground mining method. Pit shell 3 was therefore selected to define the limits of open pit mining.

Figure 16.11 Optimization Results - Western BRX



16.5.3.3 Pit design

Conceptual pits were designed based on the selected pit optimizations. The conceptual designs for the Western BRX and Klaza zones are presented in Figure 16.12 and Figure 16.14 respectively. Representative pit design sections displaying Adjusted NSR (ADNSR) values for the mineralized material are presented in Figure 16.13 and Figure 16.15.

Figure 16.12 Klaza pit design

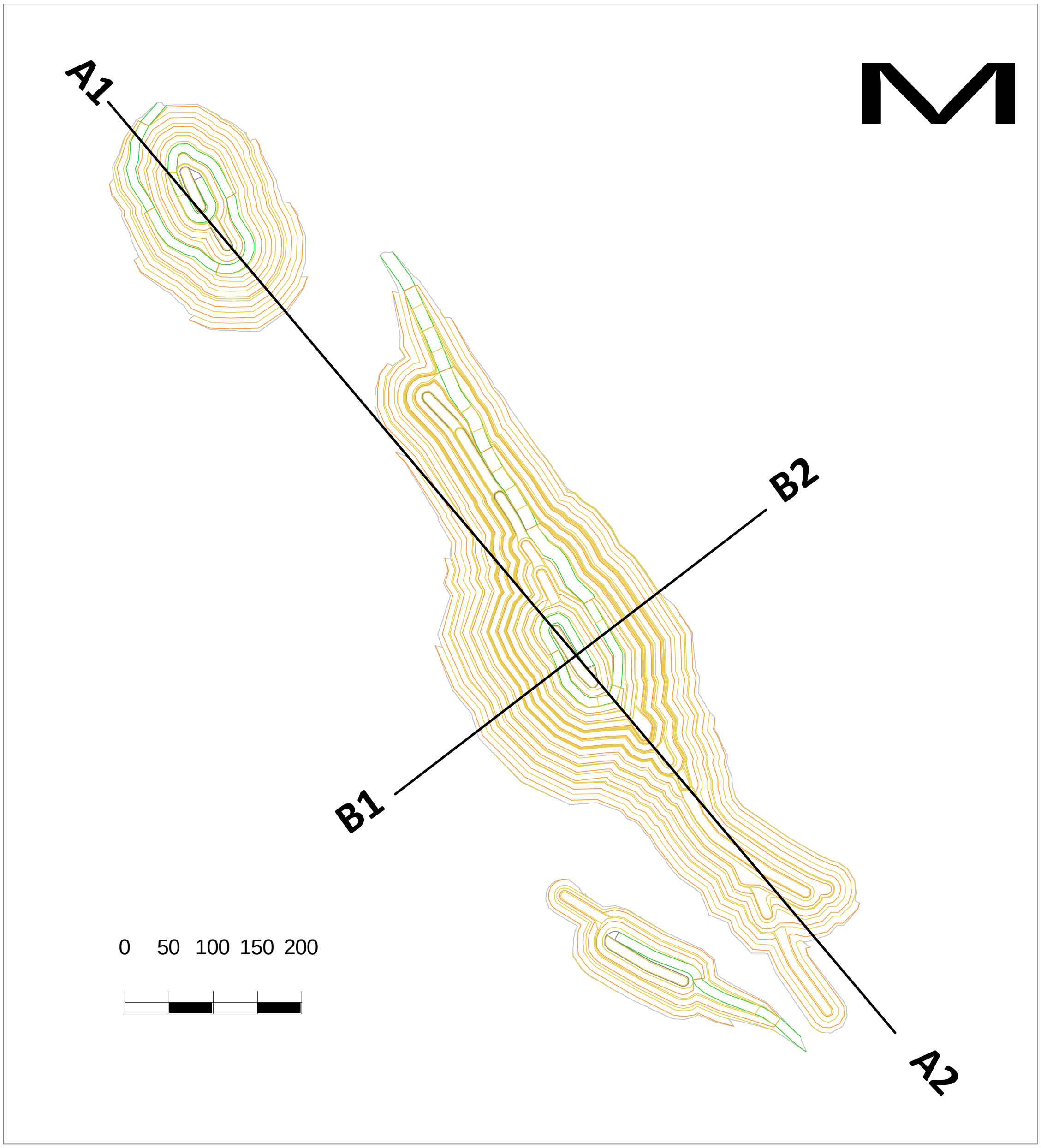
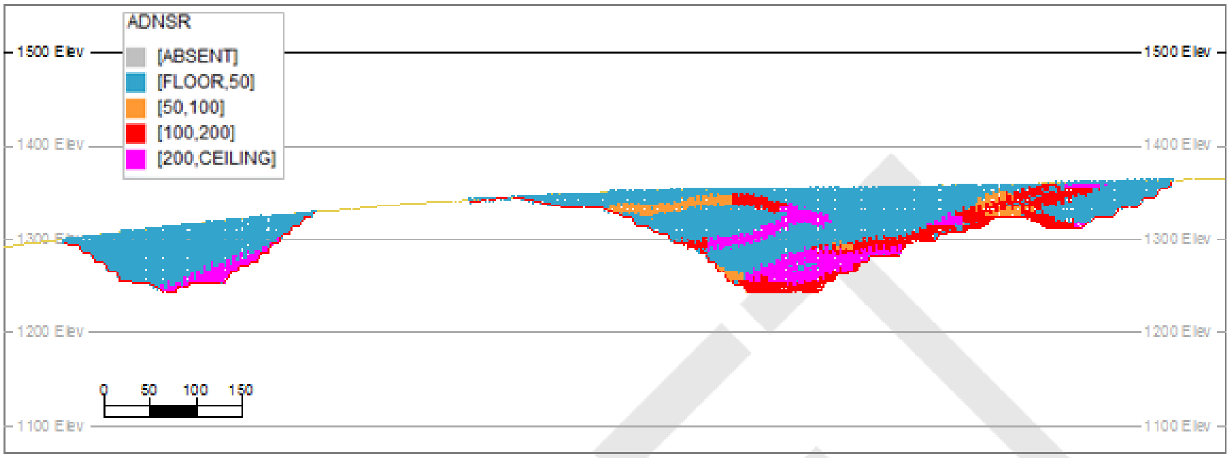


Figure 16.13 Klaza pit sections showing ADNSR in C$/t

Klaza section A1-A2



Klaza section B1-B2

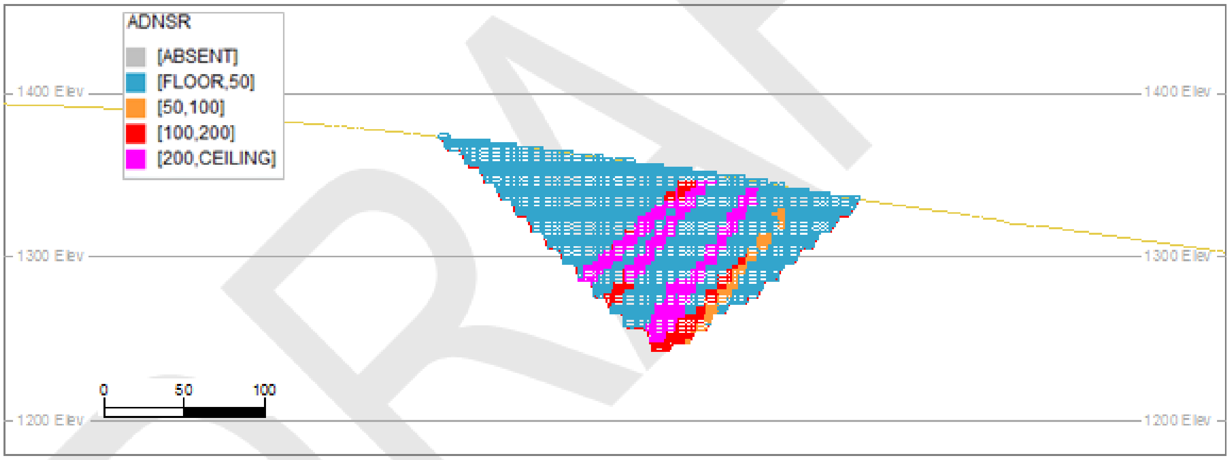


Figure 16.14 Western BRX pit design

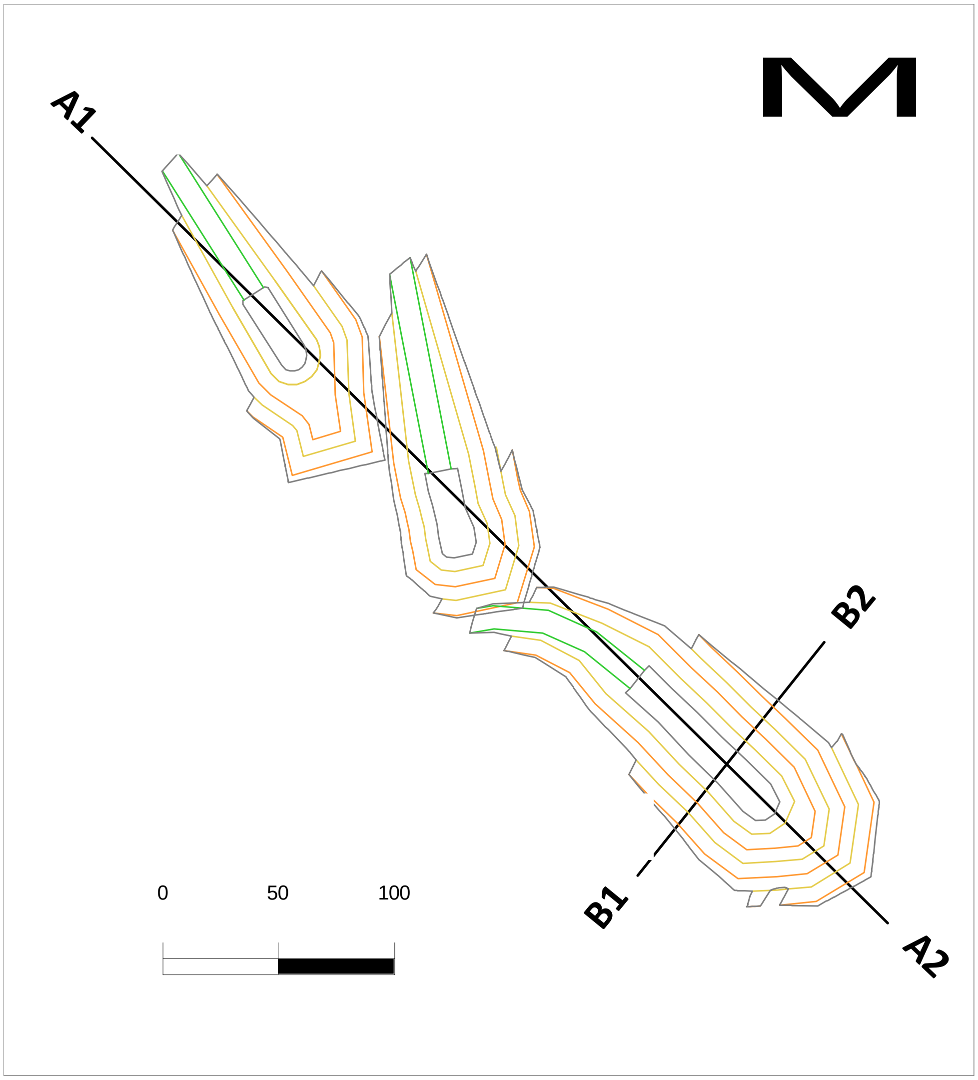
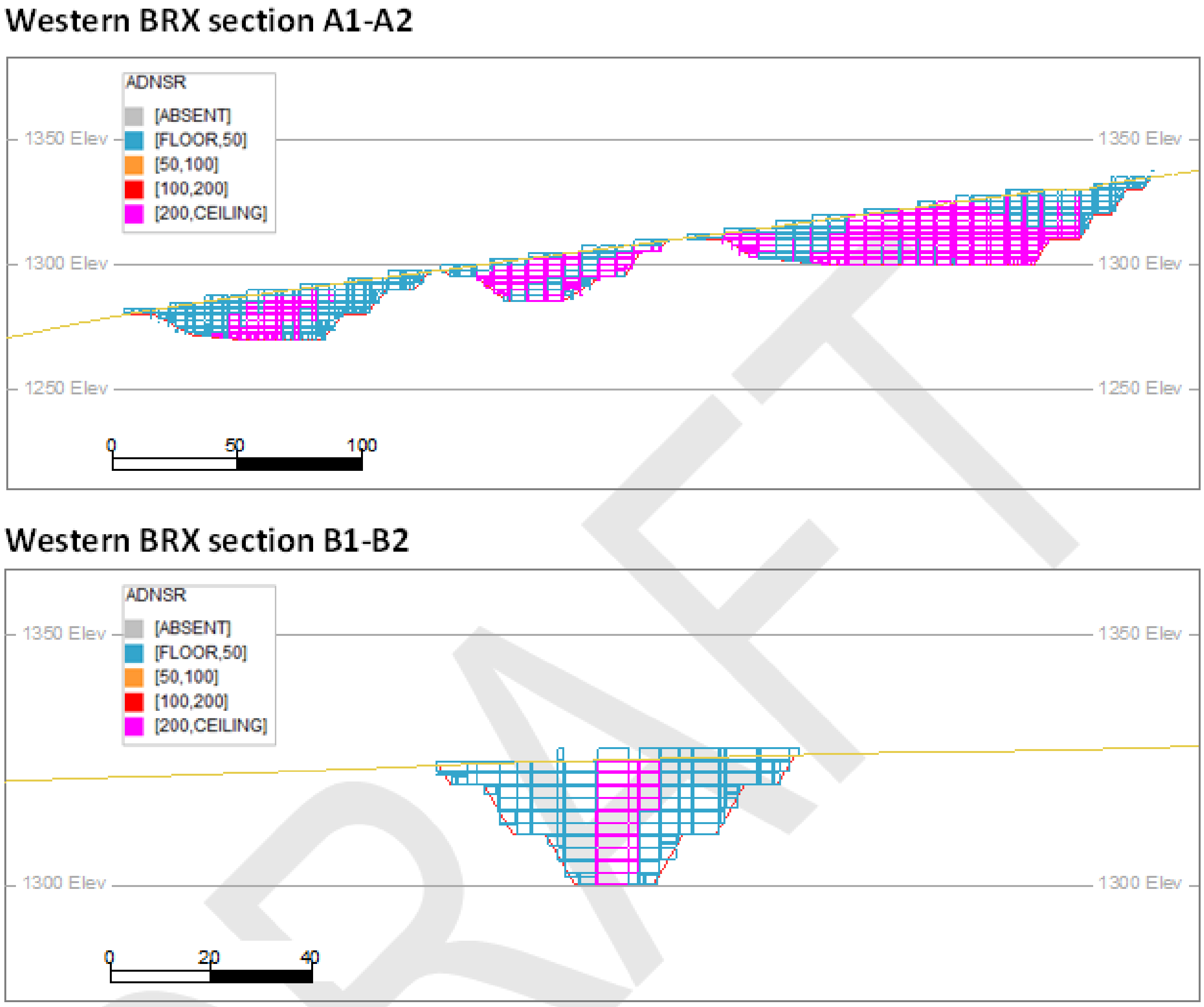


Figure 16.15 Western BRX sections showing ADNSR in C$/t



Indicative tonnes and grades contained within the conceptual pit designs are presented in Table 16.16.

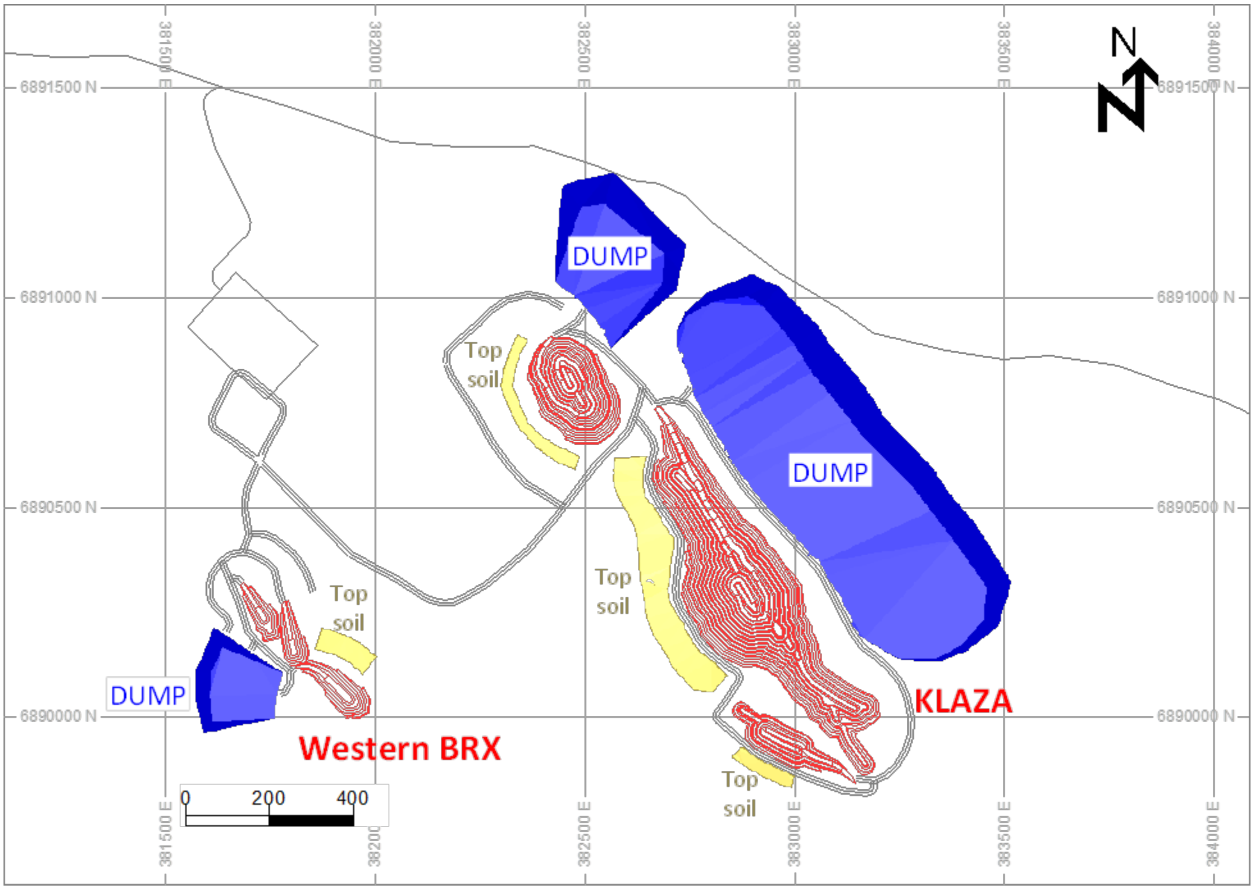
Table 16.16 Open pit projected tonnes and grades

Zone	Mineralized Material Tonnes (Mt)	Waste Material Tonnes (Mt)	ADNSR (C$)	AuEQ (g/t)	Au (g/t)	Ag (g/t)	Pb (%)	Zn (%)	Arsenic (%)
Klaza	1.3	16.9	202	4.0	3.3	52.5	0.7	1.1	0.4
Western BRX	0.04	0.6	382	7.8	6.7	92.3	0.6	0.8	1.7
Total	**1.31**	**17.5**	**205**	**4.0**	**3.4**	**51**	**0.7**	**1.1**	**0.5**

16.5.4 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 37 degree rill angle and to a maximum height of approximately 55 m. Top soil stockpiles have been designed to handle the volume generated by removing 30 cm from the surface area of the pits.

Figure 16.16 Layout of open pit infrastructure



16.5.5 Open pit equipment

The open pit operations are projected to work on a four panel, two shift roster. Features of this roster are:

- 20 hours per day, 365 days per year, two shifts per day
- 12 hours per shift
- 4 days on, 4 days off

The above roster and the projected mining schedules (see Section 16.8) were used to derive peak equipment requirements. Open pit primary equipment requirement at peak production is summarized in Table 16.17.

Table 16.17 Equipment selection

Equipment type	Peak No. required
30 t Excavator	3
Front-end loader	1
28 t Articulated dump truck	5
Top-hammer drill	3
Dozer (300HP)	2
Grader (CAT 140M)	1
Water truck	1
Total	25

16.5.6 Open pit labour and supervision

Contractor personnel numbers were estimated for all mining departments, including management and supervision, technical staff, operators and maintenance. The total number of personnel required was estimated based on the production throughput of the operation and the equipment numbers.

Total operator numbers were calculated for the number of machines on site at any given time. Equipment such as trucks, excavators, drills and dozers were considered to be manned at all times. The number of maintenance personnel was derived using a maintenance labour factor of 0.5 for the main pieces of equipment.

Contractor management and senior staff were assumed to work on a 5-day week, Monday to Friday.

Unskilled and some skilled positions are anticipated to be filled by the local workforce. AMC has assumed that the workforce will be accommodated in nearby Carmacks.

The peak requirements for the open pit workforce are summarized in Table 16.18.

Table 16.18 Open pit contractor workforce

Open pit workforce	Peak No. required
Fixed - Production	
Project Manager	1
Mine Superintendent	1
Clerk	1
Safety/Training Coordinator	1
Storeman	1
Mine Production Supervisor	4
Sub Total	**9**
Fixed - Maintenance	
Maintenance Supervisor	1
Leading Hand Mechanic / Fitter	1
Apprentice	1
Boilermaker	1
Light Vehicle Mechanic	1
Sub Total	**5**
Mining Operations - Variable numbers	
Truck operator	20
Excavator operator	12
Ancillary equipment operator	16
Drill operator	12
Fitter	22
Shot firer	1
Shot crew	1
Sub Total	**84**
Total Personnel	**98**

16.6 Open pit to underground interface

In order to optimize the open pit to underground interface, AMC has generated incremental pit shells and determined open pit value on 30 m increments and compared this to the value generated by underground stopes on a level by level basis. If the pit value, after accounting for operating and development costs, is higher than the value of the underground, the pit is selected. This analysis is repeated as the pit deepens until the optimal depth is reached.

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.

It was assumed that a 30 m crown pillar that will be partially extracted (70%) at the end of the mine life will be left beneath the BRX pits. With the Klaza pits it was assumed that the process tails will be placed in the pits after depletion. For Central Klaza, AMC has assumed that a double lift pillar will be left (60 m) fully intact beneath the pit. AMC recommends a more detailed evaluation of the crown pillar in future study work. Options such as use of a cemented plug at the bottom of the pit could facilitate improved recovery of the crown pillar. In any event, partial recovery may be achieved by drilling and blasting 15 m to 20 m upholes on a retreat basis.

16.7 Underground mining

16.7.1 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, 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.

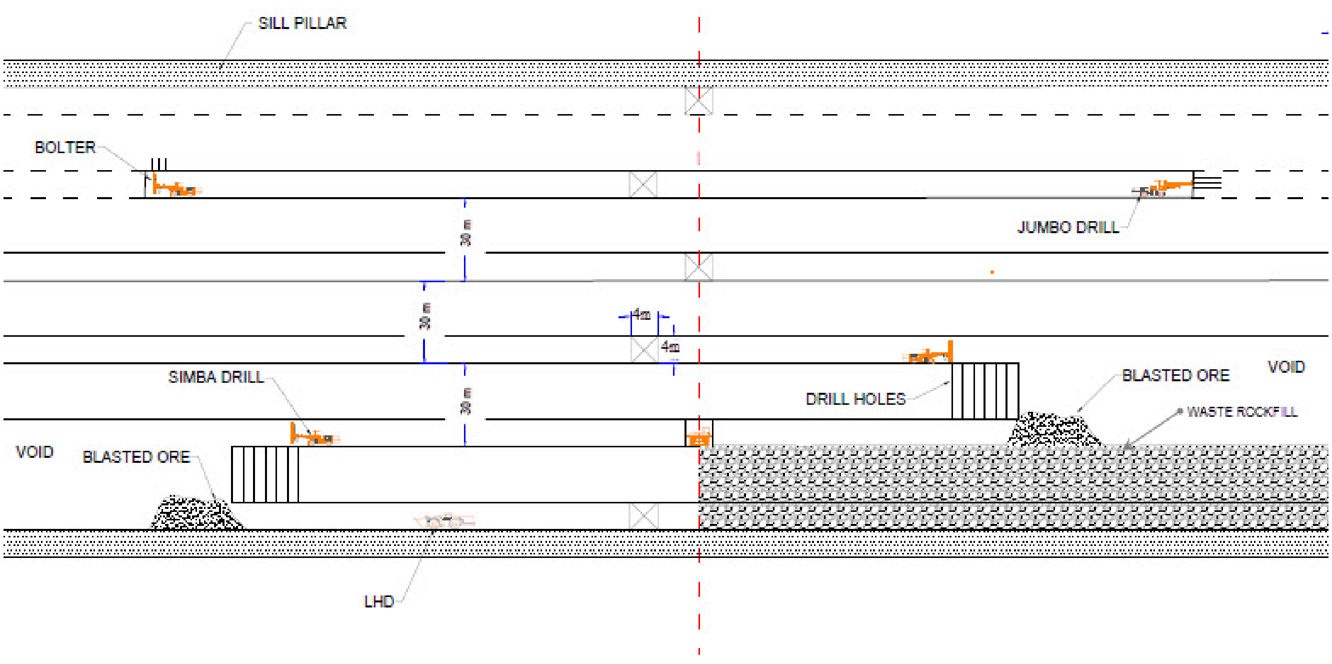
Relative to current overall knowledge of the Klaza deposits, AMC considers that the most appropriate method is mechanized longhole open stoping. Longhole open stoping methods generally provide relatively high productivity within reasonable cost limits. It is also the most common method applied to narrow veins requiring a fair amount of selectivity, whilst maintaining reasonable production rates.

Based on geotechnical recommendations, an inter-level spacing of 30 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 the underground 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



16.7.2 Dilution and mining recovery factors

Geotechnical evaluation of dilution for longhole stopes using the equivalent linear overbreak slough (ELOS) technique indicates less than 0.5 m from the hangingwall. 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 mineralized rock from the rock fill floor.

AMC has applied a dilution of 0.25 m to the hangingwall and 0.1 m to the footwall, with mucking dilution assumed to be 0.3 m. A mining recovery factor of 95% 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 C$105 for the BRX and Klaza zones. When generating stopes, a minimum stoping width of 2 m was used. For MSO design, a dilution skin of 0.25 m on the hangingwall and 0.1 m on the footwall 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 floor mucking dilution was then added to the reported tonnes and grade.

16.7.3 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 vertical tonnes per metre to project production ranges.

Production rate based on Taylor's rule of thumb, is estimated at 330 ktpa for BRX and 350 ktpa for Klaza for a combined annual production rate of 680 ktpa.

Annual Production Rate = 5 * 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 200 to 250 ktpa for BRX and 300 to 350 ktpa for Klaza, for a total annual production rate of 500 to 600 ktpa.

AMC recommends that the BRX and Klaza deposits be considered for mining as two virtually independent operations at a combined production rate of 550 ktpa. This production rate is well supported by the detailed production scheduling.

The Eastern BRX zone is located approximately 1 km east and along strike of the Western BRX zone. The mineralization is different to both the Western and Central BRX and Klaza zones, with higher grades of copper. Additional work is required to fully define the economics of this zone and further discussion on the potential of Eastern BRX is provided in Section 24. The Eastern BRX zone is not included in the economic assessment of the Western and Central BRX and Klaza zones.

16.7.4 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 value stopes were selected, the wireframes were combined into stopes 24 m in length. Stopes overlapping with the pit were removed and crown pillars of 30 m and 60 m respectively for BRX and Klaza were included in the design. For BRX, AMC has assumed that the crown pillar will be partially extracted (70%) at the end of the mine life.

The Inferred Mineral Resource associated with the projected economic stopes is summarized in Table 16.19 by zone. The total underground mineralization is estimated to be 5.1 Mt at grades of 3.31g/t Au, 84g/t Ag, 0.7% Pb and 0.8% Zn with an NSR value of C$209.

Table 16.19 Underground Inferred Mineral Resource

Zone	Tonnes	NSR (C$)	AuEQ (g/t)	Au (g/t)	Ag (g/t)	Pb (%)	Zn (%)	Arsenic (ppm)
Western Klaza	662,505	205	4.38	3.14	107	0.4	0.5	4,113
Central Klaza	1,446,887	145	3.11	2.44	43	0.5	0.6	3,154
Western BRX	1,872,106	285	6.10	4.93	82	0.8	0.9	6,536
Central BRX	1,157,091	167	3.54	1.87	123	1.0	1.0	4,242
Total	**5,138,589**	**209**	**4.46**	**3.31**	**84**	**0.7**	**0.8**	**4,755**

16.7.5 Underground development

The BRX and Klaza veins are approximately parallel and 800 m apart, and as such separate declines were designed for the Klaza and BRX. Access to the Klaza zone underground mine would be via a 5 m by 5 m decline and crosscuts on each level (West and Central). The Central Klaza zone has several stopes further east along strike and these are accessed via an independent decline (Eastern Klaza). Levels are spaced at a vertical distance of 30 m floor to floor. Development (4 m by 4m) was designed to follow the vein along strike from a central access crosscut. The Klaza decline commences from the portal on surface and splits into two declines to access the Western Klaza and Central Klaza areas. Main access to the BRX zone has a similar design with a main (5 m by 5 m) access decline that splits into two declines accessing 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	5,309	1,096	511	588	8,889	2,848
BRX	6,494	1,236	749	767	11,712	2,809
Total	11,803	2,332	1,260	1,355	20,601	5,657

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	36.5 m
Raises to surface for ventilation	4 m diameter
Allowance for sumps, loading bays	25% of crosscut length
Allowance for passing bays	13% of decline length (15 m every 150 m)

Isometric views of the BRX and Klaza proposed underground mines are provided in Figure 16.8 and Figure 16.9.

Figure 16.18 BRX underground mine design

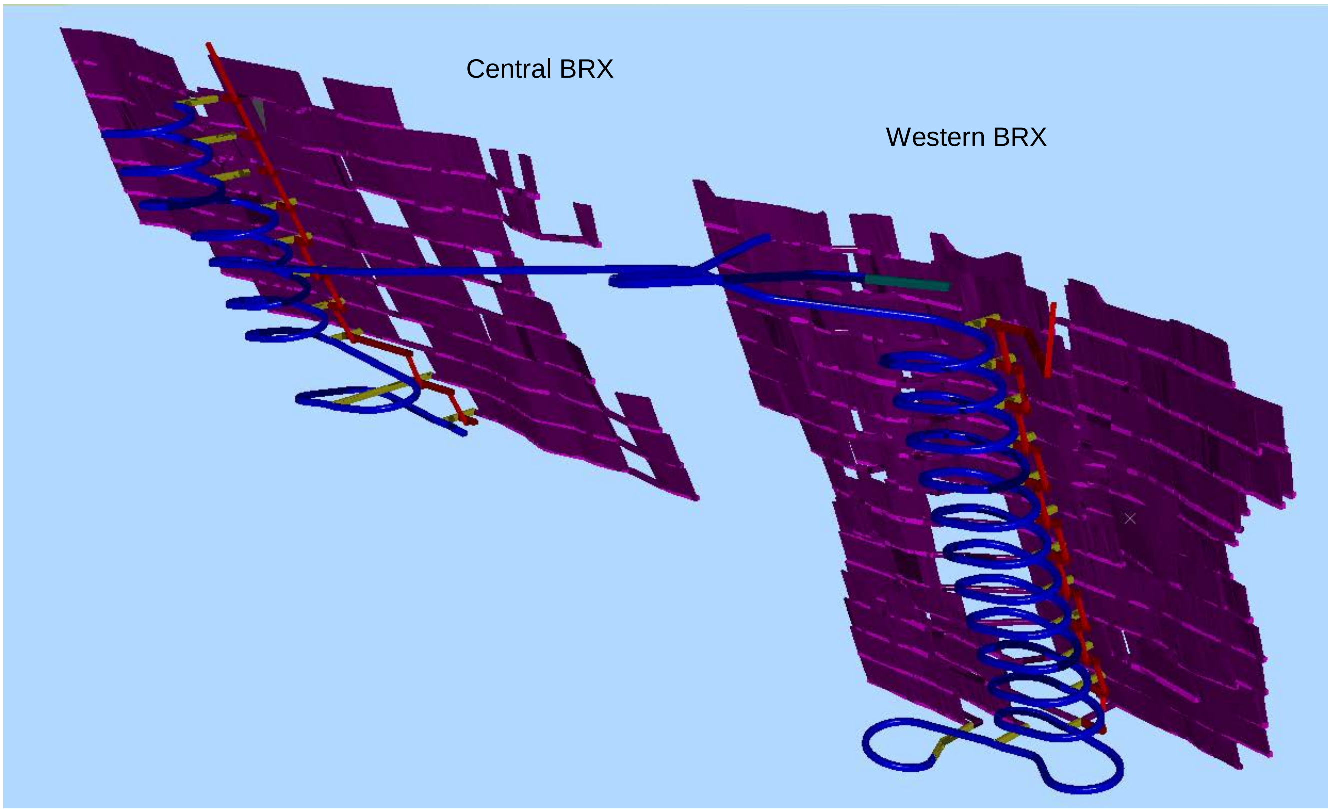
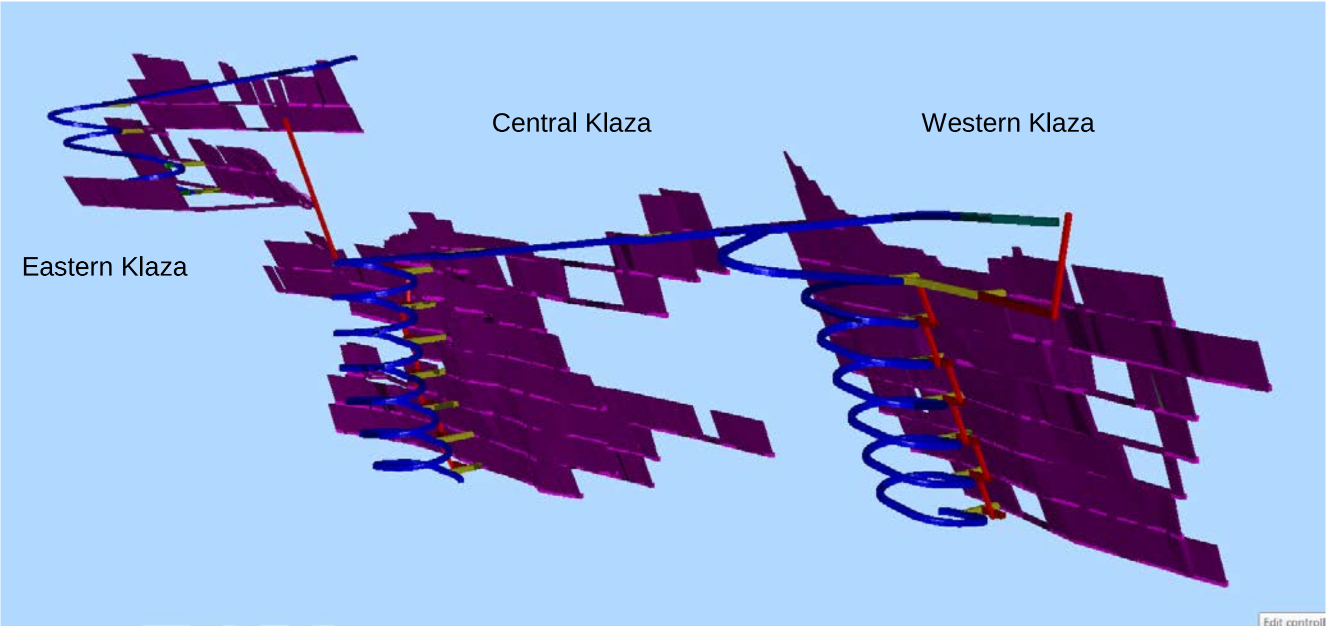


Figure 16.19 Klaza underground mine design



16.7.6 Proposed infrastructure

The proposed underground mine services will include a small maintenance shop for minor and urgent repairs, fuel and lubricant storage, and a small explosives magazine at each mine.

Compressed air will be supplied by mobile electric compressors. The compressors will be relocated to active mining levels as needed.

During development the decline will be equipped with power for distribution underground as well as a four inch pipeline for mine service water and a six inch pipeline for dewatering. Telecommunications will be provided by a conventional leaky feeder system.

16.7.7 Ventilation

AMC has undertaken a preliminary estimate of the ventilation requirements based on the underground equipment rating and anticipated utilization. This estimate has been checked against benchmark data for ventilation quantities. 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 at the portal, with the heat ducted to the intake airflow.

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, a combined total of approximately 290m³/s is planned for ventilation of both 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.

The primary surface exhaust fans required for the ventilation system are as follows:

- Western BRX exhaust fan – 250 hp

- Central BRX exhaust fan – 100 hp
- Western Klaza exhaust fan - 150 hp
- Central Klaza exhaust fan – 200 hp
- Eastern Klaza exhaust fan – 50 hp

The proposed ventilation strategies for BRX and Klaza are shown in Figure 16.20 and Figure 16.21.

Figure 16.20 Proposed BRX ventilation system

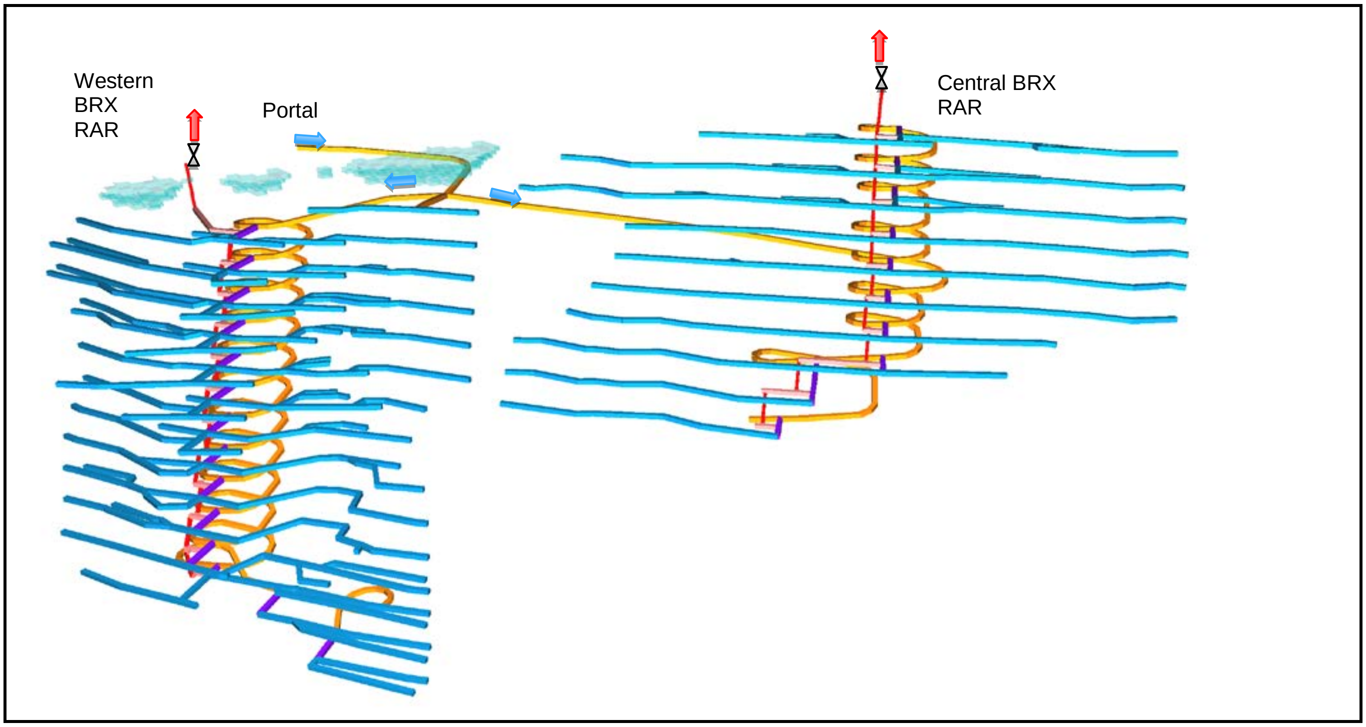
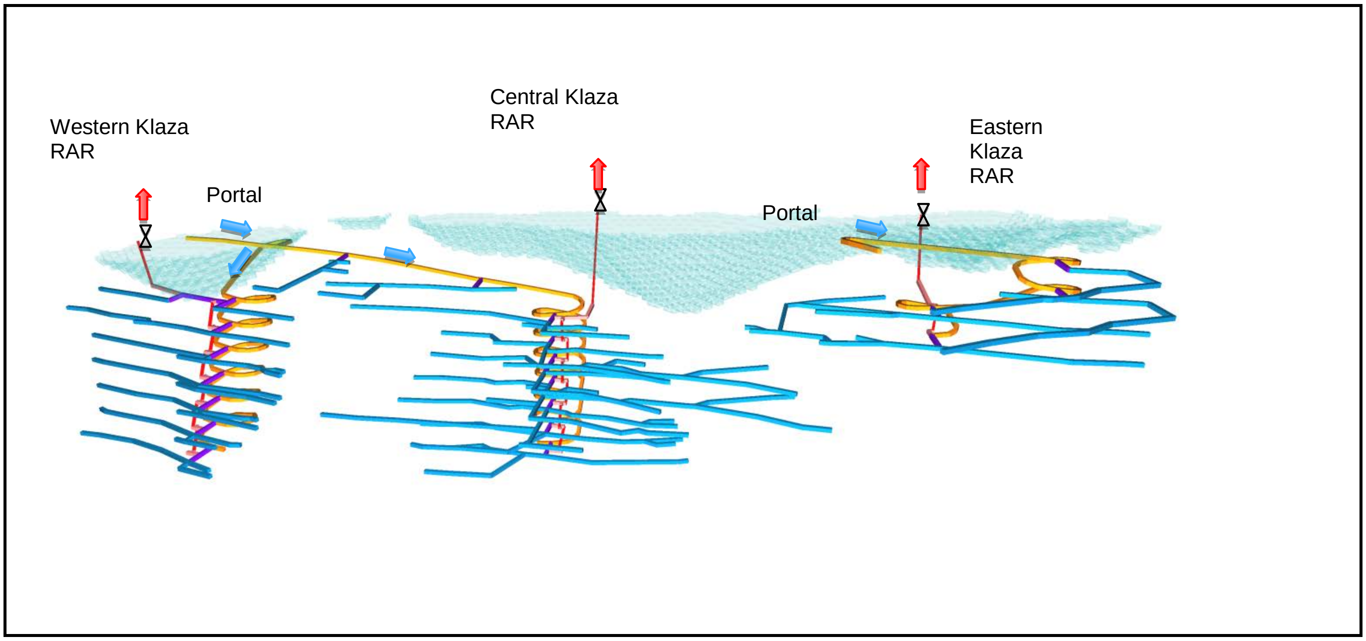


Figure 16.21 Proposed Klaza ventilation system



16.7.8 Secondary egress

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 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.

16.7.9 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 550 ktpa is summarized in Table 16.22.

Table 16.22 Equipment selection

Underground mobile equipment	No of units
Production drill rig	4
2 Boom Jumbo Drill - Development	4
Diesel LHD - Production	3
Diesel LHD - Development	3
UG Haul Truck (40 t)	5
UG Water Truck	1
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
Mobile compressors	5

16.7.10 Underground labour and supervision

Labour will be sourced from the local community and area around Klaza. The mine workforce will be encouraged to live in Carmacks and a daily bus service will be provided to drive the workforce 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 Engineer	2
Production Engineer	2
Ventilation Engineer	1
Senior Surveyor	1
Surveyor	2
Senior Geologist	1
UG Mine Geologist	2
Technicians (OP)	2
Sub-Total	**17**

The total labour and supervision personnel numbers including technical services are summarized in Table 16.24.

Table 16.24 Underground labour and supervision

Management	Number
Mining Manager	1
Administration Assistant	1
Clerk	1
Supervision	
Mine General Foreman	2
Shift Supervisor	12
Labour	
Workplace Trainer/ Safety	3
Jumbo Operator	12
Production Driller	12
Charge / Scale	9
Loader operator	18
Service Crew	15
Grader Driver	3
Truck Driver	15
Nipper	6
Maintenance	
Senior Maintenance Engineer	1
Maintenance Planner	2
Mechanical Foreman	2
Electrical Foreman	2
Leading Hand Mechanics	3
Shift Mechanics	9
Electricians	3
Millwright	3
Shift Serviceman	6
Electrical Leading Hand	3
Shift Electricians	6
Pump Fitters	3
Electrical Assistant	3
Total Staff (Including technical services)	**41**
Total Labour	**132**
Total	**173**

16.8 Projected life of mine (LOM) development and production schedule

16.8.1 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. In addition, a focus to get the open pits mined early in the mine life was maintained in order to enable placement of process tailings in the pits. The pit and underground areas are ranked by value as shown in Table 16.25.

Table 16.25 Ranking of zone value

Zone	Value (C$/t)	Ranking
Klaza Pit 1	142	3
Klaza Pit 2	82	4
Klaza Pit 3	39	7
Western BRX pit	254	1
Klaza West UG	72	5
Klaza Central/East UG	14	8
BRX West UG	159	2
BRX Central UG	41	6

Using the rankings shown above and the other referenced considerations, the planned initial development and mining sequence is to mine out the shallow BRX pits while the main Klaza pit and main access declines to the underground mines are developed. Underground development will target opening up the high grade Western BRX and Western Klaza zones as the priority. Underground mineralization is projected to be mined at a rate that, together with open pit production, will generally fill the mill from around Year 2 through to Year 10. The Central Klaza zones and Eastern Klaza zones and the Central BRX zone will be brought into production as the open pit is depleted so as to maintain full production at 550 ktpa.

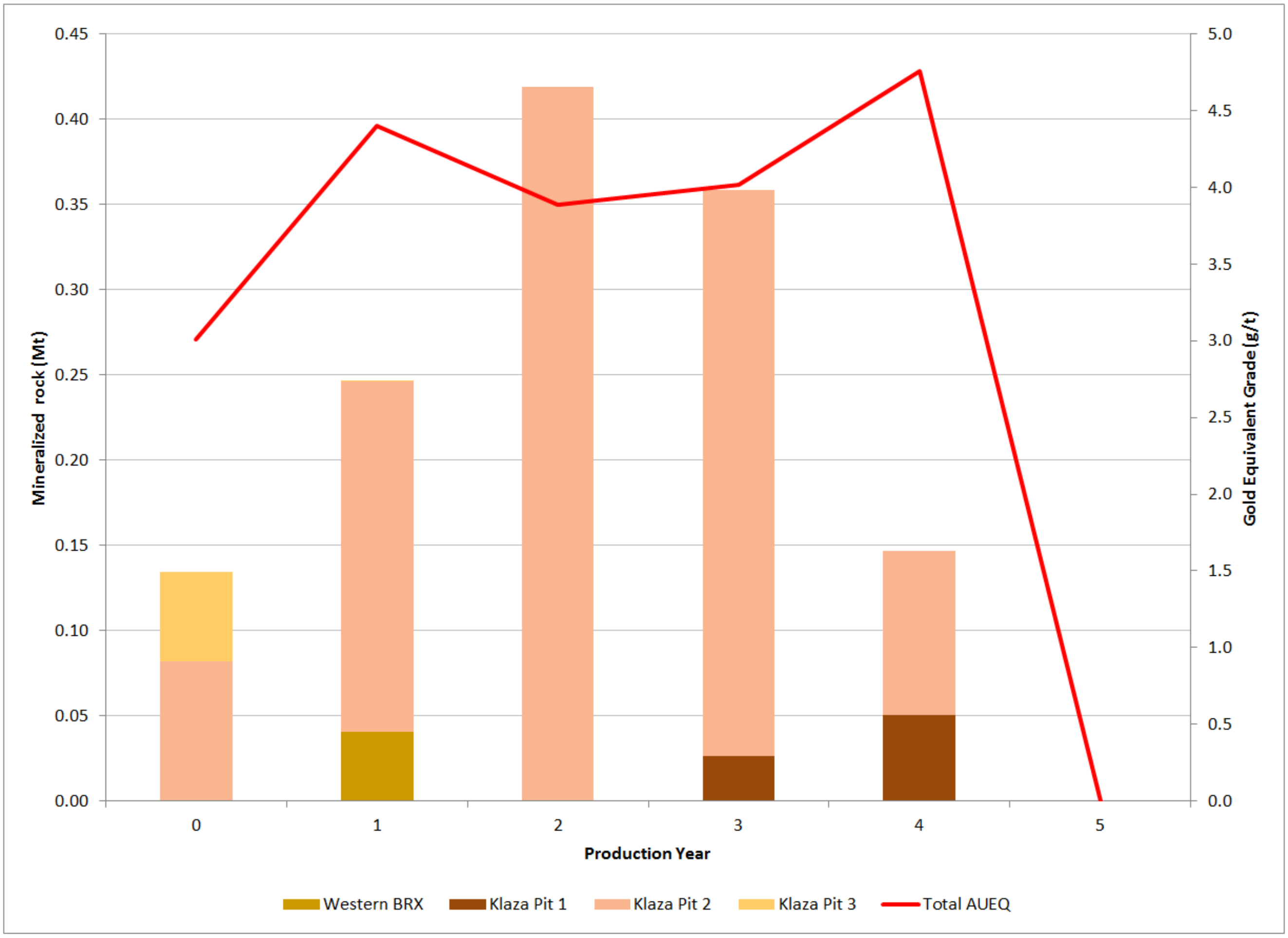
16.8.2 Conceptual open pit production schedule

The proposed open pit production schedule extends over a five year period and is summarized in Table 16.26 and Figure 16.22.

Table 16.26 Conceptual open pit production schedule

OP Production	YR0	YR1	YR2	YR3	YR4	YR5	OP Totals
Waste (Mt)	4.3	4.3	4.0	3.9	1.0		17.5
Mineralized rock (kt)	134	247	419	358	147		1,305
Mill feed (kt)		380	373	245	144	163	1,305
Mill feed AuEQ (g/t)		3.9	4.2	5.0	4.8	1.7	4.0
Mill feed Au (g/t)		3.4	3.6	4.1	3.7	1.5	3.4
Mill feed Ag (g/t)		39	46	69	99	21	51
Mill feed Pb (%)		0.6	0.8	0.9	0.7	0.3	0.7
Mill feed Zn (%)		1.2	1.2	1.2	0.9	0.3	1.1
Mill feed As (%)		0.6	0.5	0.5	0.4	0.2	0.5

Figure 16.22 Projected open pit production



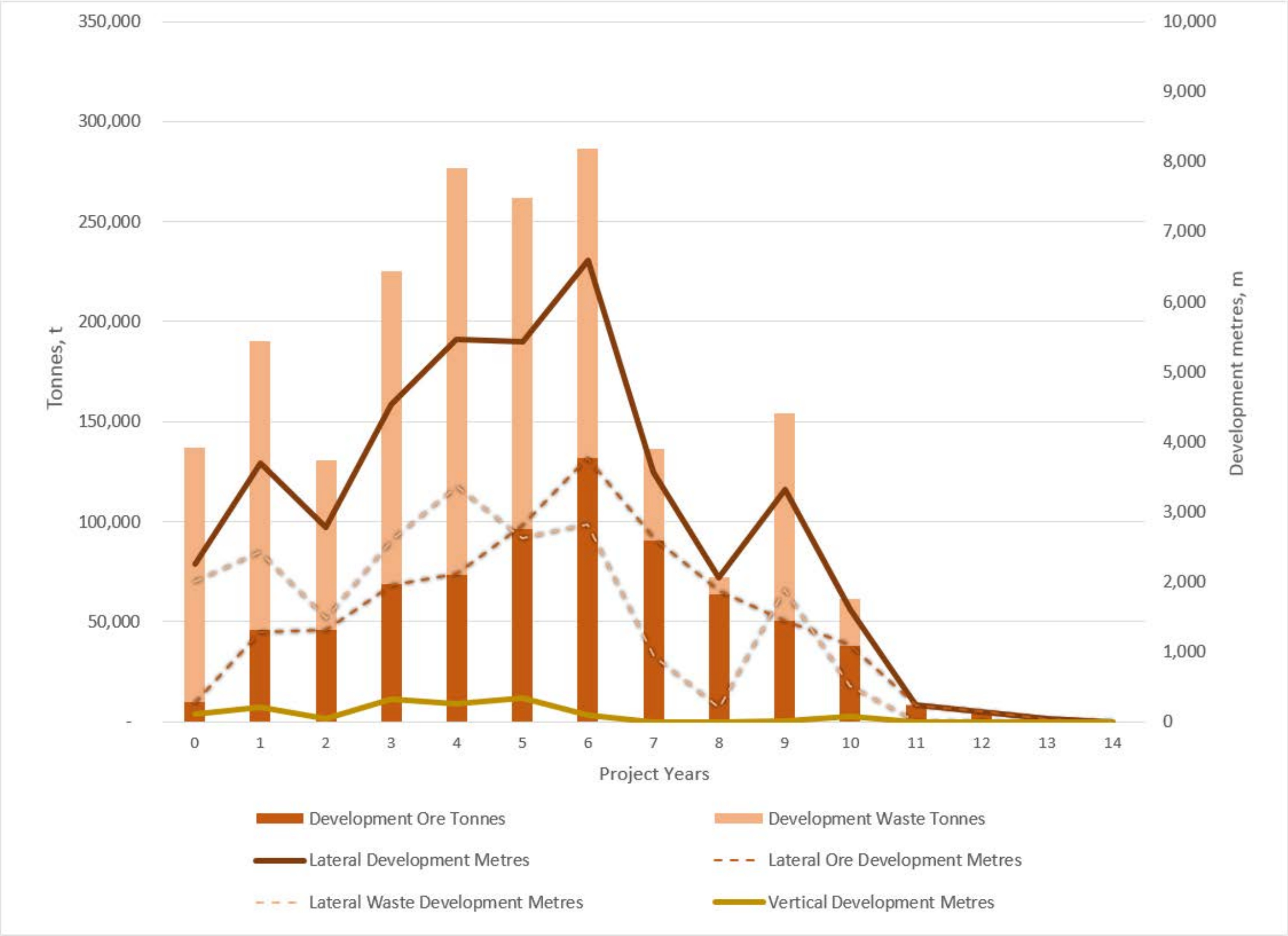
16.8.3 Projected underground development schedule

The projected underground development schedule is summarized in Table 16.27 and Figure 16.23.

Table 16.27 Proposed underground development schedule

Description	YR0	YR1	YR2	YR3	YR4	YR5	YR6	YR7
Access development (m)	1,990	2,418	1,468	2,582	3,359	2,624	2,824	932
Vein development (m)	264	1,272	1,314	1,946	2,108	2,796	3,762	2,626
Vertical development (m)	115	199	33	318	251	341	98	
Description	**YR8**	**YR9**	**YR10**	**YR11**	**YR12**	**YR 13**		**Total**
Access development (m)	200	1,887	501	9				20,793
Vein development (m)	1,860	1,433	1,100	228	145	34		20,887
Vertical development (m)		10	69					1,435

Figure 16.23 Projected underground development



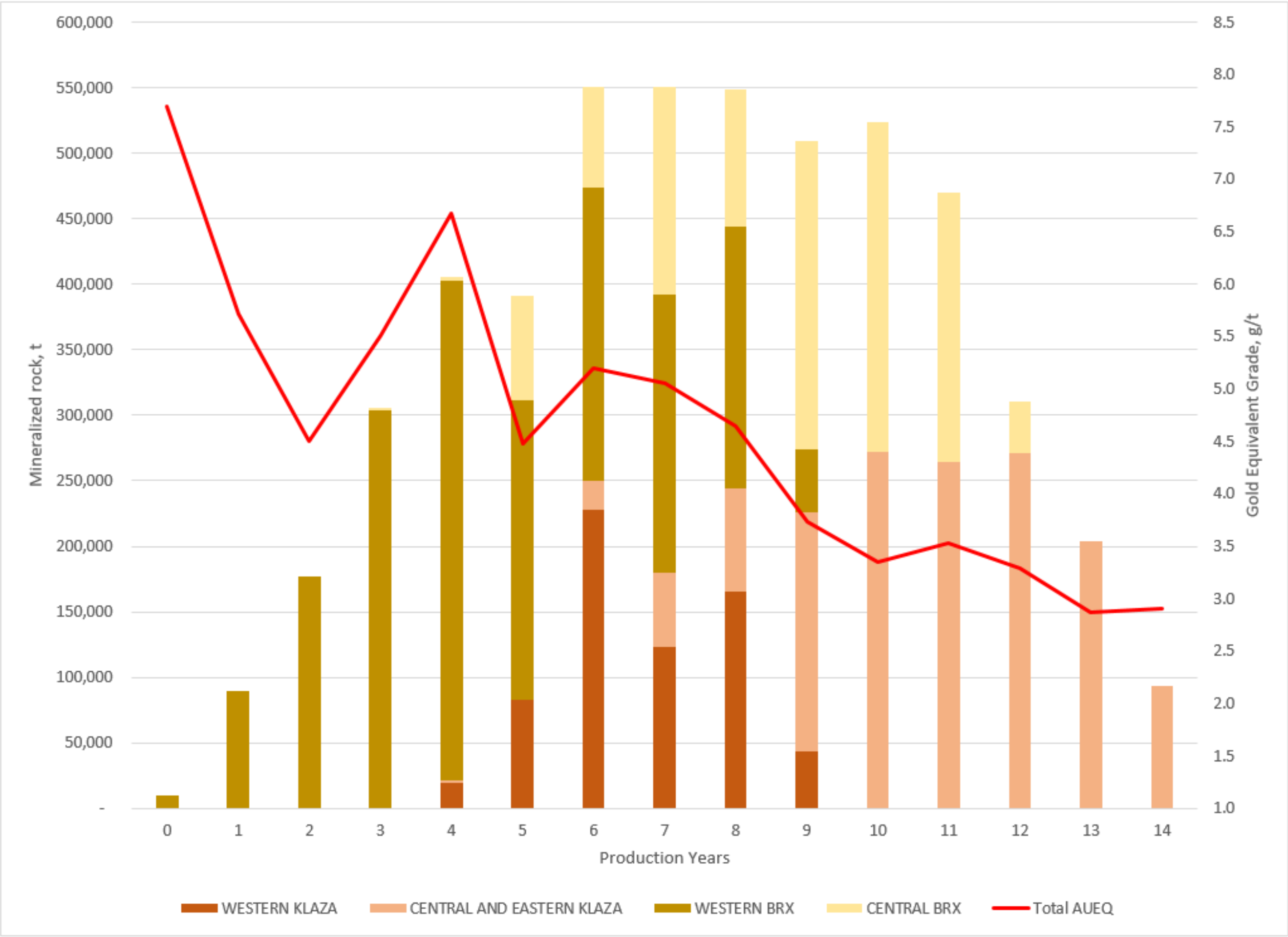
16.8.4 Conceptual underground production schedule

The projected underground production schedule is summarized in Table 16.28 and Figure 16.24.

Table 16.28 Projected underground production schedule

UG Production	YR0	YR1	YR2	YR3	YR4	YR5	YR6	YR7
Mineralized rock (kt)	9	89.9	176.7	305.2	405.8	391.4	550.9	550.3
AuEQ (g/t)	7.7	5.7	4.5	5.5	6.7	4.5	5.2	5.1
Au (g/t)	6.6	4.5	3.7	4.5	5.2	3.4	4.0	3.7
Au (g/t)	73	94	52	74	101	72	96	103
Pb (%)	0.8	0.9	0.6	0.5	1.1	0.8	0.6	0.7
Zn (%)	0.8	0.5	0.7	0.8	1.0	0.8	0.7	0.8
As (%)	0.5	0.5	0.7	0.6	0.7	0.5	0.5	0.5
UG Production	**YR8**	**YR9**	**YR10**	**YR11**	**YR12**	**YR13**	**YR14**	**Total**
Mineralized rock (kt)	548.7	509.3	523.8	470.2	310.0	203.5	93.0	5,139
AuEQ (g/t)	4.6	3.7	3.4	3.5	3.3	2.9	2.9	4.5
Au (g/t)	3.4	2.5	2.2	2.3	2.6	2.2	2.5	3.3
Ag (g/t)	95	95	82	86	44	50	24	84
Pb (%)	0.6	0.8	0.8	0.9	0.6	0.3	0.2	0.7
Zn (%)	0.8	0.8	0.9	0.8	0.5	0.6	0.7	0.8
As (%)	0.6	0.4	0.4	0.4	0.3	0.3	0.1	0.5

Figure 16.24 Projected underground production



16.8.5 Waste rockfill

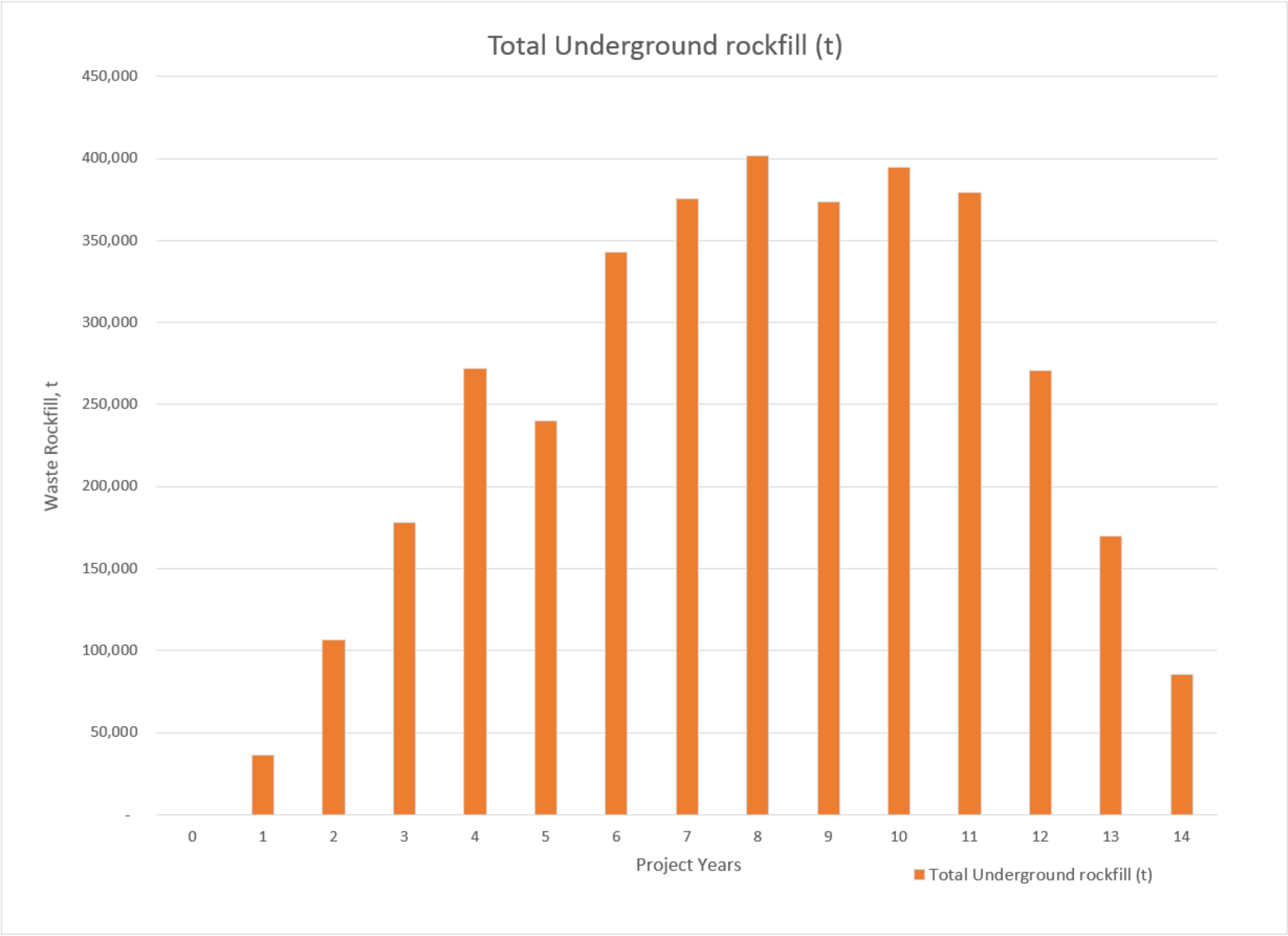
Stopes will be backfilled with waste rock on a sequential basis (stopes blasted nominally 24 m along strike and then filled). Approximately 1.2 Mt of waste is planned to be produced from underground over the life of the mine. Where possible, waste rock generated underground will be backfilled directly 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 16.29 and shown in Figure 16.25.

Table 16.29 Projected annual underground waste rock fill schedule

Quantity	**YR0**	**YR1**	**YR2**	**YR3**	**YR4**	**YR5**	**YR6**	**YR7**
Waste rock fill (kt)		36	107	178	272	240	343	376
Quantity	**YR8**	**YR9**	**YR10**	**YR11**	**YR12**	**YR13**	**YR14**	**Total**
Waste rock fill (kt)	402	374	395	380	271	170	85	3,627

Figure 16.25 Projected underground waste rock fill requirements



16.8.6 Conceptual combined production schedule

The projected combined life of mine production schedule for the open pit and underground is summarized in Table 16.30. The conceptual schedule is shown together with the AuEQ grade in Figure 16.26. The projected combined open pit and underground mine design is shown in Figure 16.27.

Table 16.30 Conceptual life of mine production schedule

Production	**YR0**	**YR1**	**YR2**	**YR3**	**YR4**	**YR5**	**YR6**	**YR7**
Waste (kt)	4,433	4,398	4,128	4,048	1,176	165	155	45
Mineralized rock (kt)	144	337	596	664	552	391	551	550
AuEQ (g/t)	3.3	4.8	4.1	4.7	6.2	4.5	5.1	5.1
Production	**YR8**	**YR9**	**YR10**	**YR11**	**YR12**	**YR13**	**YR14**	**Total**
Waste (kt)	8	104	23	1				18,686
Mineralized rock (kt)	549	510	524	470	310	204	93	6,444
AuEQ (g/t)	4.6	3.7	3.4	3.5	3.3	2.9	2.9	4.4

Figure 16.26 Conceptual life of mine production schedule and AuEQ grade

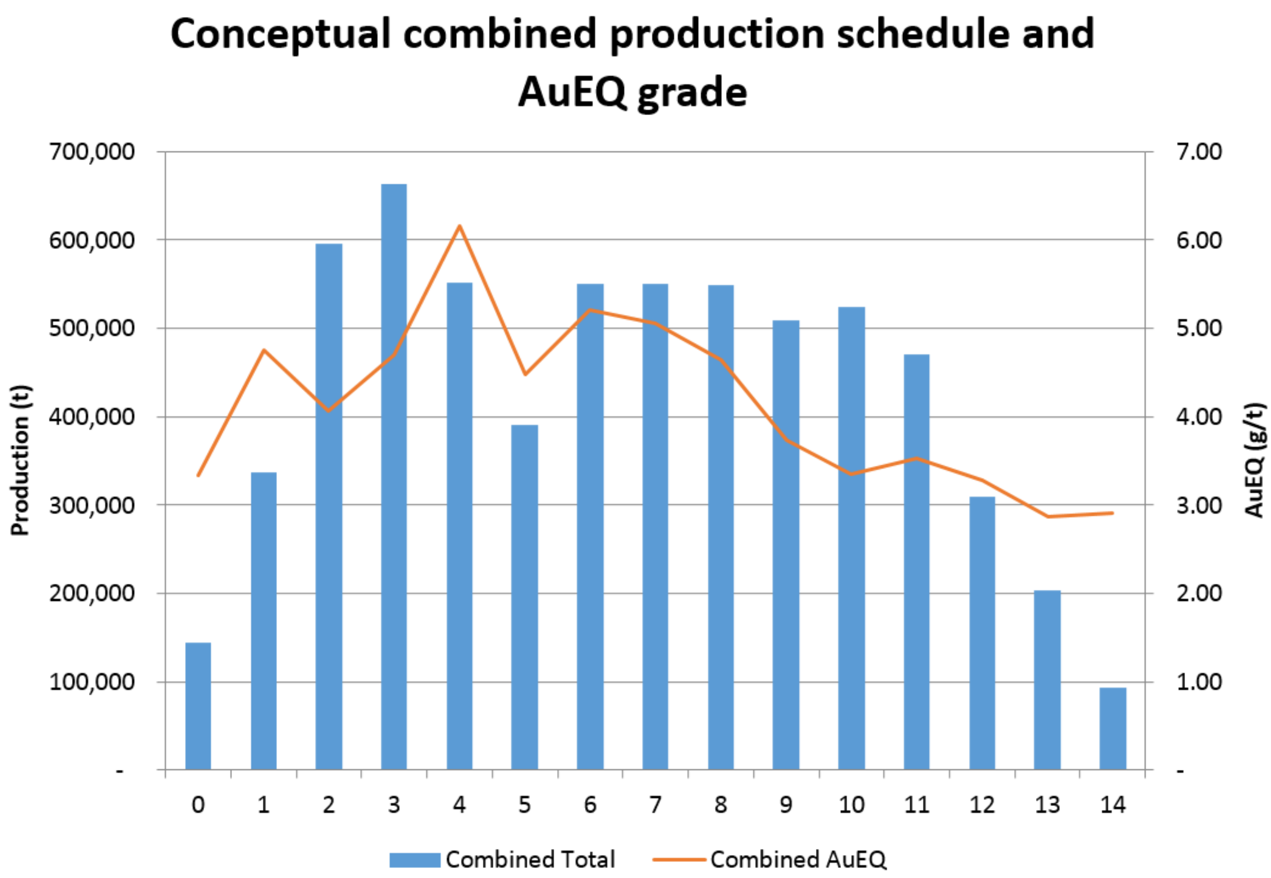
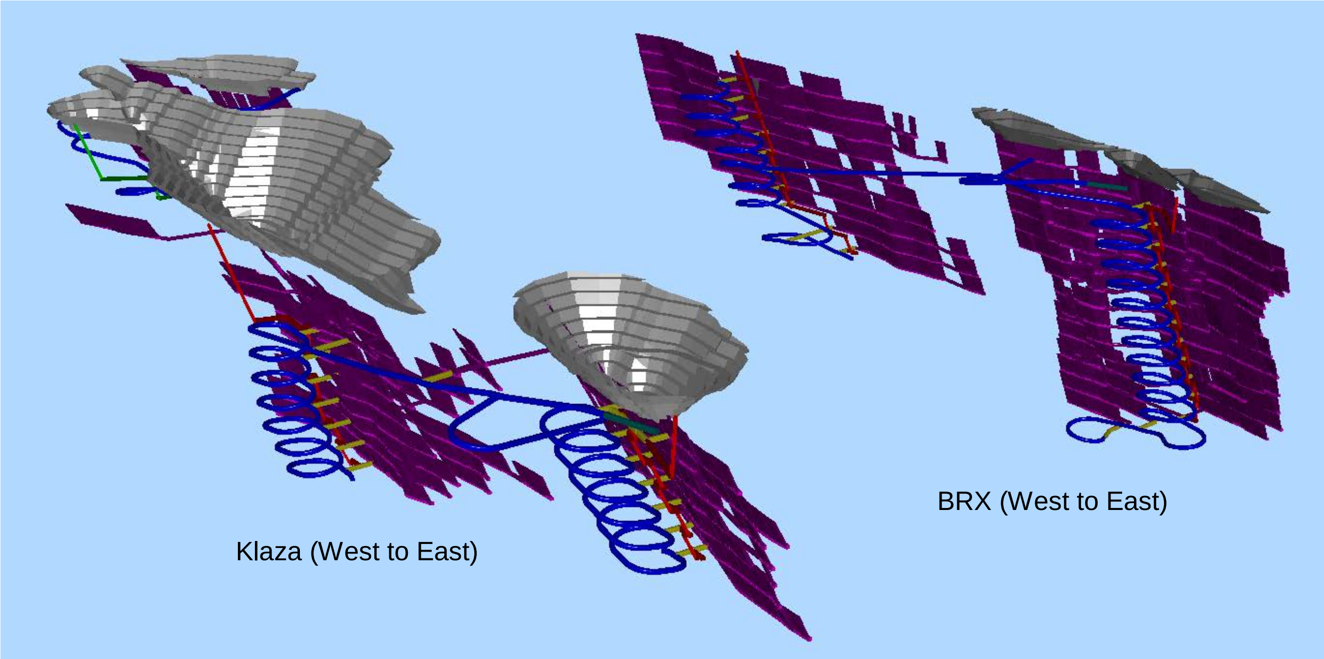


Figure 16.27 Projected open pit and underground mine design



16.8.7 Projected process plant feed schedule

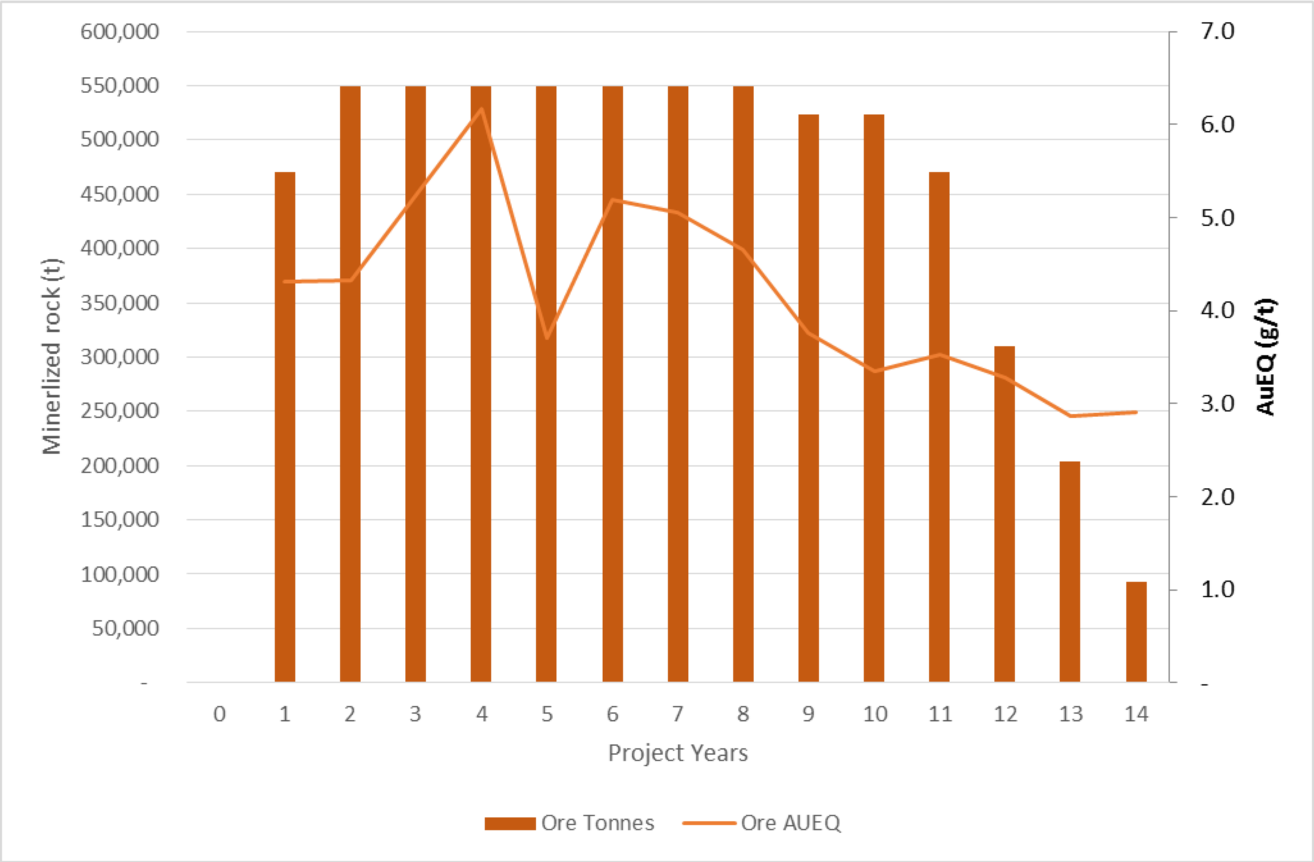
Projected production from the open pit and underground is stockpiled in YR0, during the construction of the process plant. It has been assumed that the process plant would be capable of reaching 85% capacity (470 ktpa) during YR1 and 100% capacity (550 ktpa) in YR2.

The conceptual process feed is summarized in Table 16.31 and shown in Figure 16.28.

Table 16.31 Projected process plant feed schedule

Production	**YR0**	**YR1**	**YR2**	**YR3**	**YR4**	**YR5**	**YR6**	**YR7**
Mill feed (kt)		470	550	550	550	550	550	550
Aueq (g/t)		4.3	4.3	5.2	6.2	3.7	5.2	5.1
Au (g/t)		3.7	3.6	4.3	4.8	2.9	4.0	3.7
Ag (g/t)		49	49	72	100	57	95	102
Pb (%)		0.6	0.8	0.7	1.0	0.6	0.6	0.7
Zn (%)		1.1	1.0	1.0	1.0	0.7	0.7	0.8
As (%)		0.5	0.5	0.6	0.6	0.4	0.5	0.5
Production	**YR8**	**YR9**	**YR10**	**YR11**	**YR12**	**YR13**	**YR14**	**Total**
Mill feed (kt)	550	523	524	470	310	204	93	6,444
Aueq (g/t)	4.7	3.8	3.4	3.5	3.3	2.9	2.9	4.37
Au (g/t)	3.4	2.5	2.2	2.3	2.6	2.2	2.5	3.33
Ag (g/t)	95	95	82	86	44	50	24	77
Pb (%)	0.6	0.7	0.8	0.9	0.6	0.3	0.2	0.7
Zn (%)	0.8	0.8	0.9	0.8	0.5	0.6	0.7	0.8
As (%)	0.6	0.4	0.4	0.4	0.3	0.3	0.1	0.5

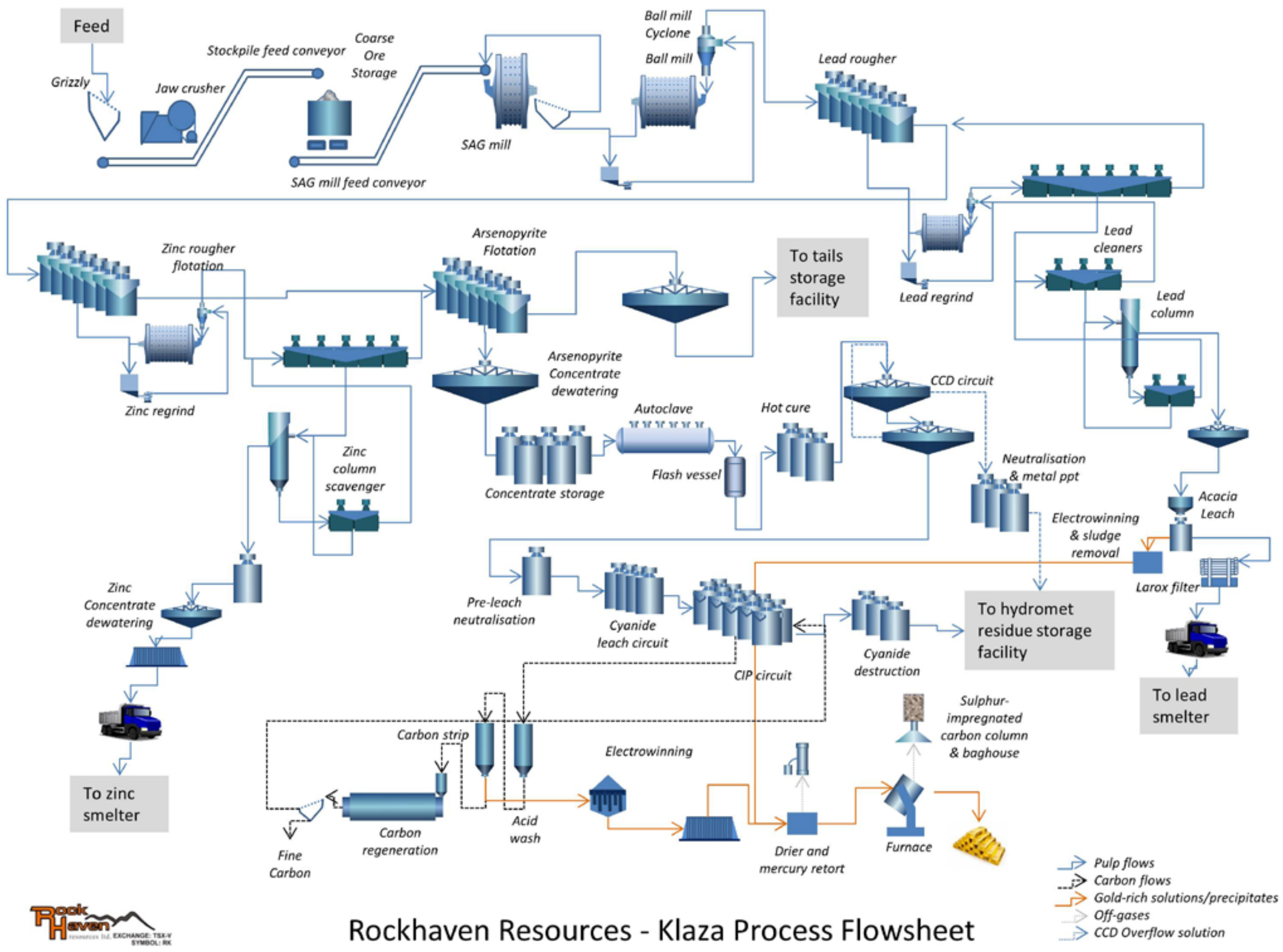
Figure 16.28 Projected process plant feed



17 Recovery methods

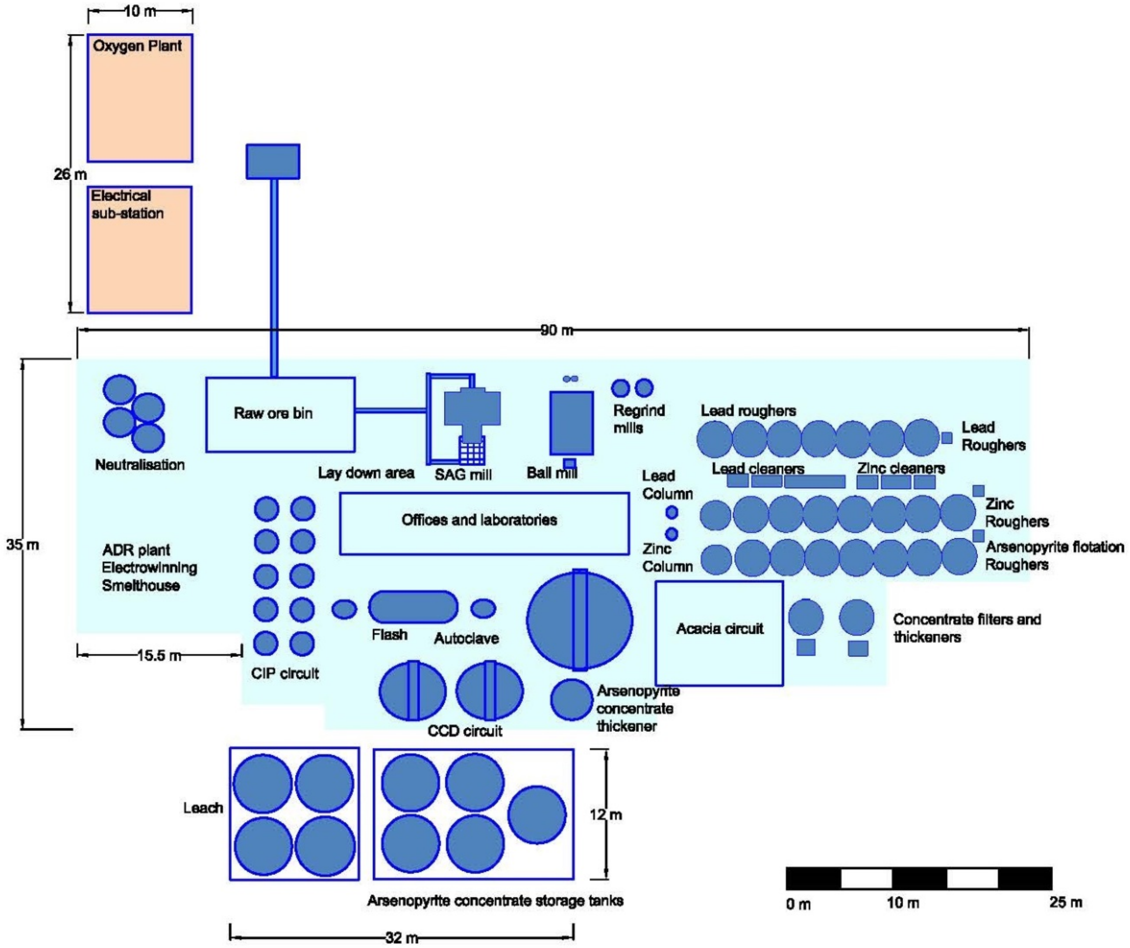
This section describes the process approach with some of the key design data tabulated. The overall process flowsheet is shown in Figure 17.1.

Figure 17.1 Overall process flowsheet



The layout for the projected processing facility is shown in Figure 17.2.

Figure 17.2 Projected processing plant layout



17.1 Mineralized rock characteristics

For the basis of the overall process design, the mineralized rock characteristics have been assumed as follows:

Table 17.1 Site conditions and mineralized rock characteristics

Parameter	Unit	Design Value	Reference
Specific gravity of mineralized rock	t/m^3	2.80	BCR-G
Bulk density of crushed mineralized rock	t/m^3	1.8 – 2.0	BCM
Moisture content of ROM mineralized rock	% 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 a mineralized rock bin with 24 hours' live capacity (1,650 t). Material from the mineralized rock 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 68 tph.

Crushed mineralized rock is reclaimed from the mill feed storage bin and fed to a single 800 kW 4.9 m diameter by 3.0 m long (2.6 m EGL) SAG Mill. 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 1,100 kW 3.7 m diameter by 5.8 m long (5.6 m EGL) 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 mineralized rock. 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. The concentrate 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 are 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 bank 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 are 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 are 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 flocculants. 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 are 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 system plant to recover a portion of the gold and silver prior to shipment to smelter, see Section 17.11.

17.5 Zinc flotation

A zinc flotation circuit is employed to concentrate zinc. Lead rougher and first cleaner tails, together with lime, are pumped from the lead rougher tails pump box to the zinc conditioner where they are 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 by 4.3 m long (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% 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 flocculants. 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 mineralized rock. Zinc rougher and first cleaner tails, together with lime, are pumped from the zinc rougher tails pump box to the arsenopyrite conditioner where they are 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 are laundered to the final flotation tails pump box, where they are pumped to the tails thickener. The concentrate from the arsenopyrite rougher is pumped to the POX circuit feed thickener, designed to thicken the concentrate to 55% solids; and 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 recirculation system.

17.7 Flotation tailings

The arsenopyrite flotation tailings are 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 flotation Tailings Storage Facility (TSF). Water is returned from the TSF to supplement process water, as required.

17.8 Pressure oxidation circuit

Underflow from the arsenopyrite thickener continuously feeds a set of surge capacity tanks providing five days (120 hours) of surge capacity storage, ahead of the pressure oxidation (POX) circuit. This is to allow up to five 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) is into a POX feed tank, where return water from the POX CCD circuit would be added to decrease the percentage of 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 shows pressure oxidation design criteria.

Table 17.2 Pressure oxidation design criteria

Parameter	Unit	Design value	Reference
Autoclave			
Feed flow rate	t/h m^3/h	5.3 36.4	BCM 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 carbon-in-pulp leach

The hot solution (90°C) from the flash vessels is treated through a hot cure stage to cure the basic iron sulphates (BIS). The hot cure acid conditions the POX product for six 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, in order to provide the six hours of residence time required. Neutralization of this liquor is completed using cheaper limestone to reduce cost. The neutralized liquor is directed to a lined Hydromet Residue Storage Facility (HRSF). Overflow from this pond will flow into the main TSF, to capture gypsum and iron precipitates and neutralize any residual acid.

The second thickener underflow will be neutralized with lime to pH 10 and the slurry transferred to a series of 4 cyanide leach tanks, 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 10 m^3 KEMIX pump cell tanks. The total CIP residence time is 12 hours. Each pump cell tank has its own contained carbon load (0.5 t), with only the passing slurry cascading through each cell. Sequentially, as the carbon in the 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 will be included on the CIP tails before pumping to cyanide destruction.

CIP tailings and acid wash water is treated to detoxify the cyanide using the INCO Air/SO_2 process before pumping to the HRSF, reducing the Weak Acid Dissociable (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) is 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 moulds, in which the slag separates from the gold. Gold doré bars are removed from the moulds, and 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 of lead concentrate per day, and consists of:

1. Concentrate storage cone
2. Consep Acacia Dissolution Module
3. Slimes recovery module
4. Electrowinning module
5. Cathode wash module

The Acacia plant modules are controlled using a PLC system, with its own control centre and MCC connected to all the modules and their instrumentation and drives. High gold recoveries from the 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 cyanide 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, flocculants, caustic soda, 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 envisaged. 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 mineralized rock 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 Programmable Logic Controllers (PLCs). Process control cubicles are located in the Motor Control Centres ("MCCs") 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 (PCs) would be located in the Main Control Room (MCR). The PCs 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 in the plant.

17.15 Laboratories

The major equipment in the support laboratories will include ICP, AA, LECO 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.

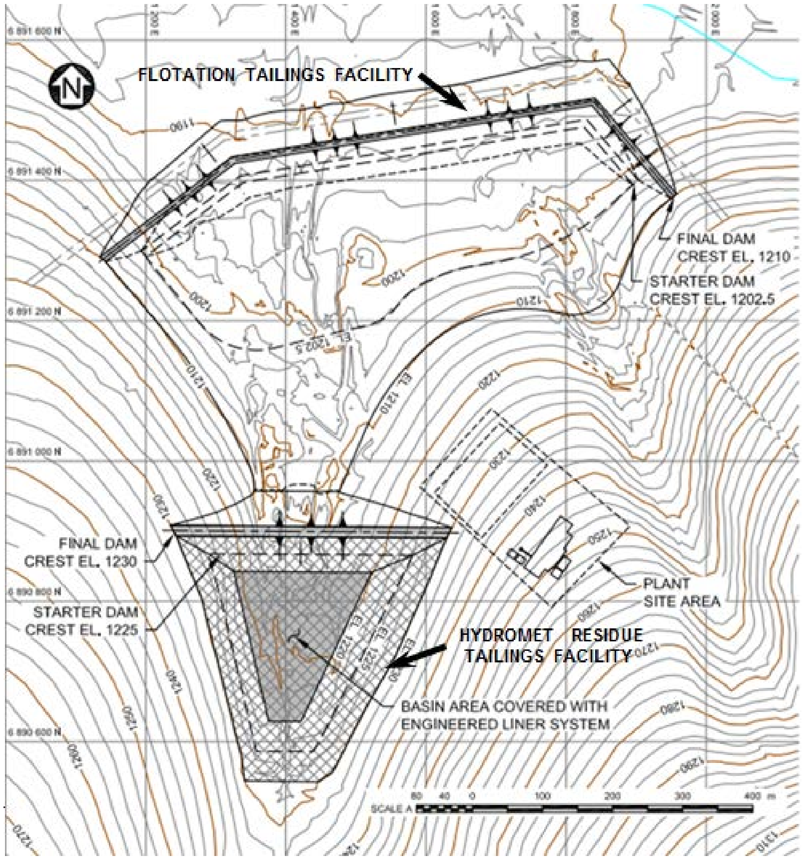
The tailings management strategy has been developed to manage the different tailings streams in separate facilities based on the tailings geochemical properties. Two tailings streams requiring disposal will be produced:

- Hydromet residue
- Flotation tailings

The hydromet residue will be managed in a separate facility sized to contain the production schedule volume. The flotation tailings will be managed using a surface facility with a containment structure until approximately Year 6, when the open pits will be available to receive tailings. Detailed descriptions of the facilities are provided in the section below.

The Hydromet Residue and initial Flotation Tailings Facility layouts are shown in Figure 17.3.

Figure 17.3 Hydromet Residue and Flotation Tailings Facilities overall layout



17.16.2 Hydromet Residue Storage Facility

The Hydromet Residue Storage Facility is optimally positioned adjacent to the plant and directly up gradient from the Flotation Tailings Facility. This location was the most viable and practical based on a number of economic and operational factors.

The hydromet residues are expected to be relatively stable, but high in arsenic 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 Flotation Tailings Facility. 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 2.5-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 1,225 m – Start-up (tailings storage Years 1 to 5)
- Stage 2 – Elevation 1,230 m - Constructed in Year 5 (tailings storage Years 6 to 14)

The embankment will be constructed and operated as a geosynthetic-faced rockfill dam. 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 of the facility is 65,000 m^2.

The Stage 2 embankment configuration features a dam roughly 20 m high and 400 m in length, with a total fill volume estimated to be 260,000 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.

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 Flotation Tailings Facility 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 Flotation Tailings Facility 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 Corrugated Polyethylene Tubing (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 1,203 m – Start-up (tailings storage Years 1 and 2)
- Stage 2 – Elevation 1,207 m - Constructed Year 2 (tailings storage Years 3 and 4)
- Stage 3 – Elevation 1,211 m - Constructed Year 4 (tailings storage Years 5 and 6)

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 Stage 3 embankment configuration features a dam roughly 25 m high and 1 km in length, with a total fill volume estimated to be 952,000 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 Open pit flotation tailings disposal

The flotation tailing deposition strategy will be modified in Years 7 to 14 to take advantage of the available space for tailings solids storage in the mined-out Klaza open pits. The mining sequence has indicated that the Open Pits designated Klaza 1 and Klaza 2 will be available for tailings storage starting in Year 7.

The filling capacity of the Klaza 1 and Klaza 2 open pits was evaluated and the results are summarized as follows:

- Open Pit Klaza 1 - Elevation at haul road daylight 1,290 m, storage capacity = 670,000 m^3 (871,000 tonnes)
- Open Pit Klaza 2 – Elevation at haul road daylight 1,320 m, storage capacity = 2,100,000 m^3 (2,730,000 tonnes)

The pits would be filled with tailings to the pit rim and supernatant water decanted to the Flotation Tailings Facility via a reclaim piping system. Excess storm-water flows would be routed along the haul road collection ditches to the Flotation Tailings Facility for use in the process.

Closure of the pits involves capping the final tailings surface with a waste rock and overburden blanket

17.16.5 Water management

The key facilities for the water management plan are:

- Open pits
- Underground mine dewatering
- Mill (including fresh and process water tanks)
- Tailings storage facility (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 Flotation Tailings Facility. 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 flotation tailings facilities 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/d.

- 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, lunch rooms, 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 Mine dry

The mine dry will also be an assembly of construction industry grade, portable trailers. Space for 200 lockers and baskets will be provided along with showers and laundry facilities. The mine dry will be connected to the offices. Personnel will be transported from the dry directly to the mine portals.

18.1.3 Sewage treatment

Sewage treatment at the mine site will consist of a vacuum truck removing sewage from various locations at the site and delivering to a central facility near the process plant. Grey water will be handled as part of the tailings stream.

18.1.4 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.

18.1.5 Power

18.1.5.1 Generation

A trade-off study was conducted to investigate providing electric power using diesel generators or grid power from the territorial utility. The results indicate that grid electrical power provides more value to the project over the life of the mine.

The diesel generation option would require generators with spare capacity and fuel tanks. The capital cost of the tanks and generators was estimated from public pricing and unit rates and benchmarked against other projects. Fuel was assumed to be trucked in from Edmonton. The electrical consumption over the life of the mine was estimated and used to prepare an operating cost. The discounted (5%) total cost (capital plus operating) of power over the life of the mine using diesel generators was estimated to be C$210M.

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 18.1). 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 using the same assumptions as the generator option. The discounted (5%) total cost (capital plus operating) of power over the life of mine using the local grid was estimated to be C$100M.

The key power assumptions were:

- Discount rate of 5%.
- Overall cost of utility power C$0.105 per kW-hr (includes allowance for cost of the transmission line).
- Cost of diesel at C$1.00 per litre.
- Utility capacity exists (or will exist) to supply the mine and process plant.

These results indicate that there is a major economic benefit using electrical power supplied from the local grid. The transmission line would be provided by the utility and paid off over the life of mine as a surcharge on the unit cost of power.

18.1.5.2 Distribution

The main electrical supply will consist of a 73 km, 138 kV transmission line from the territorial power grid at Carmacks. The main feeders will be brought to the sub-station and primary disconnect near the process plant, as this is the location of the largest power load.

Lower voltages tapped from the sub-station will then be distributed via buried cables to the mine portals, oxygen plant, and open pit mine complex.

Each of the two underground mines will have a local sub-station and disconnect that further reduces the voltage down to 4,160 V for the mine supply. Buried cables will carry the power to surface facilities such as exhaust fans. Main feeder cables will distribute power down the declines.

At each mining level there will be a disconnect switch and junction box that will allow a portable sub-station to provide power on the level. Portable sub-stations will be moved between levels to support mining activities, with lower voltage tailing cables being moved along with them. These sub-stations will provide ground fault interruption and voltage reduction to supply mobile equipment and lighting.

Permanent sub-stations will be used for stationary power loads such as mine dewatering pump stations or maintenance shops, with a centrally located sub-station using shorter distribution networks to provide efficient local supply.

18.1.6 Water supply

Water for the mine site and processing plant will be supplied from the Klaza river. 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.

18.1.7 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.

18.1.8 Surface mobile equipment

The mine will designate an emergency vehicle (ambulance) for evacuation to medical care via the access road. A fire truck will also be located near the ambulance. A vacuum truck, flat deck, and mechanics vehicles will be also be required.

A telehandler and small forklift for moving equipment and supplies around the processing plant and warehouse will be provided.

A small 35 tonne crane for maintenance tasks in the processing plant, surface handling, and other tasks can be procured with an operator on rent as needed for shutdown maintenance tasks. The low usage of such equipment would not necessitate having a permanent qualified operator on site.

18.1.9 Accommodations

The work force will be encouraged to live in the village of Carmacks and a daily bus service will be provided to drive them 73 km to the mine and back. There is no allowance for a mine camp in the project estimates. Kitchen facilities will be available for dispensing tea and coffee in the mine offices. Washrooms and lockers have been allowed for at the main office. Carmacks has a current population of approximately 450. AMC has estimated a total site workforce of 255 people and 90 open pit contractors. Approximately 185 staff on site and 60 contractors are projected during normal production, including management and technical staff.

18.1.10 Fire detection and suppression systems

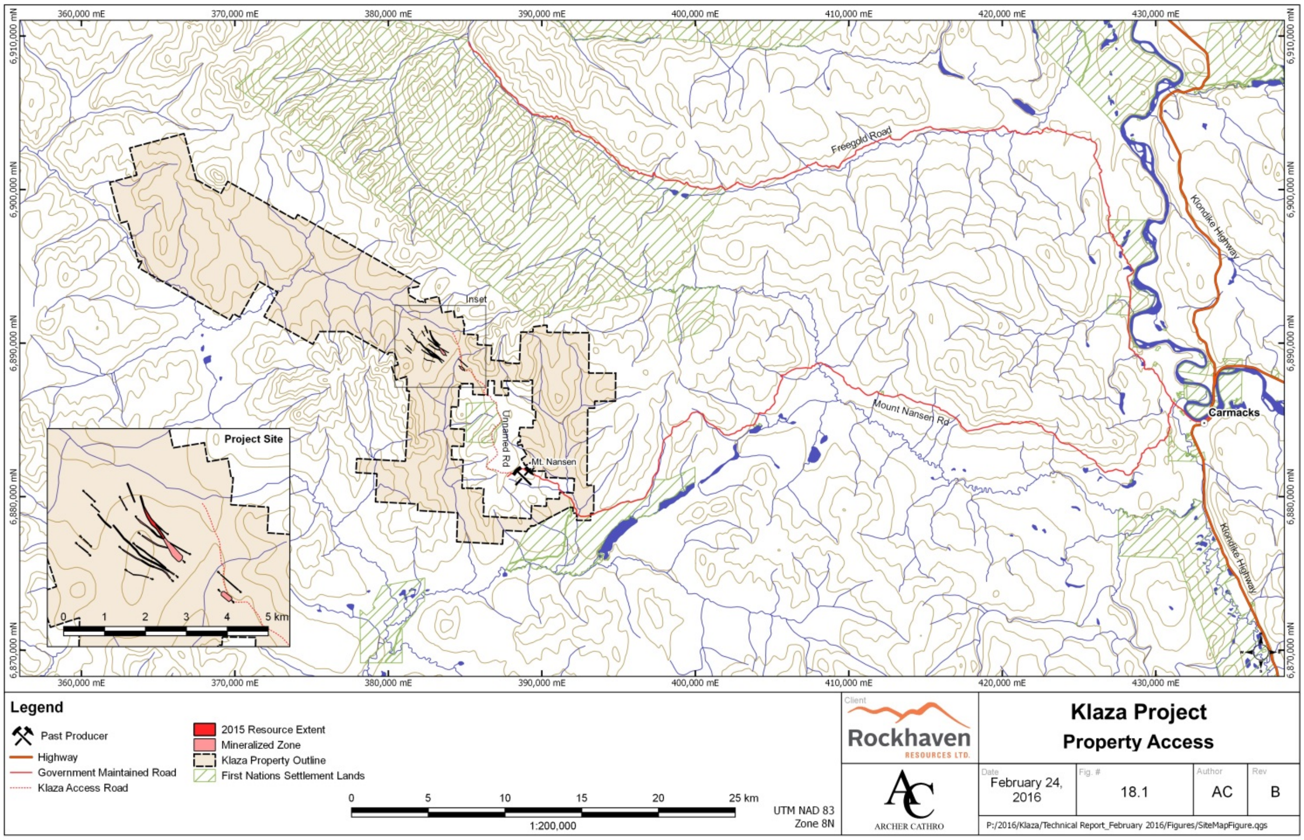
Each of the mine ventilation systems will be provided with an ethyl mercaptan (stench gas) system (activated manually or remotely) in order to warn underground personnel in the event of an emergency. Radio contact via the leaky feeder system provides an alternative method of communication. The supply, 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.

18.1.11 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. The road shoulders and crown must be maintained in good condition as the entire work-force at the plant is intended to reside at Carmacks and commute daily. Figure 18.1 shows the Mount Nansen road from Carmacks to the Klaza project.

Figure 18.1 Klaza project access road



18.1.12 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 HDPE piping systems, partially or completely buried, to the plant for processing as part of the tailings system.

18.1.13 Explosives

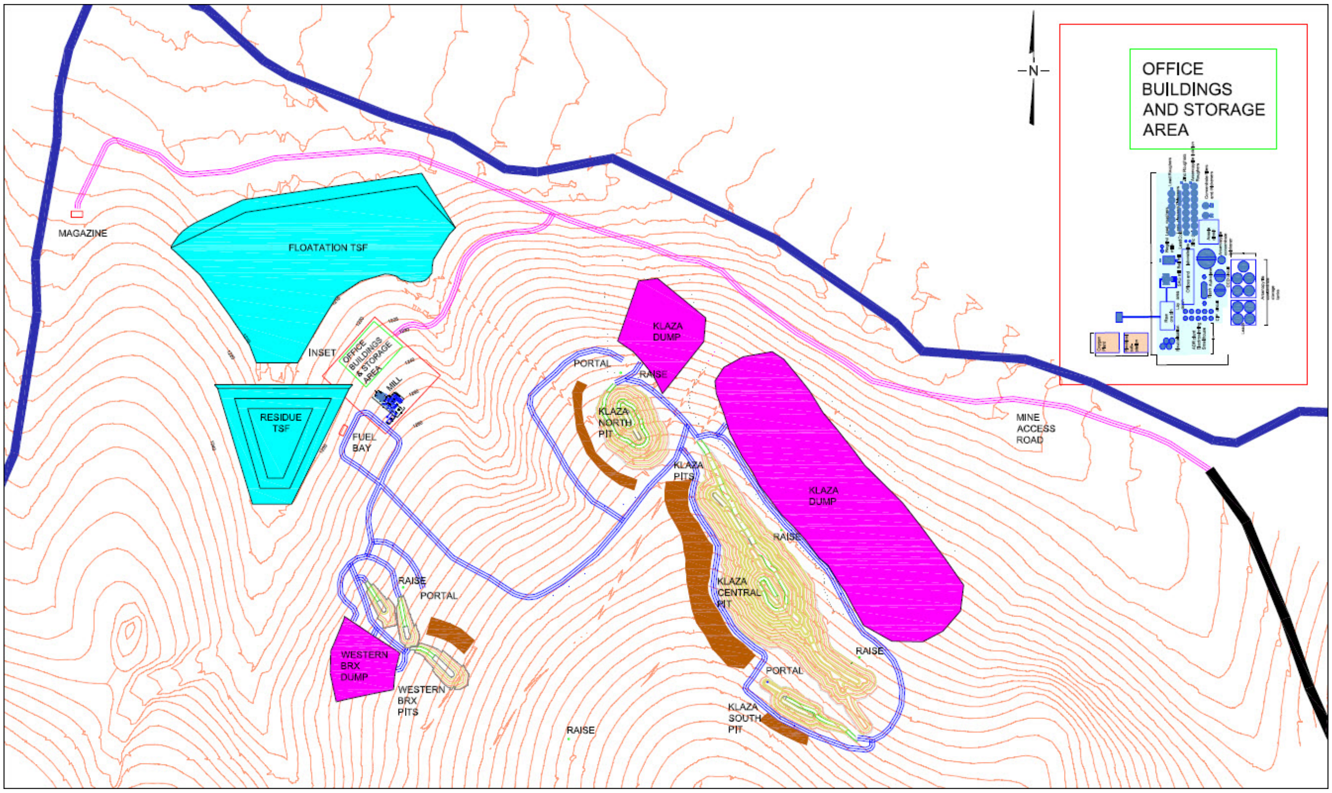
Explosives will be transported into the underground mines and two shifts of capacity stored in simple, secure underground facilities. The majority of explosives will be stored on surface as part of the open pit mine operations. The underground mines will continue to use the open pit explosives facilities when the open pit phase of the project is completed.

18.1.14 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 18.2.

Figure 18.2 General layout of the surface infrastructure for Klaza



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 cap magazines.

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.

A single bay maintenance shop with a jib crane will be provided in each mine. The intent of this bay is to enable routine tasks such as lubrication and changing of filters, and minor repairs to keep the equipment in a serviceable condition and return it quickly into service. Any significant maintenance will be conducted on surface in the main workshop.

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 totes will be provided underground. Storage will also be provided for lubricants and waste oil. A small location equipped with fire doors, fire detection, and air operated pumps will dispense the products near the maintenance bay.

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.

Compressed air will be supplied by mobile electric compressors. These will be situated on active levels and at the maintenance bay. The compressors will be relocated to active mining levels as needed.

Mine water will be supplied via a four inch 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 distribution system of steel pipe, together with hoses will be laid out on the operating levels.

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

18.2.1 Dewatering

The mine dewatering system will consist of staged 50 hp submersible dirty water pumps at nominally 60 m vertical intervals. Each sump will have two pumps to provide continual redundancy. A six inch steel 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). The six inch standard weight steel line is sized so that, with two pumps, a significant increase to the dewatering volume could be handled; with one pump, a flow that results in a carrying velocity in the line is still generated.

Dewatering facilities will be installed as the mine development proceeds and pumps will be relocated as areas are abandoned and the pumps are no longer required. Carrying cost of the pumps is presumed to be part of sustaining capital in later years and only a portion of the dewatering system cost is capitalized in this analysis.

18.2.2 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 vent 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 Personnel movement

Personnel will travel to and from Carmacks on daily buses. The closest large airport is Whitehorse International Airport located 175 km south of Carmacks. The airport has capability for large and small planes and is serviced with scheduled daily flights to larger centres in Canada. The village of Carmacks has a small municipal airfield that is used for emergency services such as fire-fighting, and for private aircraft.

18.5 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.

18.6 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.

18.7 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.

18.8 Tailings disposal

Tailings disposal is discussed under Section 17.

18.9 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 permafrost 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 Metal prices

The project NSR values have been generated using the metal prices shown in Section 22.2.

For the PEA it is assumed that the metal prices will remain stable over the project life.

19.2 Product grades and volumes

The NSR values were developed using the grades, recoveries and production volumes described in Section 16.

19.3 Marketing

19.3.1 Overview

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 metal reporting to the concentrates would 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.

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 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.

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.

19.3.2 Lead concentrate treatment terms

Treatment terms for the lead and zinc concentrates used to estimate the NSR values have been advised by HMH 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%

19.3.3 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

19.3.4 Transportation

19.3.4.1 Overview

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.

19.3.4.2 Total transportation costs

On the basis of a moisture content of 8% and using a C$:US$ exchange rate of 0.75:1, the total transportation cost is projected to be US$154.56 per dry metric ton.

19.3.5 Net smelter/mine returns – summary

NSR values are summarized in Table 19.3.

Table 19.3 Projected NSR values for concentrates

	NSR	Net Return FOB Mine
Lead concentrates (US$/dmt concentrate)	US$2,970.77	US$2,807.01
Zinc concentrates (US$/dmt concentrate)	US$1,208.61	US$1,050.14
Lead concentrates (US$/t of mineralized material)		US$30.30
Zinc concentrates (US$/t of mineralized material)		US$18.40
Total (US$/t of mineralized material)		**US$48.70**

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, LQ00305 and LQ00305a, 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, 2015)
- Yukon Lands Viewer including Species at Risk and Wildlife (Yukon Department of Energy, Mines and Resources, 2015a)
- Yukon Mining Map Viewer (Yukon Department of Energy, Mines and Resources, 2015b)
- Environment Yukon - Water Resources Branch (hydrometric stations)

20.1.2 Existing environmental studies and data

Table 20.1 (overleaf) 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, 2013	Site specific for all of 2013. It is assumed additional raw data is available for subsequent years.
	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 - 2015	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.	Yukon Lands Viewer and Yukon Mining Map Viewer	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)	Yukon Department of Energy, Mines and Resources, 2015.	See Section 20.1.3.5
	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.	Yukon Department of Energy, Mines and Resources, 2015.	
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

Environmental Consideration	Study name and details	Data source	Additional information/discussion
	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	Yukon Environmental and Socio-economic Assessment Board (YESAB), November 2015.	

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 *Yukon Environmental and Socio-Economic Assessment Act* (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. Additional data collection will be required to satisfy the requirements under YESAA and the *Yukon Waters Act* if the project moves ahead toward the environmental assessment 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 additional to these helicopter wildlife surveys, the Yukon Environmental and Socio-Economic Assessment Board (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 Research Ltd. 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.

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 environmental assessment (EA). The Yukon Environmental and Socio-economic Assessment Board (YESAB) administers Environmental Assessment (EAs) in the Yukon. The Project will be subject to an EA under the *Yukon Environmental and Socio-economic Assessment Act* (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 1,500 tpd to 1,650 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:

a. Class III Mining Land Use Approval LQ00434, expiring 6 December 2020.

20.2.2.2 Quartz mining license

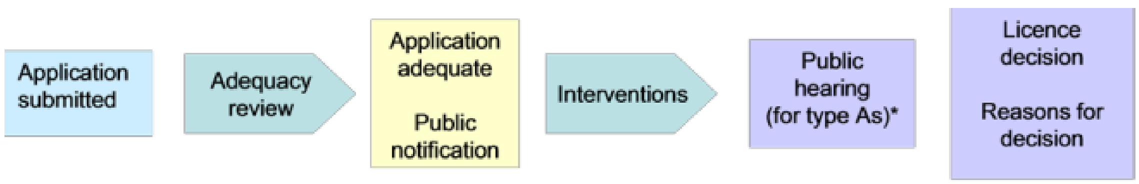
Under the *Quartz Mining Act* a Quartz Mine License is issued and administered by the Yukon Department of Energy, Mines and Resources, and enables the Government of Yukon to regulate mining developments. Any operator who wishes to construct a facility or do physical work in support of the commercial 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 in order to proceed with Licensing. Figure 20.1 depicts the process for a Water License application.

Figure 20.1 Water License application process



The Project will require a Type A Water License; 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 Little Salmon Carmacks First Nation (LSCFN) and Selkirk First Nation (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 Exploration Benefits Agreement 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 this 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

There has been limited contact with the Selkirk First Nation 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 Selkirk First Nation should any activity be planned in their traditional territory.

20.4 Closure & 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

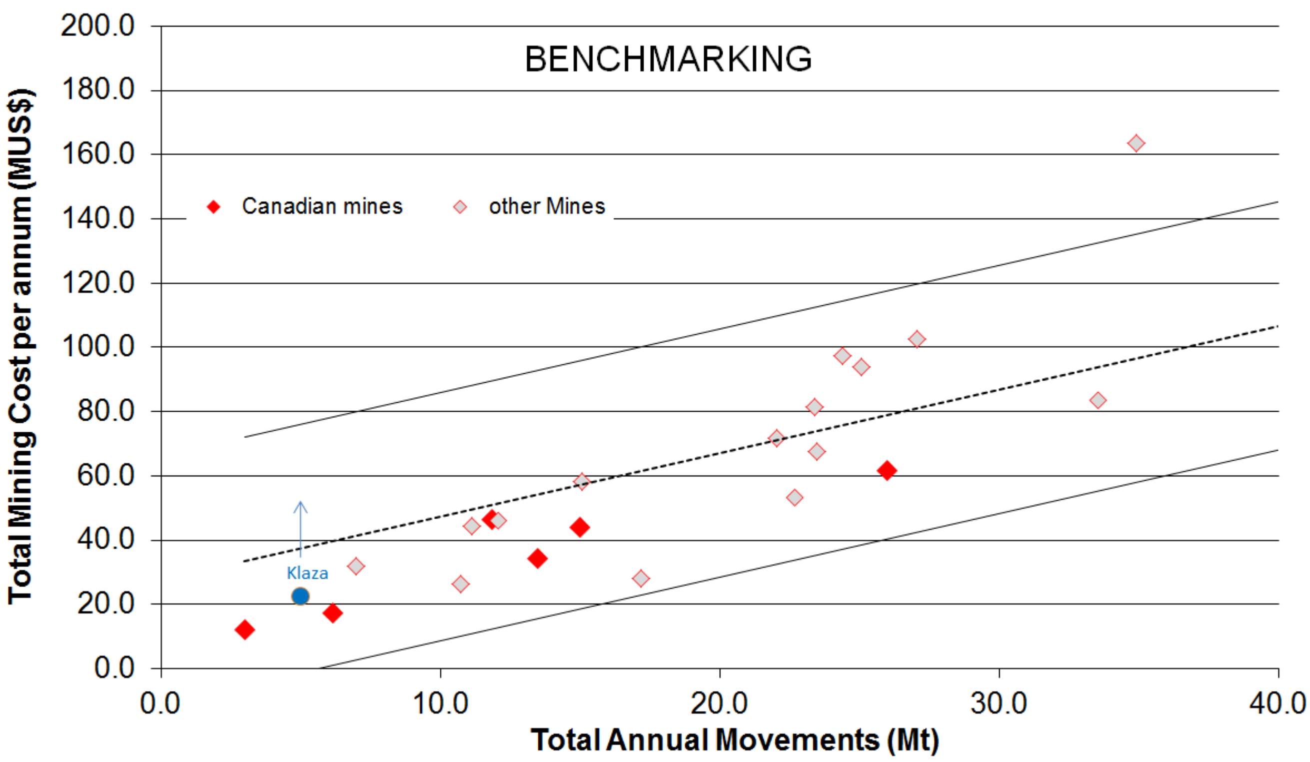
21.1 Operating cost estimate

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 life of mine average mining cost is approximately C$4.5/t mined.

Figure 21.1 Open pit benchmarking costs



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

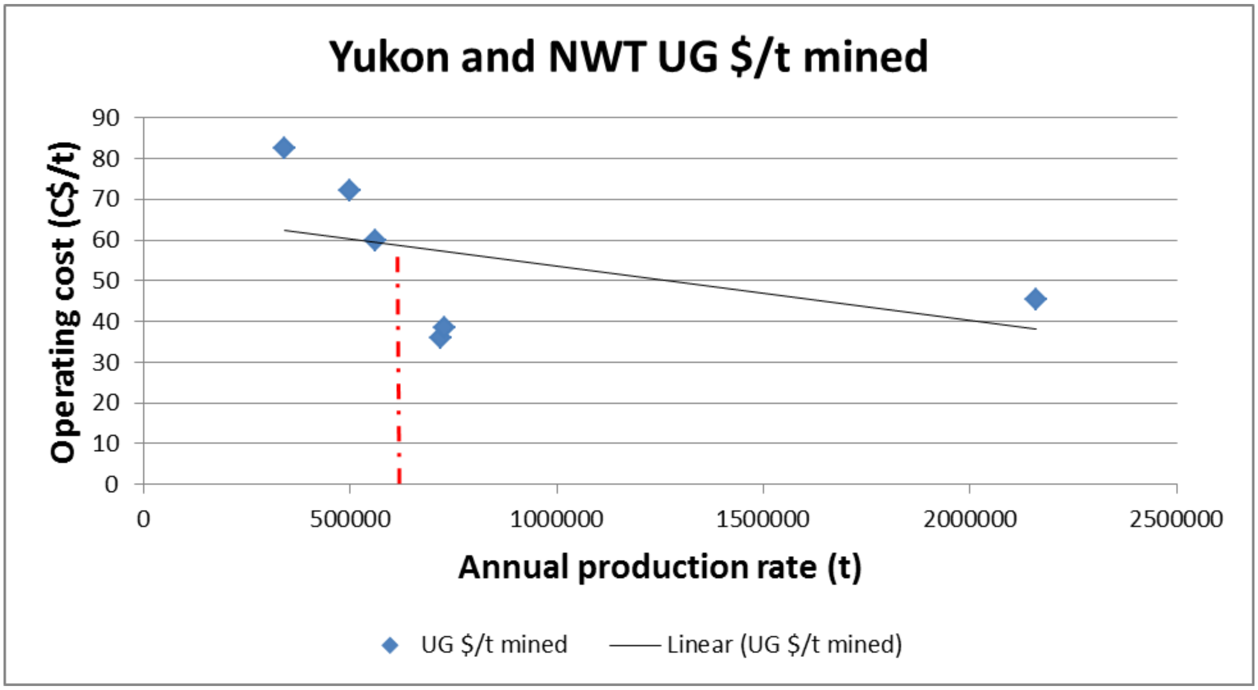
Category	%	Klaza estimate C$/t
Labour	18	0.8
Power	19	0.8
Consumables	20	0.9
Maintenance	25	1.1
Other	18	0.8
Total C$/t	**100**	**4.5**

Totals may not add up exactly due to rounding.

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 Yukon and NWT underground benchmark mining costs (C$/t)



AMC has used a benchmark mining cost of C$52.5/t for mineralized material extracted by stoping and C$100/t when extracted by development (tonnage ratio of 7.5:1 for projected LOM). Cost for waste development is estimated at C$6,000/m, for an overall mining cost of C$58.40/t. When considered against comparable local mining costs and accounting for variations in mining method, 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		Klaza estimate	%
	$/t	%	$/t	
Labour	24.2	46	32.1	55
Power	7.9	15	5.8	10
Consumables	14.7	28	15.2	26
Services	3.7	7	3.5	6
Other	2.1	4	1.8	3
Total C$/t	**52.5**	**100**	**58.4**	**100**

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 14-year life of mine, 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: grinding, milling, flotation concentrate filtration and load-out; and secondly, the hydromet sections, which include pressure oxidation (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 provide 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 C$/t	Hydromet C$/t	Total C$/t
Labour	7.99	6.14	14.12
Power	4.41	0.63	5.03
Consumables and Reagents	8.58	10.70	19.28
Maintenance supplies	1.63	1.40	3.03
Total C$/t	**22.37**	**19.33**	**41.47**

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.

Table 21.4 Estimated mill operating costs – Fixed and POX driven

Category	Mineral Processing C$/t	Hydromet C$/t	Total C$/t
Fixed daily costs	9.62	7.54	17.16
Fixed costs per tonne	12.98		12.98
POX feed rate driven costs		11.33	11.33
Total C$/t	**22.60**	**18.87**	**41.47**

The total process operating cost estimate is summarized in Table 21.5. AMC notes the total processing cost shown in this table is generated using the actual arsenic feed grade and yearly throughput as provided to BCM by AMC. This results in a variable cost per tonne over the life of the mine operation.

Table 21.5 Process operating cost estimate

	Annual mean daily tons	1471		1288	1500	1500	1500	1500	1500	1500	1500	1433	1435	1288	1500	1115	1115
			PRODUCTION YEAR														
		Life of Mine	0	1	2	3	4	5	6	7	8	9	10	11	12	13	14
Total Mine Production																	
Mill feed	kt	6,444	0	470	550	550	550	550	550	550	550	524	524	470	310	204	93
AUEQ	g/t	4.37	0	4.31	4.33	5.25	6.16	3.71	5.19	5.06	4.65	3.76	3.35	3.53	3.29	2.87	2.91
Au	g/t	3.33	0	3.65	3.65	4.31	4.81	2.88	3.96	3.71	3.43	2.48	2.17	2.31	2.62	2.19	2.47
Ag	g/t	77.00	0	48.70	48.90	71.50	100.30	57.10	95.10	102.50	94.90	95.40	82.10	85.70	44.50	50.30	23.90
Pb	%	0.70	0	0.64	0.76	0.69	0.97	0.64	0.61	0.72	0.59	0.75	0.76	0.87	0.56	0.29	0.23
Zn	%	0.84	0	1.08	1.01	0.69	1.00	0.69	0.71	0.84	0.75	0.80	0.94	0.81	0.48	0.57	0.74
As	ppm	4,741	0	5,272	5,487	5,573	6,130	4,317	4,707	4,875	5,700	4,063	3,833	3,985	3,392	3,229	1,345
	Mass pull to POX, %	5.90		6.60	6.90	7.00	7.70	5.40	5.90	6.10	7.10	5.10	4.80	5.00	4.20	4.00	1.70
	Fixed cost per tonne (C$/t)	$12.98	$	$12.98	$12.98	$12.98	$12.98	$12.98	$12.98	$12.98	$12.98	$12.98	$12.98	$12.98	$12.98	$12.98	$12.98
	Fixed daily costs (C$/t)	$19.24	$	$21.98	$18.87	$18.87	$18.87	$18.87	$18.87	$18.87	$18.87	$19.75	$19.73	$21.97	$18.87	$25.39	$25.39
	POX feed rate driven costs (C$/t)	$10.33	$	$11.49	$11.96	$12.14	$13.36	$9.41	$10.26	$10.62	$12.42	$8.85	$8.35	$8.68	$7.39	$7.04	$2.93
	Total process opex (C$/t)	$42.56	$	$46.45	$43.81	$44.00	$45.21	$41.26	$42.11	$42.48	$44.27	$41.59	$41.06	$43.64	$39.25	$45.41	$41.30

21.1.4 Tailings storage

This cost estimate was generated by Knight Piésold.

The operating costs summary in Table 21.6 show annual costs over the 14-year life of mine.

Table 21.6 Tailings storage facility operating cost

Description	Unit	Value	Unit Cost (C$)	Total Cost (C$/year)
Power – Reclaim pumping	Mwh	200	105	21,000
Power – Seepage pond pumping	Mwh	100	105	10,500
Environmental compliance	year	1.00	50,000	50,000
Engineering support and reporting	year	1.00	50,000	50,000
Total operating cost				**131,500**

21.1.5 General and administration (G&A)

G&A costs generally cover site administration and corporate costs. AMC benchmarked its estimate of $12/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 lower because of the relatively easy access to the site and the assumption that the workforce would be bused to and from the mine (not a fly-in fly-out with site camp operation as per the benchmark operation).

21.2 Total operating cost

The total operating cost estimate is summarized in Table 21.7

Table 21.7 Total operating cost estimate

Description	LOM Average Cost (C$/t)	Total LOM cost (C$M)
Mining cost	59.65	384
Processing & Tailings Storage cost	43.37	279
General and Administration cost	12.00	77
Total operating cost	**115.02**	**741**

Totals may not add up exactly due to rounding

21.3 Capital cost estimate

The capital cost estimate is split into project capital (first four years) 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 (five 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 (December 2015). Equipment numbers were estimated to meet the production target of 550 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 (2015) undertaken by AMC.

21.3.2.1 Underground development

Cost for development is estimated at C$6,000/m for lateral waste development and C$7,500/m for vertical development. The underground capital cost estimate for development is C$135.5M and is summarized in Table 21.8.

Table 21.8 Underground development cost estimate

Capital development costs	Length (m)	Unit cost (C$/m)	Project capital (C$M)	Sustaining capital (C$M)	Total cost (C$M)
UG Lateral Development (waste)	20,793	6,000	50.75	74.01	124.76
UG Vertical Development	1,435	7,500	4.99	5.77	10.76
Total	**22,228**		**55.74**	**79.78**	**135.52**

21.3.2.2 Underground mobile equipment

The underground capital cost estimate for mobile equipment is C$31.9M and is summarized in Table 21.9.

Table 21.9 Underground mobile equipment cost estimate

Description	Unit cost (C$)	Total cost (C$)
Longhole Production Drill (4)	1,136,298	4,545,192
2- Boom development Jumbo (4)	1,125,984	4,503,936
Scoops production (3)	1,073,084	3,219,252
Scoops development (3)	1,073,084	3,219,252
40T Trucks (5)	1,121,174	5,605,870
Bolter (2)	1,009,782	2,019,564
Cable Bolter (2)	1,131,125	2,262,250
Water Truck (1)	449,698	449,698
Explosives Loader (2)	419,780	839,560
Boom Truck (2)	377,010	754,020
Lube/Fuel Truck (2)	391,730	783,460
Personnel Carrier (4)	359,240	1,436,960
Scissor Lift (4)	381,130	1,524,520
Utility Vehicle (5)	59,300	296,500
Grader (1)	404,492	404,492
TOTAL		31,864,526

21.3.2.3 Underground infrastructure

The underground infrastructure capital cost estimate is C$17.4M 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 (C$)
Electrical	
Underground electrical distribution and equipment	5,100,000
Communications system and hardware	1,000,000
Ventilation	
Klaza West exhaust set-up: 150 hp	300,000
BRX West exhaust set-up: 250 hp	500,000
BRX Central exhaust set-up: 100 hp	200,000
Klaza Central exhaust set-up: 200 hp	400,000
Klaza East exhaust set-up: 50 hp	100,000
Distribution fans on six operating levels	1,500,000
Mine air heaters including enclosure (propane)	1,000,000
Dewatering	
Western BRX pumps, sumps, switchgear, piping	1,400,000
Central BRX pumps, sumps, switchgear, piping	500,000
Western Klaza pumps, sumps, switchgear, piping	400,000
Central Klaza pumps, sumps, switchgear, piping	900,000
Supply system	300,000
Other Underground Infrastructure	
Workshops	800,000
Magazines	100,000
Fuel/lube storage	400,000
Refuge stations	700,000
Lunch rooms	200,000
Escape way ladders	200,000
Mine rescue equipment	100,000
Lamp/battery rooms	100,000
Underground air compressors	200,000
Portal Infrastructure	1,000,000
TOTAL	**17,400,000**

21.3.3 Process plant

The process plant capital cost estimate is C$91.1M 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 was taken from industry standards, and typical capital cost estimation factors.

The process plant building cost was estimated by AMC (C$17M). 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 (C$)
Crushing	3,300,000
Grinding	13,500,000
Flotation	10,400,000
Dewatering	3,200,000
Autoclave, CCD Neutralization	22,700,000
Acacia Leach	2,100,000
Leach and CIP	4,500,000
Carbon handling and refinery	2,400,000
Reagents	4,800,000
Ancillaries	7,200,000
Building	17,000,000
Total	**91,100,000**

21.3.4 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 is C$13.5M, and is summarized in Table 21.12. The major component of this cost estimate is site roads and refurbishment of access roads, mine office, 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 21.12 Surface infrastructure cost estimate

Description	Total cost (C$)
Surface ancillary equipment	940,175
Power and communications	1,700,000
Water	300,000
Roads	1,431,040
Buildings, fuel and gas storage	6,600,000
Water supply / treatment	250,000
Fuel and oil storage	500,000
Storage bins	2,000,000
TOTAL	**13,721,215**

21.3.5 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 Flotation Tailings facilities will involve a progressive capping of the facility with a waste rock and overburden blanket. Closure of the pits involves capping the final tailings surface 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.

21.3.6 Sustaining capital

Capital costs for ongoing underground development after the project period (first four 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 life of mine is estimated to be C$96M.

21.3.7 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 C$11M.

21.3.8 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 C$34M.

21.3.9 Total capital cost estimate

The total capital cost is estimated to be C$358M and is summarized in Table 21.13 The capital cost estimate is split into project and sustaining capital and detailed by year in Table 21.14.

Table 21.13 Total capital cost estimate

Description	Total cost (C$M)
Underground lateral development	125
Underground vertical development	11
Floatation tailings storage & residue tailings storage	10
Underground mine infrastructure	17
Mobile equipment	32
Processing plant	91
Surface infrastructure	14
Capital indirects	11
Contingency	34
Additional 5% sustaining for equipment rebuilds	13
Total capital cost	**358**
Project capital	**262**
Sustaining capital	**96**

Table 21.14 Project and sustaining capital cost estimate

Capital cost estimate	Yr0	Yr1	Yr2	Yr3	Yr4	Yr5	Yr6	Yr7	Yr8	Yr9	Yr10	Yr11	Yr12	Yr13	Yr14	Total
Project capital (C$M)	85	102	32	42	-	-	-	-	-	-	-	-	-	-	-	262
Sustaining capital (C$M)	-	-	-	-	24	21	20	7	2	13	5	1	1	1	1	**96**
Total (C$M)	**85**	**102**	**32**	**42**	**24**	**21**	**20**	**7**	**2**	**13**	**5**	**1**	**1**	**1**	**1**	**358**

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$0.75. The cost estimate was prepared with a base date of Year 0 and does not include any escalation beyond this date. 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 30% 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 C$150M pre-tax NPV and C$86M post-tax NPV at 5% discount rate, pre-tax IRR of 20% and post-tax IRR of 14%. Project capital is estimated at C$262M with a payback period of 7 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	6,444
Total Waste Production	kt	18,686
Gold Grade (1)	g/t	3.3
Silver Grade (1)	g/t	77
Lead Grade (1)	%	0.7%
Zinc Grade (1)	%	0.8%
Gold Recovery (1)	%	94%
Silver Recovery (1)	%	88%
Lead Recovery (1)	%	83%
Zinc Recovery (1)	%	84%
Gold price	US$/oz	1,200
Silver price	US$/oz	16.00
Lead price	US$/lb	0.80
Zinc price	US$/lb	0.85
Gold Payable (2)	%	97%
Silver Payable (2)	%	81%
Lead Payable (2)	%	62%
Zinc Payable (2)	%	52%
Payable Gold metal	kg	19,606
Payable Silver metal	kg	353,457
Payable Lead metal	Tonnes	23,233
Payable Zinc metal	Tonnes	23,792
Revenue split by Commodity (Gold)	%	74%
Revenue split by Commodity (Silver)	%	18%
Revenue split by Commodity (Lead)	%	4%
Revenue split by Commodity (Zinc)	%	4%
Total Net Revenue	C$M	1,365
Capital Costs	C$M	358
Operating Costs (Total) (3)	C$M	741
Mine Operating Costs (4)	C$/t	59.7
Process and Tails Storage Operating Costs	C$/t	43.4
Operating Costs (Total) (3)	C$/t	115.0
Operating Cash Cost (AuEQ)	US$/oz AuEQ	651.5
Total All In Sustaining Cost (AuEQ)	US$/oz AuEQ	965.9
Payback Period (5)	Yrs	7
Cumulative Net Cash flow (6)	C$M	266
Pre-tax NPV (7)	C$M	150
Pre-tax IRR	%	20
Post-tax NPV (7)	C$M	86
Post-tax IRR	%	14

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

Table 22.2 Klaza production and cash flow forecast

Mine Production	**Unit / Yr**	**0**	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**	**12**	**13**	**14**	**Total**
Open pit - (Klaza, BRX Central, West)	kt	-	380	373	245	144	163	-	-	-	-	-	-	-	-	-	**1,305**
Underground - (Klaza, BRX Central, West)	kt	10	90	177	305	406	391	551	550	549	509	524	470	310	203	93	**5,139**
Total mined – Mineralized rock	kt	10	470	550	550	550	554	551	550	549	509	524	470	310	203	93	**6,444**
Open pit - waste (Klaza, BRX Central, West)	kt	4,306	4,253	4,044	3,892	973	-	-	-	-	-	-	-	-	-	-	**17,467**
Underground - waste (Klaza, BRX Central, West)	kt	127	144	85	157	203	166	155	46	8	104	23	0	-	-	-	**1,219**
Total mined - Waste	kt	4,433	4,398	4,128	4,048	1,176	166	155	46	8	104	23	0	-	-	-	**18,686**
Total Development - Lateral	m	2,254	3,690	2,782	4,527	5,467	5,420	6,586	3,558	2,059	3,320	1,601	236	145	34	-	**41,680**
Total Development - Vertical	m	115	199	33	318	251	341	98	-	-	10	69	-	-	-	-	**1,435**
Stockpile (Klaza, BRX Central, West)	kt	10	10	10	10	10	14	15	15	14	-	-	-	-	-	-	**108**
Total Mill Feed	kt	-	470	550	550	550	550	550	550	550	523	524	470	310	203	93	**6,444**
Gold	g/t	-	3.7	3.7	4.3	4.8	2.9	4.0	3.7	3.4	2.5	2.2	2.3	2.6	2.2	2.5	**3.3**
Silver	g/t	-	49	49	72	1100	57	98	102	95	95	82	86	44	50	24	**77**
Lead	%		0.6%	0.8%	0.7%	1.0%	0.6%	0.6%	0.7%	0.6%	0.8%	0.8%	0.9%	0.6%	0.3%	0.2%	**0.7%**
Zinc	%		1.1%	1.0%	1.0%	1.0%	0.7%	0.7%	0.8%	0.8%	0.8%	0.9%	0.8%	0.5%	0.6%	0.7%	**0.8%**
Recoveries																	
Overall Gold Recovery	%	-	94%	94%	94%	94%	94%	94%	94%	94%	94%	94%	94%	94%	94%	94%	**94%**
Overall Silver Recovery	%	-	88%	88%	88%	88%	88%	88%	88%	88%	88%	88%	88%	88%	88%	88%	**88%**
Overall Lead Recovery	%	-	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%	**83%**
Overall Zinc Recovery	%	-	84%	84%	84%	84%	84%	84%	84%	84%	84%	84%	84%	84%	84%	84%	**84%**
Total Payable Metal																	
Gold	kg	-	1,570	1,834	2,167	2,419	1,450	1,993	1,867	1,724	1,187	1,042	993	742	407	211	**19,606**
Silver	kg	-	16,316	19,180	28,033	39,309	22,366	37,261	40,168	37,210	35,559	30,624	28,728	9,822	7,296	1,584	**353,457**
Lead	t	-	1,554	2,166	1,955	2,764	1,813	1,736	2,030	1,688	2,020	2,066	2,122	900	308	110	**23,233**
Zinc	t	-	2,228	2,423	2,391	2,413	1,663	1,702	2,018	1,815	1,836	2,162	1,677	648	511	304	**23,792**
Overall Gold Payable	%	-	97%	97%	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%	81%	**81%**
Overall Lead Payable	%	-	62%	62%	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%	52%	**52%**
Total Net Revenue	**C$M**	**-**	**101**	**118**	**141**	**164**	**98**	**136**	**133**	**123**	**95**	**85**	**80**	**49**	**28**	**13**	**1,365**
Operating Costs																	
Mining	C$M	15	26	32	37	30	25	32	32	32	30	31	27	18	12	5	384
Processing & Tailings Storage	C$M	-	22	24	24	25	23	23	23	24	22	22	21	12	9	4	279
General & Administration	C$M	-	6	7	7	7	7	7	7	7	6	6	6	4	2	1	77
Total Operating Cost	C$M	15	53	63	68	62	54	62	62	63	58	59	54	34	24	11	741
Capital Costs																	
Project Capital	C$M	85	102	32	42	-	-	-	-	-	-	-	-	-	-	-	262
Sustaining Capital	C$M	-	-	-	-	24	21	20	7	2	13	5	1	1	1	1	96
Total Capital Cost	C$M	85	102	32	42	24	21	20	7	3	13	5	1	1	1	1	358
Undiscounted Cash flows (pre-tax)	C$M	(100)	(54)	23	32	78	23	55	64	57	24	22	25	13	3	1	266
Undiscounted Cash flows (post tax)	C$M	(100)	(54)	20	22	59	20	41	50	45	18	18	20	12	3	1	176
Discounted Cash flows (pre-tax)	C$M	(95)	(49)	20	26	61	18	39	44	37	15	13	14	7	2	1	150
Discounted Cash flows (post-tax)	C$M	(95)	(49)	17	18	46	15	29	34	29	11	11	11	6	2	1	86

Totals may not add up exactly due to rounding.

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 15% positive or negative change in metal prices, operating costs, capital costs and corporate tax 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 mining operating costs and total capital costs. The NPV is moderately sensitive to changes in processing operating costs and changes in the silver price. Sensitivity to changes in the lead price, zinc price and corporate tax rate is minimal. Note in Figure 22.1 and Figure 22.2, lead price and zinc price follow the same line, and so do mining operating costs and total capital costs.

Table 22.3 Klaza economic sensitivity analysis (pre-tax)

Item	Value	Unit	Pre-tax NPV (C$M)	Pre-tax IRR %
Base Case (NPV @ 5%)			**150**	**20%**
Gold price - fall 15%	1,020	US$/oz	41	9%
Gold price - increase 15%	1,380	US$/oz	259	30%
Silver price - fall 15%	13.60	US$/oz	125	18%
Silver price - increase 15%	18.40	US$/oz	175	22%
Lead price - fall 15%	0.68	US$/lb	144	19%
Lead price - increase 15%	0.92	US$/lb	156	20%
Zinc price - fall 15%	0.72	US$/lb	144	19%
Zinc price - increase 15%	0.98	US$/lb	156	20%
Mining operating cost - decrease 15%	50.7	C$/t	191	24%
Mining operating cost - increase 15%	68.6	C$/t	109	16%
Processing operating cost - decrease 15%	36.6	C$/t	179	22%
Processing operating cost - increase 15%	49.5	C$/t	121	17%
Total Capex - decrease 15%	304	C$M	188	26%
Total Capex - increase 15%	411	C$M	111	15%

Table 22.4 Klaza economic sensitivity analysis (post-tax)

Item	Value	Unit	Post-tax NPV (C$M)	Post-tax IRR %
Base Case (NPV @ 5%)			**86**	**14%**
Gold price - fall 15%	1,020	US$/oz	9	6%
Gold price - increase 15%	1,380	US$/oz	163	22%
Silver price - fall 15%	13.60	US$/oz	68	12%
Silver price - increase 15%	18.40	US$/oz	104	16%
Lead price - fall 15%	0.68	US$/lb	82	14%
Lead price - increase 15%	0.92	US$/lb	90	15%
Zinc price - fall 15%	0.72	US$/lb	81	14%
Zinc price - increase 15%	0.98	US$/lb	90	15%
Mining operating cost - decrease 15%	50.7	C$/t	116	17%
Mining operating cost - increase 15%	68.6	C$/t	55	11%
Processing operating cost - decrease 15%	36.6	C$/t	107	16%
Processing operating cost - increase 15%	49.5	C$/t	64	12%
Total Capex - decrease 15%	304	C$M	116	19%
Total Capex - increase 15%	411	C$M	55	10%
Corporate tax rate - decrease 15%	25%	%	95	15%
Corporate tax rate - increase 15%	35%	%	76	13%

Figure 22.1 Sensitivity analysis – pre-tax NPV at 5% discount rate

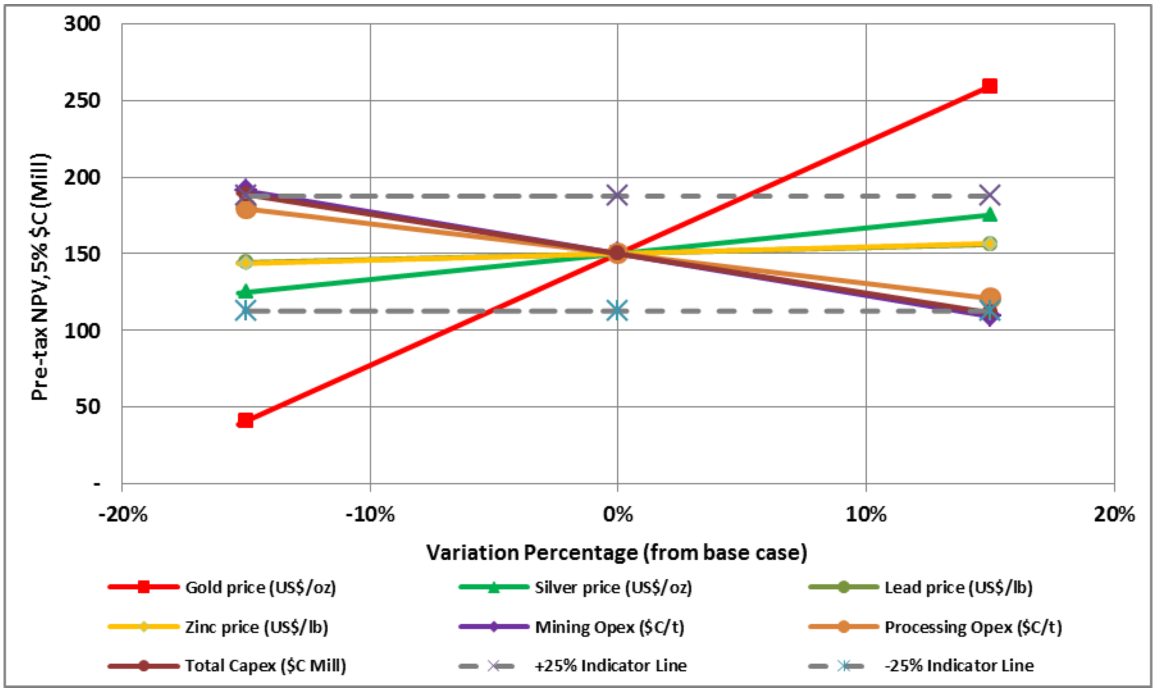
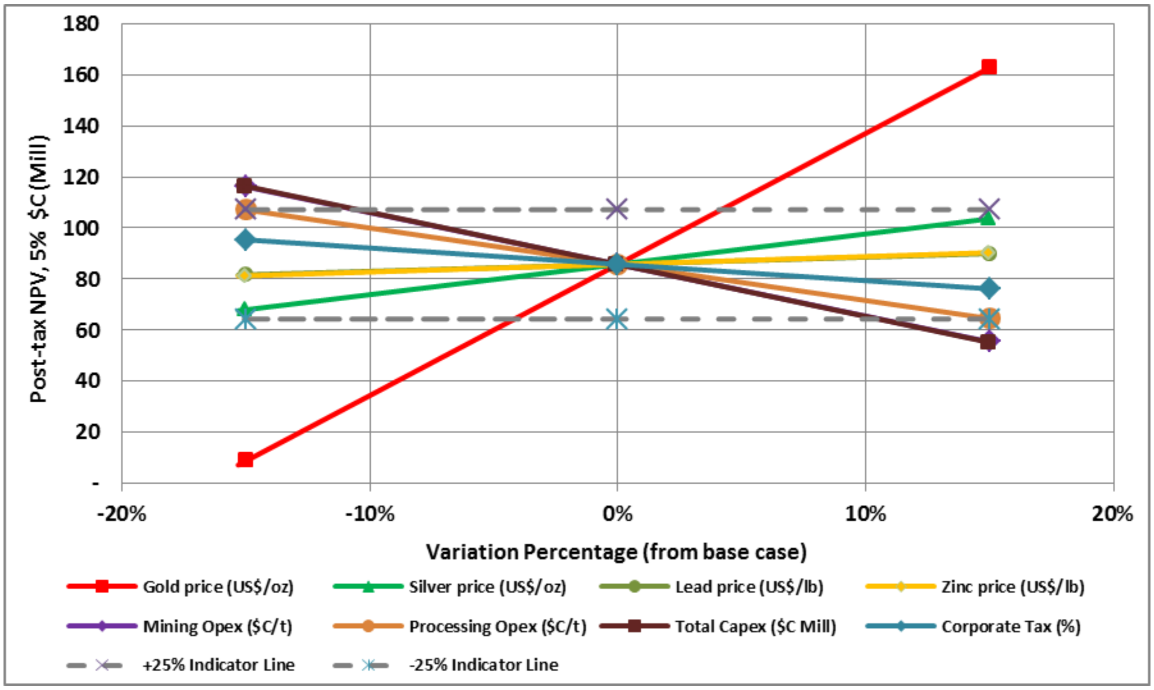


Figure 22.2 Sensitivity analysis – post-tax NPV at 5% discount rate



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 qualified person has not verified the information provide below for the 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 northwest of the BRX and Klaza zones and another operation lies immediately southeast 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, but none of the placer claims overlap any of the Mineral Resources on the Property.

Production figures for these placer gold operations are spotty, with total cumulative production reported at 2,692 ounces. The best cumulative production period took place during the past six years (2009-2014) with a reported total of 1,267 ounces (Bond, 2014). The available figures do not attribute ounces to specific operations, but rather state production by area or region.

Creeks and tributaries draining the southern and southeastern portion of the ridge hosting the mineralized structural zones on the Property, flow into Nansen Creek. Placer gold was discovered in 1899, but early production figures are not available. Total gold production from Nansen Creek from 1980 to present is reported at 26,646 ounces (Bond, 2014).

The Mount Nansen lode gold-silver property is located five kilometres southeast of the Property. 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 northwesterly 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.

There are a number of other gold-silver showings on properties owned by other parties within five kilometres 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

The PEA only considers the western and central portions of the BRX and Klaza zones, all of which contain similar mineralization and will be processed in a similar fashion. The Eastern BRX zone is located approximately one kilometre east and along strike from the Western BRX zone. The Eastern BRX zone contains gold-silver-copper rich mineralization. Preliminary evaluation of the Eastern BRX zone indicates that the Inferred Mineral Resources could be extracted by open pit followed by underground mining methods. An Inferred Mineral Resource of approximately 1 Mt at an AuEQ grade of approximately 3.7 g/t has been identified as potential for future consideration and exploration.

The deposit is open at depth and along strike and is seen as potential upside for the future.

No detailed technical work or economic assessment has been carried out on this resource at this time.

25 Interpretation and conclusions

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.

25.1 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.

The Mineral Resource for the Klaza Property is entirely classified as an Inferred Mineral Resource. An Inferred Mineral Resource is that part of a Mineral Resource for which tonnage, grade and mineral content can be estimated and reasonably assumed, but not verified. Conceptual economic projections based on an Inferred Mineral Resource must, therefore, be regarded with an appropriate degree of uncertainty, recognizing that it cannot be assumed that all or any part of an Inferred Mineral Resource will be upgraded to an Indicated or Measured Mineral Resource as a result of continued exploration.

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 TEST WORK	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 test work; further review of test work 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 test work; further review of test work done to date.
	Grinding	Mill throughput	Final mineralized rock size of 70 microns P80 not achieved or coarser grind size adequate.	Reduction in gold recovery.	Ongoing sampling and metallurgical test work; further review of test work done to date.
	Gold Recovery	Gold recovery	Gold recovery lower than test work indicates.	Reduced revenue and negative impact on Project economics.	Ongoing sampling and metallurgical test work; further review of test work done to date. Mineralized rock batching from specific deposits.
	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 test work.
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	Assume travel to and from Carmacks daily.	Unable to establish sufficient housing for mine workforce in Carmacks; inefficient costly operation.	Camp site is a potential alternative, explore availability of resources locally.
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.

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 TEST WORK	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 test work; further review of test work 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	Production Schedule	Optimization	Optimize the open pit to underground interface, sequence and timing of various zones.	Opportunity to explore increased production levels particularly in the early life of the mine, to make use of the full capacity of the mill.	Alternate production plans and assessment of project zones by value considering sequence and timing.
RECOVERY METHOD	Pre-concentration Pressure oxidation 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. Pressure oxidation tends to be more power-intensive than bioleaching.	Ongoing sampling and metallurgical test work; further review of test work 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

26 Recommendations

26.1 Geology and Mineral Resources

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

- Conduct an infill drilling program over the summer of 2016 with the primary aim of converting Inferred Mineral Resources to Indicated.
- Update the Mineral Resource estimate on completion of the drill program.
- Assuming positive drill program results, undertake a Pre-Feasibility Sstudy (PFS) based on the new Mineral Resource estimate.
- Expansion of the current Mineral Resource is highly prospective to depth and along strike. Additional drilling designed to extend the Mineral Resources down-dip and along strike should be assigned a high priority.
- Conduct further delineation drilling of the BRX East area, which is immediately east along strike of the current Mineral Resources, to determine whether it is well enough mineralized to warrant further geological and engineering work.
- A CRM reflecting the average grade of the Mineral Resource estimate also be included in the QA/QC program.
- Going forward, duplicate samples are to be taken only from mineralized material.

26.2 Hydrology

AMC concurs with the Tetra Tech EBA recommendations for further hydrogeological assessment at Klaza:

- Seasonal groundwater monitoring should be continued for the existing monitoring and observation wells. A minimum of one year of baseline groundwater data will be required for a project proposal submission under YESAA to assess seasonal changes in groundwater quality and quantity. A future Type A Water Licence application will require a minimum of two consecutive years of baseline data.
- All monitoring wells should be surveyed for their location and elevation of the top of the PVC casing in the summer of 2016 with an accuracy of about ±1 cm or better.
- Given the inferred extent of permafrost at the site, a thorough understanding of the interaction between permafrost and groundwater will be a critical aspect of the hydrogeological baseline characterization at the site; this is required for the environmental assessment and regulatory approval process, as well as for mine engineering purposes. Collecting additional ground temperature and hydrogeological data from the existing observation and monitoring wells is recommended, as well as the drilling of additional wells as required. The data should be used to update the preliminary conceptual hydrogeological model with an emphasis on permafrost-groundwater interaction.
- As the mine planning progresses, additional monitoring wells should be installed in the areas up and down gradient of proposed mine infrastructure, including any proposed infrastructure that may affect groundwater quality and/or quantity during mine construction, operation, or closure (e.g., open pits, underground workings, tailings facility, waste rock dump(s), mineralized rock stock piles, mill, fuel and chemical storage, and camp facilities).
- The groundwater and surface water baseline data collection should be integrated and both data sets be interpreted with respect to groundwater-surface water interaction.

26.3 Geotechnical

A better understanding of the factors affecting open pit and stope stability and the proposed mining method should be gained 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.
- Hydrogeological characterization of the site as per recommendations above.

- 2D-3D modelling with updated parameters to assess stope, open pit, and crown pillar dimensions and stability (between open pit and underground).
- A geotechnical diamond drilling program with oriented core should be carried out to assist in increasing the geotechnical understanding of the test area. Triple tube drilling is important in order to achieve a sufficiently high drilling quality; this is the international standard for geotechnical drilling.
- A laboratory testing program should be performed 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).
- A Ground Control Management Plan (GCMP) should be developed, with ongoing geotechnical data collection and interpretation, and updated annually. Geotechnical design should reference updated parameters as reported in the GCMP, and as confirmed through field mapping.
- 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 PFS:

- Re-evaluate open pit and underground mining opportunities for the updated Mineral Resource estimate based on an Indicated Mineral Resource.
- 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.
- Re-evaluate currently planned production rate based on the PFS Mineral Resource estimate.
- Optimize development and production schedules.
- Undertake geotechnical drilling within the proposed crown pillars to better define pillar sizes, particularly for the proposed Klaza Pit 1, where tailings would be stored.
- Project groundwater inflow to the proposed pits and underground mines from updated hydrogeological modelling.
- Undertake first principles 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:
  - Mine workforce would be based in Carmacks and bused to and from site on a daily basis.
  - Sufficient local grid power is available.

26.5 Processing and metallurgical testwork

BCM recommends the following for the Klaza project:

- Eastern BRX process development: Should the BRX East mineralization be furthered investigated for prospective mining viability, a process needs to be developed to produce separate copper, lead and zinc concentrates, mainly to allow for payment of the precious metals that would be associated with each of these products. Selective flotation of arsenopyrite should also be explored as, although the arsenic grades are low in Eastern BRX, the arsenopyrite could still contain a significant amount of refractory gold.
- Pre-concentration development: Visually, Klaza materials appear to be candidates for pre-concentration at a coarse crush size. Pre-concentration could allow for the delivery of more of the waste to the mill as the waste with whatever mineralization it may hold only needs to carry the crushing and pre-concentration costs. Waste may host values contained in narrow stringers that could become recoverable with such a process. Pre-concentration may also reduce the size of the grinding and flotation circuits, which could reduce capital costs.
- Demonstration of workability of flowsheet on pre-concentrate: If pre-concentration is proven, then the effect of pre-concentration on downstream metallurgy needs to be established.

- Grindability testing: Grindability data will be needed on the individual zones to ensure the mill as designed can handle the different materials.
- Upgrade testing on lead and zinc concentrates: Presently the lead and zinc concentrates leave opportunity for improvement by (a) lowering the arsenic grade in the lead concentrate and (b) raising the zinc grade in the zinc concentrate.
- Evaluate POX vs BIOX: If the project advances to PFS, pressure oxidation should be evaluated vs bioleaching. Pressure oxidation tends to be more power-intensive than bioleaching, but uses less cyanide. As it is likely that both will be effective, it is recommended that this trade-off is conducted initially as a paper exercise so as to identify if bioleaching offers any potential advantage over pressure oxidation, before testwork is initiated to validate the workability of bioleaching.
- Evaluate the Albion process: If the concentrates are amenable to Albion, it is possible that this would offer both capital and operating cost savings over POX. However Albion is not yet a widely used process in refractory gold processing, with the first operation going into production in Armenia in 2016.
- Validate hot cure: The hot cure process has been assumed for the PEA but has not been tested. This needs to be proven in the laboratory, and the impact on lime and limestone consumption needs to be determined.
- Prove-up use of limestone for neutralization: A study is needed to characterize (assay) the candidate limestone sources and to test their effectiveness in neutralization of the hot cure solutions. Further study should be conducted to validate costs of limestone supply once sources are confirmed.
- Testing of the chemical stability of the residues: This testwork is best driven by the project's environmental considerations and would account for the regulatory requirements from a permitting perspective.
- More variability flotation testing: A better picture of the spatially-driven variability in metallurgy needs to be established; this would allow a better prediction of mine life metallurgy to be developed.
- Development of a process water management strategy: The complex process envisaged potentially has several water recirculation systems. These need to be balanced, which for the PFS would likely entail a paper exercise supported with some testwork using recycled water.

26.6 Tailings storage facility

- Evaluate additional tailings storage potential within the Klaza open pits KL1 and 2 by constructing containment dams across the low point of the pit and increasing storage capacity.
- Complete rheology testing and geotechnical testing of the tailings streams.
- Complete geotechnical investigations to evaluate foundation conditions and construction material sources.
- Complete geotechnical investigations at the Klaza KL1 and 2 open pits to determine suitability for tailings storage.

26.7 Environmental

- Continue ongoing and constructive dialogue with Little Salmon/Carmacks First Nation and Selkirk First Nation to keep them informed of project progress.

26.8 Proposed budget for recommendations

An approximate budget for the recommended work described above is presented in Table 26.1.

Table 26.1 Proposed budget for recommendations

Parameter	Cost (C$000's)
Diamond drilling – 30,000 m (including consumables and mobilization)	3,200
Labour for exploration drilling	750
Exploration camp, field gear, rentals, food and consumables	500
Assay and analytical	520
Excavator and fuel	110
Office and Senior supervision	350
Hydrological monitoring and modelling	200
Geotechnical drilling,testwork and modelling and interpretation	200
Metallurgical and Mineralogical studies	360
PFS (Geology and mining components)	800
Logistics, airfares, ground transportation and shipping	75
Expediting and safety	75
Environmental baseline studies to stage ready for EA	600
Aerial photos and other studies	170
Consultants management fee	390
Contingency @ 15%	1,245
Total (excluding Taxes)	**9,545**

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28 Certificates of Qualified Persons

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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 with AMC Mining Consultants (Canada) Limited, 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 PEA for the Klaza Au-Ag Deposit, Yukon, Canada for Rockhaven Resources Ltd., with an effective date of 26 February 2016, (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 of the Association of Professional Engineers and Geoscientists of British Columbia (License #37418) and Alberta (Reg. #52751). I have practiced my profession for a total of 22 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 18th to 19th August 2015 for 2 days.
5. I am responsible for Sections 4 through 12 inclusively, 14 and 23, as well as part of Section 1, 25 and 26 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 "NI43-101 Techncial Report describing Updated Diamond Drilling, Metallurgical Testing and Mineral Resources on the Klaza Property Yukon, Canada", dated 9 December 2015.
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: 26 February 2016
Signing Date: 29 February 2016

Original signed and sealed by

(Signed) "*Adrienne Ross*"

Adrienne Ross, P.Geo.

Knight Piésold CONSULTING

CERTIFICATE OF AUTHOR

I, Bruno Borntraeger, P.Eng (BC)., of Vancouver, British Columbia, do hereby certify that:

1. I am currently employed as a Specialist Geotechnical Engineer with Knight Piesold 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 PEA for the Klaza Au-Ag Deposit, Yukon, Canada for Rockhaven Resources Ltd., with an effective date of 26 February 2016, (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 Association of Prefessional Engineers and Geoscientists of British Columbia (License #20926), and a member of the the Association of Professional Engineers Yukon. I have practiced my profession continuously for 25 years. I have been directly involved in geotechnical enfgineering, mine waste and water management, mine development with practical experience from feasibility studies, detailed engineering, construction, operations and closure.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I have not visited project site.
6. I am responsible for parts of Sections 1, 17 and 26 in the Technical Report.
7. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101.
8. I have not had prior involvement with the property that is the subject of the Technical Report.
9. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.
10. As of the effective date of the Technical Report and the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: 26 February 2016
Signing Date: 29 February 2016

Original signed and sealed by

(Signed) "*Bruno Borntrager*"

Bruno Borntrager, P. Eng. (BC)

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 PEA for the Klaza Au-Ag Deposit, Yukon, Canada for Rockhaven Resources Ltd., with an effective date of 26 February 2016, (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 30 years. I have experience in mineral processing operations management, and plant support and flowsheet development from roughly 400 projects located worldwide.
4. 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.
5. I have not visited the Property, subject of this Technical Report.
6. I am responsible for Sections 13, 17 and 19 and parts of Sections 1, 25 and 26 of the Technical Report.
7. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101.
8. I have had prior involvement with the Property that is the subject of the Technical Report.; I co-authored the Technical Report entitled "NI43-101 Technical Report describing Updated Diamond Drilling, Metallurgical Testing and Mineral Resources on the Klaza Property Yukon, Canada", dated 9 December 2015 as well as "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.
9. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.
10. As of the effective date of the Technical Report and the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: 26 February 2016
Signing Date: 29 February 2016

Original signed and sealed by

(Signed) "*Christopher Martin*"

Christopher Martin, C.Eng.

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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 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 PEA for the Klaza Au-Ag Deposit, Yukon, Canada for Rockhaven Resources Ltd., with an effective date of 26 February 2016, (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 member in good standing of the Association of Professional Engineers and Geoscientists of 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, feasibitly 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 18th to 19th August 2015 for 2 days.
5. I am responsible for Sections 2, 3, 15, 20, 22, 24, 27 and parts of Sections 1, 16, 21, 25 and 26 of the Technical Report.
6. I am independent of the Issuer and related companies as described in Section 1.5 of NI 43-101.
7. I have had previous involvement with the property that is the subject of the Technical Report; completed a Scoping Study on the Property.
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: 26 February 2016
Signing Date: 29 February 2016

Original signed and sealed by

(Signed) "*Gary Methven*"

Gary Methven, P.Eng.

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CERTIFICATE OF AUTHOR

I, Philippe Lebleu, P.Eng., of Vancouver, British Columbia, do hereby certify that:

1. 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.
2. This certificate applies to the technical report titled "Technical Report and PEA for the Klaza Au-Ag Deposit, Yukon, Canada for Rockhaven Resources Ltd., with an effective date of 26 February 2016, (the "Technical Report") prepared for Rockhaven Resources Limited.("the Issuer").
3. I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia (registration #41544), and a member of the Australian Institute of Mining and Metallurgy. I graduated from The Royal School of Mines, Imperial College in London, England with a Masters of Mining Engineering with Rock Mechanics in 1999. I have extensive operational experience in iron ore, copper and aggregates in Canada, Australia and Malaysia, with mines moving up to 90Mtpa. My consulting experience spans various commodities and includes project and site work in the Americas, Africa and arctic environments.

   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 parts of Sections 1, 16, 21 and 26 of the Technical Report.
6. I am independent of the Issuer and related companies as described in Section 1.5 of NI 43-101.
7. I have had previous involvement with the Property that is the subject of the Technical Report.
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: 26 February 2016
Signing Date: 29 February 2016

Original signed and sealed by

(Signed) "*Philippe Lebleu*"

Philippe Lebleu, P.Eng.

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CERTIFICATE OF AUTHOR

I, William Hughes, P.Eng., of Vancouver, British Columbia, do hereby certify that:

1. I am currently employed as a Principal Mechanical / Infrastructure Engineer 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 PEA for the Klaza Au-Ag Deposit, Yukon, Canada for Rockhaven Resources Ltd., with an effective date of 26 February 2016, (the "Technical Report") prepared for Rockhaven Resources Limited.("the Issuer").
3. I graduated from the University of Saskatchewan in Saskatoon, Canada with a Bachelors of Science – Mechanical Engineering in 1989. I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia (License #110586), and Saskatchewan (Reg#09130). I have experience in the mining industry consisting of practical problem solving for maintenance and capital projects. I have designed and constructed mine clarification and dewatering systems, ventilation systems, materials handling, hoisting, and surface infrastructure. I have extensive experience in maintenance programs and the analysis of operating costs versus capital costs in order to optimize preventative maintenance and asset management.

   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, 5, 25 and 26 of the Technical Report.
6. I am independent of the Issuer and related companies as described in Section 1.5 of NI 43-101.
7. I have not had prior involvement with the Property.
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: 26 February 2016
Signing Date: 29 February 2016

Original signed and sealed by

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