NI 43-101 TECHNICAL REPORT
AKIE PROJECT
BRITISH COLUMBIA, CANADA

JDS Energy & Mining Inc.

ZincX Resources TSX V: ZNX

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NOTICE

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

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

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1 Executive Summary

1.1 Introduction

The Akie Project (also known as the Cardiac Creek Project, Cardiac Creek, Akie or the Project) is 100% owned by ZincX Resources Corp. (ZincX), a public company which trades on the TSX-Venture Exchange (TSX-V) under the symbol ZNX. ZincX was formerly known as Canada Zinc Metals Corp. (CZM or Canada Zinc Metals) and the new company name was adopted 7 May 2018.

JDS Energy & Mining Inc. (JDS) was commissioned by ZincX to compile a Preliminary Economic Assessment (PEA) for the Akie Project. This Technical Report summarizes the results of the PEA and is prepared according to the guidelines of the Canadian Securities Administrator's National Instrument 43-101 (NI 43-101) and Form 43-101F1.

JDS managed the PEA and completed the mining, mineral processing, metallurgical testing, infrastructure, and economics sections of the report. JDS was assisted by several ZincX designated sub-contractors to provide report information as noted below:

- Sim Geological Inc. (SGI): property description, geology and mineral resources;
- Knight Piésold Ltd. (KP): mine closure, environment and permitting, mine waste and water management; and
- Lorax Environmental Services Ltd. (Lorax): geochemistry and hydrogeology.

The PEA is preliminary in nature and includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them to be categorized as Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. There is no certainty that the project presented in the PEA will be realized.

1.2 Project Description and Ownership

Akie is a zinc-lead-silver property located in north-central British Columbia, Canada. It is 100% owned by ZincX, a junior exploration company headquartered in Vancouver, BC.

The Akie property is situated within the southernmost area (Kechika Trough) of the regionally extensive Paleozoic Selwyn Basin. This sedimentary basin is known for its proliferation of sedimentary exhalative (SEDEX) zinc-lead-silver and stratiform barite deposits.

1.3 Geology and Mineralization

The Akie property is located within the Gataga District of the Kechika Trough. The trough represents a narrow, elongated extension of the large sedimentary basin known as the Selwyn Basin. The Selwyn Basin is host to numerous SEDEX-type mineral deposits (e.g. Howards Pass, Tom, Jason, Faro). Upper Devonian to Mississippian basinal facies clastic sedimentary rocks known as the Earn Group are a regional target for SEDEX type zinc-lead-silver deposits within the Kechika Trough. The Earn Group can be sub-divided into the Warneford, Akie and Gunsteel Formations. The prospective black siliceous shale of the Gunsteel Formation is the primary host to most of the deposits, prospects and occurrences within the district and is the primary target of exploration activities. Known deposits in the district include Cardiac Creek (the subject of this report), North Cirque, South Cirque, and Driftpile. Advanced prospects include, Elf, Fluke, Mt. Alcock and Bear / Spa.

The geology of the Akie property can be split into east and west segments by Silver creek. To the west of Silver creek, rocks of the Kechika Group and Road River Group are imbricated and in thrust contact with an approximately 500 metre (m) thick panel of Earn Group rocks comprised primarily of the Gunsteel Formation shales that host the Cardiac Creek deposit. This panel of prospective rocks represents an eastern limb of an overturned syncline and the steeply dipping western limb of a large anticline that straddles Silver Creek. This panel of Earn Group rocks can be traced along the entire length of the Akie property with an approximate strike length of 8 kilometres (km) that extends onto the adjacent properties (Elf and Fluke).

Discovery of the Cardiac Creek deposit in 1994 by Inmet Mining Corp. (Inmet Mining or Inmet) is recent in comparison to the other known deposits and occurrences all of which were discovered prior to 1980. The deposit was discovered by prospecting along a steeply inclined mountain creek dubbed Cardiac Creek. In general, the Cardiac Creek deposit is situated proximal to the base of the Gunsteel Formation and near the contact between the Gunsteel Formation and Road River Group. The contact is typically separated by a thin sliver of debris flow associated with the Paul River Formation. The deposit is interpreted to be a SEDEX-type body of Pb-Zn-Ag mineralization represented by a “sheet-like” tabular body of stratabound sulphides interbedded with black siliceous shales that trends NW-SE, striking at 130 degrees and dipping at 70 degrees to the southwest. The portion of the deposit that comprises the mineral resource presented in this report measures approximately 1,500 m along strike and extends to 850 m below surface. The deposit ranges in thickness from 5 m to 50 m and the mineralized horizon can be traced over 8 km from the Bear Valley Creek down to the Akie River. The mineralogy of the deposit is dominated by pyrite, barite, sphalerite and galena. Based on the mineralization, character and textures the deposit can be broken into two main mineral facies, the Proximal Facies dominated by thick pyrite rich laminar sulphide beds and the Cardiac Creek Zone facies characterised by sphalerite and galena rich sulphide bands present within the thick laminar bedded pyrite.

1.4 Metallurgical Testing and Mineral Processing

Historical metallurgical testing was performed on Cardiac Creek samples by SGS Mineral Services (SGS) in 2005, 2007 and 2008, and G&T Metallurgical Services Ltd. (G&T) in 2009 and 2010.

The most recent test program to evaluate the Cardiac Creek deposit was completed by Base Metallurgical Laboratories Ltd. (Base Met) in 2018. The program included mineralogy, comminution, dense media separation (DMS), and rougher/cleaner Pb and Zn sequential flotation. Five variability composite samples, representing the deposit were tested to develop a preliminary recovery flowsheet and associated flotation conditions. From the five variability composites a global composite was created, and locked cycle testing was completed.

QEMSCAN analysis using PMA shows that approximately 39% of the material content is sulphide, mainly pyrite, sphalerite and galena, and the rest is gangue. Mineralogy indicates galena and sphalerite liberation ranges from 27% to 38% at approximately 80% passing (P_{80}) 56 microns.

Comminution testing found that the Bond ball mill work index for the five variability composites ranged from 14.2 kWh/t to 18.1 kWh/t. The global composite can be classified as moderately hard with a bond ball mill work index of 16.9 kWh/t.

Based on the results from Base Met (2018), a dense media separation (DMS) circuit will be used to pre-concentrate the sulphide minerals at an SG of 2.8, rejecting 25% of the material as waste prior to Pb and Zn sequential flotation. Saleable Pb and Zn concentrates can be produced with a primary grind size of 80% passing (P_{80}) 56 µm, and rougher concentrate regrind sizes of 10 µm for Pb and 15 µm for Zn. For the global composite, locked cycle flotation test results achieved recoveries of 46.2% Pb and 88.8% Zn at concentrate grades of 45.1% Pb and 52.4% Zn. A summary of the BL0148-LCT21 results are shown in Table 1-1.

Table 1-1: Locked Cycle Testing Results BL0148-LCT21

Product	Weight (%)	Assay (% or g/t)						Distribution (%)					
		Pb	Zn	Fe	S	C	Ag	Pb	Zn	Fe	S	C	Ag
Feed (Float feed after DMS)	100	1.9	9.6	9.3	18.8	2.0	17.0	100	100	100	100	100	-
Pb Con	1.9	**45.1**	6.5	10.4	26.4	2.3	43.0	**46.2**	1.3	2.1	2.7	2.2	4.8
Zn Con	16.4	2.9	**52.4**	5.3	35.2	0.7	19.0	25.5	**88.8**	9.3	30.7	6.0	18.3

*Note: Lead and Zinc results are a weighted average from Lock Cycle Tests BL0148 LCT21 cycles D&E
Source: Base Met (2018)

These results were used to predict the estimated Pb and Zn concentrate grades and recoveries for the economic model.

1.5 History, Exploration and Drilling

Drilling on the Akie property by Inmet Mining Corp. (from 1994 to 1996), Canada Zinc Metals and ZincX (2005 to 2017) have identified a significant body of baritic zinc-lead-silver SEDEX mineralization known as the Cardiac Creek deposit. This drilling has defined a large tabular body of mineral resources that has approximate dimensions of 1,500 m in strike length, a dip extent of 850 m and true thicknesses approaching 50 m. The deposit is hosted by siliceous, carbonaceous, fine-grained clastic rocks of the middle to late Devonian Gunsteel Formation.

1.6 Mineral Resource Estimates

The previous resource estimate was described in a Technical Report dated 28 June 2016 (available on www.sedar.com) with an effective date of 16 May 2016. It outlined an Indicated resource of 19.6 million tonnes (Mt), averaging 8.17% Zn, 1.58% Pb, and 13.6 g/t Ag (at a 5% Zn cut-off grade), and an Inferred resource of 8.1 Mt, averaging 6.81% Zn, 1.16% Pb, and 11.2 g/t Ag (at a 5% Zn cut-off grade). Since the June 2016 estimate of mineral resources, ZincX has completed a drilling program comprising of eight diamond drill holes that have increased the area of the deposit that is delineated with 100 m spaced drill holes, resulting in an increase in resources in the Indicated category. The estimate of mineral resources is summarized in Table 1-2.

Table 1-2: Estimate of Mineral Resources (5% Zinc Cut-off)

		Average Grade			Contained Metal		
Category	Tonnes (million)	Zn (%)	Pb (%)	Ag (g/t)	Zn (Mlbs)	Pb (Mlbs)	Ag (Moz)
Indicated	22.7	8.32	1.61	14.1	4,162	804	10.3
Inferred	7.5	7.04	1.24	12.0	1,169	205	2.9

Notes:
1. Mineral resources are not mineral reserves because the economic viability has not been demonstrated.
2. The effective date of the mineral resource is November 2017.
Source: Sim (2017)

The estimate of mineral resources incorporates all drilling conducted by ZincX on the Cardiac Creek deposit since 2005 plus 29 holes drilled by Inmet Mining Corp. between 1994 and 1996. Currently, there are 151 drill holes on the Akie property with a total core length of 64,352 m. Of these 151 drill holes, 116 of them, totaling 51,978 m, are within close enough proximity of the block model to contribute to the estimation of the mineral resources. The remaining 35 drill holes test the zone over a total strike length of almost 7 km, or they test other exploration targets on the property.

The mineral resource estimate presented in this report has been generated from drill hole sample assay results and the interpretation of a geological model which relates to the spatial distribution of zinc, lead and silver. Interpolation characteristics have been defined based on the geology, drill hole spacing and geostatistical analysis of the data. The resources have been classified by their proximity to the sample locations and are reported, as required by NI 43-101, according to the CIM Definition Standards for Mineral Resources and Mineral Reserves (2014). Extensive analysis of the drill sample database shows that it is sound and reliable for the purposes of resource estimation. The resource model has been developed in accordance with accepted industry standards resulting in a mineral resource defined within the Indicated and Inferred categories.

The resources, shown in Table 1-2, are summarized based on a 5% zinc cut-off grade which is based on assumptions derived from operations with similar characteristics, scale and location. The distribution of Indicated and Inferred mineral resources above a cut-off grade of 5% Zn, occurs as a continuous zone which is favourable with respect to selectivity and other factors when considering possible mining options. The current resource extends to a maximum depth of 850 m below surface. The true thickness of the base case resource typically ranges between 8 m and 50 m, with an average of about 20 m. The shape, scale and location of the deposit indicates that it is potentially amenable to underground mining methods and, as a result, the stated resource is considered to exhibit reasonable prospects for eventual economic extraction.

1.7 Mining Methods

The Akie deposit will be mined using mechanized longitudinal long-hole as the sole mining method.

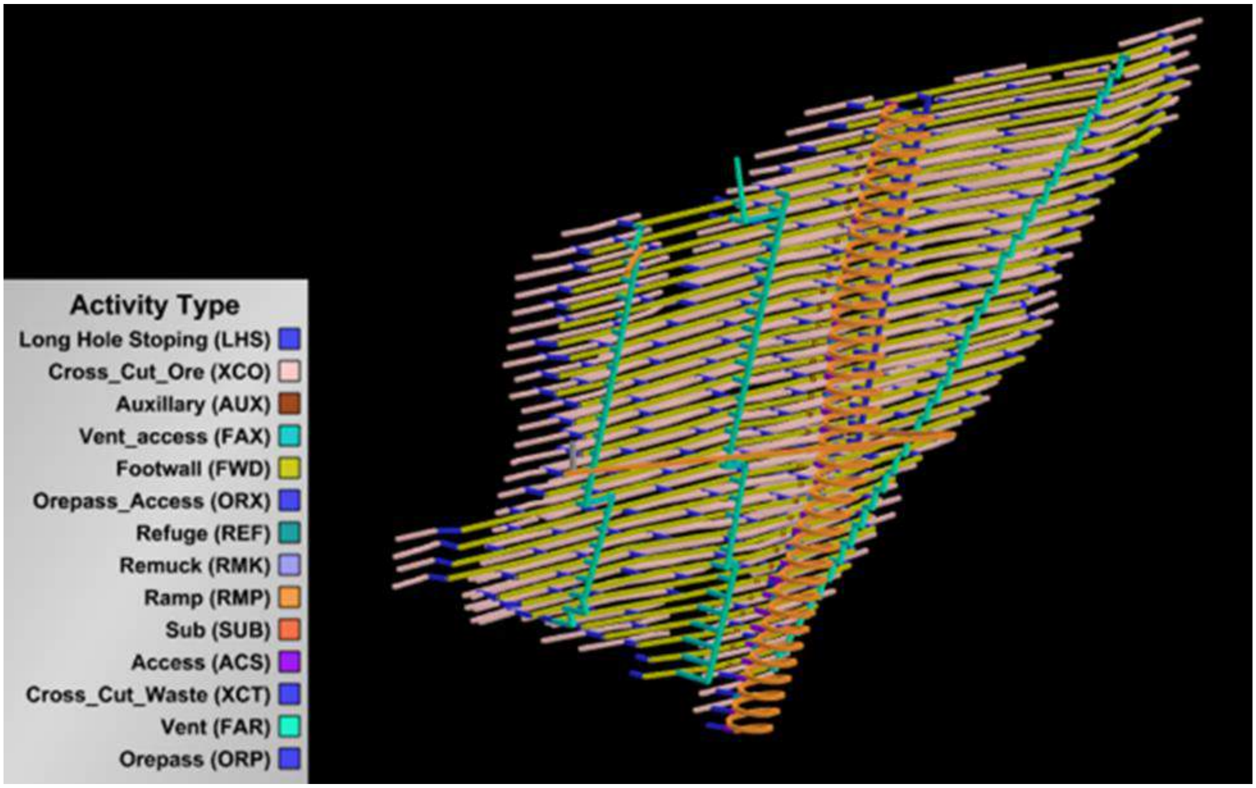
The mine will be accessed using one primary decline, driven at 5.5 m wide (mW) x 6.0 m high (mH) from Portal One, located at 1055 mASL. This decline will be sized to accommodate the necessary ventilation ducting and services and will be used for all haulage from the mine. It will also act as a fresh air feed into the mine, with a primary fan and heater located at the portal.

A second portal will be collared up-slope at 1220 mASL to provide secondary access and egress for the mine and to act as a secondary fresh air feed. It will also be equipped with ventilation fans and heaters. The high air flow requirement in this heading will prohibit its use for regular vehicular access in and out of the mine.

Vertical development will include one 4 m diameter raise-bored production pass from 920 mASL to 1320 mASL and three ventilation raises. One fresh-air raise and one return air raise will be driven by a raise bore. A second fresh air raise system will be developed by connecting a series of conventionally driven drop raises. All fresh air raises will be equipped with manway installations to act as secondary egress.

See Figure 1-1 for an oblique view of the mine layout, showing lateral and vertical development.

Figure 1-1: Akie Development - Oblique View



Source: JDS (2018)

Stope sills will be driven at 5 m x 5 m at 20 m vertical increments. Stopes will in general be a maximum of 20 m long (mL) (along strike), making a typical maximum exposed hanging wall and footwall of 20 mL x 25 mH. Where the orebody is greater than 16.0 m in width, two parallel sill drives will be used to ensure adequate drill coverage and to provide multiple extraction points for mucking.

Nominally, panels will be comprised of 30 individual stopes; six stopes along strike by five stopes high. Thus, the typical panel will have a length of 120 m along the strike and a height of 105 m, spanning five mining levels.

Stopes voids will be filled with a combination of paste, rockfill (RF) and cemented rockfill (CRF). Paste and CRF backfill will be self-standing, allowing the mining of the next, adjacent stope. Stopes less than 10 m thick, located at the outer fringe of the deposit, will be filled with uncemented RF and will incorporate permanent pillars between adjacent stopes to contain the RF.

Stope dilution is approximated by the inclusion of 1 m of hanging wall and footwall material plus 0.5 m of backfill from each exposed surface. Overall dilution was estimated at 13% rock at the grade derived from the geological model (4.7% zinc) plus 3% backfill, which is barren. Mining recovery was estimated at 95% for stopes greater than 10 m wide and 85% for stopes less than 10 m wide.

The total mineable resource is shown in Table 1-3. This does not constitute a mining reserve, as the table contains inferred resources which are not considered to be sufficiently proven geologically for reliance in an economic model.

Table 1-3: Akie Mine Plan by Resource Class

Zone	Tonnes (kt)	Zn Eq (%)	Zn (%)	Pb (%)	Ag (g/t)	NSR ($CAD)
Indicated	20,739	9.0	8.3	1.6	14.1	129.9
Inferred	5,061	7.8	7.2	1.4	13.0	112.8
Total Mine Plan	**25,800**	**8.8**	**7.6**	**1.5**	**13.1**	**126.5**

Notes:

1. Mineral Resources are estimated at a cut-off of 5.5% ZnEq. (ZnEq = Pct Zn + {0.45*Pct Pb})
2. Metal prices used for this estimate were: Zinc 1.17US$/lb; Lead 1.00US$/lb; Silver 16.95US$/oz
3. Mine planning tonnes include an additional 27.5kt of internal dilution at zero grade, which is neither inferred nor indicated.

Source: JDS (2018)

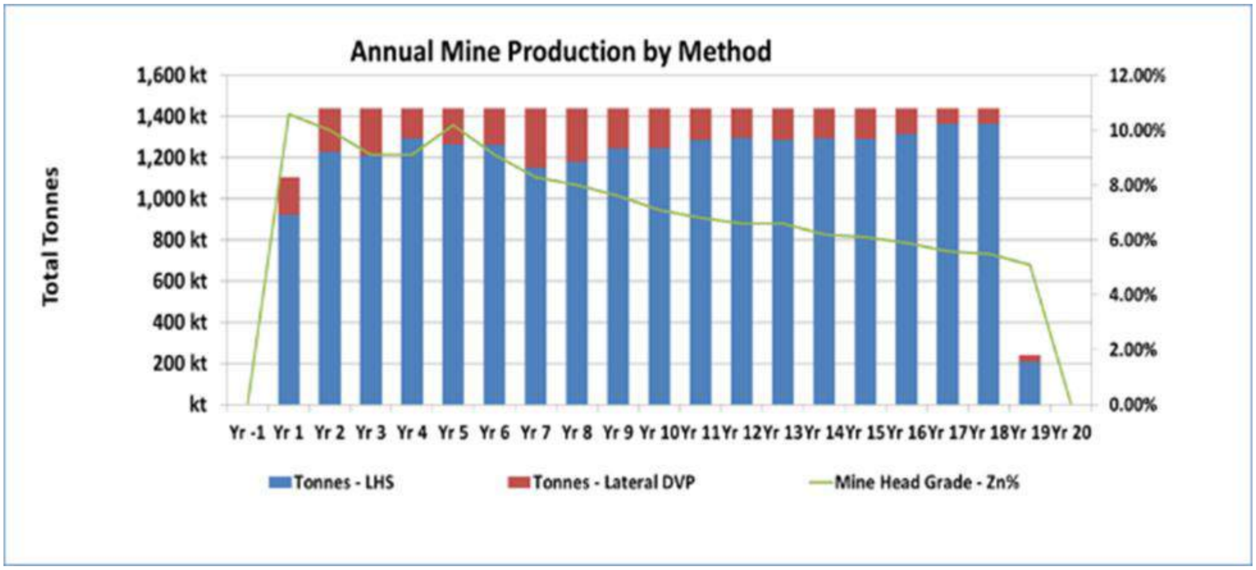
Diesel trackless equipment will be used throughout the mine. A fleet of 14 t scooptrams and 45 t haulage trucks will be used for haulage. Trucks will either be loaded by the scoop trams, or by chute from the production pass.

The mine will require a full-time work force of mining, maintenance, services, technical and administrative personnel. Mine operations will be run 365 d/a at 22 h/d through two – 11 hour shifts, allowing one hour for smoke clearing at shift change.

Two primary facilities will be located underground: the paste backfill plant and the maintenance shop.

Mine production is expected to commence in year one, with 1.1 Mt mined, approximately 80% of the steady-state production rate. The mine is expected to produce at a full production rate of 1.4 Mt/a for 17 years (Years 2 to 18) with production ending in Q1 of Year 19. A summary of the mine production schedule is presented in Figure 1-2.

Figure 1-2: Mine Production Schedule



Source: JDS (2018)

1.8 Recovery Methods

Material from the mine will feed a three-stage crushing plant followed by a dense media separation (DMS) circuit operating on average 18 h/d at a production rate of 4,000 t/d. The process plant will be fed the sink product of the DMS plant at a rate of 3,000 t/d producing saleable Pb and Zn concentrates. The process plant will operate 365 d/a at 24 h/d, with an availability of 92%.

The primary grinding circuit will consist of two identical ball mills, the first operating in open circuit and the second in reverse closed circuit with cyclones to achieve a target grind size of 80% passing (P_{80}) 56 µm. The material will then be fed to sequential Pb and Zn rougher / cleaner flotation circuits. The Pb and Zn regrind circuits will further liberate the rougher concentrates, with a target P_{80} grind sizes of 10 µm and 15 µm, respectively.

The lead and zinc flotation circuits will consist of rougher flotation followed by rougher concentrate regrind and three stages of cleaning. The final concentrates will be thickened then filtered to a target moisture content of 8%. The third cleaner lead concentrate will be bagged and loaded onto trucks. Zinc will be loaded onto trucks as a bulk concentrate for transport to the smelter. The tailings from the process will be thickened, filtered and trucked to the dry stack facility for disposal.

1.9 Project Infrastructure

The project envisions the upgrading and/or construction of the following key infrastructure items:

- Process facilities;
- Natural gas power plant and liquefied natural gas (LNG) receiving and storage facility;
- Tailings management facility (TMF);

- Water management and treatment plant;
- Permanent camp (established for the construction stage);
- Truck shop and warehouse;
- Mine dry and office complex;
- 150,000 L of on-site fuel storage and distribution;
- Industrial waste management facilities such as the incinerator; and
- Site water management facilities and structures.

1.10 Environment and Permitting

1.10.1 Baseline Environmental Studies

Environmental studies and monitoring programs in support of the Akie Project have been conducted over the past 20 years, the majority of which were conducted in 2007 and 2008. Studies included meteorology, surface water quality, hydrogeology, geochemistry, fish and wildlife, and terrain and soils. A detailed gap analysis to develop complementary baseline studies to meet current regulatory expectations will be established at the next stage of development.

1.10.2 Geochemical Considerations

Current geochemical characterization studies conclude that approximately 71% of waste rock is non-Potentially Acid Generating (NPAG), with the remaining 29% considered to be Potentially Acid Generating (PAG). Furthermore, tailings and DMS reject materials are considered to be PAG. Additional geochemical characterization studies will be required to manage waste materials moving forward.

1.10.3 Social and Community

The Akie Project lies within an area of overlap between the respective traditional territories of the Tsay Keh Dene and Kwadacha First Nations, the two communities closest to the Akie Project. ZincX and its predecessors have engaged with both First Nations and provided economic benefits to both communities through community funding, employment, and direct engagement of contractors.

A formal tripartite Exploration, Cooperation and Benefit Agreement was signed in 2013 between ZincX's predecessor, Canada Zinc Metals, and the Kwdacha and Tsay Keh Dene First Nations. This agreement covers all exploration and related activities on shared territory. The agreement is also designed to mitigate any effects of exploration programs on the traditional lands of these First Nations.

1.10.4 Environmental Assessment and Permitting

The Akie Project will need to undergo a Provincial and Federal Environmental Assessment, as well as obtain a number of Provincial and Federal Permits and Authorization. No municipal or regional permits are required for operation of any camps or potable water supplies. The key Provincial and Federal Agencies that will assess the project include:

- BC Environmental Assessment Office;

- BC Ministry of Energy, Mines and Petroleum Resources;
- BC Ministry of Environment and Climate Change Strategy;
- BC Ministry of Forests, Lands and Natural Resource Operations;
- Canadian Environmental Assessment Agency;
- Fisheries and Oceans Canada; and
- Natural Resources Canada.

1.10.5 Mine Closure

The conceptual reclamation and closure plan for the Akie Project will involve an active closure period and a post-closure period, in which all mine components will be prepared for permanent closure. Closure will be completed in a manner that will satisfy physical, chemical and biological stability, as well as follow the applicable regulatory framework. The primary objective of the closure and reclamation initiatives will be to return the surface facilities to a self-sustaining condition with pre-mining usage and capabilities as much as practicable.

1.11 Capital and Operating Cost Estimates

1.11.1 Capital Costs

The capital cost estimate was compiled using a combination of quotations, database costs and factors. The estimate is derived from engineers, contractors, and suppliers who have provided similar services to existing operations and have demonstrated success in executing the plans set forth in the study.

The capital cost (CAPEX) estimate includes the costs required to develop, sustain, and close the operation after an anticipated operating life of 19 years. The construction schedule is based on a 24-month build period. The intended accuracy of this estimate is +/- 35%.

The high-level CAPEX estimate is shown in Table 1-4. The sustaining capital is carried over operating Years 1 through 19, and closure costs are projected for Year 20.

Table 1-4: Summary of Life of Mine Capital Costs

Area	Pre-Production (M$)	Sustaining (M$)	Closure (M$)	Total (M$)
Mining	58.2	260.0	-	318.2
Site Development	7.5	0.7	-	8.2
Mineral Processing	78.8	11.8	-	90.6
Tailings Management	5.0	8.3	-	13.3
On-Site Infrastructure	55.1	6.3	-	61.4
Off-Site Infrastructure	1.0	0.2	-	1.2
Indirect Costs	28.0	5.1	-	33.2
EPCM	17.4	1.5		18.8
Owners Costs	5.6	0.0	-	5.6
Closure Costs	0.0	8.9	8.9	8.9
Subtotal Pre-Contingency	256.7	302.7	**8.9**	559.4
Contingency	45.7	12.9	0.8	58.5
Total Capital Costs	**302.3**	**315.6**	**9.7**	**617.9**

Note:
- Sums may appear incorrect due to rounding.

Source: JDS (2018)

1.11.2 Operating Costs

The operating cost estimate (OPEX) is based on a combination of experience, reference projects, budgetary quotes and factors, as appropriate with a PEA study.

Preparation of the OPEX is based on the JDS philosophy that emphasizes accuracy over contingency and utilizes defined and proven project execution strategies.

Total LOM operating costs are estimated to be C$2,014 M or an average unit cost of C$102.38 /t milled. The LOM costs are summarized in Table 1-5. UG mining costs average C$38.13 /t mined or C$50.05 /t processed.

Table 1-5: LOM Total Operating Cost Estimate

Description	Total Estimate ($ M)	Average Unit Cost ($/t processed)
UG Mining	984.7	50.05
Processing	651.7	33.13
Tailings & DMS rejects	56.5	2.87
G&A	321.3	16.33
Total Operating Costs	**2,014.1**	**102.38**

Source: JDS (2018)

1.12 Economic Analysis

This 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, and, as such, there is no certainty that the PEA results will be realized. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

An engineering economic model was developed to estimate annual cash flows and sensitivities of the project. Pre-tax estimates of project values were prepared for comparative purposes, while after-tax estimates were developed to approximate the true investment value. It must be noted that tax estimates involve many complex variables that can only be accurately calculated during operations and, as such, the after-tax results are approximations to represent an indicative value of the after-tax cash flows of the project. Base case metal prices used are calculated by averaging London Metal Exchange (LME) values for each of the prior three years with projected LME contract futures for the coming two years. The summary of results is presented in Table 1-8.

1.12.1 Main Assumptions

Table 1-6 outlines the metal prices and exchange rate used in the economic analysis.

Table 1-6: Metal Prices and F/X Rate

Parameter	Unit	Base Case Value	Spot Price Value
Lead Price	US$/lb	1.00	1.08
Zinc Price	US$/lb	1.21	1.42
Silver Price	US$/oz	16.95	16.95
Exchange Rate	US$:C$	0.77	0.77

Source: JDS (2018)

No preliminary market studies were completed on the potential sale of lead and zinc concentrates from the Akie Project. The terms selected are in-line with current market conditions.

No contractual arrangements for shipping, port usage, or refining exist at this time.

This PEA has assumed that all zinc and lead concentrates are transported and smelted in Trail, BC.

Table 1-7 outlines the terms assumed for the economic analysis.

Table 1-7: Net Smelter Return Assumptions

Assumptions & Inputs	Unit	Value
Lead Concentrate		
Metal Recovery to Concentrate	% Pb	46.2
	% Zn	1.3
	% Ag	4.8
Pb Concentrate Grade Produced	% Pb	45.1
Minimum Deduction	% Pb/t	3.0
	g/t Ag	50
Metal Payable	% Pb	95
	% Ag	95
Pb Treatment Charge	US$/dmt conc.	140
Ag Refining Charge	US$/oz	1.50
Moisture Content	%	8.0
Pb Concentrate Transportation Cost to Trail, BC	C$/wmt	231
Zinc Concentrate		
Metal Recovery to Concentrate	% Pb	0.0
	% Zn	88.8
	% Ag	18.3
Zn Concentrate Grade Produced	% Zn	52.4
Minimum Deduction	% Pb/t	0.0
	%Zn/t	8.0
	g/t Ag	93.31
Metal Payable	% Pb	0.0
	% Zn	85
	% Ag	85
Zn Treatment Charge	US$/dmt conc.	190
Ag Refining Charge	US$/oz	0.50
Moisture Content	%	8.0
Zn Concentrate Transportation Cost to Trail, BC	C$/wmt	180

Source: JDS (2018)

1.12.2 Results

The economic results for the Project based on the assumptions outlined in Section 1.13.1 are shown in Table 1-8.

Table 1-8: Economic Results

Parameter	Unit	Value
Working Capital	C$M	15.8
Pre-Tax Cash Flow	LOM C$M	1,328
	C$M/a	72
Taxes	LOM C$M	458
After-Tax Cash Flow	LOM C$M	870
	C$M/a	47
Pre-Tax $NPV_{7\%}$	**C$M**	**649**
Pre-Tax IRR	**%**	**35.0**
Pre-Tax Payback	**Years**	**2.6**
After-Tax $NPV_{7\%}$	**C$M**	**400.6**
After-Tax IRR	**%**	**27.0**
After-Tax Payback	**Years**	**3.2**

Source: JDS (2018)

1.12.3 Sensitivities

Sensitivity analyses were performed on metal prices, exchange rate, mill feed grade, capital costs, and operating costs as variables. The value of each variable was changed in plus and minus 5% increments independently while all other variables were held constant. Although the same sensitivity range was used for each variable, some parameters are likely to experience more fluctuation in value over the LOM (i.e. CAPEX versus metal prices). The results of the sensitivity analyses are shown in Table 1-9.

Sensitivities were also completed specific to Zinc price and FX rate, while keeping all other metal prices constant. See Table 1-10 and Table 1-11 for results.

Table 1-9: Sensitivity Results (Pre-Tax $NPV_{7\%}$)

Parameter	-15%	-10%	-5%	Base	+5%	+10%	+15%
Metal Price	185	340	494	649	803	958	1,112
C$:US$ FX	879	802	725	649	572	495	418
Mill Feed Grade	321	430	539	649	758	867	976
OPEX	807	754	701	649	596	543	490
CAPEX	720	696	672	649	625	601	577

Source: JDS (2018)

The analysis revealed that the project is most sensitive to metal price, followed by mill feed grade, exchange rate, and operating costs. The Project showed the least sensitivity to capital costs.

Table 1-10: Sensitivity, Pre-Tax NPV 7%, Zn Price

Parameter	US$1.01/lb	US$1.11/lb	Base Case (US$1.21/lb)	US$1.31/lb	US$1.41/lb
Pre-Tax NPV 7%	181	415	649	862	1,116

Source: JDS (2018)

Table 1-11: Sensitivity, Pre-Tax NPV 7%, FX Rate

Parameter	0.73	0.75	Base Case (0.77)	0.79	0.81
Pre-Tax NPV 7%	728	688	649	607	569

Source: JDS (2018)

1.13 Conclusions

It's the conclusion of the QPs that the PEA summarized in this technical report contains adequate detail and information to support the positive economic outcome shown for the project. Standard industry practices, equipment and design methods were used in the PEA.

The Akie Project contains a substantial zinc, lead and silver resource that can be mined by underground methods and recovered with DMS and conventional flotation processing.

Based on the assumptions used for this preliminary evaluation, the project is considered to be economic and should proceed to the pre-feasibility (PFS) stage.

There is also a likelihood of improving the project economics by identifying additional mineral resources within the development area that may justify increased mine production or extend the mine life. Further study and/or design work may identify additional opportunities to improve project economics.

The most significant potential risks associated with the Project are uncontrolled dilution, operating and capital cost escalation, the obtaining of permits, environmental compliance, unforeseen schedule delays, changes in regulatory requirements, ability to raise financing and metal prices. These risks are common to most mining projects, many of which can be mitigated with adequate engineering, planning and pro-active management.

To date, the QPs are not aware of any fatal flaws for the Project.

1.14 Recommendations

It's recommended that the project proceed to the pre-feasibility study stage in line with ZincX's desire to advance the project. It's also recommended that environmental work and permitting continue as needed to support ZincX's project development plans and the work programs defined in Section 27.

It is estimated that a pre-feasibility study and supporting field work would cost approximately $30.4 M. A breakdown of the key components of the next study phase is as follows in Table 1-12.

Table 1-12: Cost Estimate to Advance to Pre-feasibility Study Stage

Component	Estimated Cost ($C M)	Comment
Resource Drilling	5.0	Conversion of indicated to measured resources. Drilling will include holes combined for resource, geotech and hydrogeology purposes.
Metallurgical Testing	0.6	Comminution, DMS, flotation optimization, variability testing, tailings dewatering, concentrate filtration, mineralogy, minor element analysis.
Underground Development	20.0	Access for underground drilling and possible bulk sample.
Geochemistry	0.5	Acid Base Accounting (ABA) tests and humidity cell testing to determine acid generating potential of rock and tailings.
Waste & Water Site Investigation	0.8	Site investigation drilling, sampling and lab testing.
Geotechnical, Hydrology & Hydrogeology	1.0	Drilling, sampling, logging, test pitting, lab tests, etc.
Engineering	1.5	PFS-level mine, infrastructure and process design, cost estimation, scheduling & economic analysis.
Environment	1.0	Baseline investigations including, water quality, fisheries, wildlife, weather, traditional land use & archaeology.
Total	**30.4**	Excludes corporate overheads and future permitting activities.

Source: JDS (2018)

2 Introduction

2.1 Basis of Technical Report

This PEA Technical Report was prepared for ZincX by JDS, SGI, KP and Lorax; collectively referred to as the Consultants.

This document has been prepared following the guidelines of the Canadian Securities Administrator's NI 43-101 and Form 43-101F1.

The quality of information, conclusions, and estimates contained herein is consistent with the level of effort involved in the Consultant's services, based on:

- Information available at the time of preparation;
- Data supplied by outside sources; and
- The assumptions, conditions, and qualifications set forth in this report.

Given the nature of the mining business, these conditions can change significantly over relatively short periods. Consequently, actual results may vary significantly. The user of this document should ensure that this is the most recent Technical Report for the property as it may not be valid if a new Technical Report has been issued.

2.2 Scope of Work

This report summarizes the work carried out by the Consultants, all of which are independent of ZincX. The scope of work for each company is listed below and when combined, makes up the total Project scope.

JDS scope of work included:

- Compile the Technical Report that also includes the data and information provided by other consulting companies;
- Mine planning, underground design, and production schedule;
- Mining equipment selection and cost estimation;
- Determine mine geotechnical criteria and establish stope sizes;
- Provide recommendations on the execution and development of the metallurgical test work program;
- Interpret the past and current test work results and develop the Project process design criteria;
- Develop an appropriate process flowsheet and preliminary plant mass and water balance;
- Preparation of layouts, drawings, lists, and other deliverables to support the plant design basis;
- Prepare an operating cost estimate for the process plant;

- Design required plant infrastructure, estimate power requirements, and identify proper sites, plant facilities, and other ancillary facilities;
- Estimate OPEX and CAPEX for the Project;
- Prepare a financial model and conduct an economic evaluation including sensitivity and Project risk analysis; and
- Interpret the results and make conclusions that lead to recommendations to improve value, reduce risks, and move toward a pre-feasibility level study.

SGI scope of work included:

- Establish a Mineral Resource estimate for the Project following NI 43-101 guidelines; and
- Summarize geology, mineralization and drilling information.

KP scope of work included:

- Assess tailings management alternatives;
- Design the tailings a management facility (TMF) and determine which methodology would be feasible;
- Develop the mine rock management plan;
- Determine the Project water balances and establish water management plans; and
- Summarize waste disposal operating and post closure requirements and plans.

Lorax scope of work included:

- Summarize climate and hydrologic monitoring, hydrogeology and geochemistry data.

2.3 Qualification Person Responsibilities and Site Inspections

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

None of the QPs or any associates employed in the preparation of this report has any beneficial interest in ZincX. The QPs are not insiders, associates, or affiliates of ZincX. The results of this Technical Report are not dependent upon any prior agreements concerning the conclusions to be reached, nor are there any undisclosed understandings concerning any future business dealings between ZincX and the QPs. The QPs are being paid a fee for their work in accordance with normal professional consulting practice.

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

Table 2-1: QP Responsibilities

Qualified Persons	Company	Report Section(s)
Michael Makarenko, P. Eng.	JDS Energy & Mining Inc.	1.1, 1.2, 1.9, 1.11 to 1.14, 2, 3, 18 (except 18.6,18.7,18.8),19, 21 to 29
Kelly McLeod, P. Eng.	JDS Energy & Mining Inc.	1.4, 1.8, 12.2, 13, 17
Richard Goodwin, P. Eng.	JDS Energy & Mining Inc.	1.7, 12.3, 15, 16 (except 16.2)
Michael Levy, P. Eng.	JDS Energy & Mining Inc.	16.2
Jim Fogarty, P. Eng.	Knight Piésold Ltd.	1.10, 18.6, 18.7, 18.8, 20 (except 20.2)
Robert Sim, P. Geo.	Sim Geological Inc.	1.3, 1.5, 1.6, 4 to 12 (except 12.2 and 12.3), 14
Bruce Mattson, M. Sc., P. Geo.	Lorax Environmental Services Ltd.	20.2

Source: JDS (2018)

QP site visits were conducted as follows:

- Michael Makarenko, P. Eng., completed a site visit 8 to 9 August 2017;
- Robert Sim, P. Geo., completed a site visits 16 to 17 October 2007, 18 to 20 September 2013 and 8 to 9 August 2017; and
- Kelly McLeod, P. Eng., Richard Goodwin, P. Eng., Michael Levy, P. Eng., Jim Fogarty, P. Eng., and Bruce Mattson, M. Sc., P. Geo. did not visit the site and relied upon the observations of QPs Makarenko and Sim.

2.4 Sources of Information

The sources of information include data and reports supplied by ZincX personnel as well as documents cited throughout the report and referenced in Section 29. In particular, background Project information was directly taken from the technical report titled "*NI 43-101 Technical Report Mineral Resource Estimate for the Akie Zinc-Lead-Silver Project, British Columbia, Canada*" with an effective date of 16 May 2016 produced by Sim Geological Inc.

2.5 Units, Currency and Rounding

The units of measure used in this report are as per the International System of Units (SI) or "metric" except for Imperial units that are commonly used in industry (e.g., ounces (oz) and pounds (lb.) for the mass of precious and base metals).

All dollar figures quoted in this report refer to 2018 Canadian dollars (C$) unless otherwise noted.

Frequently used abbreviations and acronyms can be found in Section 26. All coordinates are reported using the UTM datum projection North American Datum 1983 (NAD83). This report includes technical information that required subsequent calculations to derive subtotals, totals and weighted averages. Such calculations inherently involve a degree of rounding and consequently introduce a margin of error. Where these occur, the QPs do not consider them to be material.

This report may include technical information that requires subsequent calculations to derive sub-totals, totals and weighted averages. Such calculations inherently involve a degree of rounding and consequently introduce a margin of error. Where these occur, JDS does not consider them to be material.

3 Reliance on Other Experts

The QPs opinions contained herein are in large part based on information provided to the consultants by ZincX throughout the course of the investigations. JDS has relied upon the work of other consultants in Project areas in support of this Technical Report.

The QPs used their experience to confirm the information supplied by ZincX and was suitable for inclusion in this Technical Report and adjusted any information that required amending.

Neither JDS nor the authors of this Technical Report are qualified to provide extensive comment on legal issues associated with the ownership or control of the Akie property. As such, portions of Section 4 dealing with the types and numbers of mineral tenures and licences, the nature and extent of ZincX's title and interest in the Akie property, the terms of any royalties, back-in rights, payments, or other agreements and encumbrances to which the property is subject, are descriptive in nature and are provided exclusive of a legal opinion.

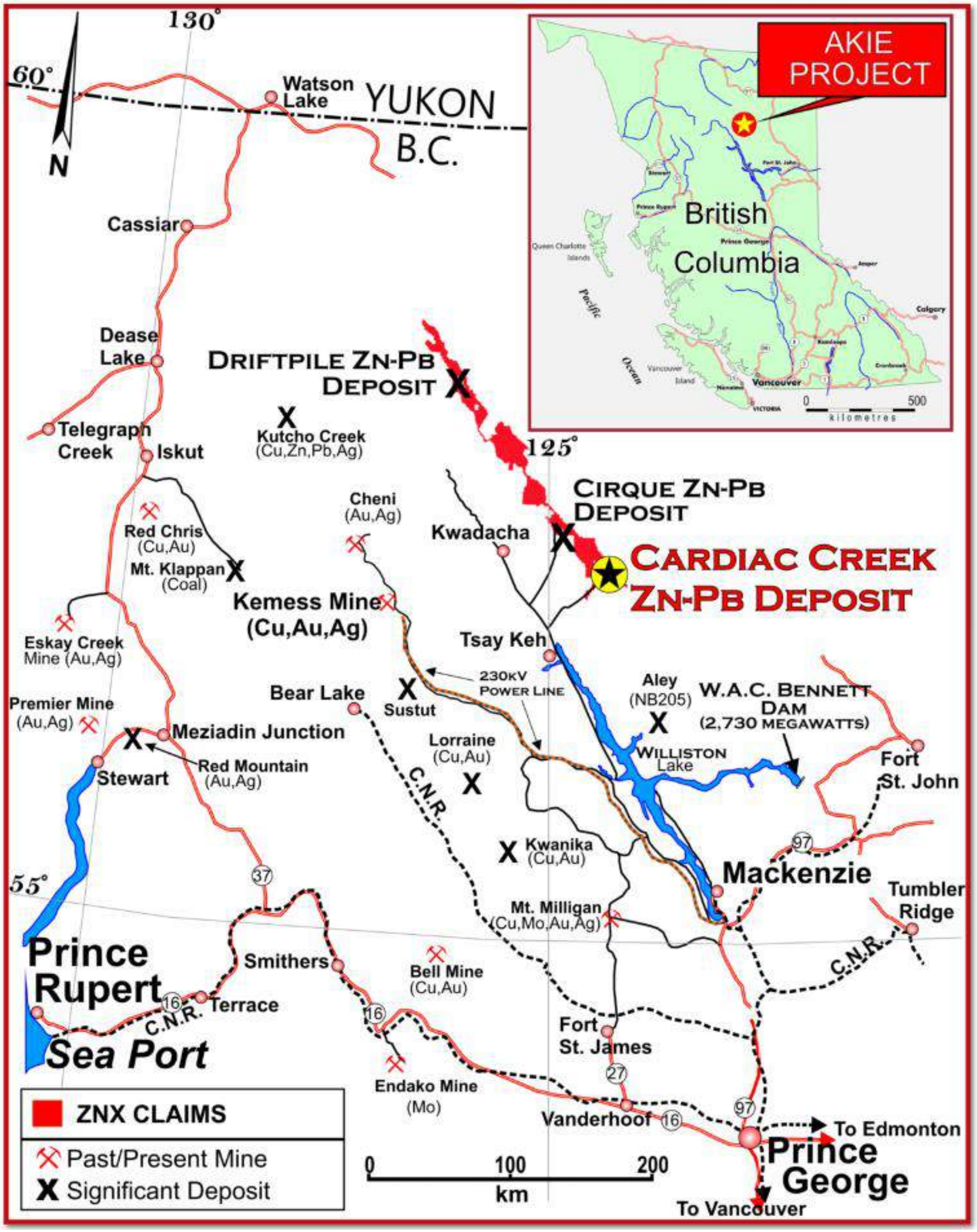
4 Property Description and Location

4.1 Property Description and Location

The Akie property is located in north-central British Columbia within the western ranges of the northern Rocky Mountains (Figure 4-1). The local First Nation communities of Tsay Keh Dene and Kwadacha (Fort Ware) are located approximately 50 km and 55 km from the property, respectively. The town of Mackenzie is located at the southern end of the Williston Lake reservoir some 250 km south of the property and the urban centre of Prince George is located approximately 410 km south of the Akie property. The property is divided by Silver Creek, which drains into the prominent Akie River that runs along the southeastern boundary of the property. The Akie River feeds into the Finlay River which in turn drains into the Williston Lake reservoir near the community of Tsay Keh Dene. All exploration activities were conducted out of the Akie exploration camp shown in Figure 4-2.

The property is centred on UTM coordinate 388550mE and 6360660mN and is located within NTS map sheet 94F/7 and TRIM map sheets 094F036, 094F037 and 094F046. The discovery outcrop of the Cardiac Creek deposit is situated within Cardiac Creek, located at UTM coordinates 389074mE and 6360045mN. The Cardiac Creek deposit is assigned as Minfile #094F 031 in the provincial BC Mineral Database System.

Figure 4-1: Akie Property Location Map



Source: ZincX (2018)

Figure 4-2: Aerial View of the Akie Camp

Source: ZincX (2018)

4.2 Mineral Tenure

The Akie property consists of 46 claims totaling approximately 11,583.4 ha (Figure 4-3). The Cardiac Creek deposit is situated on claims 324823 and 324825. All the claims are in good standing until the 21 October 2027. The claims comprising the Akie property are shown in Table 4-1. Currently, the Akie property is controlled by ZincX Resources who maintains 100% ownership. Some of the claims in Table 4-1 are listed under Ecstall Mining Corp (Ecstall Mining) which is a wholly owned subsidiary of ZincX Resources.

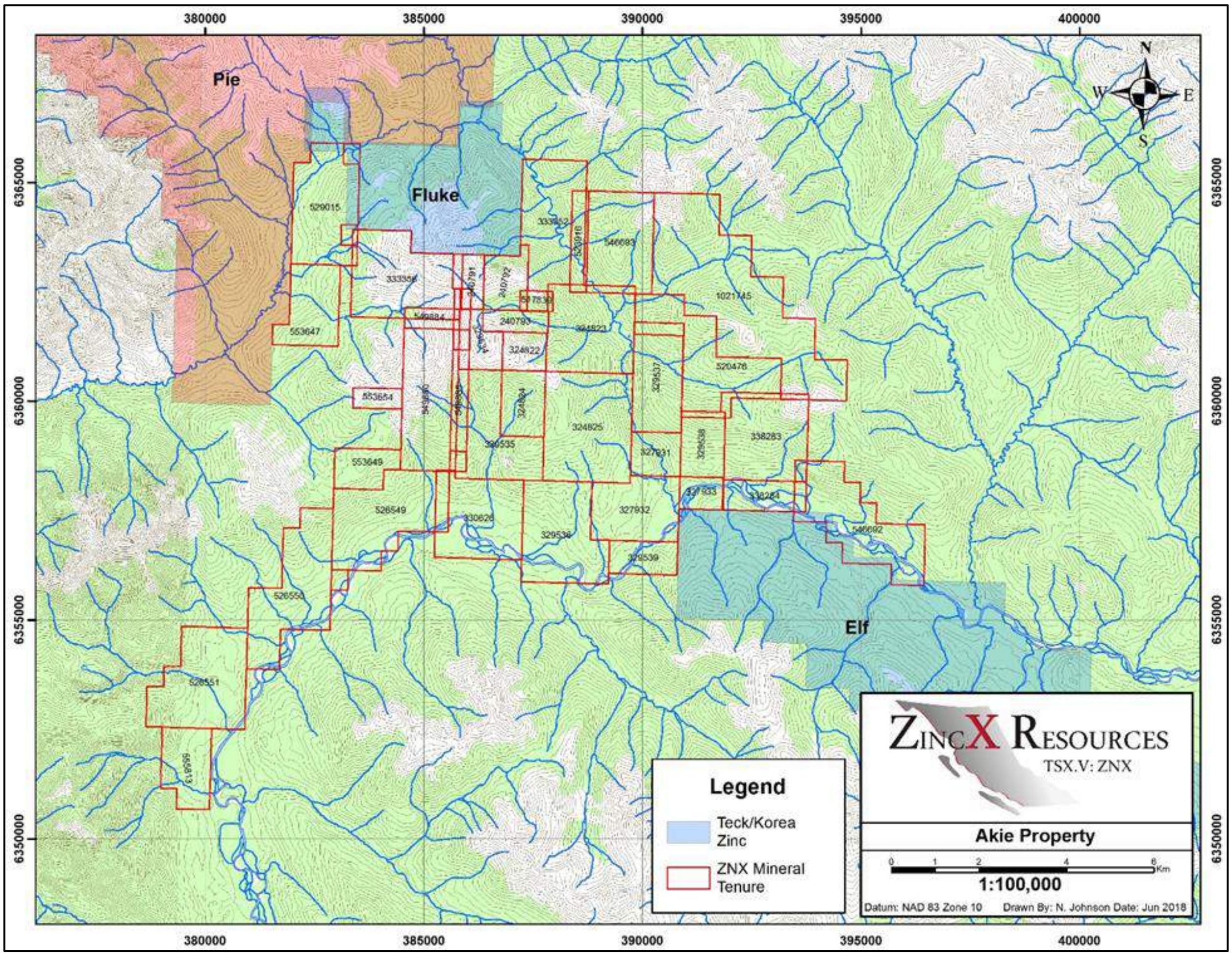
Table 4-1: Akie Property Tenure Listing

Tenure Number	Claim Name	Owner (%)	Expiry Date	Area (ha)
240791	AKIE 1	Ecstall Mining Corp.	21 Oct 2027	75.00
240792	AKIE 2	Ecstall Mining Corp.	21 Oct 2027	150.00
240793	AKIE 3	Ecstall Mining Corp.	21 Oct 2027	75.00
324822	AKIE 4	Ecstall Mining Corp.	21 Oct 2027	100.00
324823	AKIE 5	Ecstall Mining Corp.	21 Oct 2027	400.00
324824	AKIE 6	Ecstall Mining Corp.	21 Oct 2027	150.00
324825	AKIE 7	Ecstall Mining Corp.	21 Oct 2027	500.00
327931	AKIE 8	Ecstall Mining Corp.	21 Oct 2027	150.00
327932	AKIE 9	Ecstall Mining Corp.	21 Oct 2027	300.00
327933	AKIE 10	Ecstall Mining Corp.	21 Oct 2027	100.00
329534	AKIE 11	Ecstall Mining Corp.	21 Oct 2027	400.00
329535	AKIE 12	Ecstall Mining Corp.	21 Oct 2027	500.00
329536	AKIE 13	Ecstall Mining Corp.	21 Oct 2027	500.00
329537	AKIE 14	Ecstall Mining Corp.	21 Oct 2027	375.00
329538	AKIE 15	Ecstall Mining Corp.	21 Oct 2027	150.00
329539	AKIE 16	Ecstall Mining Corp.	21 Oct 2027	200.00
330626	AKIE 17	Ecstall Mining Corp.	21 Oct 2027	400.00
549885	AKIE 20	Ecstall Mining Corp.	21 Oct 2027	87.25
333352	AKIE 21	Ecstall Mining Corp.	21 Oct 2027	450.00
333353	AKIE 22	Ecstall Mining Corp.	21 Oct 2027	225.00
552382	AKIE 23	Ecstall Mining Corp.	21 Oct 2027	17.44
333356	AKIE 25	Ecstall Mining Corp.	21 Oct 2027	500.00
338283	AKIE 18	Ecstall Mining Corp.	21 Oct 2027	400.00
338284	AKIE 19	Ecstall Mining Corp.	21 Oct 2027	300.00
517839	CURE	Ecstall Mining Corp.	21 Oct 2027	34.88
520476	AKIE 30	Ecstall Mining Corp.	21 Oct 2027	436.14
523916	AKIE FR.	Ecstall Mining Corp.	21 Oct 2027	87.18
523920	AKIE FR 2	Ecstall Mining Corp.	21 Oct 2027	17.44
526549	AKIE AX 1	Ecstall Mining Corp.	21 Oct 2027	436.57
526550	AKIE AX 2	Ecstall Mining Corp.	21 Oct 2027	436.75

Tenure Number	Claim Name	Owner (%)	Expiry Date	Area (ha)
526551	AKIE AX 3	Ecstall Mining Corp.	21 Oct 2027	436.98
529015	AKIE 31	Ecstall Mining Corp.	21 Oct 2027	366.10
529025	AKIE 31A	Ecstall Mining Corp.	21 Oct 2027	17.44
529026	AKIE 31B	Ecstall Mining Corp.	21 Oct 2027	17.43
546692	AKIE 41	Ecstall Mining Corp.	21 Oct 2027	436.54
546693	AKIE 40	Ecstall Mining Corp.	21 Oct 2027	348.69
549880		Ecstall Mining Corp.	21 Oct 2027	366.47
549884		Ecstall Mining Corp.	21 Oct 2027	52.33
549887	IN	ZincX Resources	21 Oct 2027	17.46
549888	AK	ZincX Resources	21 Oct 2027	17.45
553647		ZincX Resources	21 Oct 2027	226.76
553649		ZincX Resources	21 Oct 2027	122.21
553654	1.1	ZincX Resources	21 Oct 2027	52.35
555813	HSH	Ecstall Mining Corp.	21 Oct 2027	192.36
557781	ROME	Ecstall Mining Corp.	21 Oct 2027	17.47
1021745	SITKA	ZincX Resources	21 Oct 2027	942.00

Source: ZincX (2018)

Figure 4-3: Akie Property Claim Map



Source: ZincX (2018)

4.3 Royalties, Agreements, and Encumbrances

As far as the author is aware, the property is not subject to any royalties, back-in-rights, or other payments and encumbrances and the property is not subject to any known environmental liabilities.

4.4 Environmental Liabilities and Considerations

The company extended the Akie FSR (Forest Service Road) in 2008 a total distance of 14 km to access a planned portal site for an underground exploration drill program. The planned and permitted underground exploration drill program is designed to update the current resource to the measured category. The program is permitted under Mines Act Permit MX-13-116 which remains active and is fully bonded for reclamation responsibility. Discharge from the site is permitted under Environmental Management Act Discharge Authorization 106429. In 2011 a small, temporary waste rock dump (WRD) area was developed to store overburden, excavated rock from road construction, and road from the decline development. A sedimentation pond and water collection ditches were constructed in the WRD area. Road crossings were completed over T Creek and Cardiac Creek. Approximately 4,000 m^3 of Gunsteel Formation waste rock from the road construction is stored in a covered temporary pile in the WRD area. The final 1.2 km of the road from the waste rock site to the planned portal has been cleared and grubbed and the sub-grade built but requires final grading for use.

Environmental liabilities include removal of the trailer camp, deactivation of approximately 14 km of road including the removal of several bridges; and covering the waste rock dump with impervious till and monitoring runoff for a period of time post-closure, anticipated to be no more than five years. The project area was last monitored on 16 to 18 July 2018 and all water management conveyance structures were in good condition. Water quality at compliance stations has been assessed against discharge limits, British Columbia Water Quality Guidelines, and Site-Specific Management Triggers. For the 2018 monitoring year, the water quality at the T-Creek d/s (Akie25) and Silver Creek d/s (Akie05) stations complied with the requirements of Effluent Permit 106429.

4.5 Other Significant Factors and Risks

As of the effective date of this report, JDS is unaware of any other significant factors and risks that may affect access, title, or the right or ability to perform work on the Akie Project.

5 Accessibility, Climate, Local Resources, Infrastructure and Physiography

5.1 Accessibility

The Akie property and exploration camp (Figure 4-2) are directly accessible via an extensive network of forest service roads originating near Mackenzie. The camp, located at the southwestern edge of the property, is situated at the 24.5 km mark on the Akie Mainline Forest Service Road (FSR). In 2008, CZM, now known as ZincX extended the Akie Mainline FSR to the 41.5 km mark into the south-central area of the property. Additional road construction extends from the 41.5 km mark to several planned infrastructure sites and to the proposed portal site located downslope from the Cardiac Creek massive sulphide showing. Gravel airstrips located in the First Nation communities of Tsay Keh Dene and Kwadacha provide access by fixed-wing aircraft and the camp and property can be accessed using chartered helicopter services based in either Mackenzie or Prince George.

5.2 Climate and Physiography

5.2.1 Climate

The region has a variable climate with temperatures ranging from 15°C to 30°C in the summer and -10° to -30°C in the winter. Precipitation can be variable from year to year with moderate rainfall in summer, with temporary snow accumulations at higher elevations and moderate snow accumulations in the winter months.

5.2.2 Physiography

The Akie property is characterized by northwest-southeast trending ridgelines bounded by the east-west running Akie River valley to the southeast. Elevation ranges from 850 m within the valley to 2,200 m at the peaks. Ridges and mountain tops above the tree line have either no vegetation or are covered by alpine meadows. The remainder of the property is thickly forested, characterized by lodgepole pine and black spruce covering the mountain slopes, and alder, willows and birch bordering creeks and rivers.

Abundant unnamed mountain streams and creeks feed into the larger Silver Creek which runs parallel to the ridgelines, divides the property, and ultimately drains into the Akie River.

5.3 Local Resources and Infrastructure

5.3.1 Roads

The region is well-connected by an extensive network of all-weather forestry service roads originating near Mackenzie. The Akie Mainline FSR provides direct access into the central area of the property. The mainline has been extended to the Cardiac Creek deposit and provides access to the planned point of entry for underground access. The paved, provincial highway system can be accessed in Mackenzie.

5.3.2 Air

Several gravel airstrips are located along the shores of the Williston Lake reservoir and Finley River basin; the closest is located at the village of Tsay Keh Dene, approximately 50 km southwest of the property. Regularly scheduled charter flights provide service to the communities of Tsay Keh Dene and Kwadacha during the work week.

5.3.3 Electricity

The hydroelectric W.A.C. Bennett Dam located on the Peace Reach of the Williston Lake reservoir supplies power to the nearby Kemess copper-gold mine via the Kennedy substation located near Mackenzie, BC. On-site, diesel-fueled generators provide electricity to the Akie camp.

5.3.4 Water

Barge services operating out of Mackenzie on the Williston Lake reservoir provide intermittent water services to the local communities and the forestry industry.

5.3.5 Rail

Mackenzie provides the closest access to rail service.

6 History

Exploration on the Akie property has been intermittent since the late 1970's, marked by short periods of intense activity. Exploration activities have included prospecting, mapping, large-scale soil sampling programs, litho-geochemical sampling, limited geophysical surveys and diamond drilling.

As of 31 December 2017, there are 151 drill holes on the Akie property totalling 64,288 m. The following subsections chronicle the key historical exploration activities conducted on the Akie property and the results. This information is based primarily on publicly filed assessment reports with the BC Ministry of Energy and Mines and internal company reports. Except for minor edits Sections 6.1 through to 6.3 summarizing the early exploration history of the Akie property have been taken unabridged from the 2016 NI 43-101 report by Sim. The information remains current.

6.1 Exploration History

The exploration history of the Akie property has been sporadic since the early 1970's with all of the work being completed over three periods of time; the late 1970's to early 1980's, the mid 1990's and from 2005 to present. Exploration work has consisted of grassroots prospecting, sampling and mapping through to drilling and geophysical surveys. The following table (Table 6-1) outlines a summary of exploration activities that have occurred on the property.

Table 6-1: Akie Exploration History

Year	Operator	Exploration Work
1978	RioCanex Ltd.	Stakes the area based on anomalous Pb values in regional stream sediment samples. The claims were staked as the Dog claims.
1979-1981	RioCanex Ltd.	Conducted extensive soil sampling program identified a series of ill-defined Pb, Zn, Ag, and Ba anomalies. This work was complimented with VLF-EM survey.
1985	RioCanex Ltd.	Allowed Dog claims to lapse.
1989	Ecstall Mining Corp.	Staked Akie claims 1 to 3 covering ground previously known as Dog claims.
1992	Ecstall Mining Corp.	Ecstall options property to Inmet Mining Corp. (Minnova Inc., Metall Mining Inc.).
1992	Inmet Mining Corp.	Conducts small scale soil sampling program over Fluke Ridge and identifies a significant Pb, Zn, Ag, and Fe anomaly.
1994	Inmet Mining Corp.	Conducts; extensive soil sampling program, preliminary mapping, VLF/resistivity survey and magnetometer surveys which result in identification of numerous Pb, Zn, Ag, and Ba anomalies. Prospecting discovers Cardiac Creek showing. A drill program (12 DDH's = 3,753.20 m) discovers the mineralized horizon now known as the Cardiac Creek deposit. Claims were expanded to include Akie 4 to 17.
1995	Inmet Mining Corp.	Additional drilling (7 DDH's = 5,314 m) continues to define the Cardiac Creek deposit.
1996	Inmet Mining Corp.	Additional drilling (10 DDH's = 4,483.10 m) continues to test the deposit and other property targets. A historical non 43-101 compliant resource for the Cardiac Creek deposit is calculated at 12 Mt @ 8.6% Zn, 1.5% Pb, 17.1 g/t Ag (MacIntyre, 2005).

Year	Operator	Exploration Work
1996	Inmet Mining Corp.	Allows option on property to lapse.
2005	Ecstall Mining Corp.	Options the property to Mantle Resources Inc.
2005	Mantle Resources Inc.	Commissions Don MacIntyre to complete a 43-101 compliant report on the Akie property and conducts drill program (4 DDH's = 1,998.90 m). Discovers the high-grade core to the Cardiac Creek deposit.
2006	Mantle Resources Inc.	Additional drilling on Cardiac Creek deposit (11 DDH's = 4,480.37 m)
2007	Mantle Resources Inc.	Additional drilling on Cardiac Creek deposit (12 DDH's = 6,526.26 m). Mapping and sampling also conducted
2008	Canada Zinc Metals Corp.	Completes takeover of Ecstall Mining Corp. and acquires 100% ownership of Akie property. Company changes name to Canada Zinc Metals Corp. A NI 43-101 compliant inferred resource is calculated for the Cardiac Creek deposit of 23.6 Mt @ 7.6% Zn, 1.5% Pb, 13 g/t Ag at a 5% Zn cut-off (MacIntyre & Sim, 2008). Additional drilling on the deposit and North Lead anomaly which encounters mineralization. (14 DDH's = 6,226.15 m). Mapping also completed and new road and trails were constructed to within 3 km of the deposit.
2009	Canada Zinc Metals Corp.	Prospecting discovered the GPS Zone (GPS) bedded barite showing in black shales similar to the Gunsteel Formation shales along western edge of Akie property. Minor mapping, silt and soil sampling completed.
2010	Canada Zinc Metals Corp.	Additional drilling on the Cardiac Creek deposit and other property targets (11 DDH's = 6,124.51 m). New style of mineralization encountered over 1.17 m in the drilling similar to the Nick Ni-Mo deposit in the Yukon. Continued road development reaches to within 1.5 km of the deposit.
2011	Canada Zinc Metals Corp.	Road development reaches deposit at the proposed underground portal site. Additional drilling on the deposit and other property targets (12 DDH's = 5,667.80 m).
2012	Canada Zinc Metals Corp.	Hydrogeochemistry survey completed. Revised NI 43-101 resource calculated for the Cardiac Creek deposit. Indicated: 12.7 Mt @ 8.38% Zn, 1.68% Pb, 13.7 g/t Ag and Inferred: 16.3 Mt @ 7.38% Zn, 1.34% Pb, 11.6 g/t Ag at a 5% Zn cut-off. (Sim, 2012)
2013	Canada Zinc Metals Corp.	Additional drilling on the Cardiac Creek deposit and other property targets (9 DDH's = 4,599.31 m). Additional soil sampling conducted on the eastern & western areas of the property. Prospecting discovered the Sitka Ba-Zn-Pb showing.
2014	Canada Zinc Metals Corp.	Additional drilling on the Cardiac Creek deposit (8 DDH's = 2,855.12 m)
2015	Canada Zinc Metals Corp.	Additional drilling on the Cardiac Creek deposit (11 DDH's = 5,347.18 m)
2016	Canada Zinc Metals Corp.	Revised NI 43-101 resource calculated for the Cardiac Creek deposit. Indicated: 19.6 Mt @ 8.17% Zn, 1.58% Pb, 13.6 g/t Ag, Inferred: 8.1 Mt @ 6.81% Zn, 1.16% Pb, 11.2 g/t Ag at a 5% Zn cut-off (Sim, 2016)
2017	Canada Zinc Metals Corp.	Additional drilling on the Cardiac Creek deposit (8 DDH's = 4,807.75 m). Metallurgical samples taken from 2017 drill core. Updated resource calculated for the Cardiac Creek deposit. Indicated: 22.7 Mt @ 8.32% Zn, 1.61% Pb, 14.1 g/t Ag and Inferred: 7.5 Mt @ 7.04% Zn, 1.24% Pb, 12.0 g/t Ag at a 5% Zn cut-off. (Sim, 2017)

Source: ZincX (2018)

6.2 Ownership

6.2.1 RioCanex Inc. (1978 – 1981)

In 1978, based on elevated lead values in regional stream sediment sampling, RioCanex Inc. (RioCanex) staked the Dog claims 1 to 8 (Hodgson, 1979) in the central area of what now comprises the present-day Akie property. Initial reconnaissance work involved the collection of 167 stream sediment samples that returned consistently elevated zinc values (ranging from > 1,000 ppm to 19,000 ppm) and nominal lead values (Hodgson, 1979). Follow-up work on the property consisted of preliminary mapping and a single line of soil sampling conducted to the northwest of the Cardiac Creek deposit. A total of 51 soil samples were collected which indicated the presence of anomalous zinc and lead soils overlying prospective shale of the Gunsteel Formation (Hodgson and Faulkner, 1979).

In 1980, additional mapping was completed on the Dog claims (Hodgson, 1980). In 1981, a large-scale soil sampling program was undertaken by RioCanex, apparently based on the single line of soil sampling conducted in 1979. A cut grid covering the property was established and a total of 1,490 soil samples were collected. In conjunction with the soil program, a 34.1 line km ground-based very-low frequency electromagnetic (VLF-EM) survey was completed (Hodgson, 1981). The results identified a broad area of zinc and silver values across the property within a 100 m to 500 m wide zone of elevated lead values trending northwest-southeast across the property (Hodgson, 1981). The VLF-EM survey confirmed the northwest-southeast-trending orientation of the underlying strata (Hodgson, 1981). Exploration efforts were unable to identify any occurrences of mineralization on the property, despite the mention of a barite-pyrite showing in an internal company report (Hodgson, 1980). Based on the exploration results, RioCanex subsequently let the Dog claims lapse (Wells, 1992).

6.2.2 Ecstall Mining Corp. (1989 – 1992)

In 1989, Ecstall Mining Corp. (Ecstall Mining) re-staked Dog claims 1 to 3 which were renamed the Akie Claims 1-3. No exploration work was completed during this period (Wells, 1992).

6.2.3 Inmet Mining Corp. (1992 – 1996)

In early 1992, Inmet Mining Corp., previously known as Minnova Inc. and Metall Mining Corporation, optioned the Akie claims from Ecstall Mining and proceeded to explore for SEDEX-style mineralization from 1992 to 1996. Based on the early exploration results, the Akie claims were subsequently expanded to Akie claims 1 to 17. During this time, Inmet Mining executed several exploration programs that included prospecting and mapping, soil sampling, litho-geochemical sampling, geophysical surveys and diamond drilling.

6.2.3.1 Prospecting and Mapping

During the 1994 exploration season, prospecting activities discovered a gossanous outcrop of laminated sulphides. Chip sampling across the widest observed sulphide bed returned values of 16.0% Zn and 2.80% Pb over 40 cm. This outcrop is now known as the Cardiac Creek discovery showing (Baxter, 1995). In addition to the Cardiac Creek showing, prospecting also identified two nodular barite showings: the Waterfall Barite showing on the southeastern edge of the Akie property; and the Fluke Ridge showing on the northwestern edge of the property. Mapping was also completed across the property at 1:10,000 scale.

The mapping was concentrated along the Akie Main Grid with limited mapping conducted on the Akie Reconnaissance Grid (Figure 6-1).

6.2.3.2 Soil Sampling

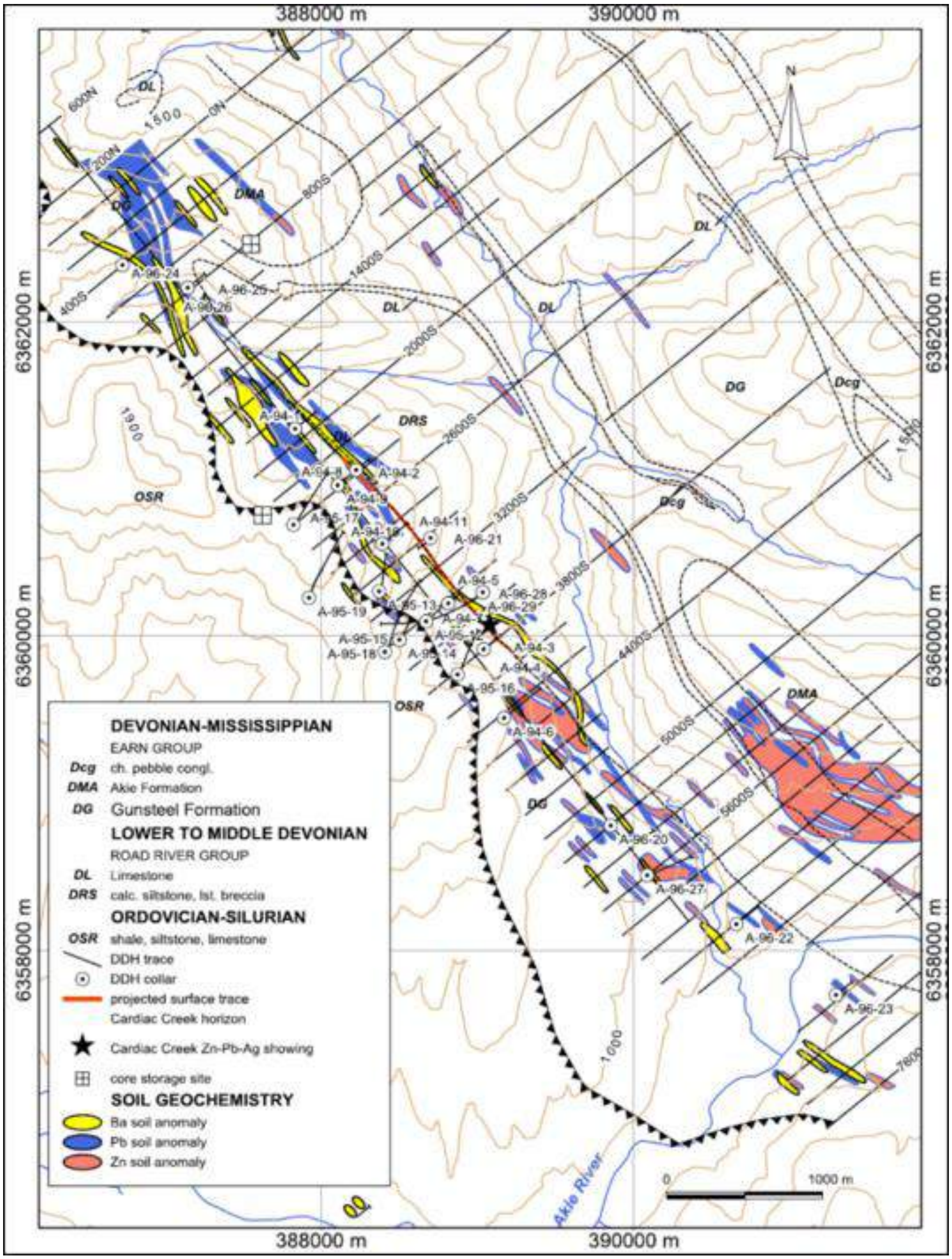
Between 1992 and 1996, a series of soil sampling programs were executed across the Akie property (Wells, 1992; Baxter, 1995, 1996a, 1996b and 1996c). An extensive cut grid was established across the property with two primary areas of the interest: the Akie Main Grid and the Akie Reconnaissance Grid. The Akie Main Grid consisted of 200 m spaced lines from line 600N to line 7600S (Baxter, 1995, 1996a and 1996b). The grid extended from the northwest to the southeast edge of the property and covered the main panel of Gunsteel Formation shale that is host to the Cardiac Creek deposit and the stratigraphically important Gunsteel Formation shale / Road River Group contact. The Akie Reconnaissance Grid consisted of broad 400 m to 600 m spaced lines and represented extensions of the Akie Main Grid lines onto the eastern side of the Akie property (Baxter, 1996c). Follow-up work on the Akie Reconnaissance Grid consisted of 200 m spaced infill lines. In total, 3,071 samples were taken during these programs (Baxter, 1995, 1996a, 1996b and 1996c). The results of this work outlined several distinct soil anomalies on the Akie Main Grid while several small anomalies were defined on the Akie Reconnaissance Grid. Some of the anomalies on the Akie Main Grid were subsequently drill tested while others remained open for further exploration. Baxter (1996c) described the western panel of Gunsteel Formation covered by the Akie Main Grid to be highly anomalous, hosting several multi-element soil anomalies.

The Akie Main Grid anomalies shown in Figure 6-1 are briefly described here:

- South Zinc Anomaly is represented by a 2,000 m by 500 m area of highly elevated zinc values (up to 1.12% Zn). Internally, there are additional discontinuous barium, lead, cadmium, iron, manganese and arsenic anomalies. The anomaly is situated proximal to the important Gunsteel Formation shale / Road River Group contact (MacIntyre and Sim, 2008). This is the largest soil anomaly present on the Akie property, but it was not drill tested by Inmet Mining;
- Fluke Ridge Anomaly (now generally referred to as the North Lead Anomaly) is defined by a lead anomaly that measures approximately 200 m by 1,000 m long, with minor internal barium, arsenic and iron anomalies. This anomaly is partially attributed to a nodular barite showing along the ridge and a massive sulphide lens enriched in lead that was intersected in drill hole A-96-24. In general, lead enrichment within the hanging wall shale of drill hole A-96-24 was found to be poor (MacIntyre and Sim, 2008);
- The Cardiac Creek deposit is flanked by two anomalies: an 1,800 m long lead and barium anomaly with minor arsenic, silver, cadmium and zinc along the northwestern end of the deposit, and a 1,600 m to 2,200 m long lead and zinc anomaly with minor barium, cadmium, iron, arsenic and silver anomalies along the southeastern end of the deposit. The southern anomaly was drill tested by Inmet Mining with three holes that intersected minor distal or fringe-style mineralization. The northern extent of this anomaly remains open for testing (MacIntyre and Sim, 2008); and
- The Waterfall Barite Anomaly is a barium, lead and manganese anomaly with minor zinc, arsenic, manganese and iron extending primarily from lines 7000S to 7600S at the southeastern end of the Akie property. It is associated with a nodular barite occurrence. This has not been drill tested. This anomaly can be extrapolated to the northwest to line 5200S, although with a weaker signature,

which was tested by drill holes A-96-20, A-96-23 and A-96-27. No significant mineralization was intersected (MacIntyre and Sim, 2008).

Figure 6-1: Inmet Compilation Map Showing Key Soil Anomalies, Geology, and Drill Hole Locations



Source: MacIntyre and Sim (2008)

6.2.3.3 Litho-geochemistry

In association with prospecting, mapping, soil sampling and drilling programs, a total of 284 whole rock litho-geochemical samples were collected from grab rock samples and drill core. Samples were analyzed for major and trace elements using ICP and atomic absorption methods at Min-En Laboratories Ltd. of Vancouver. This was completed to identify areas of elemental enrichment or depletion due to the interaction of metal-enriched hydrothermal fluids with the host rocks of the Cardiac Creek deposit (MacIntyre and Sim, 2008).

6.2.3.4 Geophysics

Using the existing cut grid, ground-based geophysical surveys conducted by Pacific Geophysical Ltd. of Vancouver were completed across the mapped panel of Gunsteel Formation shale. This included magnetometer and VLF-resistivity surveys (Baxter, 1995). The magnetic signature was found to be flat across the survey area and no significant anomalies were recognized. The VLF-resistivity survey was able to delineate the approximate lithological contacts between the Gunsteel Formation shale and the Road River Group calcareous siltstone due to the contrast between their individual resistive characteristics (Baxter, 1995).

6.2.3.5 Drilling

From 1994 to 1996, Inmet Mining completed three separate drilling programs. During this time, 29 holes were drilled totaling 13,551 m; 25 were completed to their intended depths and four were abandoned due to ground conditions or excessive deviation of the drill stem. The details of these drilling programs are summarized in the Drilling Section of this report.

6.3 Canada Zinc Metals Corp. and ZincX Resources Corp. (2005 – 2018)

In mid-2005, Mantle Resources Inc. optioned the Akie property from Ecstall Mining in a bid to acquire 65% ownership. In late 2007, Mantle Resources acquired 100% of the property through acquisition of Ecstall Mining. In early 2008, Mantle Resources was renamed Canada Zinc Metals Corp. Subsequently Canada Zinc Metals Corp. changed its name to ZincX Resources Corp. in early 2018. Since 2005, the company has been actively defining the Cardiac Creek deposit as well as exploring for additional SEDEX-style mineralization on its extensive land holdings in the Kechika Trough. Exploration programs have consisted primarily of diamond drilling, although geophysics; prospecting and mapping; soil, silt, rock and water sampling have also been conducted across the property. This work will be summarized in the Exploration Section of this report.

6.4 Historical Production

There has been no commercial production from the Cardiac Creek deposit.

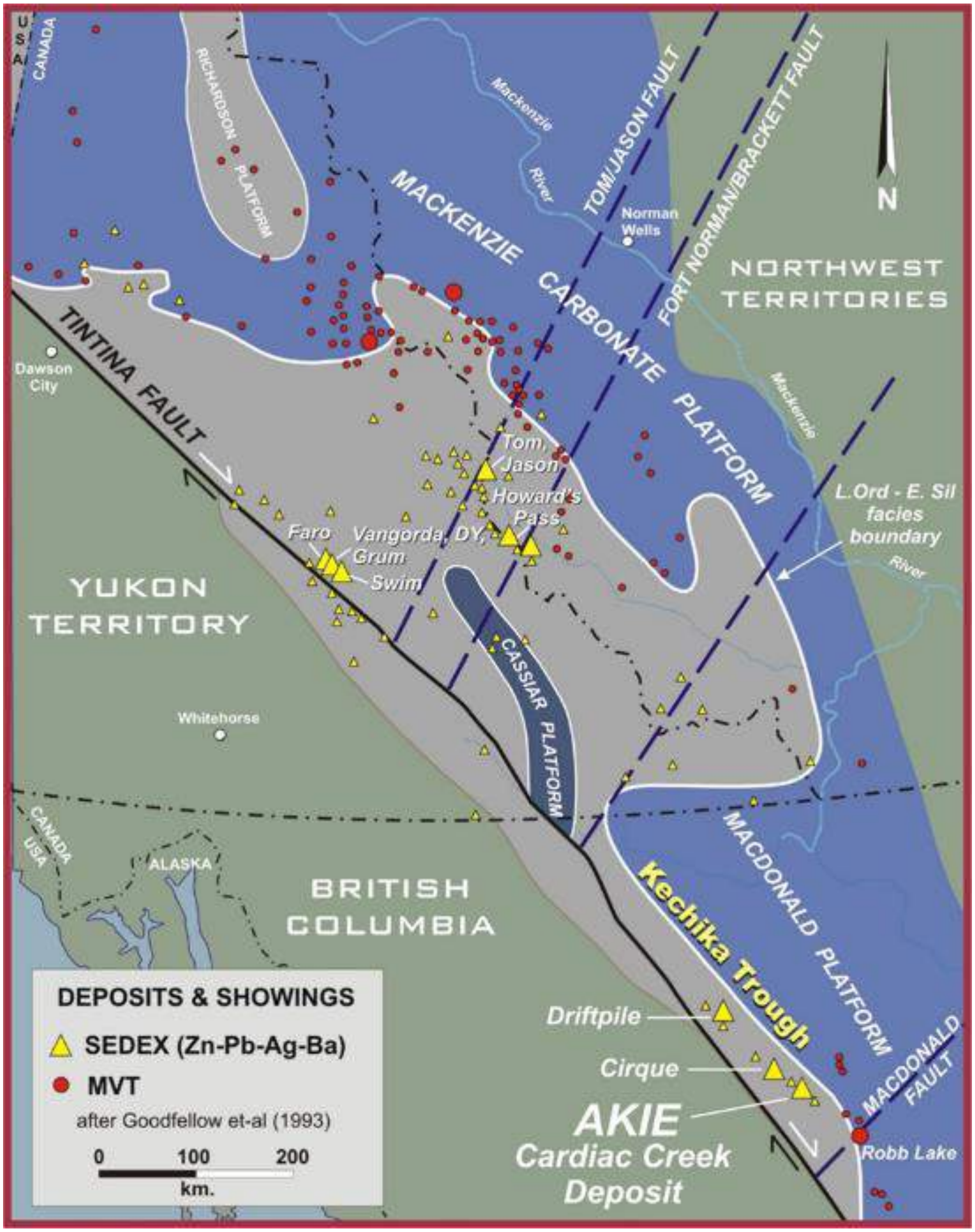
7 Geological Setting and Mineralization

7.1 Regional Geology

The regional geology in the vicinity of the Akie property has been described in detail by Don MacIntyre in the NI 43-101 report entitled "*Geological Report on the Akie Property"* prepared for Mantle Resources (now ZincX Resources) in 2005. For a comprehensive review of the geology of the Akie River district, the reader is referred to the 1998 BC Ministry of Energy and Mines Bulletin 103 entitled "*Geology, Geochemistry and Mineral Deposits of the Akie River Area, Northeast British Columbia*" by Don G. MacIntyre. The following summarizes the information contained within these reports.

The Akie property is situated within the Rocky Mountain fold and thrust belt of northeastern British Columbia and hosted in the central portion of the Kechika Trough. The trough is interpreted to be the southeastern extension of the expansive sedimentary Selwyn Basin bounded by shallow water sedimentary rocks characteristic of the Cassiar (west) and MacDonald platforms (east) (MacIntyre, 1998). The trough is situated along the ancestral continental margin of North America and is host to clastic and carbonate rocks ranging in age from the late Cambrian to late Triassic (MacIntyre, 2005) (Figure 7-1).

Figure 7-1: Geological Setting of Selwyn Basin and Kechika Trough



Source: Goodfellow et al. (1993)

A generalized stratigraphic column can be seen in Figure 7-2 depicting the key geological units.

7.1.1 Windermere Supergroup and GOG Group (Proterozoic to Cambrian)

The oldest rocks exposed in the Kechika Trough are the Proterozoic to early Cambrian coarse-grit units thought to be representative of the Windermere Supergroup and the early to late quartzites and massive limestone correlative to the Gog Group (MacIntyre, 2005). These rocks are not exposed in the vicinity of the Akie property. They are restricted to the northern and northeastern edge of the Kechika Trough and to the immediate west of the property (Gog Group) (MacIntyre, 2005). The grit units of the Windermere Supergroup are thought to act as important aquifers for fluids involved in the formation of sediment and carbonate-hosted lead-zinc-silver deposits of the Selwyn Basin and Kechika Trough (MacIntyre, 2008).

7.1.2 Kechika Group (Cambrian to Ordovician)

A thick, approximately 1,500 m succession of cream-coloured to light-grey weathered, talcy, phyllitic mudstone and wavy-banded nodular limestone characterize the rocks of the Kechika Group (MacIntyre, 2005; Demerse and Hopkins, 2008). Thin beds of green weathered tuffs (MacIntyre, 2005) and thin felsic dykes have been noted within the Kechika Group rocks which are indicative of volcanic activity during deposition of these rocks. The Kechika Group rocks are prominent in the southern Kechika Trough thinning northwards where they are rare to absent (MacIntyre, 2005). These rocks are common in the western half of the Akie property.

7.1.3 Skoki Limestone (Ordovician)

Locally, in the vicinity of Pesika Creek and the Kwadacha River (the southern and eastern sections of the Kechika Trough, respectively), an approximately 500 m thick buildup of thinly bedded limestone of Ordovician age overlies the Kechika Group rocks. These rocks are generally absent in the Northern Kechika Trough (MacIntyre, 2005). The Skoki limestone is absent on the Akie property.

7.1.4 Road River Group (Ordovician to Early Devonian)

The rocks of the Road River Group unconformably overlie those of the Kechika Group and represent a collection of fine-grained clastics rocks, carbonates and minor volcanics of Ordovician to early Devonian age (MacIntyre, 1998). They are pervasive throughout the Kechika Trough and can be informally broken into three distinct groups: The Lower Road River Group, the Ospika Volcanics and the Silurian Siltstone (MacIntyre, 2008). The Road River Group is thought to represent the transition between platform and basin rocks (MacIntyre, 2008).

The basal unit of the Lower Road River Group comprises a cream, beige to reddish-brown weathered, thin-bedded calcareous siltstone and shale interbedded with minor limestone turbidites and debris flows. This siltstone grades up section into a distinct middle to late Ordovician-aged black graptolitic shale (MacIntyre, 1998). The graptolite fossil assemblage allows for relatively easy differentiation from the lithologically similar and prospective rocks of the Devonian (MacIntyre, 2008). Locally, the shale is interbedded with black chert horizons in the vicinity of the REB massive pyrite lens in the southern Kechika Trough, and in the east, they are locally interbedded with quartz wackes, arenites and pebble conglomerates (MacIntyre, 2008).

The Ospika Volcanics are present throughout the central Kechika Trough area (Akie River, Paul River and Ospika River) and are represented by a series of discontinuous lenses and beds of green mafic flows, microdioritic sills and orange weathered ankeritic crystal lapilli tuffs that are interbedded with the rocks of the Lower Road River Group. It is suggested that these rocks were emplaced along fault structures bounding the basin due to their orientation of deposition (MacIntyre, 1998). In 2009, a diorite intrusive plug was discovered along the Del Creek which is thought to represent a bounding fault structure and a possible source for the lenses of volcanic rocks found in the area.

The upper Road River Group, represented by an early to middle Silurian Siltstone, unconformably overlies the Ordovician graptolitic black shale (MacIntyre, 2008). At the base, a 0 m to 20 m thick unit consisting of thin-bedded to cross-laminated limestone and dolostone beds is interbedded with laminated grey calcarenite, dark grey dolomitic shale, and minor debris flows. To the east, the limestone / dolostone beds are interbedded with quartz wacke and arenite. This unit is commonly referred to as the Silurian Limestone. The Silurian Limestone is overlain by a 100 m to 500 m thick tan to orange-brown weathered, dolomitic, thin-bedded to platy siltstone with minor orange weathered limestone and dolostone interbeds. The thicker bedded siltstone is commonly bioturbated, containing worm burrows and feeding trails. Minor graptolites and sponge impressions are present in the thin-bedded to platy sections (MacIntyre, 2008).

The rocks of the Lower Road River Group and the Ospika Volcanics are common in the western half of the Akie property whereas the Silurian Siltstone is situated in the central area of the property, directly underlying the prospective rocks of the Gunsteel Formation. The youngest unit of the Road River Group is informally recognized as the Paul River Formation (Pigage, 1986), and consists of deep-water marine turbidites comprising black chert, interbedded black shale with limestone debris flows, and rusty weathered, dark grey to brown weathered silty shale and siltstone (MacIntyre, 2008). In the Akie River area, the rusty weathered silty shale partially onlap the early to middle Devonian Akie and Kwadacha Reefs. These reefs can range up to 200 m thick and are characterized by medium- to thick-bedded micritic to bioclastic limestone interbedded with minor shale beds. Locally, to the east, the reefs are directly overlain by pebble conglomerates (MacIntyre, 2008).

7.1.5 Earn Group (Middle Devonian to Mississippian)

Rocks of the Earn Group conformably overlie those of the carbonate reefs as well as the Silurian Siltstone and are characterized by carbonaceous, siliceous shale, cherty argillite, phyllitic shale and coarse quartzose turbidites of middle Devonian to Mississippian age (MacIntyre, 1998). The Earn Group has been subdivided into three distinct formations: the Warneford, the Akie and the Gunsteel (Pigage, 1986; MacIntyre, 1998). These rocks are representative of a major marine transgression that halted reef growth, resulting in the onlapping of fine clastic sediments onto the MacDonald platform to the east (MacIntyre, 1998).

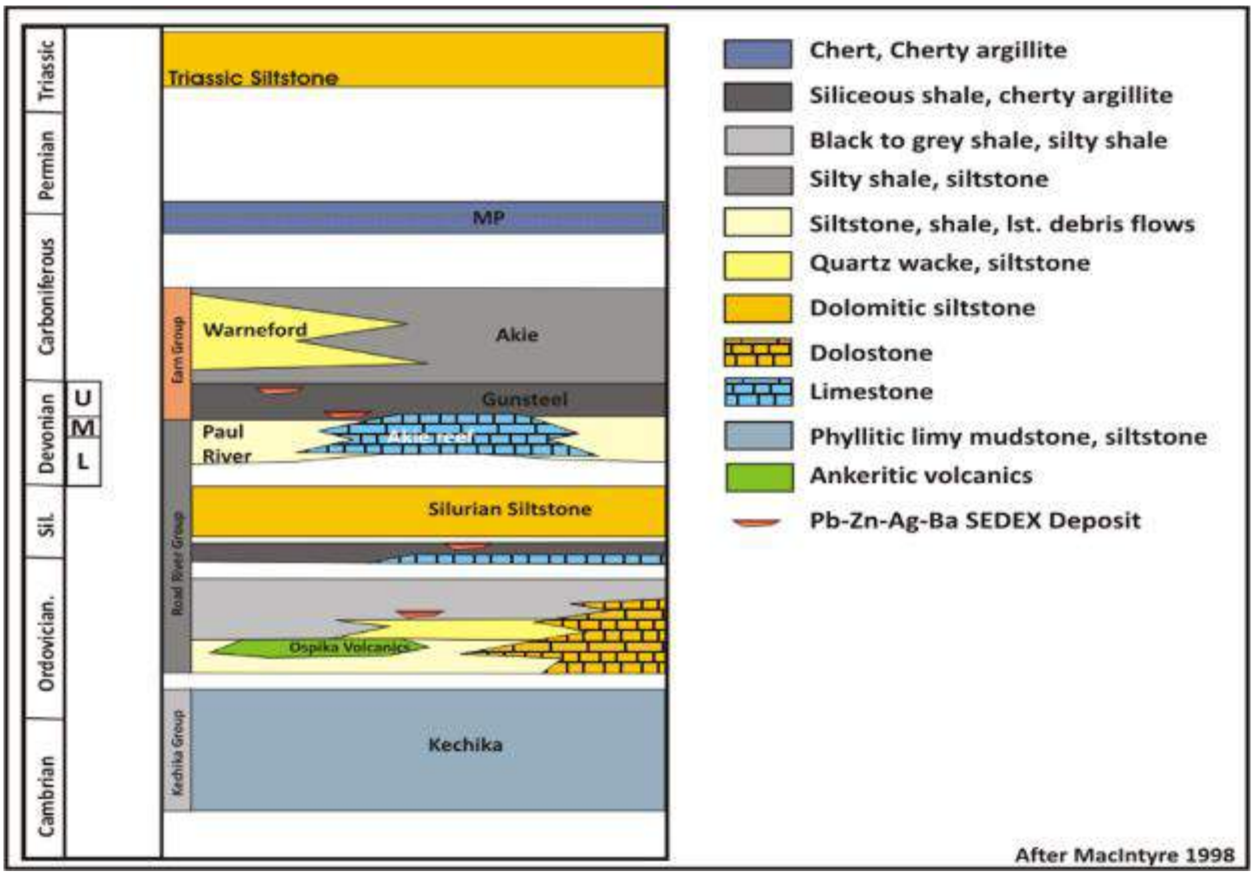
The rocks of the Gunsteel Formation are the oldest within the Earn Group of middle to late Devonian age. They weather to a distinctive “gunsteel” blue and are represented by a collection of carbonaceous and siliceous shale, argillite and cherty argillite (MacIntyre, 1998). The Gunsteel Formation is the primary group of prospective rocks within the Kechika Trough hosting the Cirque, Cardiac Creek and Driftpile deposits as well as the Fluke, Elf, Pie and Mount Alcock prospects. Occurrences of laminar pyrite and nodular barite are common and are indicative of the Gunsteel Formation rocks. They are overlain by the Akie Formation and characterized by soft, medium to dark grey phyllic shale to silty shale and siltstone which typically weather to a rusty brown, tan or silvery colour (MacIntyre, 1998; Demerse and Hopkins, 2008).

The youngest group of rocks within the Earn Group (the Warneford Formation) are interpreted to be proximal-to-medial turbidites represented by grey weathered chert pebble conglomerates, quartz wacke and siltstone and are intercalated with the soft shale of Akie Formation (MacIntyre, 1998). The rocks of the Earn Group outcrop across the majority of the Akie property.

7.1.6 Triassic Siltstone (Mississippian to Triassic)

The youngest rocks of the Kechika Trough occur in the core of a major northwest-trending synclinorium in the area northwest of the Kwadacha River. They are represented by dolomitic siltstone and limestone, similar in character to the Silurian Siltstone, but can be differentiated by the presence of Triassic brachiopods (MacIntyre, 1998). This unit is not present on the Akie property.

Figure 7-2: Kechika Trough Generalized Stratigraphic Section



Source: MacIntyre (1998)

7.2 Regional Structure

The following section is an unabridged excerpt from the previous technical report entitled "*Geology, Diamond Drilling and Preliminary Resource Estimation, Akie Zinc-Lead-Silver Property, Northeast British Columbia, Canada"* by Donald G. MacIntyre and Robert C. Sim (2008). This information remains current.

"The linear nature of the geology of the Akie River area reflects the 'thin-skinned' tectonic-style of the Rocky Mountain Fold and Thrust Belt. Northeast-directed compression resulted in detachment of the Paleozoic strata from a rigid crystalline basement and partial stacking of the detached plates along a series of imbricate thrust faults. The thrust plates, which are composed of relatively incompetent basinal facies rocks, have been internally folded during thrusting. In general, incompetent strata below overriding thrust plates have tight isoclinal folds with southwest-dipping axial planes whereas rocks in the overriding plate are asymmetrically folded and often have northeast-dipping axial planes. This style of folding may be related to the development of inversion structures similar to those described by McClay et al., (1989) in the Driftpile Creek area.

The structural-style changes from west to east across the map area. In the west, imbricate, southwest-dipping reverse faults bound asymmetric overturned folds with southwest-dipping to vertical-axial planes. To the east, large-scale upright folds occur within major synclinoriums that are bounded by outward dipping reverse faults that truncate folds within overriding anticlinoriums. Devonian strata are preserved within the synclinoriums. This structural style suggests that high-angle growth faults bounding depositional troughs in Devono-Mississippian time were reactivated during Tertiary compression and became the locus of major thrust faults in the district. That major high-angle thrust faults may be localized along much older crustal breaks is also suggested by close spatial association of Paleozoic mineralization, reef building, coarse clastic fans and volcanism to such faults.

Detailed studies of the structure of the Cirque deposit led to the recognition of two coaxial phases of deformation (Pigage, 1986). The earliest deformation, which is recognizable throughout the study area, includes northwest-trending, tight asymmetric folds that verge northeast and have gently dipping southwest limbs and steep to overturned northeast limbs. The steep limbs are often broken and offset by high-angle reverse faults, resulting in the juxtaposition of Ordovician and Silurian strata against the middle to late Devonian Gunsteel Formation shale. The high-angle reverse faults may coalesce at depth into a major detachment surface possibly rooted in the highly attenuated Kechika Formation. Shale typically has a pervasive slatey cleavage that is axial planar to the macroscopic folds; a closely-spaced fracture cleavage is found in the more competent strata.

The second phase of deformation folds the early slatey cleavage and develops a penetrative crenulation cleavage. This cleavage is axial planar to the late folds, which may have an amplitude of up to 30 m (Pigage, 1986). The folds are open to upright, trend northwest and have northeast convergence.

High-angle listric normal and reverse faults are also common in the Akie River area and generally trend parallel or at slight angles to the major high-angle thrust faults. These faults are probably related to brittle failure of thrust plates during detachment and thrusting. Displacements of up to several hundred metres have been documented at the Cirque deposit (Pigage, 1986).

North to northeast-trending high-angle faults offset earlier thrust and listric normal faults. Some of these faults have a strike-slip movement and may be synthetic shears related to an oblique compressional stress regime. This compressional event is believed to be Tertiary in age."

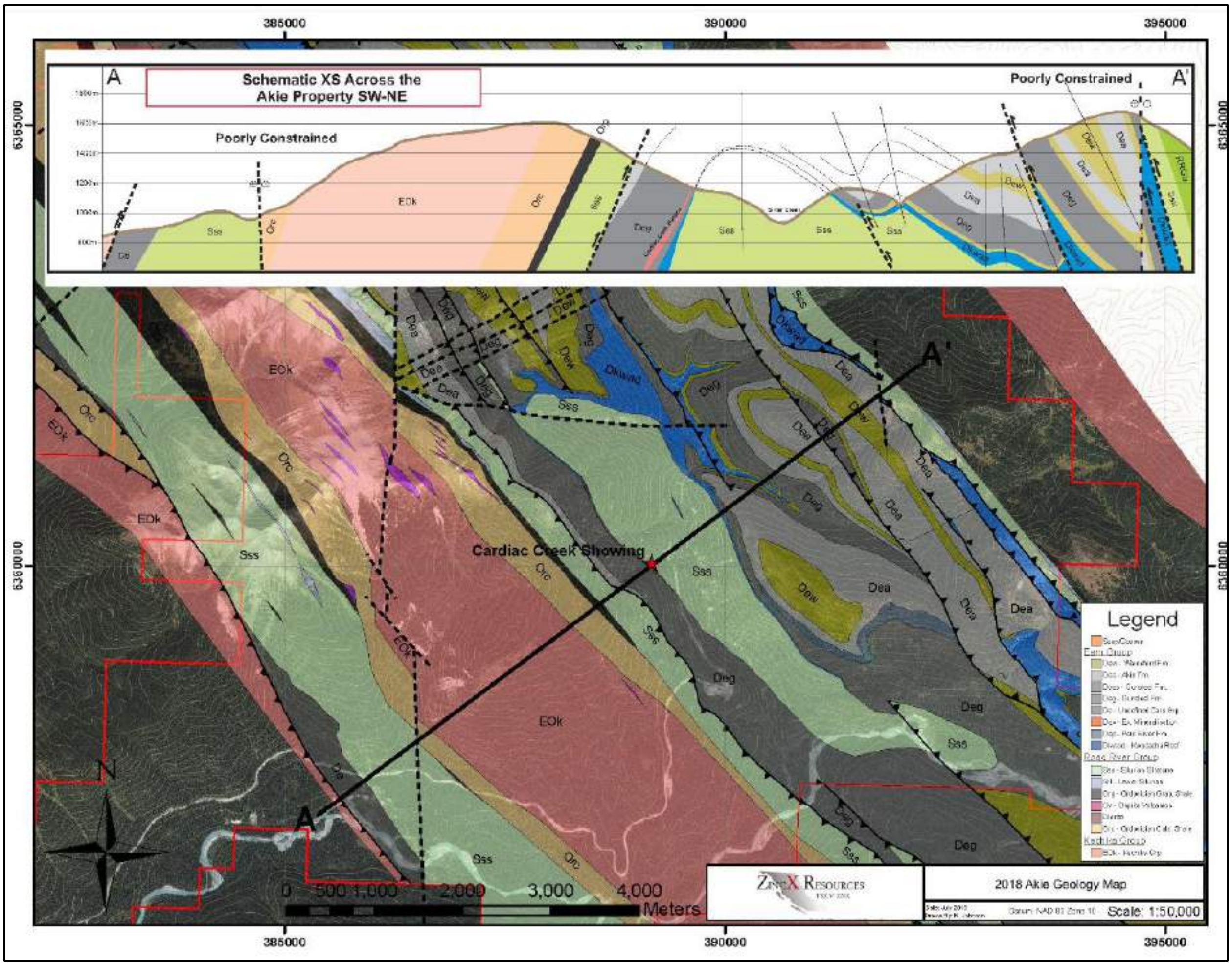
7.3 Property Geology

The geology of the Akie property can be subdivided into east and west segments by Silver Creek. To the west of Silver Creek, the wavy-bedded mudstone with nodular limestone rocks of the Kechika Group, the Ospika volcanics and siltstones, black graptolitic shales, limestones and calcareous siltstones of the Road River Group form a series of southeast-striking, southwest-dipping imbricated thrust panels that are in thrust contact with a thick, approximately 500 m panel of southeast-striking, southwest-dipping Earn Group rocks comprised primarily of Gunsteel Formation shales that are the host to the Cardiac Creek deposit. The panel of Gunsteel Formation shale is currently interpreted to represent the limb of a steeply inclined overturned syncline and the steeply dipping western limb of a large anticline that straddles Silver Creek. The Gunsteel Formation shales are underlain by the dolomitic to weakly calcareous siltstones of the Silurian Siltstone of the Road River Group. This siltstone straddles Silver Creek and represents the core of a large anticline that is central to the property. Along the eastern flanks of the antiform the Silurian Siltstone is immediately overlain by medium grey fossiliferous limestone of the Kwadacha Reef. The limestone is characterized by abundant crinoids, brachiopods, corals and other fossils (MacIntyre, 2008) and typically outcrops along the eastern banks of Silver Creek.

Erosion of the limestone by the local streams and creeks feeding into Silver Creek from the east has produced steep cliffs and gorges with waterfalls. Locally, immediately overlying the limestone is a thin lens of chert pebble conglomerate containing millimetre- to centimetre-sized grains hosted in a silty shale matrix (Baxter, 1996c). The rocks of the Gunsteel Formation are recognized above this conglomerate unit and are exposed across much of the eastern half of the property and have been folded into a number of minor synforms and antiforms. Mappable units within the Gunsteel Formation include the “Pinstripe shale” and chert pebble conglomerate. The pinstripe shale is exposed along ridge tops in the central area of the property and is characterized by black silty shale interbedded with thinly bedded, light creamy-grey siltstone (Baxter, 1996c). The eastern edge of the property is bounded by a steep east-dipping thrust fault depositing Road River Group limestone on top of the Earn Group stratigraphy (MacIntyre, 2005).

In general, the geology of the Akie property has been described as a large anticlinorium bound by outwardly dipping thrust faults (MacIntyre, 1998). Minor thrusting and faulting is observed across the property, each producing an unknown degree of displacement. The geology of the Akie property can be seen in Figure 7-3. Drilling on the Akie property has focused on the rocks of the Gunsteel Formation rather than those of the Akie, Warneford and Paul River Formations, the Silurian Siltstone and other rocks of the Road River Group.

Figure 7-3: Akie Geology Map



Source: ZincX (2018)

7.4 Property Mineralization

The following description of the style and character of the Cardiac Creek deposit and the mineralized horizon is an excerpt from the assessment report *"Summary Report on the 2015 Diamond Drilling Program, Akie Project, Akie Property"* written and compiled by N. Johnson (2016a). It is presented here unabridged except for a few minor edits. The outlined information remains current as of the date this report was issued.

Discovery of the Cardiac Creek deposit in 1994 (MacIntyre and Sim, 2008; Baxter, pers. comm.) is recent in comparison to other known occurrences of lead, zinc, silver and barium mineralization found within the Kechika Trough. In contrast, the Cirque and Driftpile deposits, as well as the Mt. Alcock, Pie, Fluke and Elf occurrences, were all discovered prior to 1980. The deposit attributes its name to its discovery by Paul Baxter and his exploration team while prospecting along Cardiac Creek (Figure 7-4). Initial drilling programs conducted by Inmet Mining from 1994 to 1996 defined a historical non-NI 43-101 compliant resource of 12 Mt @ 8.6% Zn, 1.5% Pb and 17.1 g/t Ag (MacIntyre, 2005). In 2012, the company updated the maiden

2008 NI 43-101 compliant resource calculation for the Cardiac Creek deposit. The 2012 NI 43-101 compliant resource outlined an Indicated resource of 12.7 Mt grading 8.38% Zn, 1.68% Pb, and 13.7 g/t Ag and an Inferred resource of 16.3 Mt grading 7.38% Zn, 1.34% Pb, and 11.6 g/t Ag at a cut-off grade of 5% Zn (Sim, 2012). The deposit is centrally located on the Akie property claim block, situated beneath the Cardiac and Avalanche Creek beds (which drain into the northwest-southeast-oriented Silver Creek).

Figure 7-4: Cardiac Creek Discovery Showing

Source: MacIntyre (2005)

There are two other significant mineral occurrences on the Akie property: the GPS bedded barite showing located on the western edges of the property; and the Sitka barite-sphalerite-galena-quartz vein showing located on the eastern edges of the property. The GPS showing consists of a 1 m to 2 m thick bed of massive barite with an approximate strike extent of 50 m to 100 m. It is similar in character to that of the Barite facies observed below the Cardiac Creek deposit (see Section 7.4.6 for a description). The showing is hosted within a thin panel of black shale that is lithologically similar to that of the Gunsteel Formation. The black shale is overlain by calcareous siltstone of the Road River Group. The host black shale is also directly along strike from the Cirque deposit located approximately 10 km to the northwest. The Sitka showing is a 2 m to 3 m thick barite-quartz vein with variable amounts of disseminated coarse-grained sphalerite and galena (Figure 7-5). The vein is situated along the thrust contact between the older Silurian Siltstone of the Road River Group and the prospective black shales of the Earn Group.

Figure 7-5: Sitka Showing (Barite-Quartz Vein +/- Galena & Sphalerite)

Source: CZM (2014)

7.4.1 Character

The Cardiac Creek deposit is hosted by the siliceous, carbonaceous black shale of the Gunsteel Formation. The deposit is situated towards the base of the Gunsteel Formation near the Gunsteel Formation shale / Road River Group contact and separated by a thin sliver of debris flows and silty to turbiditic shale associated with the Paul River Formation. The deposit is interpreted to be a SEDEX-type lead-zinc-silver body of mineralization. The mineralization is represented by a “sheet-like” tabular body of interbedded sulphides and shale trending northwest-southeast, striking at 130°, dipping at 70° southwest, and ranging in thickness from 5 m to 50 m. The mineralized horizon can be traced over 7 km from the Bear Valley Creek southeast to the Akie River. The known and potentially economic portion of the deposit has an approximate strike length of 1,500 m with a dip extent of at least 850 m. The sulphide mineralogy of the deposit is relatively simple, dominated by pyrite, sphalerite, and galena with barite (sulphate). Internal company petrological reports have identified a rare occurrence of Stannite (Sn oxide) (Lehne, 1995); however, no systematic petrological study of the mineralogy has taken place. Analytical data collected from drill hole sampling indicate that the Cardiac Creek deposit is enriched in the following suite of elements: Pb, Zn, Ag, Ba, Fe, Cd, Sn, Tl, Hg, S, Pd, In, and Ga.

7.4.2 Mineral Facies

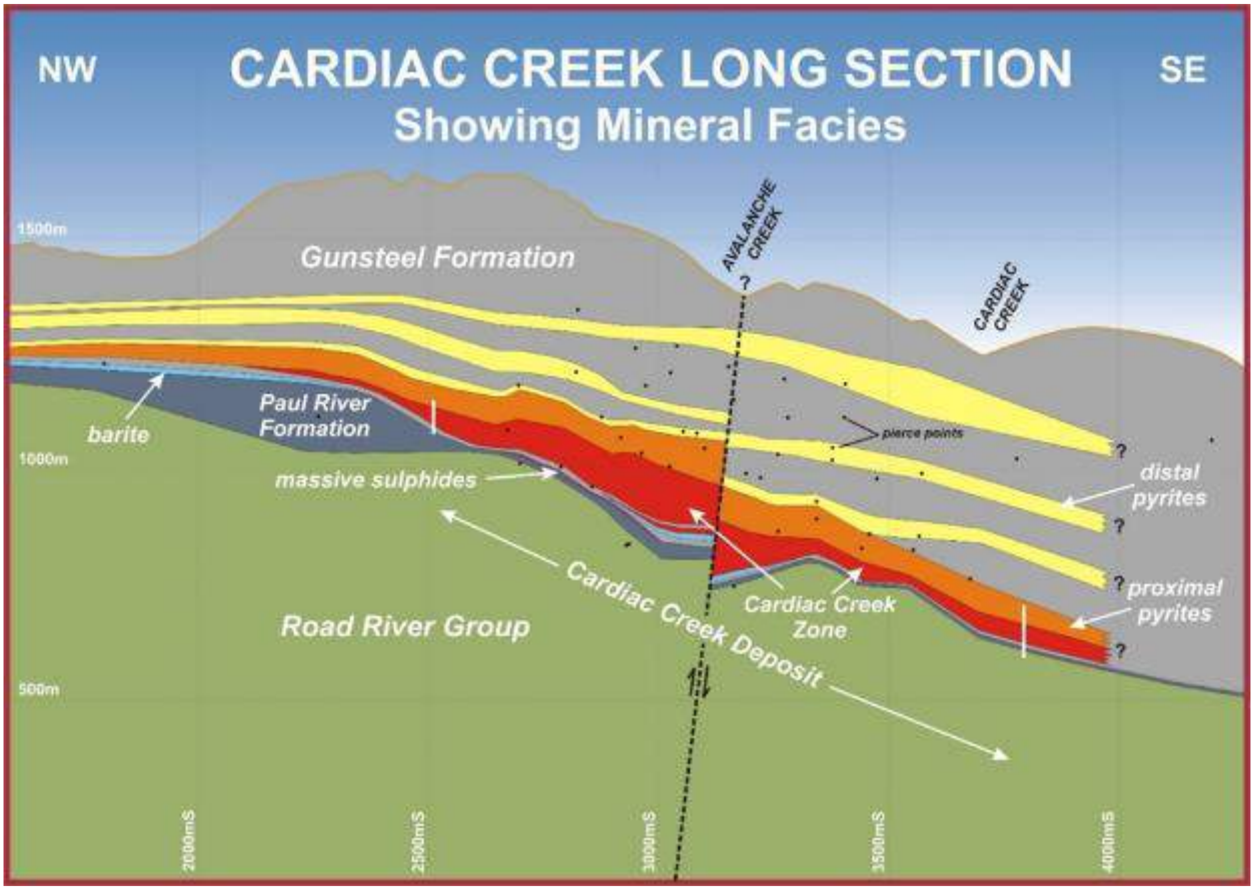
The prospective mineralized horizon associated with the Cardiac Creek deposit can be attributed to several distinct mineral facies present within the Gunsteel Formation stratigraphy: Distal, Proximal, Cardiac Creek Zone (CCZ) and Barite facies (Figure 7-6 and Figure 7-14). A schematic distribution of mineral facies across the deposit can be seen in Figure 7-7.

Figure 7-6: Mineral Facies Associated with the Cardiac Creek Deposit

		Facies	General Description
Mineralised Horizon		Distal	10 to 20cm thick bands comprised of thinly laminated pyrite with nodular barite and shale interbedded with generally featureless black massive shale beds. Not always present above proximal facies.
	Cardiac Creek Deposit	Proximal	20 to 60cm thick beds of finely laminated pyrite with lesser nodular barite and minor steel grey sphalerite bands interbedded with pyritic massive black shale beds. Contact with underlying Cardiac Creek Zone very gradational
		Cardiac Creek Zone	20cm to >1m thick beds of steel grey sphalerite, pyrite and galena interbedded with pyritic massive black shale beds. "Mottled" texture indicates high grade Zn, Pb mineralisation. Also host to sub-rounded to angular rip-up clasts.
		Barite	1 to 10m thick beds of offwhite, granular looking, massive barite generally in gradational contact with the Cardiac Creek Zone and host to minor pyrite, sphalerite and galena mineralisation. Character can change from massive to laminar/nodular to nodular bedded barite.

Source: CZM (2011)

Figure 7-7: Schematic Distribution of Mineral Facies across the Cardiac Creek Deposit

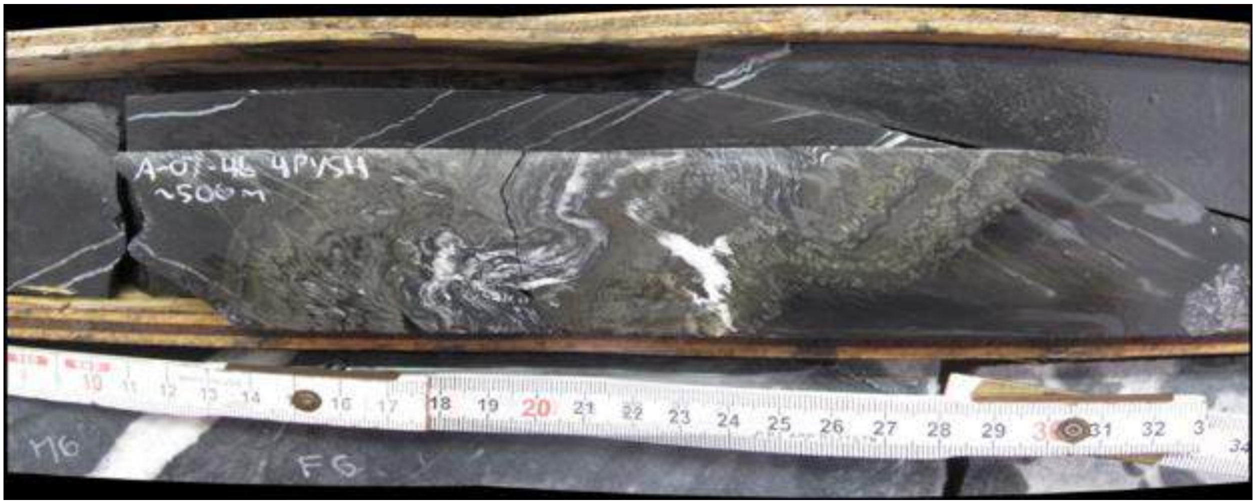


Source: CZM (2016)

7.4.3 Distal Facies

The Distal facies is interpreted to represent the distal expression of the deposit both in the immediate hanging wall and along strike. The facies are represented by 10 cm to 20 cm thick bands individually comprised of interbedded, thinly laminated, fine-grained, dull-brown pyrite, black shale and off-white nodular barite (commonly replaced by carbonate and brassy yellow euhedral pyrite) interbedded with generally featureless black Gunsteel Formation shale (Figure 7-8). The facies can vary significantly in thickness from less than 5 m to more than 100 m. The overall sulphide content ranges from 5% to 15%, and zinc and lead grades reach < 0.1% to 0.5%, and < 0.1%, respectively and the facies is not always present in the immediate hanging wall or along strike to the deposit. Several additional horizons of identical character have been recognized further into the hanging wall and are interpreted to represent separate mineral horizons possibly post-dating the Cardiac Creek mineral horizon.

Figure 7-8: Distal Facies Mineralization in A-07-46 @ 506.00 m



Source: CZM (2011)

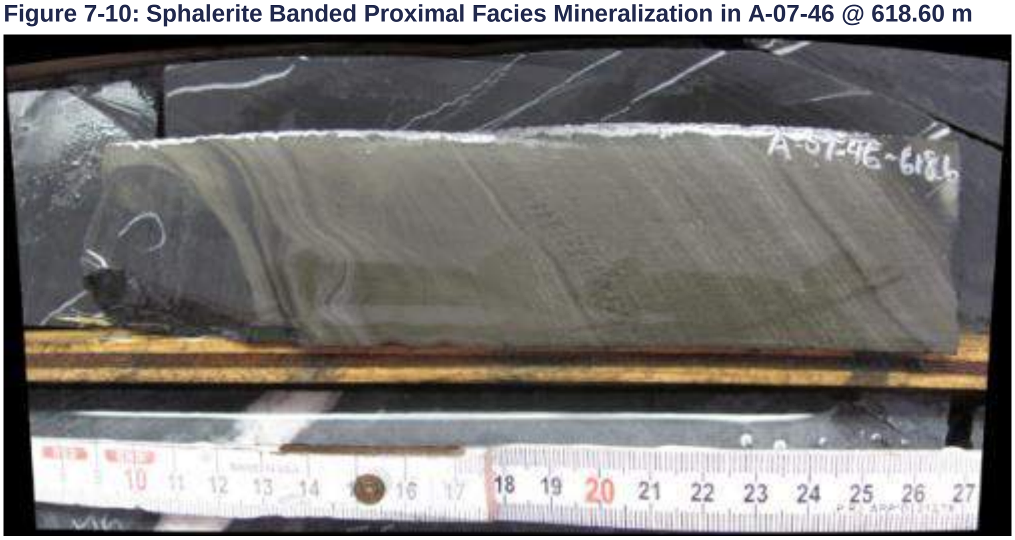
7.4.4 Proximal Facies

The Proximal Facies is interpreted to represent the upper portion of the deposit and consists of 20 cm to 60 cm thick, internally laminated, very fine-grained, dull brown pyrite beds with very minor amounts of nodular barite (generally sub-millimetre and replaced by carbonate and brassy yellow pyrite) interbedded with featureless pyritic massive black shale beds (Figure 7-9). The appearance and concentration of steel grey sphalerite bands increases towards the base of the Proximal Facies with a very gradational boundary between the Proximal and Cardiac Creek Zone facies (Figure 7-10). The determination of this boundary is subjective, but in general it is marked by the substantial increase in sphalerite banding within the pyrite beds. The facie ranges in thickness from 5 m to 30 m in which the overall sulphide content reaches 30% to 50%. Zinc and lead grades are on the order of 0.5% to 3% and up to 0.5%, respectively.

Figure 7-9: Proximal Facies Mineralization in A-07-46 @ 619.40 m

Source: CZM (2011)

Figure 7-10: Sphalerite Banded Proximal Facies Mineralization in A-07-46 @ 618.60 m



Source: CZM (2011)

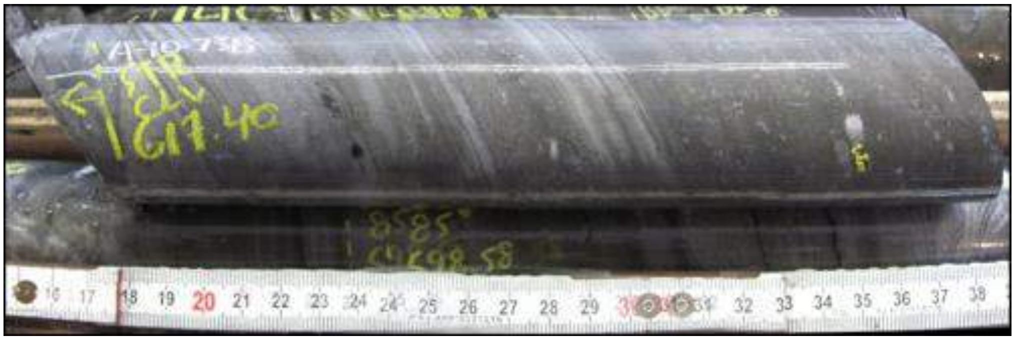
7.4.5 Cardiac Creek Zone Facies

The Cardiac Creek Zone facies represents the lower segment of the deposit and consists of 0.30 m to 2.0 m thick sulphide beds internally comprised of: laminated very fine-grained, dull-brown pyrite; very fine-grained steel-grey sphalerite bands with minor galena; and barite interbedded with generally featureless, pyritic, black Gunsteel Formation shale beds. The facies range in thickness from 5 m to 40 m, and sulphide content reaches 50% to 70%, with zinc, lead and silver grades of 3% to 30%, 1% to 5%, and 5 g/t to 30 g/t, respectively. Higher grade zinc and lead mineralization is associated with a “mottled” texture hosted within the sphalerite bands (Figure 7-11). The lower contact is gradational with the Barite facies (Figure 7-12). Also hosted within the facies are numerous angular to sub-rounded, bedded, light grey white to dark grey concretions that are hosted within the interbeds of black shale and the sulphide beds (Figure 7-10).

Figure 7-11: Mottled Textured High-Grade Cardiac Creek Zone Mineralization in A-07-47 @ 375.60 m

Source: CZM (2011)

Figure 7-12: High-Grade Sphalerite Mineralization Interbedded with Barite in A-10-73B @ 617.40 m



Source: CZM (2011)

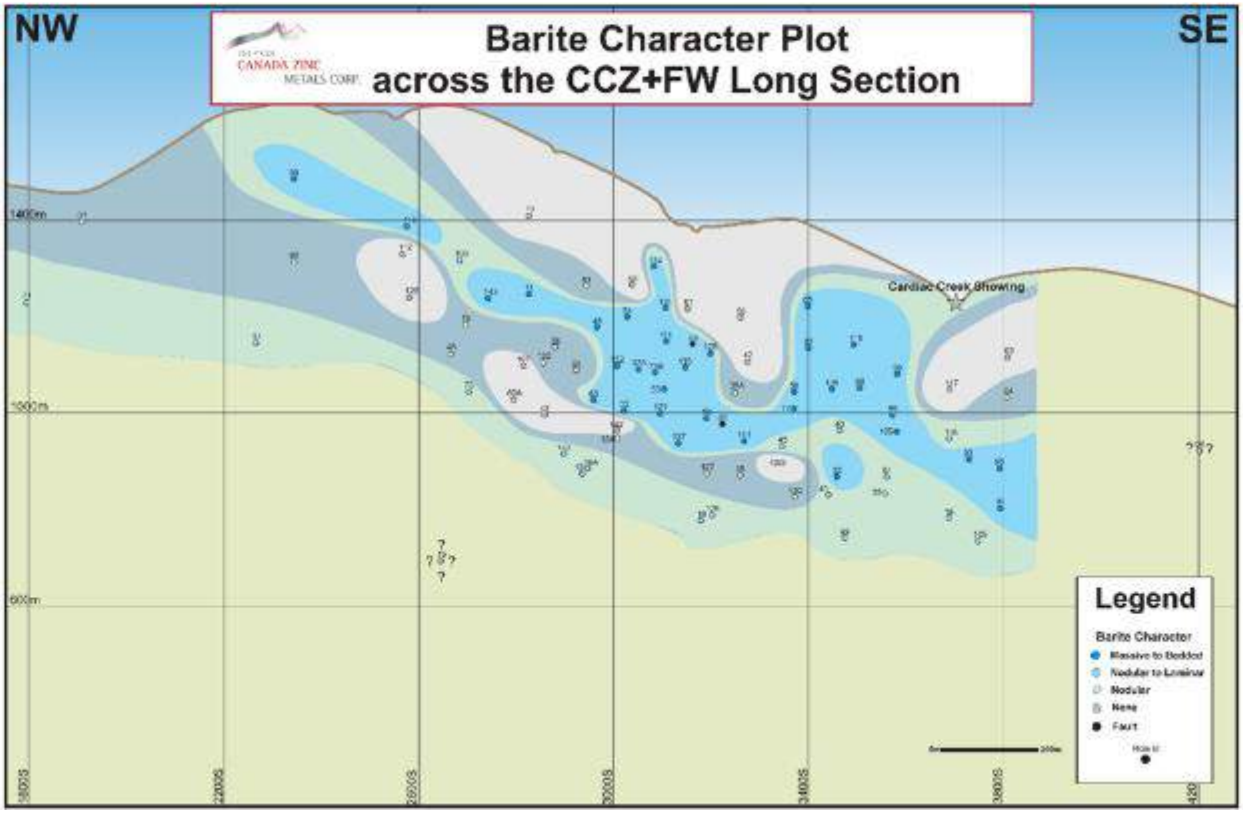
7.4.6 Barite Facies

The deposit is underlain by the Barite facies (Figure 7-5). This facies changes in character across the deposit from thickly bedded (1 m to 10 m) off-white, granular, massive beds of barite interbedded with minor pyrite, sphalerite and or galena (Figure 7-13), to thinly-bedded barite with nodular barite, to strictly nodular barite with little to no sulphide mineralization. The zinc, lead and silver grades vary substantially depending on the sphalerite or galena content.

Figure 7-13: Massive Granular Barite Bed in A-07-50 @ 574.30 m

Source: CZM (2011)

Figure 7-14: Barite Facies Character across the Deposit

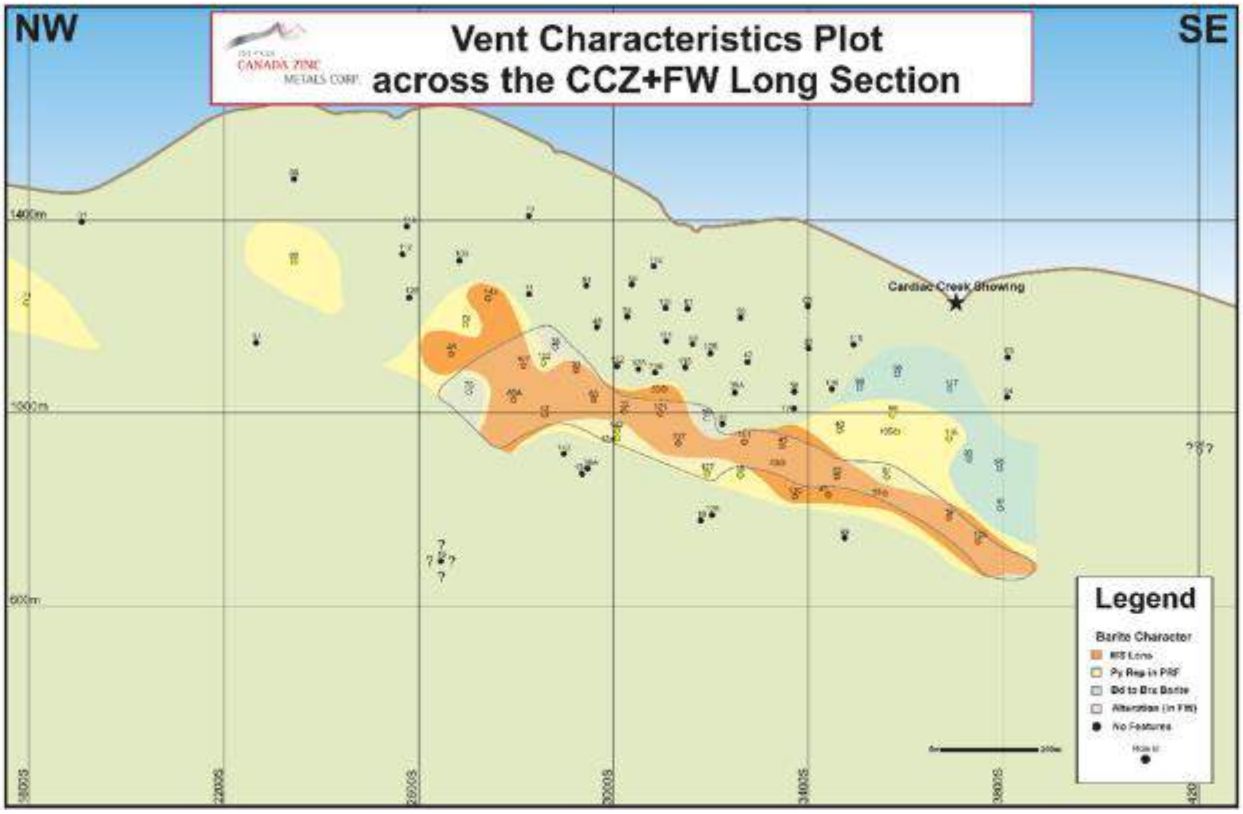


Source: ZincX (2018)

7.4.7 Vent-Proximal Characteristics

The Cardiac Creek deposit is underlain by features that are suggestive of its proximity to a possible hydrothermal vent, such as thin, crudely layered, semi-massive sulphide lens, sulphide replacement of the Paul River debris flow, and silicification, sulphide stringers and breccias, carbonate veining, barite needles and laths present within the immediate footwall rocks of the Road River Group siltstone (Figure 7-16 and Figure 7-17). These features are generally concentrated across the core of the deposit with a rough correlation to the higher-grade material (Figure 7-6).

Figure 7-15: Long-Section View Across the Cardiac Creek Deposit



Note:

Vent zone features yellow: pyrite replacement of debris flow; grey: silicification, sulphide stringers and sulphide breccias in calcareous siltstone; orange: massive sulphide lens.

Source: ZincX (2018)

Figure 7-16: Silicification & Carbonate Veining Containing Sphalerite in Road River Rocks in A-08-63 @ 484 m

Source: CZM (2011)

Figure 7-17: Sphalerite-Rich Sulphide Breccias in Road River Rocks in A-08-63 @ 479 m

Source: CZM (2011)

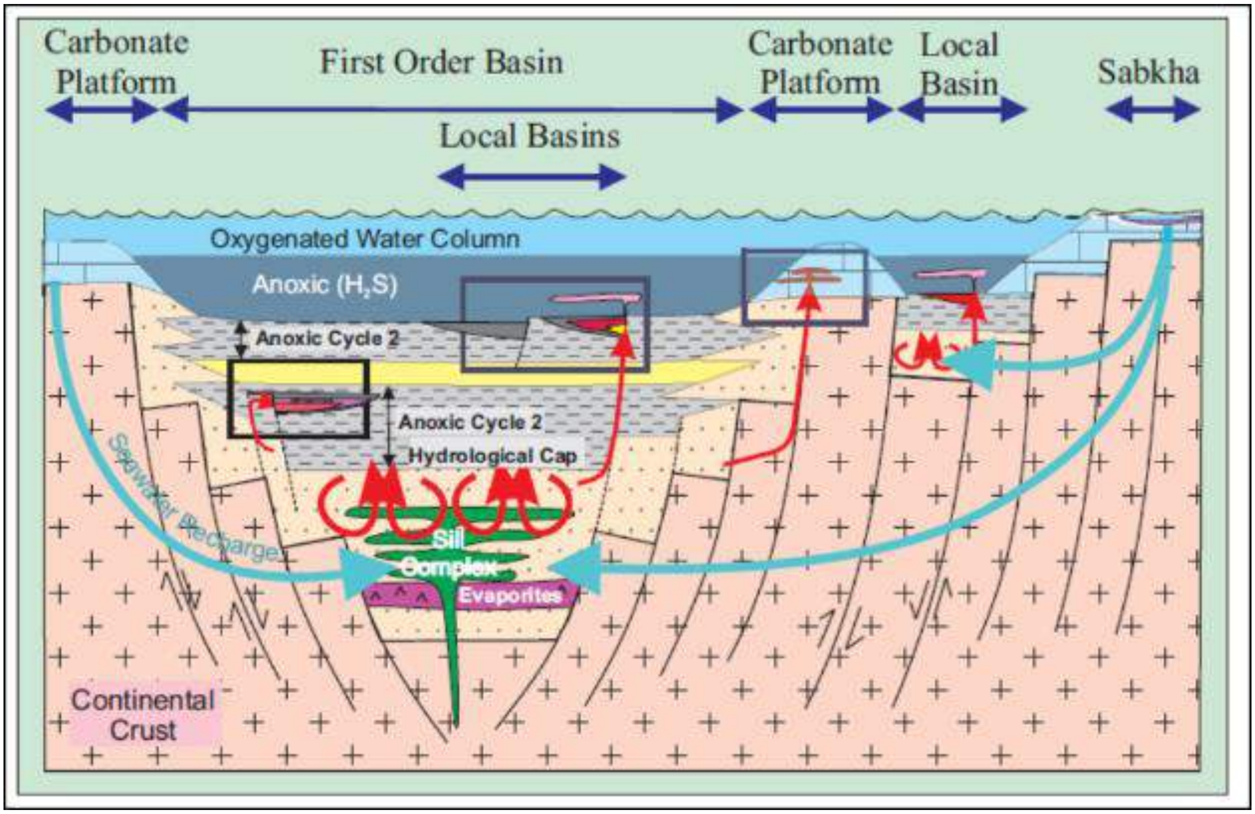
8 Deposit Types

The Cardiac Creek, Cirque, Driftpile deposits and other lead-zinc-silver occurrences within the Kechika Trough are characterized as sedimentary exhalative (SEDEX) type deposits. The following is a general summary of this deposit type and its characteristics. For a detailed review of SEDEX deposits, the reader is referred to the excellent overview paper of Canadian SEDEX deposits by Wayne D. Goodfellow and John W. Lydon (2007), "*Sedimentary Exhalative (SEDEX) Deposits from the publication Mineral Deposits of Canada: A Synthesis of Major Deposit Types, District Metallogeny, the Evolution of Geological Provinces, and Exploration Methods by the Geological Association of Canada, Mineral Deposits Division, Special Publication No. 5., 2007.*"

The lead-zinc-silver-barium deposits and occurrences found within the Kechika Trough (Cirque, Driftpile and Cardiac Creek), as well as the deposits and occurrences in the Selwyn Basin (Howards Pass, Tom, Jason, Faro and Grum), the Belt-Purcell District (Sullivan), and in Australia (HY, Century, Mount Isa and Broken Hill) and the Brookes Range in Alaska (Red Dog) all share common characteristics and are typically grouped as SEDEX deposits (Goodfellow and Lydon, 2007). The SEDEX deposit type was first proposed by Carne and Cathro (1982) in their early description of the Selwyn Basin and Kechika Trough deposits. This type of deposit shares many similar characteristics with VMS (volcanogenic massive sulphide) and MVT (Mississippi Valley Type) deposits suggesting a shared genetic link (Goodfellow and Lydon, 2007).

Much research has been completed on this type of deposit examining the geological characteristics, genetic models and the physiochemical controls (MacIntyre, 2008). From this work, a general consensus concerning the formation of SEDEX deposits has been made. It is generally agreed that SEDEX deposits are formed from the precipitation of sulphide and sulphate minerals from metalliferous brines exhaled out onto the seafloor along re-activated rift faults that generate rapidly subsiding graben or half-graben structures (MacIntyre, 2008; Goodfellow and Lydon, 2007). However, recent work by Gadd et al. (2015) on the Howards Pass deposit in the Selwyn basin is beginning to test this theory which may not apply to all SEDEX deposits in the Selwyn Basin and or Kechika Trough. The metal-bearing fluids are likely derived from dewatering of fine- to coarse-grained clastic sediments or carbonate hydrothermal reservoirs (Goodfellow and Lydon, 2007) where leaching has scavenged the zinc and lead and other elements (Figure 8-1). In the Selwyn Basin and the Kechika Trough, the coarse clastic grits of the Windermere Super Group are thought to have acted as the hydrothermal reservoir for the mineralizing fluids (MacIntyre, 2008).

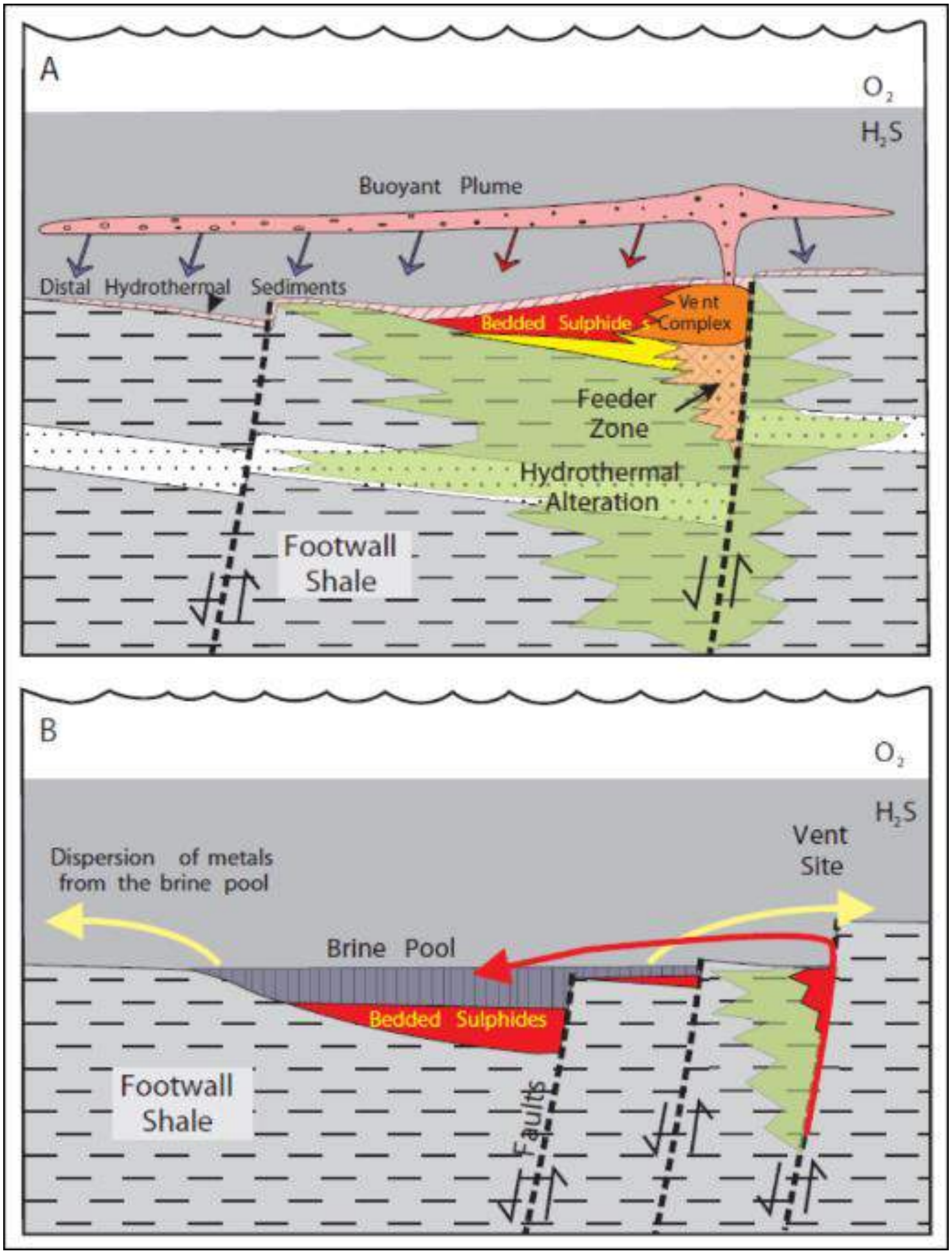
Figure 8-1: Genetic Model of SEDEX Deposit Formation



Source: Goodfellow and Lydon (2007)

Goodfellow and Lydon (2007) recognized two sub-types of SEDEX deposits: vent-proximal and vent-distal. The two types of deposits result from either a buoyant metalliferous brine that precipitates sulphides near the source fault structure or a bottom-hugging brine that precipitates sulphide mineralization within localized third order basins at a distance from the source fault structure (Figure 8-2). Examples of the vent-proximal deposits include Sullivan, Tom, Jason and Rammelsberg and are characterized by four distinct features, including bedded sulphides, a recognized vent complex, a stringer zone, and distal hydrothermal sediments (Goodfellow and Lydon, 2007). Vent-proximal deposits are typically wedge-shaped, exhibiting a moderately high aspect ratio of length versus thickness.

Figure 8-2: Vent-Proximal and Vent-Distal Sub-Types of SEDEX Deposits



Source: Goodfellow and Lydon (2007)

In contrast, vent-distal deposits have well-bedded sulphides, are generally weakly zoned and their morphology conforms to the local basin. This type of deposit is typically tabular to sheet-like in nature with very high aspect ratios (Goodfellow and Lydon, 2007).

SEDEX deposits are commonly hosted in basinal marine sediments such as fine-grained clastics, carbonaceous chert and shale representing pelagic sediments. In some cases, the shale can be interbedded with turbiditic siltstone and sandstone and localized coarse-grained sediments (Goodfellow and Lydon, 2007).

The mineralogy associated with this type of deposit is generally simple with pyrite, sphalerite, galena and barite being most common. Associated with these minerals are a suite of elements that may include: As, Bi, Ca, Cd, Co, Fe, Ga, Hg, In, Mn, Ni, P, Sb, Se, Sn, and Tl (Goodfellow and Lydon, 2007). The gold content of SEDEX deposits is quite low; however, deposits found in Anvil district of the Yukon (Vangorda, Dy) district contained mineable grades of the precious metal (Goodfellow and Lydon, 2007). These elemental enrichments commonly exhibit a refined zonation across many of the deposits allowing specific ratios to be used as exploration tools guiding exploration towards possible source vents and economic deposits (Goodfellow and Lydon, 2007). Common metal ratios include: Zn/Pb, Pb/Ag, Cu/(Pb+Zn), Pb/(Pb+Zn), Fe/Zn, Ba/Zn and SiO_2/Zn (Goodfellow and Lydon, 2007).

9 Exploration

Early exploration activities conducted by Inmet Mining from 1994 to 1996 are summarized in Section 6 - History. Exploration activities completed by ZincX, apart from the drilling which is documented in Section 10, included geological mapping, prospecting, soil and rock sampling, geophysics and an orientation hydrogeochemical sampling program. These activities contributed to a more thorough understanding of the regional setting of the Gunsteel Formation on the Akie property and provided additional target areas for follow up exploration.

9.1 Hydrogeochemical Sampling

In 2011, a total of 14 water samples were collected on the Akie property as part of an orientation study for major and trace elements in stream waters. Samples were collected from both the Akie and Pie properties. This study was designed to determine the effectiveness and applicability of field-testing for barium sulphate in stream water samples as a possible indicator for nearby SEDEX mineralization (Caron, 2007). The levels of barium sulphate in each sample were measured qualitatively in the field, and quantitatively in the laboratory.

This study returned anomalous values of SO_4 (between 50 mg/L and 100 mg/L) downstream from the GPS bedded barite showing (Sa# 860613) as expected; however, the sample (Sa# 860605) taken downstream from the Cardiac Creek showing, returned a nominal value of SO_4. However, sample #860605 returned the highest concentrations of zinc at 130.6 ppb and thallium at 0.10 ppb. Key results from the 2011 orientation survey are listed in Table 9-1. Both the qualitative approach and laboratory analysis reconcile sufficiently to suggest accuracy in the analytical results.

Based on the results from the 2011 orientation survey the program was expanded in 2012 to include several of the Kechika Trough properties, including additional sampling on the Akie property. A total of 121 samples were collected as part of the program of which 27 samples were from the Akie property (Figure 9-1). The 2012 program focused on obtaining a potential geochemical signature associated with known deposits and key showings with samples being taken immediately upstream and downstream as well as identifying new areas of interest for future exploration. On the Akie property creeks downstream of the Cardiac Creek showing, the GPS bedded barite showing and the Elf showing were all sampled. The sampling demonstrated that immediately downstream of the Cardiac Creek showing indicated elevated values of Ba, Ca, Cu, K, Na and Tl. Slight increases of Mg, Pb, Sb, Si, Sr and U were also observed. Compiling the results from all the known showings indicated that a possible geochemical signature might involve elevated values of the following elements:

Ba, Pb, Rb, Sb, Sr, U +/- Cu, P, Tl, SO_4

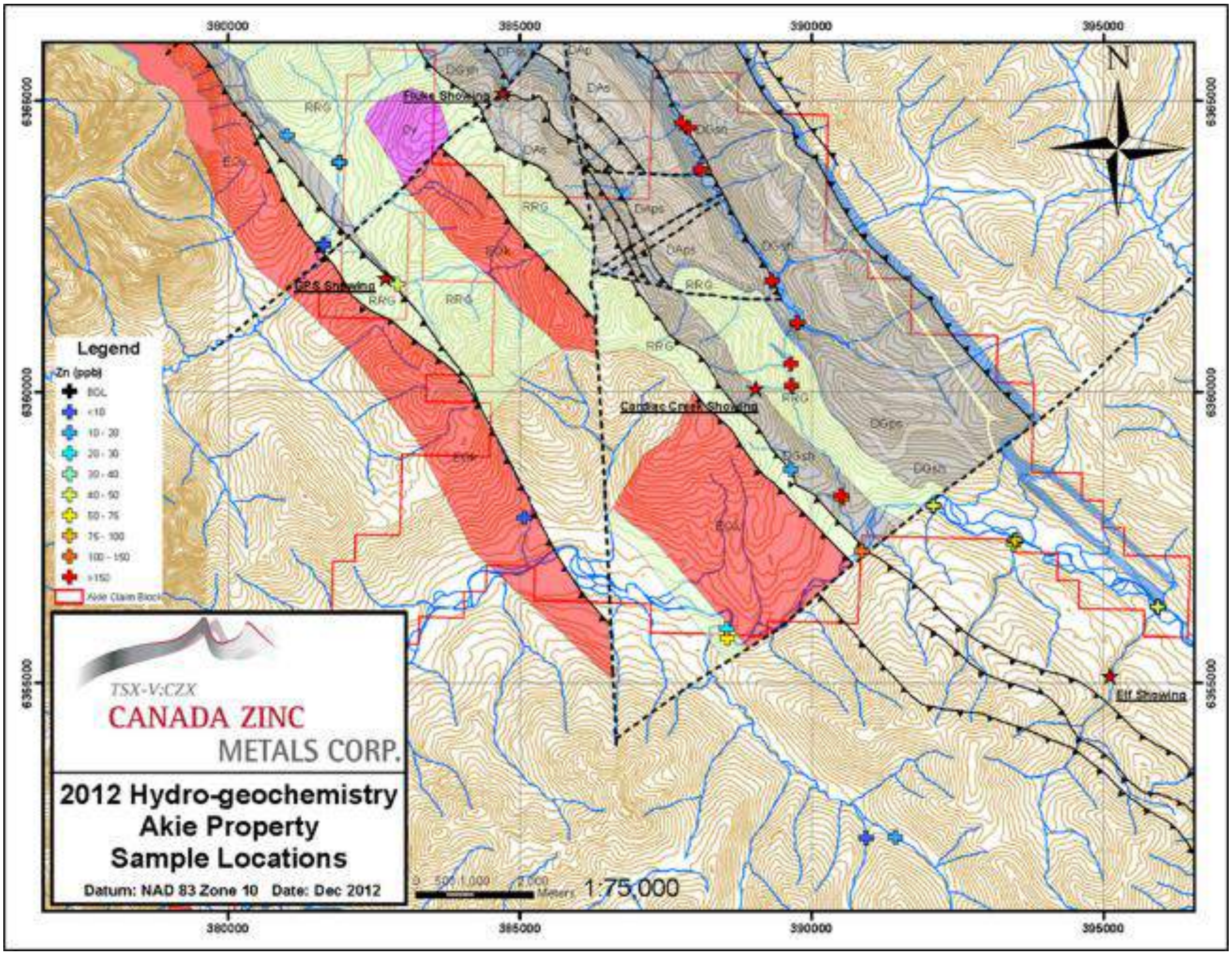
The program also produced lower than expected zinc values downstream of the known showings or deposits. It was found that zinc appeared to be an excellent vector to guide exploration to a general area rather than a specific drainage for further exploration (Johnson, 2013).

Table 9-1: 2011 Baseline Water Sampling Program Results

Comparison of Field, Colorimeter and Analysis of SO_4 Water Samples															
	SO_4 Concentration			Sample Results											
	Hach Method 8051		Ion Chromatography												
Sample #	Field Ob (mg/L)	Colorimeter (mg/L)	ACME Labs (mg/L)	Certificate #	Ba (ppb)	Zn (ppb)	Cu (ppb)	Mn (ppb)	Ni (ppb)	Co (ppb)	Tl (ppb)	La (ppb)	Pb (ppb)	Ag (ppb)	As (ppb)
860601	20+	22	19	VAN11005190	62.89	<0.5	0.3	<0.05	<0.2	<0.02	<0.01	<0.01	<0.1	<0.05	<0.5
860602	50+	56	44	VAN11005190	66.38	<0.5	0.4	0.18	<0.2	0.03	<0.01	<0.01	<0.1	<0.05	<0.5
860604	50+	54	42	VAN11005190	72.53	1.6	0.4	0.38	0.2	<0.02	<0.01	<0.01	<0.1	<0.05	<0.5
860605	50+	51	45	VAN11005190	90.37	130.6	0.8	1.10	6.7	0.04	0.1	<0.01	<0.1	<0.05	<0.5
860607	20+	44	35	VAN11005190	111.89	12.5	0.6	0.19	3.9	0<0.02	0.01	<0.01	<0.1	<0.05	<0.5
860608	20+	53	53	VAN11005190	106.37	87.1	0.6	1.38	12.2	0.15	0.06	<0.01	<0.1	<0.05	<0.5
860609	20+	43	37	VAN11005190	68.51	4.8	0.4	0.13	<0.2	<0.02	<0.01	<0.01	<0.1	<0.05	<0.5
860613	100+	83	86	VAN11005190	79.90	1.5	0.7	0.14	2.3	0.03	<0.01	<0.01	<0.1	<0.05	<0.5

Note: Samples were prepared in the field by field staff using Hach method 8051. SulfaVer® 4 reagent (BaCl) added to 10 ml WQ sub-sample in a Hach 10 ml sample.
Source: CZM (2013)

Figure 9-1: 2012 Water Sampling Program, Akie Property



Source: CZM (2013)

9.2 Prospecting and Mapping

In 2008 and 2009, mapping was completed across the property at a scale of 1:10,000. Traverses were generally restricted to the ridgelines and creeks where outcrops generally occur on the property. This work resulted in incremental gains in the understanding of the geology. The final interpretation was similar to the work completed by Inmet Mining. No additional occurrences of Cardiac Creek-style mineralization were discovered during the mapping; however, numerous iron seeps were identified along Silver Creek.

In 2009, prospecting on the northwestern edges of the property discovered a thin panel of black shale hosting a bedded barite occurrence named the GPS showing. This panel of black shale is situated directly along strike from the Cirque deposit (to the northwest) and has been tentatively identified as Gunsteel Formation shale.

In 2013, additional mapping focused on the eastern side of the Akie property and the southeast strike extension of the GPS showing to better understand the geology where little to no mapping had been completed. The mapping along strike at the GPS showing tentatively identified a strike extension of the western panel of Earn Group rocks, and mapping on the eastern side of Silver Creek better defined the geology along the eastern edges of the Akie property, specifically the contact between the Earn Group stratigraphy that is in thrust contact with older Road River Group rocks. Additionally, a sphalerite, galena-bearing barite-quartz vein named the Sitka showing was discovered along the thrust contact between Earn Group rocks and the Silurian Siltstone. The geology of the Akie property can be seen in Figure 7-3.

9.3 Rock and Litho-geochemical Sampling

As part of the mapping and prospecting programs, a total of 65 rock samples were taken across the property (including channel samples), but the primary focus was the area surrounding the GPS bedded barite showing. The channel samples that transected the barite showing returned expected barium values ranging from 3.75% to 38.29% and highly anomalous lead and zinc values of up to 149.77 ppm and 3,263 ppm, respectively. Rock sampling in close proximity to the barite showing to the northwest returned consistently anomalous zinc values over 1,000 ppm and ranging up to > 1%. This sampling is also associated with elevated lead and thallium values ranging up to 157.55 ppm and 4.41 ppm, respectively. This anomaly remains open to the northwest.

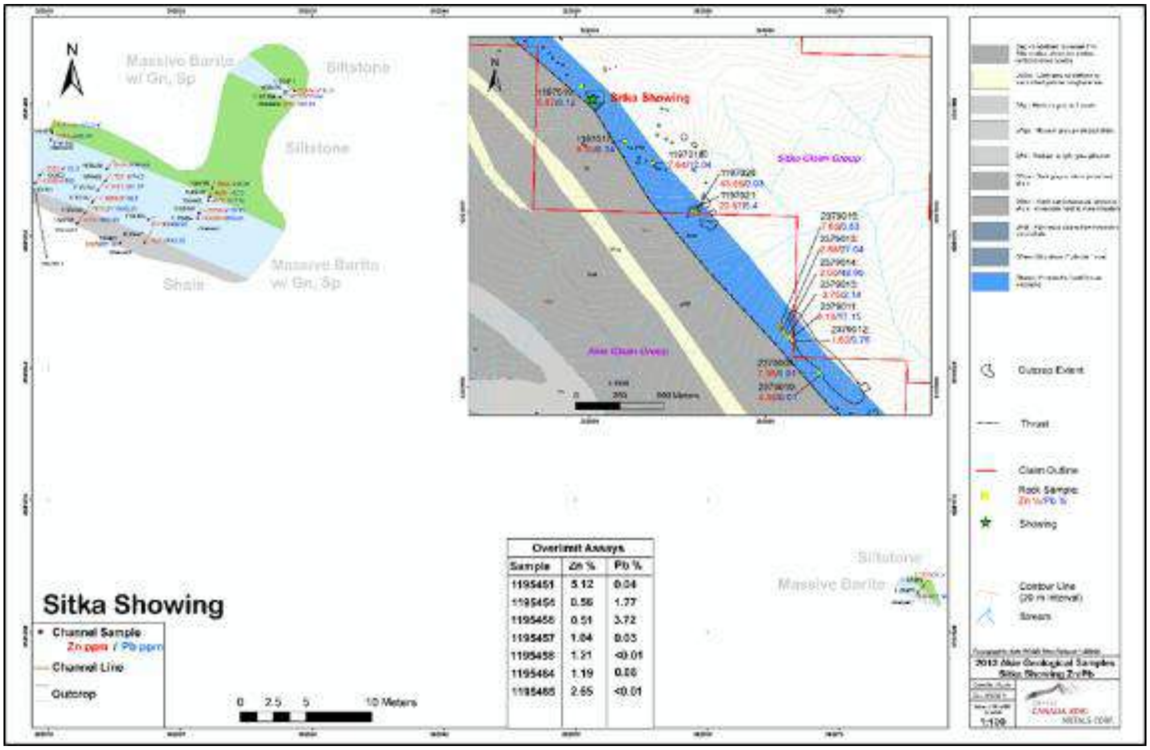
In addition to grab and channel sampling, drill hole A-07-47 was selected for litho-geochemical sampling. A total of 354 samples were taken down the entire length of the hole and analyzed for major and trace elements to identify a possible alteration signature and determine suites of elements that are either enriched or depleted through the stratigraphy. This work indicated that Zn, Pb, Ag, Ba, Cd, Fe, Sn, Tl, Hg, S, Mg, Mn, Ga, Ge and In are enriched elements associated with the deposit.

In 2011, a total of nine rock samples were collected on the Akie property from select locations. There were no significant results obtained from these samples and they did not delineate any obvious trends or geochemical patterns.

The discovery of the Sitka showing in 2013 (Figure 7-5) prompted channel sampling on the showing to be completed. A total of seven channels were cut into the showing and 23 samples were collected (Figure 9-1). The channel samples were highly anomalous in zinc with grades ranging up to 5.12% Zn with values consistently in excess of 2,000 ppm Zn. Both lead and silver grades were elevated with one sample, returning grades of 3.72% Pb and 9,442 ppb Ag. Prospecting in the vicinity resulted in the discovery of additional narrow barite-quartz veins enriched with both galena and sphalerite hosted within the fossiliferous limestones of the Kwadacha Limestone and proximal to the limestone / Earn Group contact. A total of 35 additional grab samples were taken. These grab samples returned some highly anomalous lead and zinc grades, with values reaching 48.95% Pb and 43.55% Zn.

In 2014, a total of 126 drill core litho-geochemical samples were collected from the main lithological units. The goal was to improve on the geochemical characterization of the key lithological units encountered in drilling and assist in the classification of units identified in the field during mapping.

Figure 9-2: Sitka Showing Channel Sampling and Select Grab Samples, Akie Property



Source: CZM (2014)

9.4 Soil and Silt Sampling

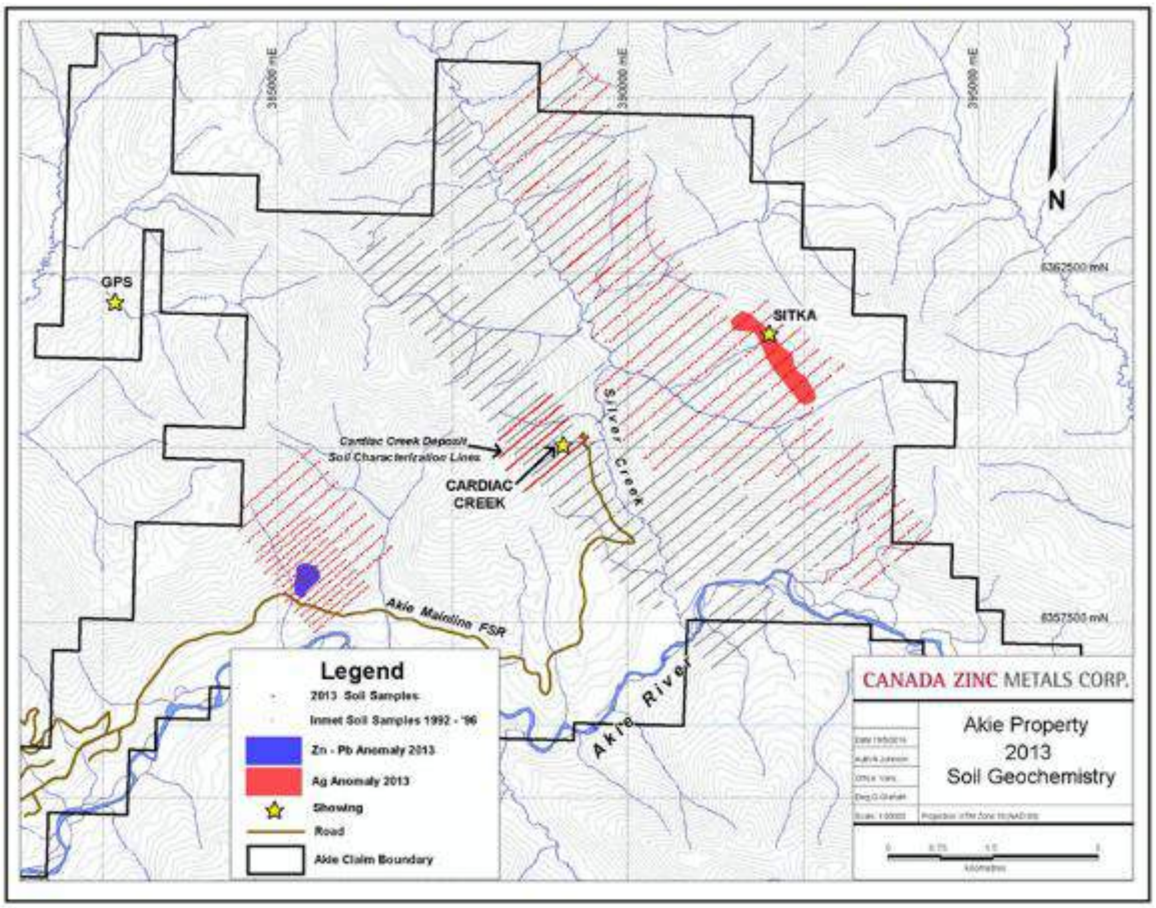
In 2008 and 2009, small soil and silt sampling programs were conducted across the Akie property and focused primarily on the GPS bedded barite showing. A total of 398 samples were collected along 100 m spaced lines at 50 m intervals. The sampling over the GPS bedded barite showing failed to define any significant soil anomalies. In 2011, a small 39 sample program expanded upon the grid at the GPS showing to close off weakly anomalous soils with zinc values ranging from 100 ppm to 250 ppm. The steep terrain and poor soil profile inhibited sampling over the prospective black shale that hosted the GPS bedded barite showing.

Associated with the soil sampling program were a total of 70 silt samples that were collected in the general vicinity of the GPS bedded barite showing. No significant anomalies were defined, although one sample immediately downstream of the GPS showing returned > 10,000 ppm Zn, > 2,000 ppm Co, > 10,000 ppm Mn, 4,017 ppm Ni and elevated copper at 184.58 ppm.

In 2013, a large soil sampling program focused on: the eastern side of the property; infilling the widely spaced Inmet Mining soil lines; the southeast strike extent of the GPS showing directly southwest of the deposit; and a select number of soil lines situated directly over the deposit testing different digestion and analytical packages. A total of 1,826 samples were taken. This program resulted in the delineation of two distinct anomalies. The sampling shows the highly variable character of silver values on the eastern side of Silver Creek with silver values consistently in excess of 2,500 ppb in the northern portion of the property.

Along the eastern edges of the property, there is a prominent large open-ended silver anomaly measuring approximately 1,400 m long by 275 m wide. This silver-rich trend is in close proximity to the Sitka barite-quartz vein showing. Values are consistently in excess of 1,000 ppb Ag and range up to 15,765 ppb Ag (Figure 9-2). The second anomaly is located directly southwest of the deposit and southeast of the GPS showing. The anomaly is roughly circular in shape and quite small, measuring approximately 300 m by 350 m with values ranging up to 1,690.2 ppm Zn and correlating with values ranging up to 291.73 ppm Pb (Figure 9-2). This anomaly is located within the recently mapped and interpreted continuation of the western panel of Earn Group rocks on the property.

Figure 9-3: 2013 Soil Geochemical Sampling Map



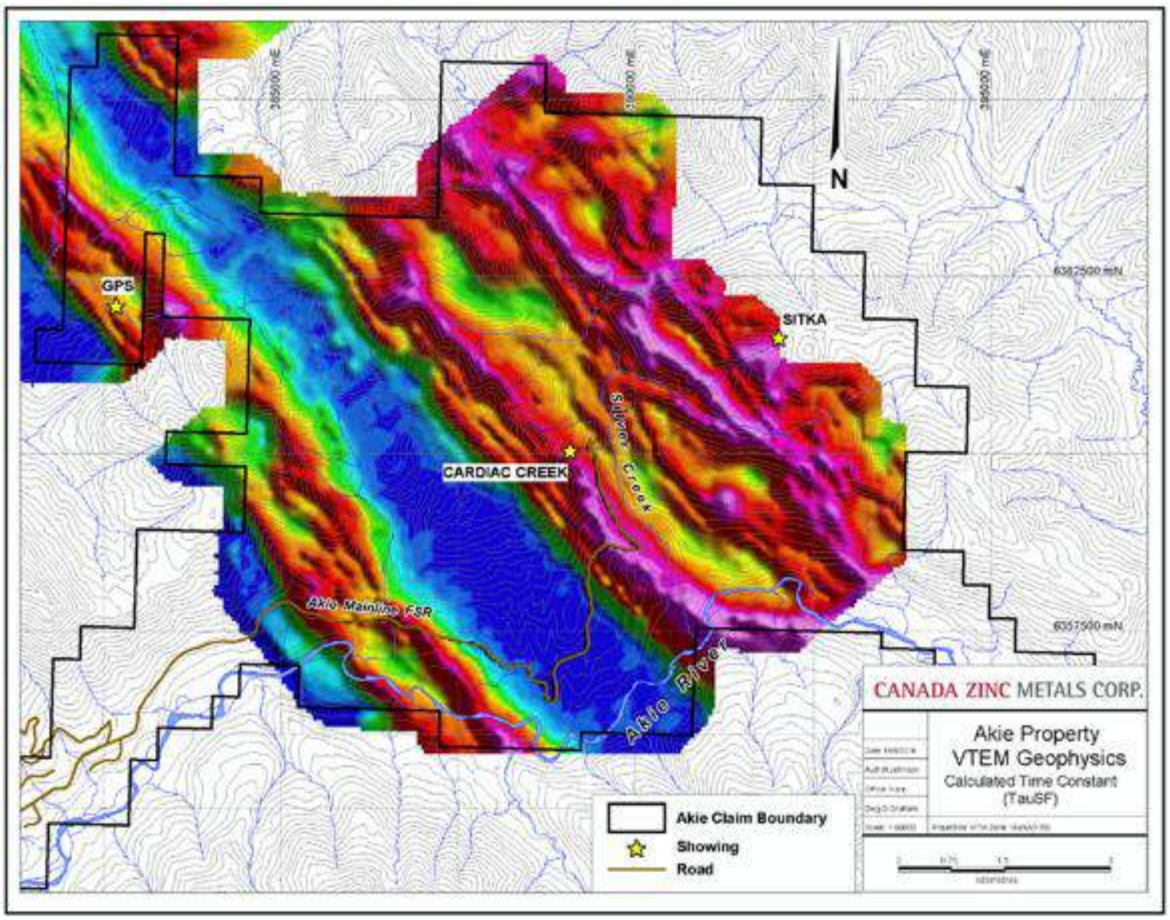
Source: CZM (2016)

9.5 Geophysics

In 2012, Canada Zinc Metals contracted Geotech Ltd. to conduct an airborne Versatile Time Domain Electromagnetic (VTEM) system survey over the Akie, Pie and Mt. Alcock properties with a line spacing of 200 m. A tighter line spacing of 100 m was flown directly over the Cardiac Creek deposit to determine whether there was a unique response from the deposit. The results of the survey provided detailed geological and structural data across much of the Akie property. The prospective Gunsteel Formation was found to produce a strong, distinct electromagnetic (EM) response correlating well with the mapped geology. Work by Condor Consulting Ltd. found that the Cardiac Creek deposit appeared to produce a slightly depressed EM response which correlated with a subdued magnetic response (Condor Consulting, 2014). Other geological units also had unique EM responses allowing for a better geological interpretation. The TauSF response over the Akie property can be seen in Figure 9-3. In 2013, the VTEM survey was subsequently expanded to include all the Company's tenure holdings.

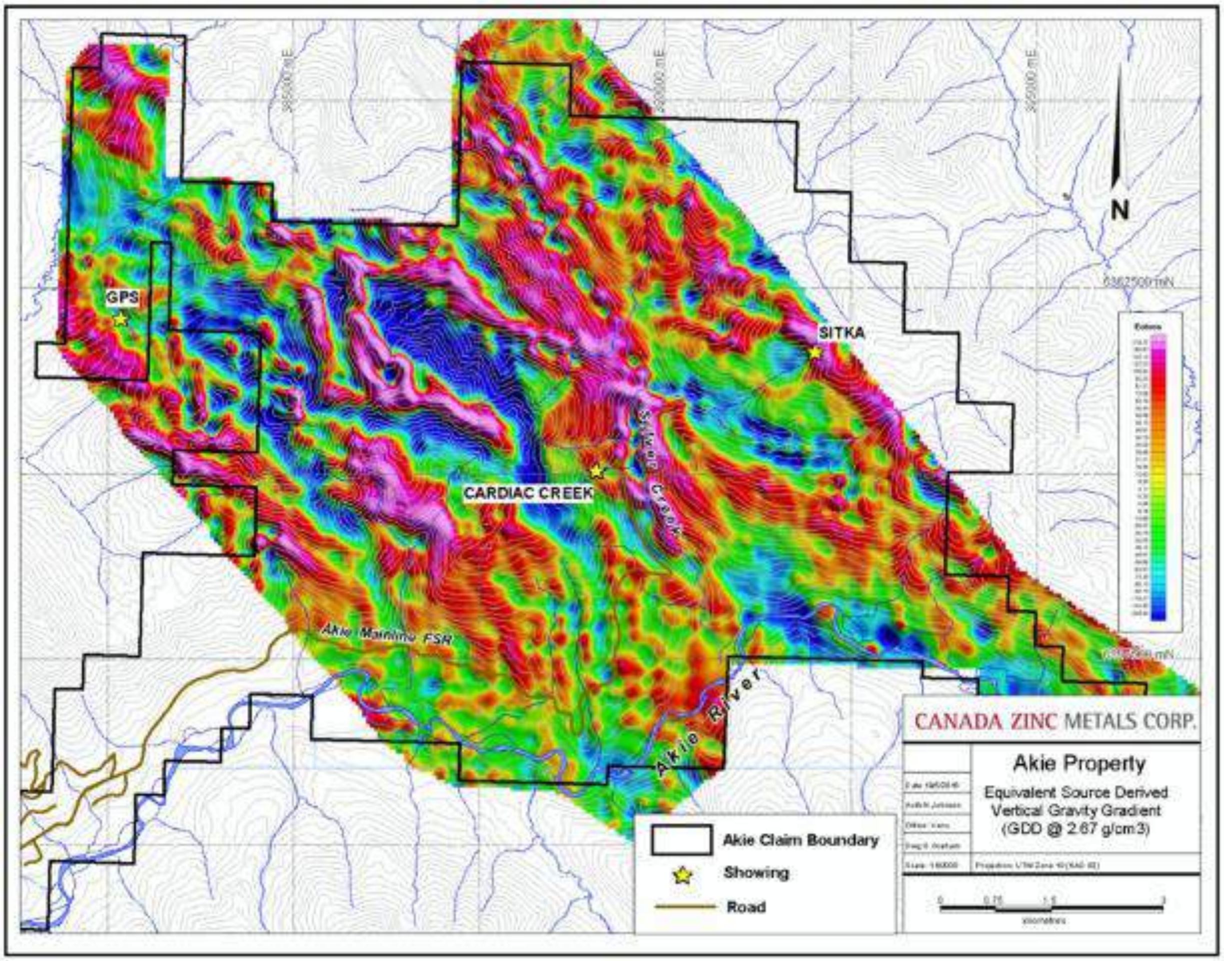
In late 2014, CZM contracted CGG to conduct an airborne gravity gradiometry survey over the Akie property with a line spacing of 200 m and flown at a nominal terrain clearance of 35 m. The results were received in early 2016. Despite a density contrast between the host Gunsteel Formation shales and the Cardiac Creek deposit, there did not appear to be a distinct response from the deposit itself. The equivalent source Vertical Gravity Gradient can be seen in Figure 9-4.

Figure 9-4: VTEM Airborne Geophysics Survey Across the Akie Property Displaying TauSF Response



Source: CZM (2016)

Figure 9-5: Airborne Gravity Gradiometry Survey Across the Akie Property Displaying the Equivalent Source Vertical Gravity Gradient Response



Source: CZM (2016)

9.5.1 Drilling

From 2005 to 2015, CZM completed nine separate drilling programs. During this time, 110 holes were drilled totaling 45,709 m; 83 drill holes were completed to their intended depths, 17 were abandoned due to ground conditions or excessive deviation of the drill stem, and 10 were drilled for geotechnical purposes. The details of these drilling programs are summarized in Section 10 of this report.

10 Drilling

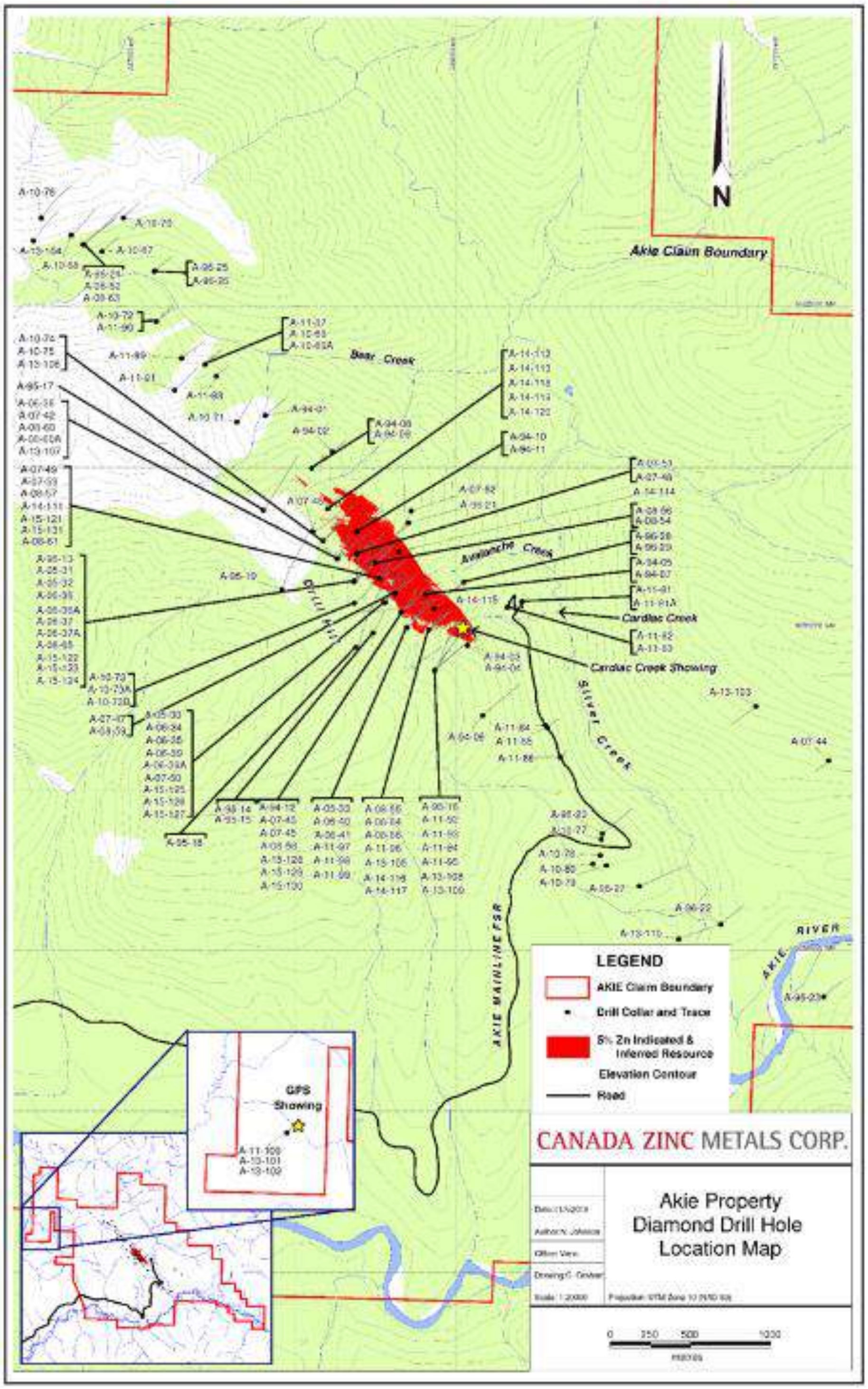
The following section summarizes the drilling activities completed on the Akie property by Inmet Mining (1994 to 1996), Canada Zinc Metals (2005 to 2017). The location of all drill holes on the Akie property can be seen in plan in Figure 10-1 and Figure 10-2. Figure 10-3 shows the locations of drilling segregated into the main campaigns. There is a total of 151 drill holes on the property with a total core length of 64,288 m.

10.1 Inmet Mining Drill Programs (1994 – 1996)

The following is a summary of the drilling activities carried out by Inmet Mining from 1994 to 1996. Assessment reports 23870, 24323, 24439 and 24703 (Baxter, 1995, 1996a, 1996b and 1996c) provide a detailed review of drilling and include drill logs, analytical results, interpretation and conclusions. These reports can be obtained in PDF format from BC's Ministry of Energy and Mines Assessment Report Indexing System (ARIS) website at: http://www.empr.gov.bc.ca/mining/geoscience/aris/pages/default.aspx.

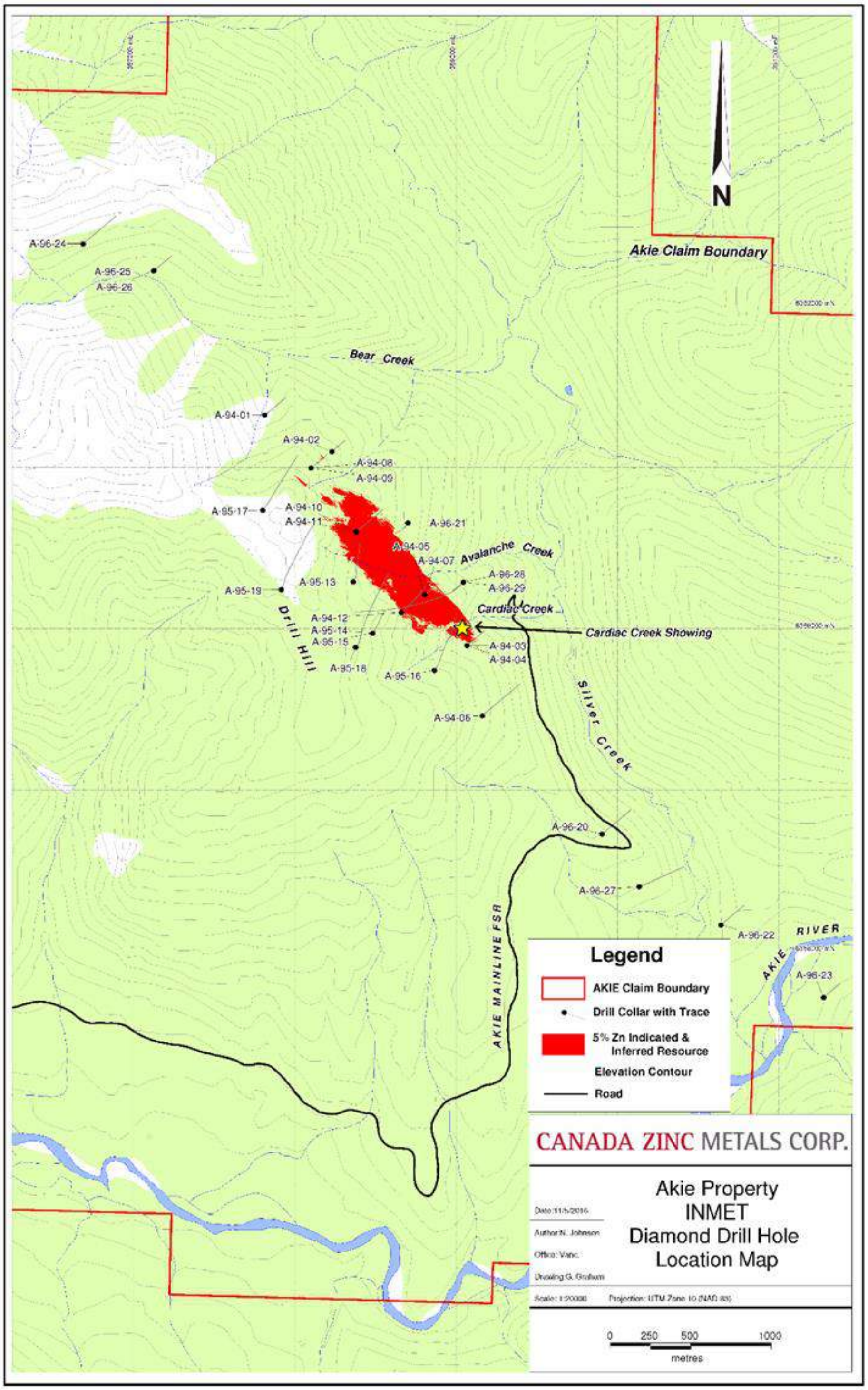
From 1994 to 1996, Inmet Mining conducted three helicopter-supported, diamond drilling programs completing 29 drill holes totaling 13,685.50 m. The details of these drill holes can be found in Table 10-1 and located on Figure 10-1.

Figure 10-1: Drill Hole Location Map for All Drilling on the Akie Property



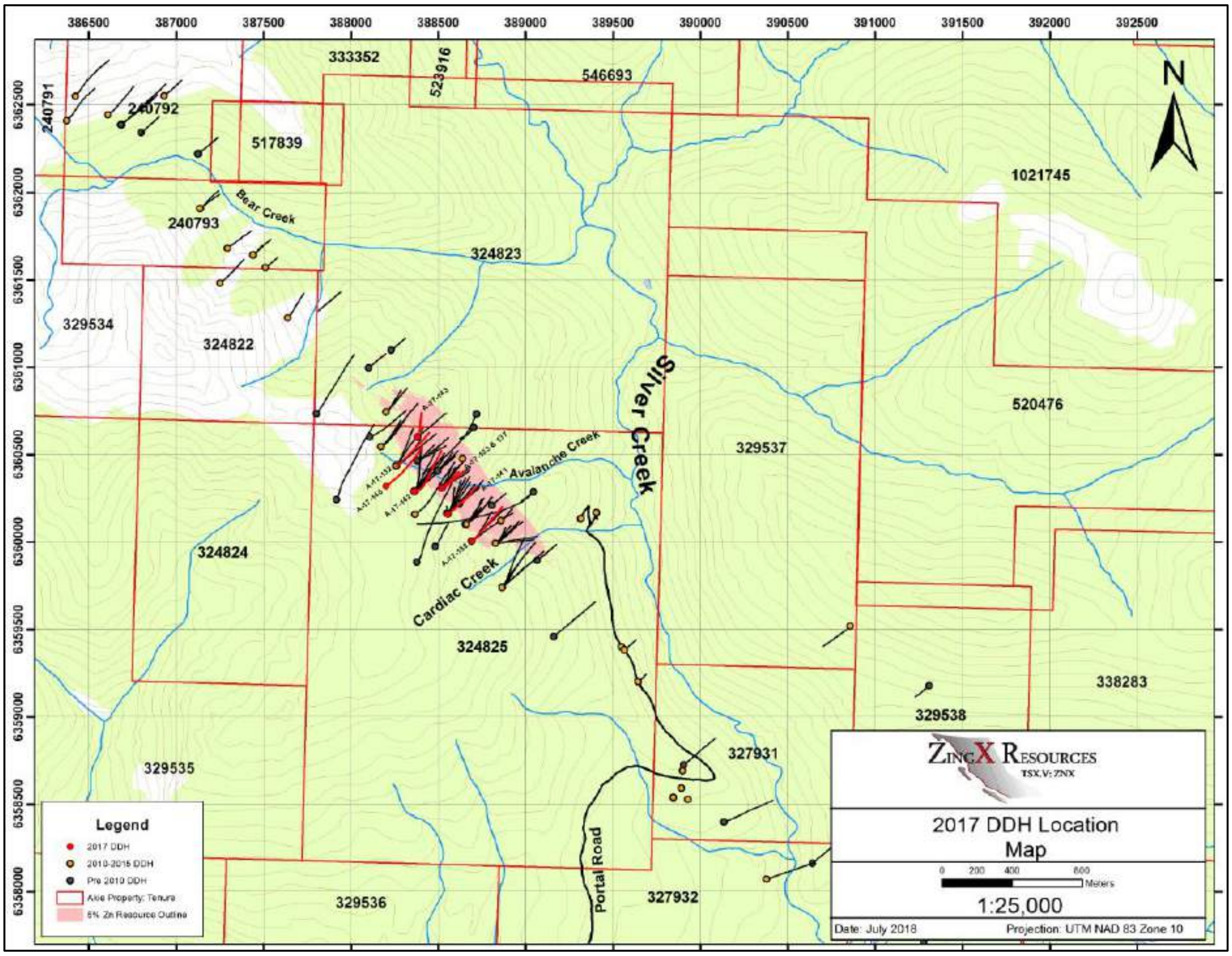
Source: CZM (2016)

Figure 10-2: Drill Hole Map for All Inmet Mining Drill Holes



Source: ZincX (2018)

Figure 10-3: Plan Map Showing Drill Locations by Campaigns



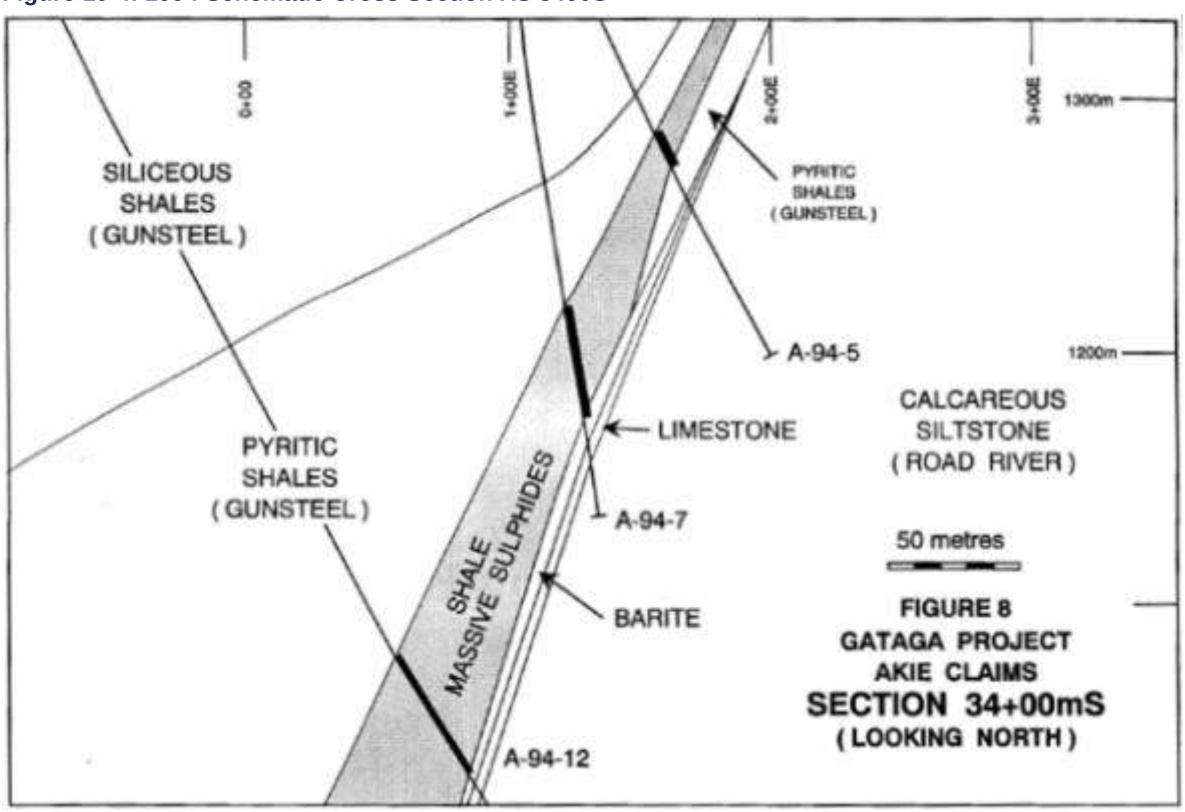
Source: ZincX (2018)

In 1994, Inmet Mining initiated a diamond drilling program to test new discoveries found through prospecting and soil sampling, including the Cardiac Creek showing and several soil anomalies. Twelve NQ-sized drill holes were completed totaling 3,753.20 m (A-94-01 to A-94-12). The drilling was conducted on approximate 400 m centres with two drill holes fanned from most setups. The drilling covered a strike length of 2.3 km across the prospective panel of Gunsteel Formation shale. These drill holes tested for mineralization primarily within 250 m of surface. Drill hole A-94-12 was an exception to this, testing for mineralization 400 m below surface.

The drilling defined a simple geological sequence with a 400 m to 500 m thick panel of Gunsteel carbonaceous siliceous shale overlying a thin layer of debris flow/limestone and a thick sequence of calcareous siltstone (Silurian Siltstone) of the Road River Group. All drill holes were terminated within footwall siltstone unless previously abandoned due to poor ground conditions or excessive drill hole deviation (Figure 10-4).

Mineralization was encountered towards the base of the Gunsteel Formation shale in most of the drill holes. This mineralization consisted of variably thick intervals of interbedded shale with pyrite, sphalerite and galena sulphides that were underlain by thin and discontinuous units of bedded barite (Baxter, 1995). Figure 10-4 is a schematic cross section through the host stratigraphy depicting the Cardiac Creek horizon towards the base of the Gunsteel Formation.

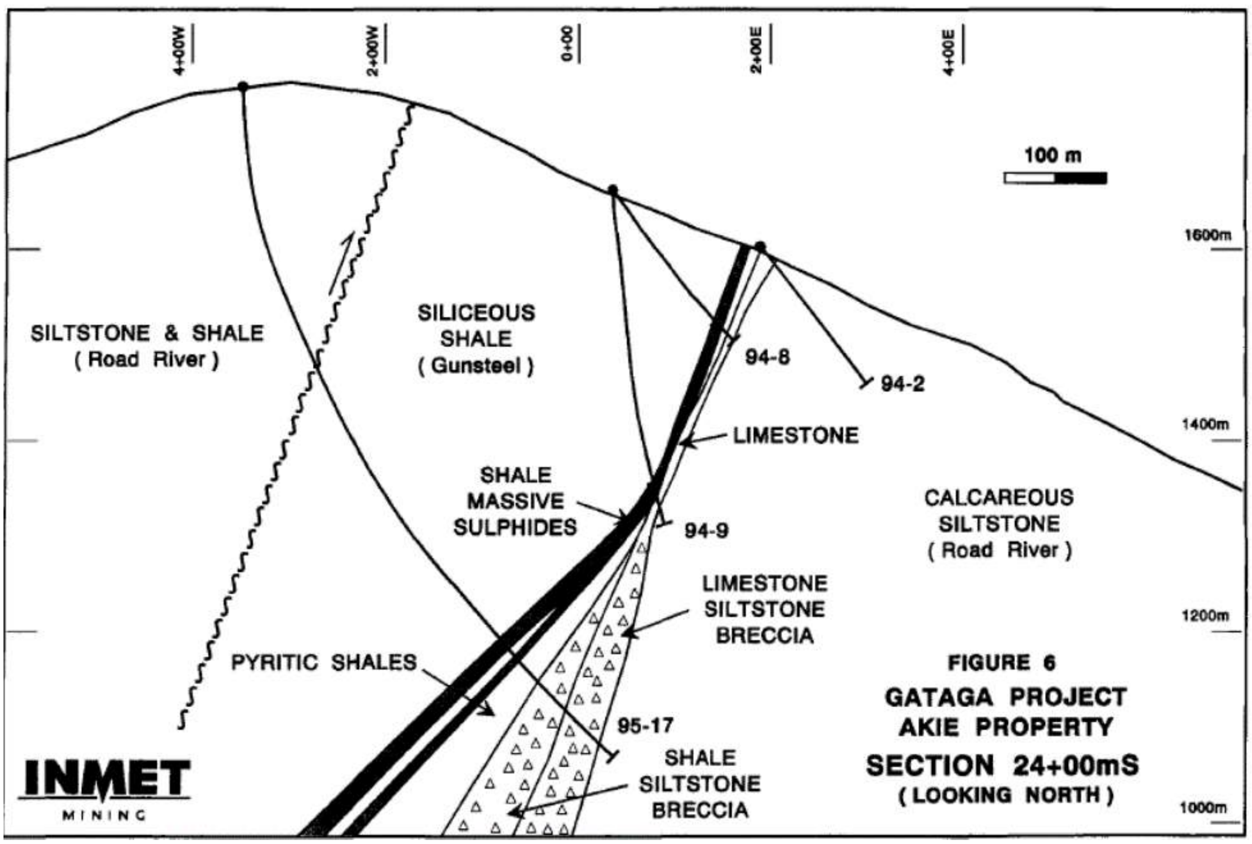
Figure 10-4: 1994 Schematic Cross Section XS 3400S



Source: Baxter (1995)

Based on the success of the 1994 drilling program, Inmet Mining conducted additional drilling campaigns in 1995 and 1996. The 1995 drilling program was primarily focused on testing the continuity of the Cardiac Creek horizon at depth (Baxter, 1996b). Seven BQ- to NQ-sized drill holes were completed totaling 5,314 m (A-95-13 to A-95-19). These drill holes were widely spaced covering a strike extent of 1.4 km and tested the mineralization at depths of approximately 500 m to 800 m below surface. Due to the location of the intended targets at depth, some drill holes were collared into the overlying Ordovician siltstone and/or graptolitic black shale present in the hanging wall thrust. These holes cut through the entire panel of Gunsteel Formation shale. However, at depth it appeared that the thin layer of debris flow present at the base of the Gunsteel Formation shale thickened (Figure 10-5). In addition to a single horizon of mineralization at the base of the Gunsteel Formation shale, drilling intercepted two apparently separate mineralized horizons. This was observed in drill holes A-95-13, A-95-16, A-95-17 and A-95-18 and can be seen in Figure 10-5.

Figure 10-5: 1995 Schematic Cross Section XS 2400



Source: Baxter (1996a)

Significant intervals of Cardiac Creek Zone-style mineralization were intercepted in all drill holes except for A-95-14 and A-95-15 which were abandoned due to excessive deviation and poor ground conditions. These mineralized intervals ranged in thickness from 2 m to greater than 34 m.

The 1994 and 1995 drilling programs tended to focus on testing the broad extents of the emerging Cardiac Creek deposit, whereas the 1996 program focused on testing primarily other property scale targets. Ten BQ- to NQ-sized drill holes (A-96-20 to A-96-29) were completed totaling 4,483.80 m and covered a strike extent of approximately 7 km (the entire length of the property). This strike extent also roughly represents the entire length of the main prospective panel of Gunsteel Formation shale on the Akie property. The 1996 drilling enhanced the general understanding of the lithology of the prospective stratigraphy.

Drill holes A-96-20, 22, 23 and 27 all tested zinc and lead soil anomalies to the southeast of the Cardiac Creek deposit (Baxter, 1996c). These holes were widely spaced and covered a strike extent of 1.7 km. Drill hole A-96-22, 1.5 km southeast of the Cardiac Creek deposit, intersected a 4.6 m thick interval of 30% to 75% laminar bedded massive pyrite which returned a 1.7 m interval grading 1.36% Zn. However, the other drill holes failed to intersect any significant mineralization (Baxter, 1996c). Drill holes A-96-24, 25 and 26 were drilled approximately 2.5 km to the northwest of the Cardiac Creek deposit and targeted a large lead soil anomaly (Baxter, 1996c) which is commonly referred to as the North Lead Anomaly. Drill hole A-96-24 intersected a 0.8 m thick interval of massive pyrite, galena and sphalerite mineralization directly overlying the debris flow present at the Gunsteel Formation shale / Road River Group contact (Figure 10-22). This 0.8 m interval graded 11.6% Zn and 9.05% Pb. Overlying the massive sulphide lens was 45 m of 5% to 12% laminar bedded pyrite interbedded with Gunsteel Formation shale (Baxter, 1996c). Follow-up drill hole A-96-24, A-96-25 and A-96-26 were drilled 400 m along strike to the southeast. Minor barite mineralization was intersected in A-96-25, but in general no significant mineralization was encountered in these two drill holes. In addition, the lithology present in holes A-96-24 to A-96-26 was dissimilar to that encountered on the Cardiac Creek deposit suggesting the presence of a large fault structure separating the two target areas (Baxter, 1996c). An example of the massive sulphides intersected in drill hole A-96-24 is shown in Figure 10-6. This was referred to as the Bear Valley Block which is now commonly referred to as the North Lead Zone or North Lead Anomaly.

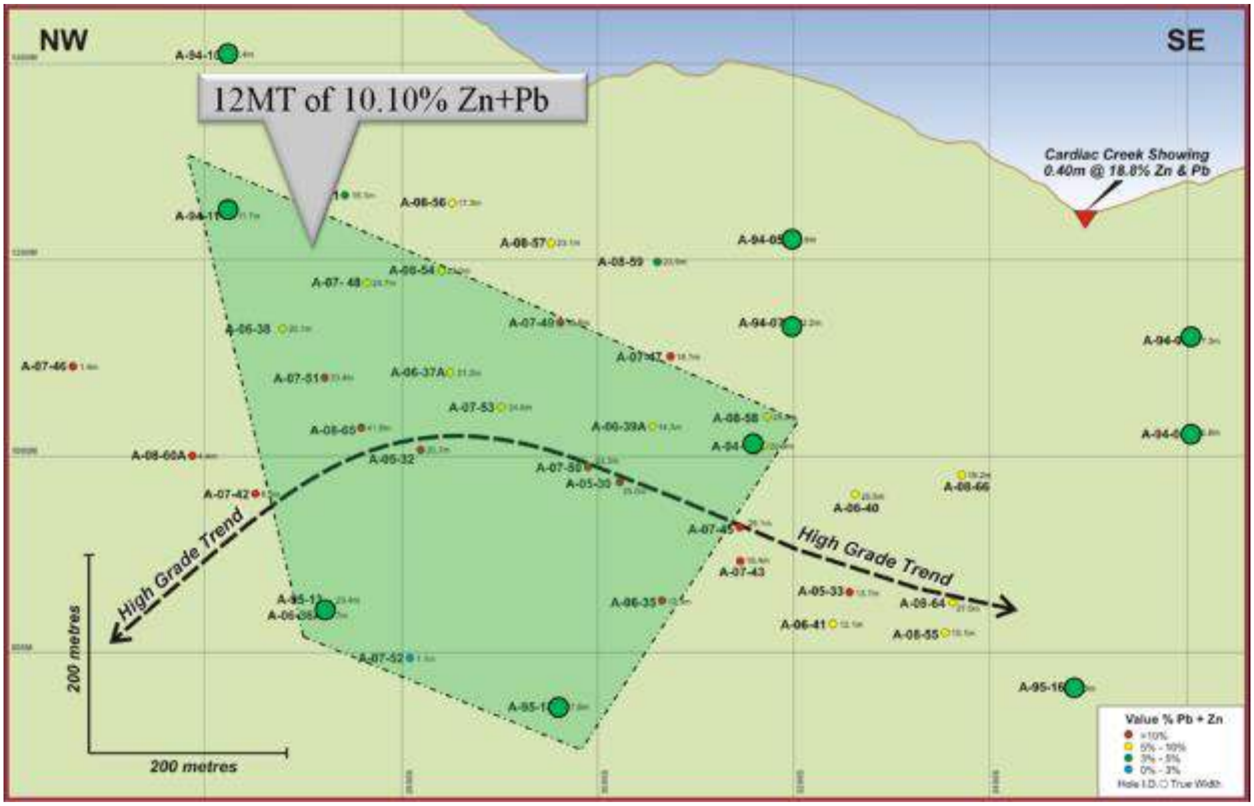
Figure 10-6: A 5 cm Piece of the 0.8 m Interval of Massive Sulphide in A-96-24

Source: CZM (2008)

Limited drilling tested the Cardiac Creek deposit in 1996. Drill holes A-95-19, A-96-21, A-96-28 and A-96-29 all attempted to obtain intersections of the Cardiac Creek Zone located approximately 1 km below surface. Drill hole A-95-19 successfully intersected a 12.6 m interval of the upper hanging wall zone comprised of 30% to 70% laminar bedded pyrite. However, the Cardiac Creek Zone was displaced, at an unknown distance, by a fault located at the Gunsteel Formation shale / Road River Group contact. Drill hole A-96-29 encountered a similar fault present at the Gunsteel Formation shale / Road River Group contact that also seemed to have offset the Cardiac Creek Zone by an unknown amount of displacement, but it was believed to be a minimum of 150 m (Baxter, 1996c). Drill holes A-96-21 and A-96-28 were both abandoned due to excessive deviation and poor ground conditions (Baxter, 1996c).

The drilling conducted on the Cardiac Creek Zone in 1994 and 1995 allowed Inmet Mining to produce a geological resource for the Cardiac Creek deposit of 12 Mt grading 8.6% Zn and 1.5% Pb (MacIntyre, 2008). This is a historical estimate of mineral resources that have not been verified by the authors of this report and should not be relied upon. The approximate outline of this historical resource can be seen in Figure 10-7.

Figure 10-7: Approximate Outline of the non-43-101 Compliant Preliminary Resource



Source: CZM (2009)

10.2 Canada Zinc Metals Drill Programs (2005 – 2015)

The following is a summary of the drilling activities carried out by Canada Zinc Metals from 2005 through 2015. This information is based on assessment reports filed by CZM as well as internal company reports which provide a detailed review of drilling and include drill logs, analytical results, interpretation and conclusions. Further information was obtained based on personal communications with CZM project geologist Nicholas Johnson. The assessment reports can be obtained in PDF format from BC's Ministry of Energy and Mines ARIS website at: http://www.empr.gov.bc.ca/mining/geoscience/aris/pages/default.aspx.

From 2005 to 2015, CZM conducted nine helicopter-supported diamond drilling programs completing 110 drill holes totaling 45,709 m. The details of these drill holes can be found in Table 10-1 and can be located in Figure 10-7.

In 2005, CZM initiated a late-season (October to December) drill program designed to test the core of the preliminary resource outlined by Inmet Mining (Vanwermeskerken and Metcalfe, 2006). Four HQ-sized drill holes were completed totaling 1,998.90 m (A-05-30 to A-05-33). The drilling was conducted on approximate 200 m centres from three setups across the centre of the deposit, with the intent of intercepting mineralization between the 900 m and 1,000 m elevation mark.

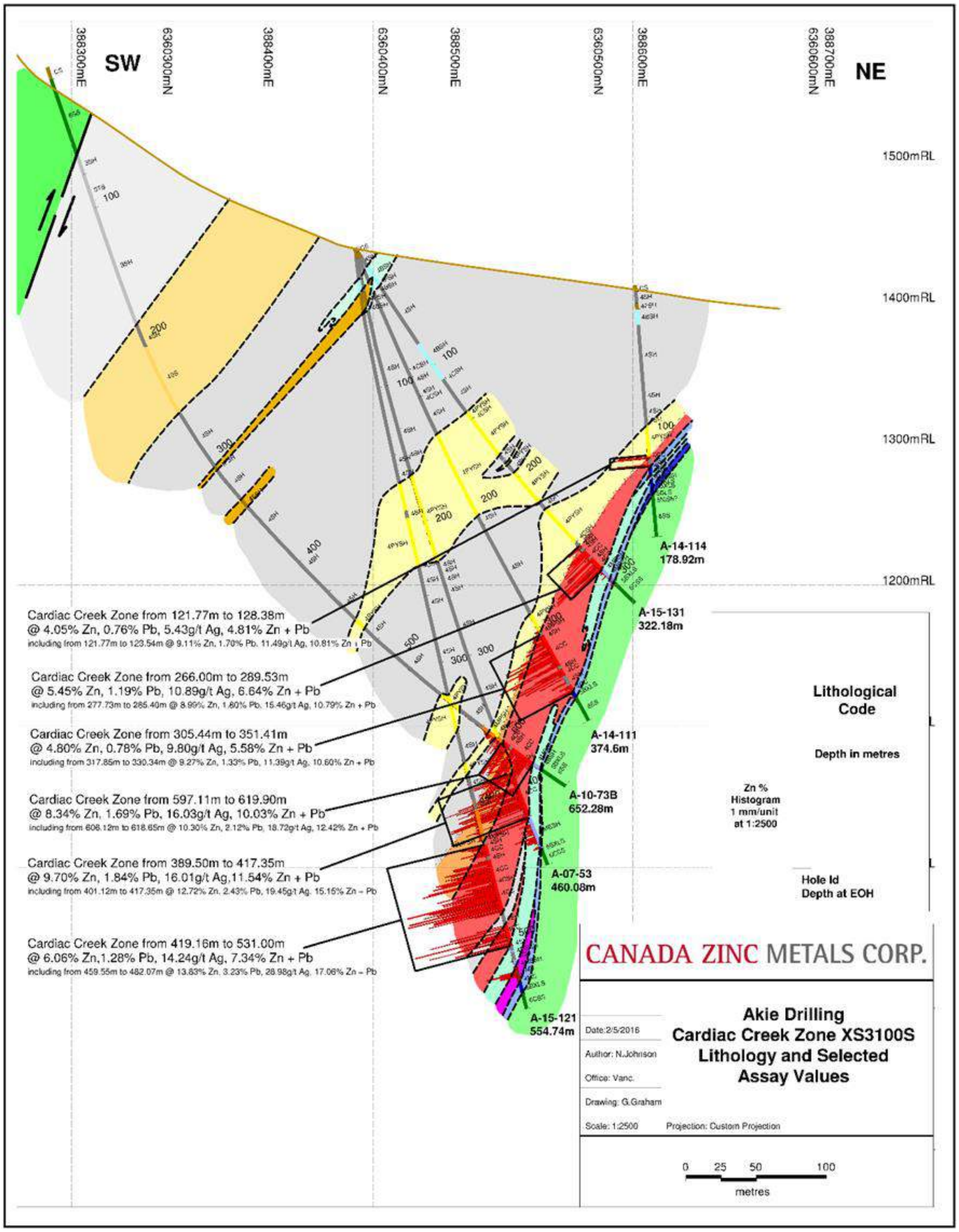
Due to the late start of the 2005 drill program and the onset of winter conditions, the logging of drill core was predominantly restricted to the mineralized sections referred to as the “Cardiac Creek Unit” (Vanwermeskerken and Metcalfe, 2006). In general, the drilling intersected the siliceous shale of the Gunsteel Formation with the Cardiac Creek Zone present towards the base of the formation. The shale was underlain by the previously known debris flows of the Paul River Formation and calcareous siltstone of the Road River Group. No new lithological units were intersected or recognized in the 2005 drilling.

Mineralization was intersected in three drill holes (A-05-30, A-05-32 and A-05-33). Drill hole A-05-31 was abandoned due to poor ground conditions. The mineralization intersected returned higher than expected grades of zinc and lead over significant widths in the three drill holes. Results from this program included: 11.87% Zn, 2.83% Pb and 23 g/t Ag over 34.05 m in A-05-30; 11.96% Zn, 2.73% Pb and 22 g/t Ag over 26.70 m in A-05-32; and 9.81% Zn, 2.20% Pb and 19 g/t over 11.50 m in A-05-33 (Vanwermeskerken and Metcalfe, 2006). In addition, the presence of massive medium-grained pyrite associated with the underlying debris flow in all three drill holes suggests the presence of a possible vent in close proximity to the deposit. The results of this program redefined the nature of the Cardiac Creek deposit and indicated the presence of a high-grade core (Vanwermeskerken and Metcalfe, 2006). Pierce points can be located in Figure 10-9.

Based on the results of the 2005 program, follow-up drilling in 2006, 2007 and 2008 began to define the high-grade core of the deposit as well as expand upon its known boundaries. Thirty-seven HQ- and NQ-sized drill holes were completed totaling 17,636.96 m (A-06-34 to A-08-66). The drilling was conducted on approximate 100 m centres with several drill holes being completed from individual setups. The drilling covered the entire strike extent of the deposit of approximately 1.2 km and straddled Cardiac and Avalanche creeks.

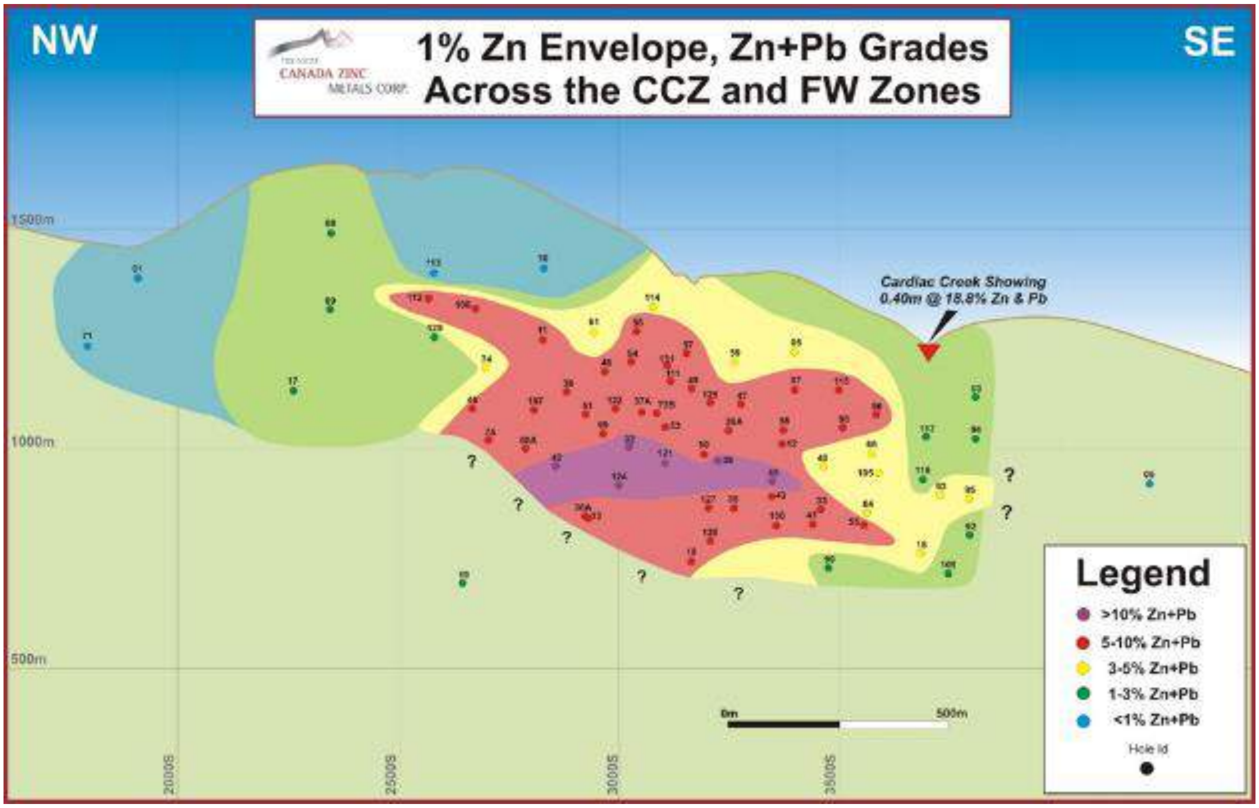
New lithological units were recognized in these drilling programs. The soft shale of the Akie Formation was present as a thin wedge situated directly below the hanging wall thrust and located stratigraphically above the siliceous shale of the Gunsteel Formation. Several sub-units were recognized within the Gunsteel Formation, including fragmental units, nodular barite units, the mineral facies, and a massive sulphide lens associated with the Cardiac Creek deposit. A schematic cross section depicting the geology can be seen in Figure 10-8.

Figure 10-8: 2015 Schematic Cross Section XS 3100S



Source: CZM (2016)

Figure 10-9: Schematic Long-Sections Depicting the Lithology and Mineral Facies on the Akie Property



Source: CZM (2016)

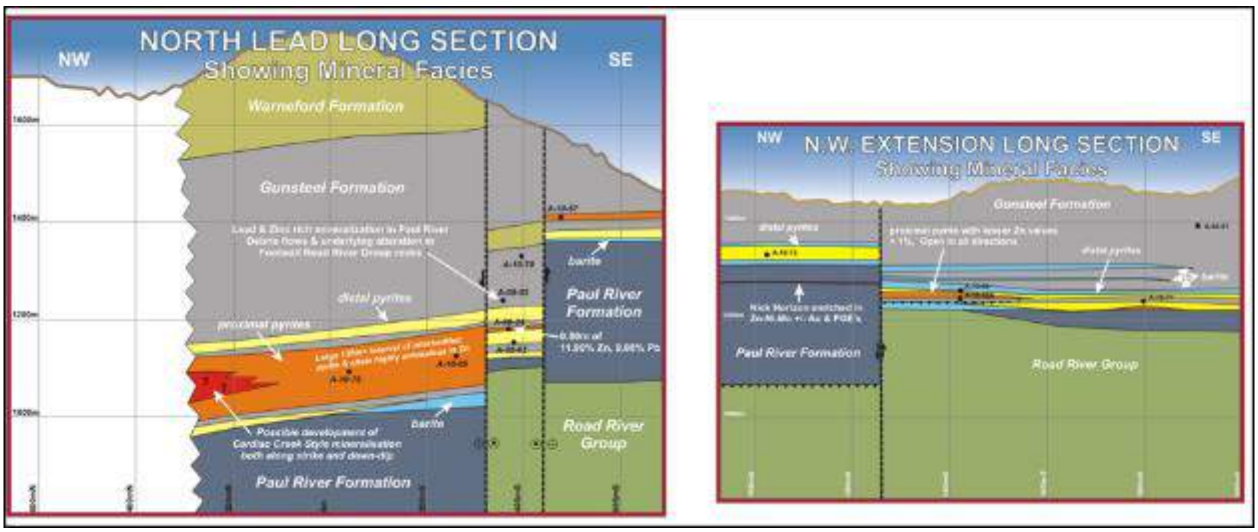
Of the 37 drill holes, 29 successfully intersected the Cardiac Creek Zone and five were abandoned due to poor ground conditions and/or excessive deviation. For due diligence purposes, drill holes A-06-36A and A-08-58 were twins of A-95-13 and A-94-12, respectively. Drilling results continued to demonstrate the high-grade nature of the deposit as well as its lateral continuity.

Three drill holes tested targets other than the Cardiac Creek deposit on the Akie property. Drill hole A-07-44 tested the edge of the South Zinc soil anomaly defined by Inmet Mining. The Gunsteel Formation shale was intersected; however, the source of the overlying zinc anomaly was not discovered, and no significant mineralization was encountered in A-07-44. Drill holes A-08-62 and A-08-63 tested the up- and down-dip extents of the massive sulphide mineralization intersected in A-96-24 (Figure 10-22) at the North Lead Anomaly. Although no massive sulphide mineralization was intersected in either drill hole, both holes did encounter thick intervals of laminated pyrite with nodular barite mineralization that returned highly anomalous zinc (0.1% to 0.6%) and lead (100 ppm to 900 ppm) values and were similar in character to the Proximal Facies mineralization of the Cardiac Creek deposit (Johnson, 2009). In addition, vent-proximal features were present in the siltstone of the Road River Group in drill hole A-08-63, including sulphide replacement of the Paul River Formation debris flows, silicification, sulphide stringers and breccias, all which provided encouraging results.

The 2010 and 2011 drilling programs focused on multiple targets across the Akie property covering an approximate strike length of 6 km, including the Cardiac Creek deposit, the North Lead Anomaly, the NW Extension and the SE Extension. Thirty-eight drill holes were collared totaling 12,856.36 m (A-10-67 to A-11-100) of which several were for geotechnical purposes and several deviated off target and were abandoned. Nine holes successfully intersected the Cardiac Creek deposit in the area of the resource block model.

In addition to the debris flows and limestone, new lithological units associated with the Paul River Formation were identified in the 2010 drilling program. This included siliceous shale interbedded with regular thinly bedded siltstone to conglomerate lenses and siliceous shale containing disrupted chert lenses and layers, and fine sub-millimetre laminations of pyrite. Following a reinterpretation of the drilling data, it was noted that the Paul River Formation generally thickens at depth and to the northwest towards the NW Extension and North Lead Anomaly targets (Figure 10-10). Brassy yellow pyrite and nodular to laminar barite mineralization typically mark the boundary between the Paul River Formation and the Gunsteel Formation.

Figure 10-10: Schematic Long-Sections Depicting the Lithology and Mineral Facies on the Akie Property



Source: CZM (2011)

Drilling at the North Lead Anomaly was conducted in 2010 with four drill holes being completed totaling 2,584.79 m. The drilling tested the up- and down-dip as well as the strike extent of the mineralization and alteration intersected in the previous drilling. While no additional alteration or vent-proximal features were intersected, thick, 125 m intervals of laminar to bedded pyrite mineralization interbedded with Gunsteel Formation shale were encountered down-dip and along strike to the northwest in drill holes A-10-68 and A-10-76 (Figure 10-9). This mineralization was highly anomalous with zinc values ranging from less than 1,000 ppm to greater than 2%.

The NW Extension target is situated between the Cardiac Creek deposit and the North Lead Anomaly. In 2010 and 2011, nine drill holes were completed on this target totaling 3,255.72 m. The 2010 program tested for Cardiac Creek-style mineralization at the 1,000 m elevation where the bulk of the high-grade mineralization occurs at the deposit. Three widely spaced drill holes were completed on approximate 400 m centres. Proximal Facies mineralization was intersected in A-10-69 over 14.87 m and returned highly anomalous zinc values consistently in excess of 2,000 ppm and reaching 1.90% (Johnson, 2011). In addition to the targeted Cardiac Creek horizon mineralization, a thin lens of sulphide mineralization was intersected in A-10-72 within the underlying Paul River Formation (Figure 10-11). This 1.17 m interval returned 2.69% Zn, 0.60% Ni and 4.36 g/t Ag and contained highly anomalous values across a diverse suite of elements, including molybdenum, copper, lead, cobalt, arsenic, uranium, cadmium, antimony, bismuth, vanadium, phosphorus, mercury, thallium, selenium, rhenium, gold and palladium. This mineralization is of similar character to the Nick Deposit in the Yukon.

Figure 10-11: "Nick" Style Mineralization Intersected in A-10-72 @ 299.57 m

Source: CZM (2011)

Follow-up drilling in 2011 tested the discoveries made in the previous year. Intervals of Proximal Facies mineralization were intersected along strike and up-dip of A-10-69; however, the mineralized horizon appeared to be offset at depth due to brittle faulting. These intervals of Proximal Facies mineralization were highly anomalous in zinc with values consistently greater than of 1,000 ppm. A thin 1.60 m massive sulphide lens underlying the Proximal Facies mineralization was intersected in A-11-88, and returned 6.99% Zn, 0.25% Pb and 2.35 g/t Ag. The NW Extension mineralization remains open along strike.

The 2011 drilling tested the up-dip potential for additional "Nick"-style mineralization; however, no additional intervals were encountered.

In addition to the exploration drilling, 10 geotechnical drill holes were completed in 2010 and 2011 totaling 516.54 m. The drilling was designed to provide engineering and hydrogeological data to support design and permitting for future underground exploration. While the drilling was intended for geotechnical purposes, near surface mineralization was intersected in drill holes A-11-84 and A-11-85 consisting of laminar to bedded pyrite and nodular barite. The mineralization appeared to represent the southeast strike extension of the Cardiac Creek horizon. In fact, the first exploration drill hole of the 2011 drilling program tested the down-dip extent of this mineralization but returned nominal results.

Drilling in 2013 focused on a number of different targets: the GPS bedded barite showing, follow-up drilling at the North Lead Anomaly, South Zinc Anomaly, the Cardiac Creek deposit and the SE Extension. A total of 4,851.41 m was completed in 10 drill holes with 1 being abandoned.

Exploratory drilling was conducted on the GPS barite bedded showing that was discovered in 2009. In 2011, an initial attempt to test the GPS showing resulted in the abandonment of hole A-11-100 due to poor ground conditions; however, the first two drill holes of the 2013 program totaling 662.64 m were successful in testing the showing. A thick sequence of Earn Group lithology was encountered consisting primarily of Akie and Paul River Formation soft shales, cherty shales and debris flows with minor intervals of Gunsteel

Formation shales. Poorly developed mineralization consisting of laminar beds of pyrite and minor nodular barite was intersected at contact between the Akie and Paul River Formation rocks which was interpreted to represent the down-dip extent of the showing. Sampling returned no significant results.

Drill hole A-13-103 followed up on the 2007 drill hole A-07-44 that attempted to determine the source of the large South Zinc Anomaly located on the eastern side of Silver Creek. Unfortunately, there was no mineralization associated with the Cardiac Creek horizon present in drill hole A-13-103 and nothing to explain the origin of the South Zinc Anomaly. A narrow interval of “Nick”-style mineralization was noted at the unconformable contact between the Kwadacha Limestone of the Paul River Formation and the underlying Silurian Siltstone of the Road River Group. This narrow interval from 252.37 m to 252.87 m is highly anomalous in a suite of elements, including Pb, Zn, Ni, U, V, P, and numerous other elements (Figure 10-12).

Figure 10-12: “Nick” Style Mineralization in Hole A-13-103 @ 252.37m

Source: CZM (2014)

A single drill hole tested the North Lead Anomaly, hole A-13-104. The down-dip extension of the mineralization observed in both holes A-10-68 and A-10-76, which represents the North Lead Zone, was present in hole A-13-104 over an extremely thick interval from 474.83 m to 646.00 m. Distal and Proximal Facies mineralization was interbedded with Gunsteel Formation shale. This appeared to be very similar in character to the mineralization encountered in the two up-dip drill holes (A-08-68 and A-10-76). The stratigraphy and mineralization were found to be recumbent to gently dipping to the southwest rather than steeply dipping as previously thought. Sampling indicated anomalous zinc grades in excess of 1,000 ppm and elevated lead, silver and thallium throughout the entire mineralized sequence. The mineralization present at the North Lead Anomaly remains open along strike to the northwest and down-dip.

The drilling on the Cardiac Creek deposit in 2013 was for infill and expansion purposes. Drill holes A-13-105 and A-13-107 were infill, and holes A-13-106 and A-13-109 both tested and expanded the known limits of the deposit. The drilling did not close off any area of the deposit; however, the grade encountered in hole A-13-109 from the Cardiac Creek horizon was weaker than expected. A total of 2,499.06 m was completed in four drill holes and one abandoned hole. The drill hole locations can be found in plan in Figure 10-7 with the collar details are included in Table 10-1. The pierce points can be seen in Figure 10-8.

Drill hole A-13-105 intersected a thick interval of mineralization grading greater than 1% Zn over 54.33 m from 357.00 m to 411.33 m that contained several higher-grade intervals. This intercept is comparable in grade and width to the surrounding intercepts in holes A-08-64 and A-08-66. Drill hole A-13-106 encountered a narrow, 12.06 m thick intersection of high-grade mineralization from 476.00 m to 488.06 m. Faulting is present along the upper and lower contacts of the mineralization suggesting that it has been displaced from depth to its current position. In addition to the Cardiac Creek Zone mineralization drill hole A-13-106 also intersected a narrow interval of "Nick"-style mineralization along the unconformable contact between the Kwadacha Limestone and the Silurian Siltstone, similar to the intercept from A-13-103 (Figure 10-13). The sample was found to be anomalous in lead, zinc, nickel, uranium, phosphorus, arsenic and other elements.

Drill hole A-13-107 intersected a broad interval of mineralization grading greater than 1% Zn over 26.61 m from 541.53 m to 568.14 m that contained a couple of higher grade intervals. The grade is comparable to the surrounding drill holes, such as A-08-60A. The last hole, A-13-109, looked to expand the deposit along the southeastern edge. A thick 40.02 m interval from 615.56 m to 655.58 m was intersected but the grades were lower than expected. Narrow intervals of low- to moderate-grade material are present within the overall envelope. A low-grade Footwall Zone and a small massive sulphide lens were also intersected from 667.66 m to 677.45 m and from 684.12 m to 685.30 m, respectively.

Figure 10-13: "Nick"-Style Mineralization in A-13-106 @ 501.13 m

Source: CZM (2013)

The final drill hole of the 2013 exploration program, A-13-110, targeted the down-dip extension of zinc-rich mineralization present in drill hole A-96-22. No mineralization was encountered along the prospective horizon. However, it was discovered that the Earn Group stratigraphy intersected in the upper portion of the drill hole was in thrust contact with a previously unknown panel of Gunsteel Formation shale present from 450.24 m to 539.39 m. Unfortunately, this new panel was barren with only very weak nodular barite and laminated pyrite present along the Gunsteel Formation and Silurian Siltstone contact.

The drilling in 2014 and 2015 concentrated on the Cardiac Creek deposit. Several different areas of the deposit were targeted, including: the northwest and southeast strike extents, the up-dip extent, down-dip of the high-grade core, and infill targets. A total of 8,365.39 m was completed in 21 drill holes and five of these were abandoned due to drill hole deviation or poor ground conditions. Drill hole locations can be found in Figure 10-6 and the collar details are shown in Table 10-1. The pierce points can be seen in Figure 10-8.

Three drill holes (A-14-112, A-14-113, and A-14-120) focused on the northwest edge of the deposit. Hole A-14-112 obtained a pierce point located approximately 130 m along strike from A-13-106 intersecting an envelope of mineralization grading 5.27% Zn+Pb and 6.87 g/t Ag over 13.70 m (true width) from 337.15 m to 356.30 m (Figure 10-14). Higher grade intervals, such as 6.59% Zn+Pb and 7.86 g/t Ag over 9.52 m (true width) from 343.00 m to 356.30 m and 7.17% Zn+Pb and 8.23 g/t Ag over 5.23 m (true width) from 349.00 m to 356.30 m were encountered. This result was followed up with holes A-14-113 and A-14-120. Stockwork veining and faulting limited the Cardiac Creek horizon in grade and thickness in hole A-14-113 and hole A-14-120 intersected 12.98 m (true width) of mineralization returning 1.59% Zn+Pb and 3.22 g/t Ag from 409.00 m to 432.82 m. The highest-grade material was present at the base of the mineralized interval with 1.27 m (true width) of 4.59% Zn+Pb and 6.90 g/t Ag from 423.87 m to 426.20 m.

Figure 10-14: High-Grade Mineralization from the Cardiac Creek Zone in Hole A-14-112

Source: CZM (2014)

A single drill hole (A-14-114) tested the up-dip extents of the deposit in the vicinity of A-08-56 and A-08-57. Hole A-14-114 intersected two narrow high-grade intervals separated by a thick shale interbed. The overall envelope of mineralization returned a grade of 4.81% Zn+Pb and 5.43 g/t Ag over 2.64 m (true width) from 121.77 m to 128.38 m. The upper interval returned 10.81% Zn+Pb and 11.49 g/t Ag over 0.76 m (true width) from 121.77 m to 123.54 m, and the lower interval returned 10.37% Zn+Pb and 11.90 g/t Ag over 0.46 m (true width) from 127.30 m to 128.38 m. Additional “Nick”-style mineralization was also encountered in A-14-114.

The objective for the 2015 drill program was to test the down-dip extents of the deposit and high-grade core. Drill holes A-15-121, A-15-124, A-15-126, A-15-127 and A-15-130 all tested this area which had seen limited drilling in the past. This drilling was very successful in achieving the intended targets.

Drill hole A-15-121 provided a pierce point located in the central core of the deposit down-dip of A-08-53 and along strike of holes A-05-30 and A-05-32. The results from this hole were very comparable to the surrounding holes with an extremely thick intersection of high-grade lead and zinc mineralization representing the Cardiac Creek Zone returning 36.89 m (true width) of 9.85% Zn+Pb and 16.38 g/t Ag, which includes a very high-grade intersection of 12.98 m (true width) of 17.06% Zn+Pb and 28.98 g/t Ag. The drill hole also contained a very high-grade Footwall Zone returning 8.86 m (true width) of 10.24% Zn+Pb and 21.51 g/t Ag. Hole A-15-121 also included a 12.46 m interval of massive sulphide dominated by pyrite with minor carbonate-sphalerite-galena mineralization. An example of the mineralization present in hole A-15-121 can be seen in Figure 10-15. Similar to hole A-15-121, drill hole A-15-124 obtained a thick intersection of high-grade mineralization returning 38.43 m (true width) grading 7.72% Zn+Pb and 12.30 g/t Ag, including 11.09 m (true width) of 17.20% Zn+Pb and 26.43 g/t Ag. Drilling down-dip of holes A-07-50 and along strike of A-06-35 produced similar results to hole A-06-35. The main interval of Cardiac Creek Zone mineralization is present from 601.13 m to 656.41 m. Drill hole A-15-127 achieved a pierce point located down-dip of A-07-50 and along strike of A-06-35. The mineralization intersected was comparable to A-06-35 returning 10.86 m (true width) of 8.53% Zn+Pb and 14.45 g/t Ag and a Footwall Zone of 6.07 m (true width) of 13.17% Zn+Pb and 21.32 g/t Ag.

Drill holes A-15-126 and A-15-130 provided additional information concerning the down-dip extent of the deposit. Drill hole A-15-126 provided a pierce point in the immediate vicinity of the historical Inmet Mining drill hole A-95-18; however, the mineralization is hosted within three distinct intervals representing a Hanging Wall Zone, the Cardiac Creek Zone and the Footwall Zone separated by thick intervals of black siliceous shale. The Cardiac Creek Zone returned 5.45% Zn+Pb and 9.79 g/t Ag over 11.72 m (true width). The final hole testing the down-dip extents of the deposit was A-15-130 which provided a pierce point directly down-dip of A-07-43. This hole was also comparable to A-06-35, returning a 12.15 m (true width) intersection grading 8.35% Zn+Pb and 12.84 g/t Ag.

Figure 10-15: Mottled Textured High-Grade Mineralization in A-15-121 @ ~480.50 m

Source: CZM (2015)

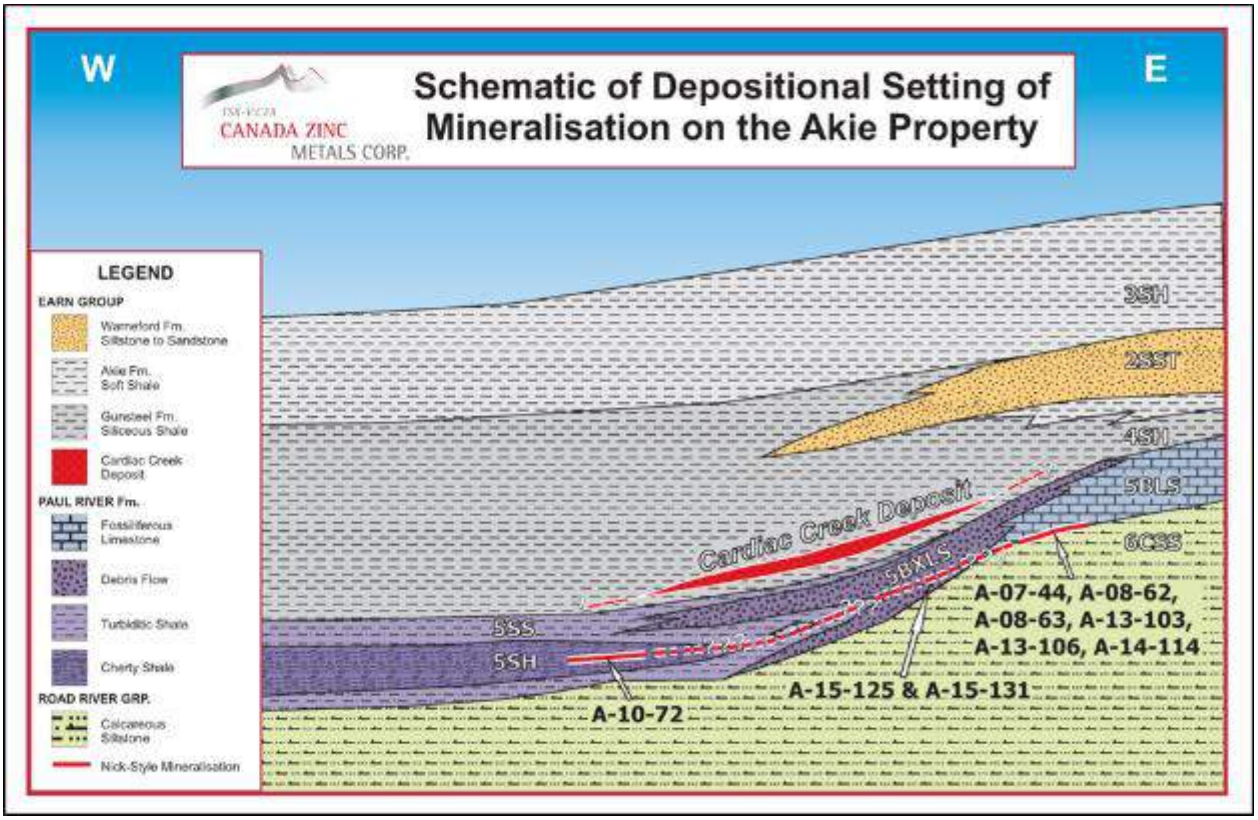
Several holes in the 2014 and 2015 drilling programs focused on filling in large gaps or holes across the deposit where the density of drilling was low. This included A-14-115, A-14-116, A-14-117, A-15-122, A-15-125, and A-15-131. Results of these holes were similar to those around them. Examples include: A-14-115 that returned 20.87 m (true width) of 6.01% Zn+Pb and 7.31 g/t Ag, including 6.66 m (true width) of 9.52% Zn+Pb and 11.71 g/t Ag; A-15-122 which returned 23.36 m (true width) of 10.31% Zn+Pb and 14.64 g/t Ag, including 12.35 m (true width) grading 13.62% Zn+Pb and 17.92 g/t Ag; and, A-15-125 which returned 20.83 m (true width) of 9.38% Zn+Pb and 12.99 g/t Ag, including 8.68 m (true width) of 15.45% Zn+Pb and 21.76 g/t Ag.

The 2015 drilling also encountered “Nick”-style mineralization in holes A-15-125 and A-15-131. Both intersections are thin and occur within the debris flows of the Paul River Formation. Subsequently, all the analytical results from 2006 to 2014 were reviewed to determine the presence of previously unrecognized intervals of “Nick”-style mineralization. Table 10-2 presents all intersections of recognized “Nick”-style mineralization encountered to date on the Akie property, their stratigraphic position and elemental enrichment. Based on the intercepts, there appears to be a core group of elements that are enriched within this style of mineralization, including:

Pb, Zn, Ni, U, V, P, La, Cr, Se

A diverse group of secondary elemental enrichment is variable due to dilution of the surrounding material within a given sample. The similarities in the suite of elements from all the occurrences suggest a genetic link despite the variation in the stratigraphic locations. The variation in stratigraphic settings for each occurrence is presented in Figure 10-10.

Figure 10-16: Depositional Setting of the Cardiac Creek and "Nick"-Style Mineralization on the Akie Property



Source: CZM (2016)

Table 10-1: Drill Hole Collar Information

Hole ID	UTM N (m)	UTM E (m)	Elev. (m)	Azimuth (°)	Dip (°)	Length (m)	Type
A-94-01	6361324*	387814*	1475*	050	-55	262.40	Exploration
A-94-02	6361098	388230	1545	050	-54	178.90	Exploration
A-94-03	6359895	389067	1252	050	-54	233.50	Exploration
A-94-04	6369895	389067	1252	050	-73	296.00	Exploration
A-94-05	6360212	388806	1345	050	-65	230.70	Exploration
A-94-06	6359460	389162	1298	050	-57	540.70	Exploration
A-94-07	6360211	388806	1344	050	-87	272.80	Exploration
A-94-08	6360997	388101	1624	050	-55	203.00	Exploration
A-94-09	6360997	388101	1624	050	-85	350.80	Exploration
A-94-10	6360601	388381	1572	050	-49	294.70	Exploration
A-94-11	6360600	388381	1570	050	-78	370.90	Exploration
A-94-12	6360101	388660	1429	050	-71	518.80	Exploration
A-95-13	6360290	388263	1526	050	-82	818.40	Exploration
A-95-14*	6359973	388482	1528	055	-79	124.10	Abandoned
A-95-15*	6359973	388482	1528	055	-84	578.20	Abandoned
A-95-16	6359741	388866	1355	050	-83	741.30	Exploration
A-95-17	6360735	387802	1726	055	-87	829.10	Exploration
A-95-18	6359884	388376	1559	055	-87	1030.50	Exploration
A-95-19	6360243	387917	1655	035	-88	1192.40	Exploration
A-96-20	6358726	389904	1074	050	-60	438.30	Exploration
A-96-21	6360657	388702	1424	203	-84	601.10	Abandoned
A-96-22*	6358163	390641	943	050	-50	282.90	Exploration
A-96-23*	6357713	391278	890	050	-50	206.70	Exploration
A-96-24	6362387	386687	1587	050	-60	541.90	Exploration
A-96-25	6362221	387128	1456	050	-45	214.60	Exploration
A-96-26*	6362219	387124	1480	050	-87	129.50	Exploration
A-96-27	6358400	390135	994	070	-62	593.80	Exploration
A-96-28	6360288	389046	1229	230	-70	211.80	Exploration
A-96-29	6360288	389046	1230	230	-75	1262.20	Exploration
A-05-30	6360161	388557	1484	050	-78	599.00	Exploration
A-05-31*	6360296	388366	1543	060	-70	132.50	Abandoned
A-05-32	6360292	388366	1526	055	-68	638.40	Exploration
A-05-33	6360006	388693	1398	060	-77.5	629.00	Exploration
A-06-34*	6360165	388550	1497	050	-86	330.50	Abandoned
A-06-35*	6360165	388550	1497	050	-74	696.00	Exploration
A-06-36	6360291	388364	1527	055	-80	75.29	Abandoned
A-06-36A	6360291	388364	1527	055	-80	791.67	Exploration

Hole ID	UTM N (m)	UTM E (m)	Elev. (m)	Azimuth (°)	Dip (°)	Length (m)	Type
A-06-37*	6360296	388366	1543	055	-65	24.99	Abandoned
A-06-37A	6360292	388365	1526	055	-65	593.45	Exploration
A-06-38	6360435	388260	1603	055	-70	599.55	Exploration
A-06-39*	6360165	388550	1497	055	-72	15.40	Abandoned
A-06-39A*	6360165	388550	1497	055	-71.5	542.24	Exploration
A-06-40	6360005	388691	1398	055	-73	535.54	Exploration
A-06-41	6360005	388691	1398	055	-83	675.74	Exploration
A-07-42	6360435	388260	1603	060	-80	712.02	Exploration
A-07-43	6360100	388659	1430	055	-81	629.72	Exploration
A-07-44	6359179	391309	1269	230	-65	221.04	Exploration
A-07-45	6360101	388659	1429	040	-78	584.00	Exploration
A-07-46	6360603	388109	1720	069	-74	730.61	Exploration
A-07-47	6360219	388623	1443	055	-72	401.12	Exploration
A-07-48	6360464	388380	1544	063	-68.5	446.84	Exploration
A-07-49	6360311	388522	1439	060	-64	387.71	Exploration
A-07-50	6360160	388554	1485	025	-78	587.22	Exploration
A-07-51	6360464	388380	1544	063	-80	513.90	Exploration
A-07-52*	6360732	388719	1424	205	-63	852.00	Exploration
A-07-53	6360311	388522	1438	060	-79	460.08	Exploration
A-08-54	6360407	388496	1469	050	-76	338.28	Exploration
A-08-55	6359993	388827	1322	050	-83	564.75	Exploration
A-08-56	6360407	388497	1467	050	-60	277.62	Exploration
A-08-57	6360313	388521	1438	060	-58	319.00	Exploration
A-08-58	6360102	388657	1431	052	-72	479.00	Exploration
A-08-59	6360220	388622	1443	055	-65	329.00	Exploration
A-08-60	6360435	388258	1604	050	-79	146.00	Abandoned
A-08-60A	6360435	388258	1605	050	-83	688.00	Exploration
A-08-61	6360465	388380	1544	065	-62	377.00	Exploration
A-08-62	6362386	386683	1589	050	-70	566.00	Exploration
A-08-63	6362386	386683	1589	050	-58	548.00	Exploration
A-08-64	6359994	388828	1323	050	-80	551.00	Exploration
A-08-65	6360295	388368	1527	042	-77	633.00	Exploration
A-08-66	6359994	388828	1322	050	-72	413.00	Exploration
A-10-67*	6362343	386800	1543	050	-78	553.83	Exploration
A-10-68*	6362445	386610	1652	050	-78	808.29	Exploration
A-10-69*	6361641	387441	1475	050	-76	236.00	Exploration
A-10-69A*	6361641	387441	1475	050	-82	335.00	Exploration
A-10-70*	6362552	386932	1656	050	-74	400.00	Exploration

Hole ID	UTM N (m)	UTM E (m)	Elev. (m)	Azimuth (°)	Dip (°)	Length (m)	Type
A-10-71*	6361283	387637	1542	050	-76	443.00	Exploration
A-10-72*	6361909	387138	1510	050	-72	533.00	Exploration
A-10-73*	6360159	388365	1566	055	-74	71.00	Abandoned
A-10-73A*	6360159	388365	1566	055	-78	32.95	Abandoned
A-10-73B*	6360159	388365	1566	055	-72	652.28	Exploration
A-10-74*	6360545	388172	1700	060	-76	645.27	Exploration
A-10-75*	6360545	388172	1700	060	-82	778.15	Exploration
A-10-76*	6362550	386423	1729	050	-82	822.67	Exploration
A-10-77*	6358691	389899	1068	050	-90	6.71	Technical
A-10-78*	6358591	389892	1057	050	-90	40.23	Technical
A-10-79*	6358529	389929	1049	050	-90	5.18	Technical
A-10-80*	6358538	389846	1051	050	-90	5.33	Technical
A-11-81*	6360166	389405	1041	050	-90	25.00	Technical
A-11-81A*	6360168	389406	1041	050	-90	10.00	Technical
A-11-82*	6360132	389315	1085	050	-90	175.26	Technical
A-11-83*	6360134	389317	1085	050	-90	30.00	Technical
A-11-84*	6359400	389549	1081	050	-90	45.73	Technical
A-11-85*	6359384	389566	1081	055	-60	173.10	Technical
A-11-86*	6359202	389644	1066	050	-83	505.05	Exploration
A-11-87*	6361641	387440	1475	050	-55	231.65	Exploration
A-11-88*	6361569	387510	1518	050	-72	299.62	Exploration
A-11-89*	6361680	387293	1506	050	-65	374.60	Exploration
A-11-90*	6361909	387137	1510	050	-62	281.64	Exploration
A-11-91*	6361481	387250	1582	050	-70	521.21	Exploration
A-11-92*	6359740	388865	1354	050	-68	648.32	Exploration
A-11-93*	6359740	388865	1354	035	-60	590.40	Exploration
A-11-94*	6359740	388865	1354	030	-48	162.46	Abandoned
A-11-95*	6359740	388865	1354	050	-62	593.45	Exploration
A-11-96*	6359993	388827	1322	050	-55	336.81	Exploration
A-11-97*	6360006	388692	1398	050	-85	99.06	Abandoned
A-11-98*	6360006	388692	1398	050	-59	471.83	Exploration
A-11-99*	6360006	388692	1398	050	-85	813.22	Exploration
A-11-100*	6362031	382560	1381	050	-70	99.06	Abandoned
A-13-101*	382560	6362031	1381	050	-55	269.75	Exploration
A-13-102*	382560	6362031	1381	050	-80	392.89	Exploration
A-13-103*	390858	6359518	1295	230	-60	373.88	Exploration
A-13-104*	386374	6362410	1650	050	-75	737.01	Exploration
A-13-105*	388828	6359993	1322	070	-75	442.87	Exploration

Hole ID	UTM N (m)	UTM E (m)	Elev. (m)	Azimuth (°)	Dip (°)	Length (m)	Type
A-13-106*	388172	6360545	1700	045	-59	531.27	Exploration
A-13-107*	388260	6360434	1600	035	-68	626.36	Exploration
A-13-108*	388865	6359740	1354	050	-77	152.10	Abandoned
A-13-109*	388865	6359740	1354	050	-78	746.46	Exploration
A-13-110*	390378	6358071	935	070	-65	578.82	Exploration
A-14-111*	388521	6360311	1438	040	-66	374.60	Exploration
A-14-112*	388200	6360744	1651	048	-75	397.46	Exploration
A-14-113*	388200	6360744	1651	048	-62	338.38	Exploration
A-14-114*	388640	6360477	1412	050	-85	178.92	Exploration
A-14-115*	388860	6360123	1317	050	-68	240.79	Exploration
A-14-116*	388827	6359993	1322	080	-62	476.40	Exploration
A-14-117*	388827	6359993	1322	080	-52	387.10	Exploration
A-14-118*	388200	6360744	1651	062	-83	62.18	Abandoned
A-14-119*	388200	6360744	1651	056	-83	36.58	Abandoned
A-14-120*	388200	6360744	1651	056	-80	461.47	Exploration
A-15-121*	388522	6360311	1438	035	-83	554.74	Exploration
A-15-122*	388362	6360290	1525	042	-64	553.21	Exploration
A-15-123*	388362	6360290	1525	042	-79	270.66	Abandoned
A-15-124*	388362	6360290	1525	045	-75	706.88	Exploration
A-15-125*	388557	6360161	1484	030	-65	461.77	Exploration
A-15-126*	388557	6360161	1484	030	-81	814.43	Exploration
A-15-127*	388557	6360161	1484	025	-76	716.28	Exploration
A-15-128*	388660	6360101	1429	030	-84	137.47	Abandoned
A-15-129*	388660	6360101	1429	030	-84	119.48	Abandoned
A-15-130*	388660	6360101	1429	035	-86	690.08	Exploration
A-15-131*	388522	6360311	1438	040	-57	322.18	Exploration

(*) Denotes un-surveyed drill hole collar.
Source: ZincX (2018)

Table 10-2: Table of "Nick"-Style Intercepts Recognized on the Akie Property since 2007

Hole ID	From/To (m)	Length (m)	Sample #	Elemental Enrichment	Stratigraphic Location
A-07-44	206.40 to 207.31	0.91	Unsampled	-	Limestone/RRG contact
A-08-62	542.00 to 542.94	0.94	855421	Cu, Pb, Zn, Ni, U, V, P, La, Cr, Se	Limestone/RRG contact
A-08-63	472.28 to 473.13	0.85	855656	Pb, Zn, U, P, Cr, Se	Limestone/RRG contact
A-10-72*	299.40 to 300.57	1.17	856376, 856377	Mo, Cu, Pb, Zn, Ag, Ni, Co, Fe, As, U, Cd, Sb, Bi, V, Ca, P, Ca, Hg, Tl, S, Ga, Se, Au Te, Ge, Sn, Y, Ce, Re, Pd, Pt	Cherty shales
A-13-103*	252.37 to 252.87	0.50	1195656	Mo, Cu, Pb, Zn, Ag, Ni, Co, As, U, Cd, Sb, V, P, La, Cr, Hg, Tl, Se, Au, Te, Cs, Ge, Y, Ce, Re, Pt	Limestone/RRG contact
A-13-106*	499.90 to 501.13	1.23	1196258	Pb, Zn, Ni, As, U, P, Se, Re, Pt	Limestone/RRG contact
A-14-114	148.30 to 149.69	1.39	269976, 269977, 269978, 269979	Mo, Cu, Pb, Zn, Ni, U, Sb, V, P, La, Cr, Hg, Tl, Se	Limestone/RRG contact
A-15-125	443.58 to 444.02	0.44	2695158	Pb, Zn, Ni, U, V, P, La, Cr	Debris flow/RRG contact
A-15-131	300.60 to 301.20	0.60	2695716	Cu, Pb, Zn, Ni, U, V, P, La, Cr, Hg, Se	Debris flow/RRG contact

(*) denotes the use of Acme Analytical Group 1F 54 element package to obtain rare earths and PGEs.
Source: CZM (2016)

10.3 2017 Exploration Program

The details from the 2017 exploration program including its objectives, results and discussion is an excerpt from the assessment report titled "*The 2017 Diamond Drilling Program on the Akie Property: Summary Report*" written and compiled by N. Johnson (2018). It is presented here unabridged except for a few minor edits. The outlined information remains current as of the date this report was issued.

The 2017 exploration program was based out of a trailer camp located at the 24.5 km mark of the Akie mainline FSR that is situated in an old Canfor forestry cut block (Figure 10-17). The seasonal camp can accommodate up to a maximum of 50 people and was opened in mid-June. Diamond drilling operations began on 28 June 2017 and continued until 19 August 2017. The camp was winterized and closed on 25 August 2017.

Figure 10-17: Camp Photograph

Source: ZincX (2018)

10.3.1 Program Objectives

The 2017 diamond drilling exploration program focused on the Cardiac Creek deposit with two primary objectives: to provide solid high-grade zone infill intercepts in the core of the deposit and test the down-dip extents of the high-grade zone with the intent of expanding the known NI 43-101 indicated and inferred resource boundaries. The mineralized material from the infill drilling will be used in subsequent metallurgical testing of the deposit.

10.3.2 Field Protocol

The exploration procedures implemented during the 2017 exploration program are outlined below. Details with respect to sample security, chain of custody, sample preparation and analyses and QA/QC of the analytical data are outlined in Section 11.

10.3.2.1 Drill Hole Numbering and Collar Locations

All the drill holes were numbered in accordance with the historical scheme with "A" (for the Akie property) dash “17” (the year) dash “132” (the next hole number in sequence). If a hole was abandoned and re-collared, the hole number shifted to the next number in sequence. The practice of suffixing the re-collared hole number with the letter “A” has been discontinued. To mark the location of a drill hole the casing remained in the ground. Occasionally, the casing from abandoned holes was pulled and used again in which case the hole was marked using a log if possible. A casing cap is then screwed into place engraved with the hole number, azimuth, dip, and depth of hole (see Figure 10-18).

10.3.2.2 Down Hole Surveys

Down hole directional surveys were taken, on average, of every 30 m to 50 m (approximately 100 ft to 150 ft) using a Reflex EZ-Shot single-shot down-hole survey tool. This survey tool provided point measurements of azimuth and dip of hole with estimated precisions of ± 0.5° and ± 0.2°, respectively. Allowing for a hypothetical depth to target of 550 m, the propagated horizontal and vertical uncertainties on a longitudinal projection or cross-section do not exceed 5 m and 2 m respectively.

Figure 10-18: Capped Casing

Source: ZincX (2018)

10.3.2.3 Core Handling & Logging

All drill core was boxed by the drill helper at the drill site. The core was flown to camp via helicopter for logging and sampling. The core is received by the geo-technician. The beginning and ending depth of each box is recorded and each box is labeled with aluminum tags. The technician measure and records the recovery and RQD characteristics of all the core. Characteristics such as lithology, veining, mineralization, alteration, etc., are recorded by geologist into the predefined logging template using a laptop computer. Selected samples are marked out by the geologist using, with a few exceptions, a maximum of 1.5 m sample length. The technician staples an aluminum tag, denoting the sample number, to the bottom of the box at the start of a given sample interval. Additional aluminum tags are stapled vertically at the start and end of each sample interval to clearly define a sample's boundaries. Drill holes are then photographed in their entirety by a technician prior to cutting of the samples for QA/QC purposes as shown in Figure 10-19.

Figure 10-19: A Core Photograph from A-17-137 Boxes 127 to 129 Prior to Sampling

Source: ZincX (2018)

Sampled intervals are cut in half by a core cutter using a diamond rock saw. The remaining core was returned to the core box as a record. The split sample is placed in a doubled-up polypropylene bag and each bag was secured with a zap strap. The samples are placed in polypropylene woven rice sacks, five samples to a sack, and kept in secure storage to await transportation to the analytical laboratory in Vancouver. The drill core is stored on-site in constructed core racks and/or cross-piled on wooden pallets as shown in Figure 10-20.

Figure 10-20: Core Storage at the Akie Property

Source: ZincX (2018)

10.3.2.4 Drilling Conditions

The drilling conditions on the Akie property can be difficult and can be attributed to several factors:

1. The fissile character of the host Gunsteel Formation shales (see Figure 10-21);
2. Poor ground conditions associated with brittle faulting encountered in the Gunsteel Formation (see Figure 10-22); and
3. Loss of water circulation down hole due to the highly fractured nature of the rock.

As a result, the rate of drilling can be quite slow. The use of drilling additives can improve production rates and core recovery leading to the successful completion of drill holes.

Figure 10-21: Fissile Character of the Gunsteel Formation Shales (Hole A-06-36A)

Source: ZincX (2018)

Figure 10-22: Bad Ground Associated with Brittle Faulting (A-08-60A)

Source: ZincX (2018)

10.4 Diamond Drilling Program (2017)

The 2017 drilling program involved eight planned drill holes with two primary objectives. The first was to obtain additional infill information within the core of the deposit. A total of five targets were outlined for this objective. The mineralized material from these intercepts was used for additional metallurgical testing of the Cardiac Creek deposit. The second objective focused on expanding the boundaries of the indicated resource both down-dip and along strike to the northwest. A total of three targets were outlined for this objective. Twelve holes were drilled totaling 5,092 m. All eight planned holes were completed to their intended depths, achieving the targeted pierce points and four were abandoned due to drill hole deviation. The drill core is stored at the company's exploration camp with the UTM coordinates of 379,335 mE, 6,351,701 mN. The details of each drill hole are found in Table 10-3. A summary of the drilling is provided in the following sections.

Table 10-3: 2017 Drill Hole Collar Details

HOLE ID	UTM E (m)	UTM N (m)	ELEV (m)	AZIMUTH (°)	DIP (°)	LENGTH (m)	Target Zone
A-17-132	388260	6360435	1603	55	-71	598.02	Cardiac Creek deposit
A-17-133	388522	6360311	1438	68	-73	413.61	Cardiac Creek deposit
A-17-134	388522	6360311	1438	50	-83	82.30	Abandoned
A-17-135	388522	6360311	1438	55	-83.5	159.11	Abandoned
A-17-136	388692	6360006	1398	45	-60	120.22	Abandoned
A-17-137	388522	6360311	1438	66	-83.5	614.78	Cardiac Creek deposit
A-17-138	388692	6360006	1398	45	-63	454.76	Cardiac Creek deposit
A-17-139	388200	6360320	1608	55	-70	43.28	Abandoned
A-17-140	388200	6360320	1608	55	-70	847.96	Cardiac Creek deposit
A-17-141	388554	6360159	1484	55	-75	651.36	Cardiac Creek deposit
A-17-142	388362	6360290	1525	68	-77	700.13	Cardiac Creek deposit
A-17-143	388380	6360600	1570	30	-83	406.91	Cardiac Creek deposit

Source: ZincX (2018)

10.4.1 Cardiac Creek Deposit

The Cardiac Creek deposit is central to the Akie property, straddling Cardiac and Avalanche Creeks. A total of 5,092 m was drilled in 12 drill holes with four drill holes abandoned due to excessive deviation. Summaries of each drill hole are presented below.

10.4.1.1 A-17-132

Drill hole A-17-132 was the first of five infill holes targeting the core of the deposit. The hole achieved a pierce point located approximately 40 m from holes 38, 51, and 107. Deviation was not an issue for this drill hole.

The drill hole collared into a thin, approximately 25 m thick, sliver of the soft medium grey shales of the Akie Formation before shifting into the prospective Gunsteel Formation siliceous shales. Near the contact with the Akie Formation intervals of disrupted silty to sandy shales were encountered to a depth of 164.66 m intermixed with minor fragmental shales before transitioning into several intervals of baritic and or fragmental shales from 164.66 m to 309.84 m. Below 309.84 m black siliceous shales are the dominate lithology interbedded with several sections of distal laminar pyrite with nodular barite as well as scattered thin intervals of chert. The hole intersected the thick beds of laminar pyrite with minor nodular barite of the Proximal Facies at a depth of 510.61 m with the upper contact of the mineralization marked by a distinct quartz-carbonate vein zone. Increasing amounts of light grey sphalerite rich bands are present within the Proximal Facies mineralization below a depth of 528.88 m. The hole transitioned into the Cardiac Creek Zone at a depth of 537.41 m which is characterized by thick beds of laminar pyrite and sphalerite rich bands. The sphalerite bands exhibit well developed mottled textures below 545.60 m. The Cardiac Creek Zone is interbedded with minor amounts of black siliceous shales. Below the Cardiac Creek Zone, the hole intersected a thin 2.76 m thick interval of debris flows of Paul River Formation. The hole ended in the calcareous siltstones of the Silurian Siltstone at a depth of 598.02 m

10.4.1.2 A-17-133

Drill hole A-17-133 was the second of five infill holes targeting the core of the deposit. The hole achieved a pierce point in an open area within the core of the deposit located approximately 55 m down dip of hole 49 and along strike of holes 53, 73B and 39A. Deviation was not an issue for this drill hole.

The hole collared into the black siliceous shales of the Gunsteel Formation interbedded with a few narrow intervals of baritic and fragmental shales occurring to a depth of 36.71 m. Alternating sequences of black siliceous shales, chert and cherty shales, and distal facies laminar pyrite with nodular barite are present from 36.71 m to 336.64 m. The thick layers of laminar pyrite characteristic of the Proximal Facies interbedded with a few thin black siliceous shale beds occur from 336.64 m to 352.51 m where the mineralization transitions into the Cardiac Creek Zone mineral facies. The zone is present to a depth of 388.37 m and is characterized by light grey sphalerite rich bands intermixed with laminar pyrite. Higher grade mottled textured sulphide bands enriched in sphalerite, galena, pyrite, quartz, carbonate and barite are prominent over approximately 20 m from 361.10 m to 381.09 m. The zone is underlain by 8.25 m of massively bedded barite interbedded with laminar pyrite. A thin 4.42 m thick interval of debris flow is present beneath the mineralization at a depth of 400.05 m. The hole ended within the calcareous siltstones of the Silurian Siltstone at a depth of 413.61 m.

10.4.1.3 A-17-134, A-17-135

Drill holes A-17-134 and A-17-135 were both planned to test a target slightly along strike of A-15-121 and down-dip. Unfortunately, both holes experienced excessive amounts of deviation and were abandoned at depths of 82.30 m and 159.11 m respectively. Hole A-17-134 was re-collared as A-17-135 and hole A-17-135 was re-collared as A-17-137.

10.4.1.4 A-17-136

Drill hole A-17-136 was planned to test a target along strike of hole A-11-98 and up-dip of hole A-06-40. Unfortunately, the hole experienced an excessive degree of flattening and was abandoned at a depth of 120.22 m. The hole was re-collared as A-17-138.

10.4.1.5 A-17-137

Drill hole A-17-137 was the third of five infill holes targeting the core of the deposit. The hole achieved a pierce point in a large open area down-dip from the core of the deposit. A pierce point was obtained located approximately 75 m down dip of hole 121 and up-dip and slightly along strike of hole 127. Deviation was not an issue in hole A-17-137 and a pierce point was obtained within 10 m of the planned target.

The geology of hole 137 is very similar in nature to that of hole 133 described above. The hole collared into the black siliceous shales of the Gunsteel Formation with a few intervals of baritic and fragmental shales. Beyond a depth of 47.34 m the hole alternated between intervals of black siliceous shales, distal facies mineralization consisting of laminar pyrite with nodular barite, and a few scattered narrow chert beds or cherty shales. At a depth of 423.05 m the Proximal Facies mineralization consisting of thick bands of laminar dull brown very fine-grained pyrite interbedded with black siliceous shales was intersected. Towards the lower contact of the Proximal Facies there is an increasing amount of light grey sphalerite rich bands within the pyritic bands. The Cardiac Creek Zone is present beyond a depth of 466.80 m. The zone is characterized by strong, well-developed mottled textured sulphides throughout enriched in light grey sphalerite, galena, quartz, carbonate and barite. The mineralization is interbedded with a few narrow and thin beds of black siliceous shales. The zone continues to a depth of 534.10 m where a thick and distinct 10.36 m bed of black shales separates the Cardiac Creek Zone from the Footwall Zone. The Footwall Zone is present from 544.46 m to 559.42 m and consists of the same style of mineralization as the Cardiac Creek Zone intermixed with beds of massive, granular barite and some laminar pyrite. A narrow 5.54 m thick pyrite dominated massive sulphide lens occurs beneath the mineralization of the Cardiac Creek and Footwall Zones that overlies a 6.24 m thick interval of debris flows of the Paul River Formation. The hole ended in the calcareous siltstones of the Silurian Siltstone at a depth of 614.78 m however within this unit there are three distinct intervals of black pyrobitumen occurring between 599.00 m to 606.86 m.

10.4.1.6 A-17-138

Drill hole A-17-138 was the fourth of five infill holes targeting the core of the deposit. The hole achieved a pierce point in a large open area along the southeastern flanks of the core of the deposit. A pierce point was obtained located approximately 85 m up-dip of hole 40 and along strike of holes 58 and 98. Unlike hole 136 deviation was accounted for and the pierce point was slightly up-dip of its planned target.

The hole collared into the black siliceous shales of the Gunsteel Formation. From the collar to a depth of 79.20 m the hole intersected intervals of baritic shale and fragmental shale interbedded with sections of barren black siliceous shales. Beyond 79.20 m there are alternating sequences of distal facies laminated pyrite with nodular barite, chert or cherty shales and black siliceous shale. This sequence of stratigraphy continues to a depth 403.30 m where a thin quartz-carbonate vein zone marks the contact with the Proximal Facies. The Proximal Facies, characterized by thick beds of laminar pyrite interbedded with the black siliceous shales, is quite thin compared to the other holes drilled in 2017. The Cardiac Creek Zone was intersected at a depth of 412.15 m and continues through to 440.85 m intermixed with a few minor chert intervals and thin shale intervals. The mineralization is predominantly banded in appearance with abundant light grey sphalerite bands which exhibit poorly developed mottled textures locally. The mineralization is underlain by a thin 2.74 m interval of the Paul River Formation debris flows and the hole ended in the calcareous siltstones of the Silurian Siltstone at a depth of 454.76 m.

10.4.1.7 A-17-139

Drill hole A-17-139 was planned to test a target down-dip of A-08-65. The hole experienced greater than intended deviation and was abandoned at a depth of 43.28 m. The hole was re-collared as A-17-140.

10.4.1.8 A-17-140

Drill hole A-17-140 was the first of three holes to test the down-dip and strike extents of the indicated resource. The hole was targeting an area down-dip of hole 65 and up-dip of hole 36A. Deviation was an issue in that the hole did not flatten as much as intended and swung to the north resulting in a pierce point in an open area located approximately 80 m from hole 36A and 100 m from hole 42. Despite the deviation encountered, the hole still achieved the intended goal of testing the edges of the indicated resource.

The hole collared into the calcareous siltstones of the Road River Group that are in thrust contact with the underlying Earn Group rocks. The Hanging Wall Thrust was encountered at a depth of 63.73 m. Immediately below the thrust, a thick section of soft aluminous shales of the Akie Formation were encountered down to a depth of 278.38 m. At the base of the Akie Formation shales the hole transitioned into the black siliceous shales of the Gunsteel Formation. For approximately 100 m from 278.38 m to 373.85 m there are a few intervals of baritic or fragmental shales interbedded with the black shales. Intervals of distal facies laminated pyrite with nodular barite along with a few minor chert beds and fragmental shale occur below a depth of 541.87 m. Just above the Proximal Facies there is an uncommon but distinct 10.98 m interval of nodular barite that occurs from 684.84 m to 695.82 m. A quartz-carbonate vein zone occurs at the base of this interval that commonly marks the upper contact with the Proximal Facies. There are four distinct intervals of both Proximal Facies and Cardiac Creek Zone style mineralization. Two hanging wall zones are present from 695.82 m to 706.18 m and 718.18 m to 723.84 m separated by thick intervals of siliceous shale. The mineralization in these two zones are comprised of thickly bedded laminated pyrite mineralization with minor light grey sphalerite bands. The Cardiac Creek Zone, present from 730.22 m to 766.46 m, is characterized by thickly bedded laminar pyrite with sections of light grey sphalerite banding. The mineralization is interbedded with thin but distinct beds of black siliceous shale and chert beds. An 8.23 m thick black siliceous shale separates the Cardiac Creek Zone from the Footwall Zone. The Footwall Zone is comprised of mottled textured bands of sulphides with minor laminar pyrite that are enriched in light grey sphalerite, galena, quartz, and carbonate. Underlying the Cardiac Creek and Footwall Zones is a 14.94 m thick interval of dull to brassy yellow laminar pyrite containing abundant irregular shaped calcareous concretions. A thick section of debris flows and turbiditic shales of the Paul River Formation are present over 28.22 m below a depth of 792.78 m. The hole ended in the calcareous siltstones of the Silurian Siltstones at a depth of 847.96 m.

10.4.1.9 A-17-141

Drill hole A-17-141 was the fifth and final drill hole to target the core of the deposit. The hole achieved a pierce point in an open area along strike of holes 45 and 30 and up-dip of hole 35. Deviation was not an issue and the intended target was achieved.

The hole collared into the black siliceous shales of the Gunsteel Formation. Numerous intervals of baritic and fragmental shales interbedded with black shales with minor chert intervals occur for the first several hundred metres of the hole to a depth of 335.00 m. Interbedded intervals of black shale, chert and distal facies laminar pyrite with nodular barite are the dominant lithologies from 335.00 m to 551.80 m. The upper contact of the Proximal Facies is strongly faulted displacing an unknown amount of the mineralization. The

Proximal Facies occurs from 551.80 m to 560.48 m. The mineralization is characterized by thick beds of laminar pyrite with an increasing amount of light grey sphalerite banding present towards the base of the interval. The Cardiac Creek Zone is strongly affected by faulting from 562.20 m to 580.27 m displacing higher-grade mottled textured sulphides enriched in sphalerite, galena, quartz, carbonate and barite. Below the faulting the mineralization is intermixed with granular massive barite beds and interbedded with black siliceous shale. Locally, the mineralization appears to be overprinted by a coarser grained wisps or stringers of red-brown sphalerite. Underlying the mineralization there is a thin massive sulphide lens comprised primarily of pyrite that occurs at a depth of 594.94 m. It is intermixed with the debris flows of the Paul River Formation. The hole ended at a depth of 651.36 m in the calcareous siltstones of the Silurian Siltstone.

10.4.1.10 A-17-142

Drill hole A-17-142 was the second of three holes to test the down-dip and strike extents of the indicated resource. The hole was targeting a large open area down-dip of hole 121. Deviation was an issue, the hole experiencing a significant amount of flattening and swing in the azimuth at a depth of about 275 m. As a result, a pierce point was obtained in close proximity to hole 124 intersecting similar lithological units and mineralization.

The hole collared into a relatively thin section of soft aluminous shales of the Akie Formation that extended to a depth 44.10 m. Disrupted silty shales, siltstones and sandstone lenses within the black siliceous shales of the Gunsteel Formation occur between 44.10 m to 178.40 m. The hole transitions into alternating sequences of baritic and fragmental shales interbedded with black siliceous shale as well as some minor chert which extend down to a depth of 363.32 m. Several intervals of distal facies laminar pyrite with nodular barite and scattered thin sections of chert are present from 363.32 m to 578.31 m. The upper contact of the Proximal Facies was marked by a thin quartz-carbonate breccia vein zone like most of the other holes drilled in 2017. The mineralization is characterized by thick beds of laminar dull brown pyrite with minor nodular barite interbedded with black siliceous shales. The Cardiac Creek Zone was intersected at 612.12 m and consisted of abundant light grey sphalerite banding hosted within the thick beds of laminar pyrite. The sphalerite banding transitions to mottled textured sulphides towards the base of the zone. A distinct 10 m thick black siliceous shale bed separates the Cardiac Creek Zone from the Footwall Zone from 632.17 m to 642.17 m. The Footwall Zone is characterized by strongly developed mottled textured sulphides enriched in light grey sphalerite, galena, quartz, carbonate, and barite with lesser amounts of laminar pyrite ending at a depth of 655.75 m. Debris flows of the Paul River Formation are present from 658.08 m to 662.12 m and the hole ended in the calcareous siltstones of the Silurian Siltstone at a depth of 700.13 m.

10.4.1.11 A-17-143

Drill hole A-17-143 was the third and final of three holes to test the down-dip and strike extents of the indicated resource. The hole was targeting a large area down-dip of hole 11 along the northwestern edges of the indicated resource. The hole experienced a significant amount of deviation in both azimuth and dip resulting in a pierce point located approximately 75 m along strike and slightly down-dip of hole 11. Despite the deviation, the area was previously untested thus providing meaningful data.

The hole collared the black siliceous shales of the Gunsteel Formation. Unlike the other holes of the 2017 program there were no distinct intervals of baritic and fragmental shales present near the top of the hole. From the collar to a depth of 217.17 m there are scattered, thin, intervals of chert and cherty shales with minor amounts of silty shale interbedded with black siliceous shales. Intervals of distal facies laminar pyrite with nodular barite are present below 217.17 m to the top of the Cardiac Creek horizon. The quartz-carbonate vein zone that marks the upper contact of the Proximal Facies was intersected at a depth of 343.51 m. The Proximal Facies mineralization is quite thin before transitioning into the Cardiac Creek Zone at 365.44 m. Thick beds of sulphides comprised of laminar dull brown pyrite with a rapid increase in light grey sphalerite banding and mottled textured sulphides characterized the mineralization of the Cardiac Creek Zone which extends to a depth of 382.95 m. Towards the base of the zone the sulphides are intermixed with massive granular beds of barite. A 6.04 m thick interval of debris flow with fossiliferous limestone boulders of the Paul River Formation is present from 384.42 m. At the base of the debris flow is a thin 18 cm interval of what appears to be “Nick”-style mineralization. The hole ended in the calcareous siltstones of the Silurian Siltstone at a depth of 406.91 m.

10.4.2 Drill Hole Results

A summary of the analytical results from the 2017 drilling program can be seen below in Table 10-4.

Table 10-4: Summary of Drill Results from the 2017 Program

Drill Hole	From (m)	To (m)	True Width (m)*	Zn (%)	Pb (%)	Ag (g/t)†	Zn+Pb (%)
A-17-132	520.29	573.08	42.43	6.41	1.08	10.6	7.49
CCZ	537.41	573.08	28.67	8.84	1.54	14.2	10.38
including	546.41	571.06	19.81	10.52	1.87	15.9	12.39
including	546.41	566.01	15.75	10.96	2.01	16.7	12.97
including	546.41	559.05	10.16	12.18	2.24	17.2	14.42
A-17-133	341.08	388.38	33.14	4.77	0.78	8.5	5.55
CCZ	351.03	387.57	25.63	5.68	0.94	9.6	6.62
including	361.90	381.10	13.48	8.00	1.40	12.9	9.40
including	367.68	381.10	9.42	10.30	1.81	16.0	12.11
A-17-137	454.40	559.44	57.79	9.72	2.07	19.1	11.79
CCZ	466.78	534.09	37.06	11.83	2.68	23.4	14.51
including	480.93	534.09	29.26	14.32	3.33	28.0	17.65
including	506.00	534.09	15.44	18.27	4.34	36.2	22.61
FW	544.48	559.44	8.20	14.41	2.36	25.3	16.77
MS	559.44	565.00	3.04	0.98	0.23	10.0	1.21
A-17-138	403.32	440.85	33.40	5.33	0.91	9.0	6.24
CCZ	412.15	440.17	24.96	6.60	1.15	10.4	7.75
including	426.27	439.52	11.82	8.50	1.57	12.3	10.07
A-17-140	694.00	776.57	59.87	2.24	0.37	4.9	2.61
HW A	694.00	706.20	8.66	1.11	0.14	4.0	1.25
HW B	718.19	723.83	4.05	3.77	0.63	7.4	4.40
CCZ	730.24	758.23	20.40	2.44	0.34	5.6	2.78
FW	766.46	776.57	7.51	7.49	1.50	13.8	8.99
including	766.46	775.16	6.46	8.50	1.71	15.5	10.21
A-17-141	555.20	587.64	23.36	8.09	1.46	15.1	9.55
CCZ	562.18	587.64	18.34	10.05	1.84	18.4	11.89
including	563.85	587.64	17.14	10.47	1.94	19.1	12.41
including	563.85	586.00	15.96	10.86	2.06	19.4	12.93
including	563.85	574.24	7.49	18.79	3.69	29.3	22.48
A-17-142	581.84	655.75	60.67	5.55	1.06	10.0	6.61
including	616.34	655.75	32.65	9.30	1.85	15.5	11.15
CCZ	616.34	632.17	13.05	6.45	1.14	11.3	7.59
including	623.33	632.17	7.30	7.09	1.38	14.0	8.47
FW	642.17	655.75	11.31	19.30	4.01	30.9	23.32
A-17-143	346.92	384.42	25.33	5.72	0.89	8.6	6.61

Drill Hole	From (m)	To (m)	True Width (m)*	Zn (%)	Pb (%)	Ag (g/t)†	Zn+Pb (%)
CCZ	352.64	382.95	20.49	6.73	1.04	9.8	7.77
including	365.99	382.95	11.50	7.17	1.27	12.4	8.44
including	371.31	382.95	7.90	8.84	1.57	15.0	10.41

True widths calculated based on the assumed orientation of the Cardiac Creek deposit with a 70-degree dip. (CCZ – Cardiac Creek Zone, FW – FW Zone, MS – Massive sulphide, NLZ – North Lead Zone, NICK – "Nick"-style mineralization) (*): Values below detection limit given half the value for the purposes of weighted averages.

Source: ZincX (2018)

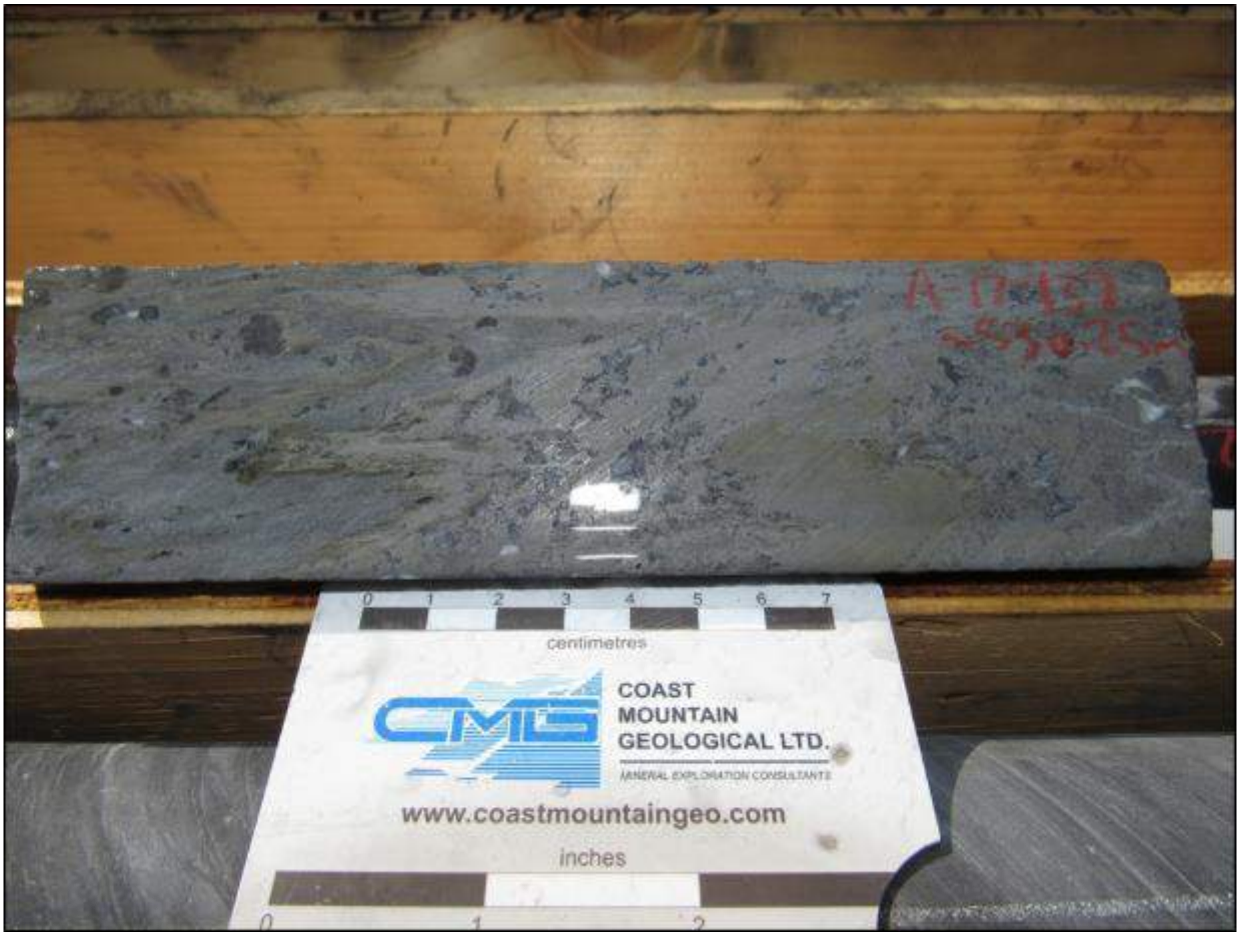
10.4.2.1 Cardiac Creek Deposit

The drilling on the Cardiac Creek deposit targeted specific areas in the core of the deposit to provide; infill information, to collect representative mineralized material for metallurgical testing, and to expand the boundaries of the current indicated resource. Drill holes A-17-132, A-17-133, A-17-137, A-17-138, and A-17-141 all provided infill information on the deposit. The intercepts in each hole returned results very similar in grade and thickness to the pierce points surrounding them (Table 10-4) with a couple exceptions.

Drill hole A-17-137 provided a pierce point located in the central core of the deposit down-dip of A-15-121. The results from this hole represent the best intersection obtained to date from the Cardiac Creek deposit with extremely thick and high-grade Pb and Zn mineralization. The mineralization associated with the both the Cardiac Creek and Footwall Zones extend from 428.62 m to 559.42 m characterized by thick beds of dull brown very fine grained laminar pyrite interbedded with black siliceous shales containing an increasing amount of light grey sphalerite banding and mottled textured sulphides enriched in sphalerite, galena, quartz, carbonate and barite with depth. The mottled textured sulphides become the dominant style of mineralization below 480.05 m reflecting the high-grade nature of the intercept (Figure 10-24). The Cardiac Creek Zone is present from 428.62 m to 534.10 m and the Footwall Zone is present from 544.46 m to 559.42 m separated by a distinct siliceous shale interbed. Both zones returned extremely high-grade Pb, Zn, and Ag results over a variety of intervals (Table 10-4). A massive sulphide lenses underlies the deposit from 559.42 m to 564.96 m and is characterized by bright brassy yellow pyrite and locally cross cut by creamy white carbonate and seams of sphalerite and galena. The lens is anomalous in Pb with values ranging from 0.16% to 0.28%, Zn values ranging from 0.17% to 2.87% and Ag values are also anomalous ranging from 5.4 g/t to 13.3 g/t. The results from the massive sulphide lens are summarized in Table 10-4.

Drill hole A-17-141 obtained a pierce point located in the core of the deposit along strike of hole 45 and along strike and slightly down-dip of hole 30. The mineralization associated with the Cardiac Creek horizon is present from 551.80 m to 587.67 m. The main zone of mineralization was thinner than expected due to a significant brittle structure which has truncated the zone with an unknown amount of displacement. It appears that this structure has offset the higher-grade portion of the zone based on the textures, style and character of the mineralization present within the structure. Despite the structure, the results are comparable to the high-grade mineralization present in the surrounding holes.

Figure 10-23: Folded High-Grade Mineralization in Cardiac Creek Zone in A-17-137 @ 530.25 m



Source: CZM (2017)

There was a total of three drill holes that tested the boundaries of the indicated resource both at depth and along strike to the northwest. They include: A-17-140, A-17-142, and A-17-143. Hole A-17-140 intersected a total of four distinct zones of mineralization, two Hanging Wall Zones, the Cardiac Creek Zone and the Footwall Zone over an extensive interval from 695.82 m through to 775.16 m. The results from the two Hanging Wall Zones were low grade and thin. The Cardiac Creek Zone returned similar results compared to the surrounding holes (e.g. 36A, 42). The highest-grade material was encountered in the Footwall Zone and included a thin interval of 10.21% Zn+Pb and 15.5 g/t Ag over a true width of 6.46 m. The deviation encountered in hole 142 positioned it in close proximity to hole 124 returning very similar results from the Cardiac Creek Zone and Footwall Zones. While the results from the Footwall Zone were similar to hole 124 the 23.32% Zn+Pb, and 30.9 g/t Ag over a true-width of 11.31 m represents the best intersection to date returned from the Footwall Zone. The final hole of the program, A-17-143 returned better than expected results along the northwestern boundary of the indicated resource with a well mineralized interval from 346.92 m to 384.42 m. The results were thicker and higher-grade than expected in this area of the deposit.

Hole A-17-143 also returned results indicative of “Nick”-style mineralization situated at the base of the debris flows, underlying what appears to be a fossiliferous limestone boulder from the Paul River Formation. A selective, 18 cm, sample from 392.86 m to 393.04 m returned anomalous values of Pb, Zn, Ni, U, V, P, La, and Cr. Selenium, an element commonly associated with this type of mineralization, is elevated with respect to the surrounding samples however it is comparable to those further up and associated with the Cardiac Creek style mineralization. This information is summarized in Table 10-5.

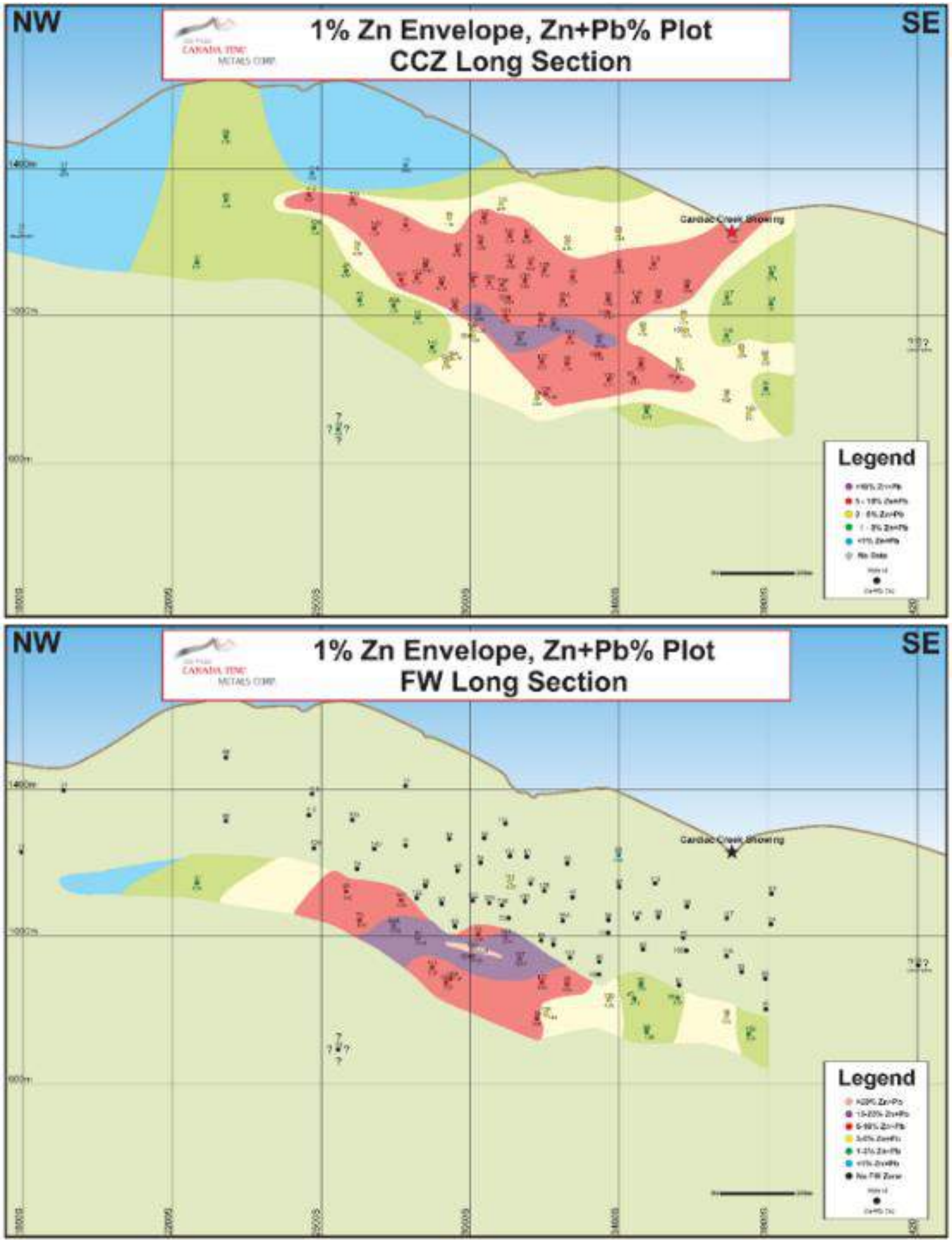
Table 10-5: 2017 “Nick”-style Mineralization Intercepts

Hole ID	A-17-143
Sample #	2697422
Interval (m)	392.86 to 393.04
Length (m)	0.18
Elemental Enrichment	Pb, Zn, Ni, U, V, P, La, Cr, Se?

Source: CZM (2017)

The pierce points from the 2017 drill holes can be seen in the Cardiac Creek and Footwall Zone long sections presented in Figure 10-24.

Figure 10-24: Long Sections Depicting Zn+Pb% Values across the (A) Cardiac Creek Zone (CCZ) and (B) Footwall Zone (FW)



Source: ZincX (2018)

10.4.3 Discussion

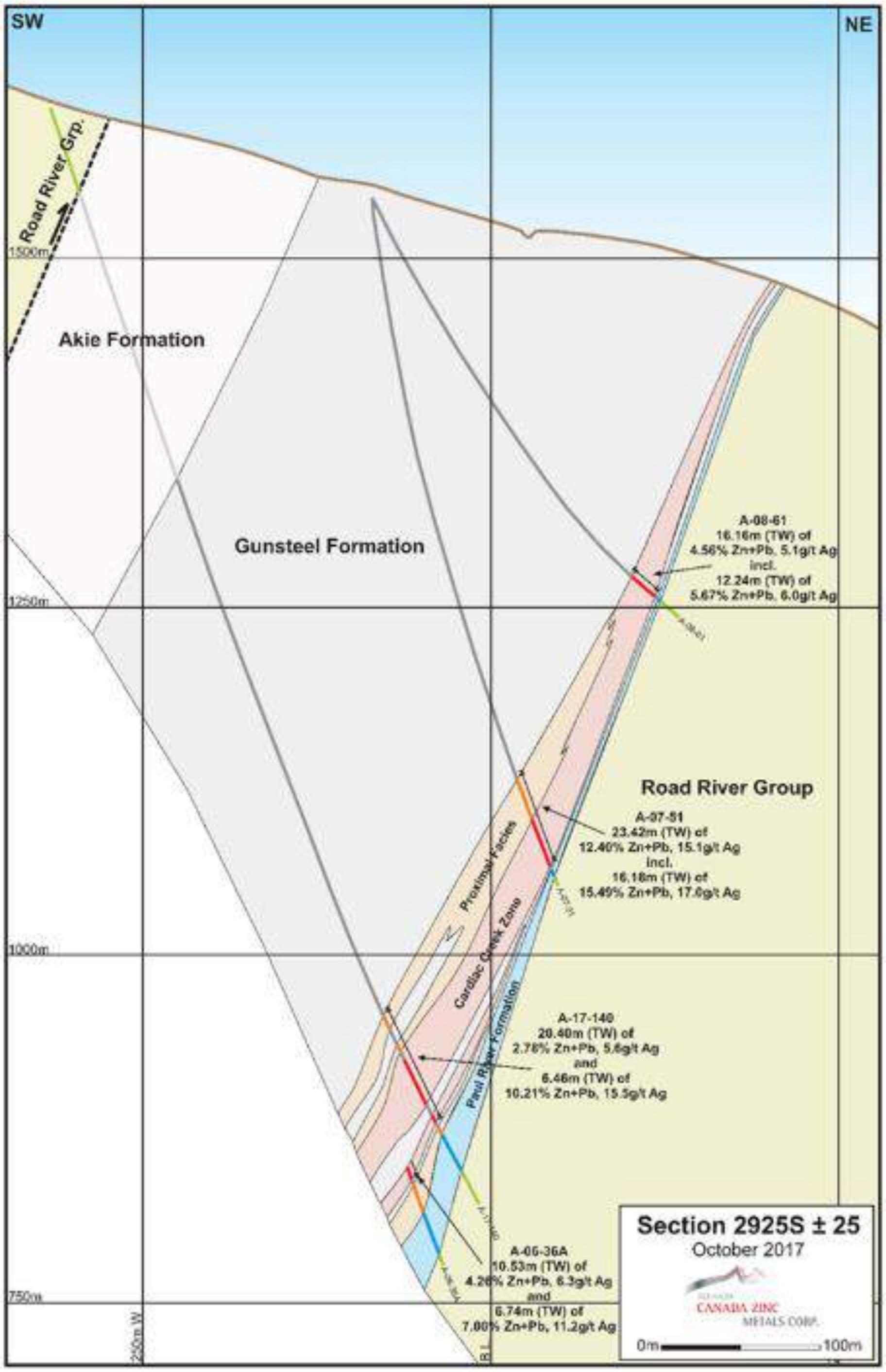
10.4.3.1 The Cardiac Creek Deposit

The objectives for the 2017 drill program focused on infill within the core of the deposit and boundary expansion of the indicated resource at a 5% Zn cut-off both along strike to the northwest and down-dip. Drill holes A-17-132, A-17-133, A-17-137, A-17-138, and A-17-141 all intersected results consistent with the surrounding drill holes, except for A-17-137 which returned the best intersection to date from the deposit. These data underlie the consistency of the high-grade core across the entire deposit.

Drill holes A-17-140, A-17-142, A-17-143 all tested the boundaries of the 5% Zn cut-off indicated resource. Drill hole A-17-140 returned similar values to the surrounding holes for the Cardiac Creek and Footwall Zones. The hole also intersected two thin Hanging Wall Zones. The first from 695.82 m to 706.18 m and the second from 718.18 m to 723.84 m. Both zones returned nominal low-grade results however the second zone of mineralization displayed some characteristics of Cardiac Creek Zone style mineralization with the presence of light grey sphalerite banding. The presence of a second distinct Hanging Wall Zone is unique that has not been observed previously. After a review of all the drilling the Hanging Wall Zone was found to be present in a total of nine drill holes spread across the down-dip portion of the deposit.

These holes include: A-95-13, A-95-18, A-06-36A, A-11-92, A-11-95, A-13-109, A-15-126, A-15-127, and A-17-140. It is unclear whether these Hanging Wall Zones possibly represent a folded limb of the Cardiac Creek Zone that has relaxed at depth or whether they are separate continuous lenses of mineralization that have merged into the Cardiac Creek Zone further up-dip because of folding and thrusting. These zones are generally very low grade and currently do not display any clear or distinct trends that might vector future drilling program.

Figure 10-25: 2017 Schematic Cross Section XS 2925S



Source: ZincX (2018)

Drill hole A-17-143 intersected a thicker and higher-grade interval of the Cardiac Creek Zone than expected. The presence and tenor of this mineralization has expanded the known limits of the high-grade core further to the northwest. In addition to the mineralization associated with the Cardiac Creek Zone a very thin 18 cm interval of the "Nick"-style mineralization was encountered below the deposit at a depth of 392.86 m (Figure 10-26). The mineralization is hosted in a thin interval of black shale underlying the fossiliferous limestone or fragments of limestone and in contact with the calcareous siltstones of the Silurian Siltstone. The small amount of sulphide appears to be associated with thin bands of flattened phosphatic chert nodules (pebbles) that are commonly associated with this type of mineralization (Figure 10-27). There is a total of 10 intercepts of this style of mineralization spread out across the Akie property. The results from this new intersection are similar to the other occurrences. A table of the previous nine intercepts can be found below in Table 10-4. The intercept from A-17-143 shows a similar enrichment to the other occurrences.

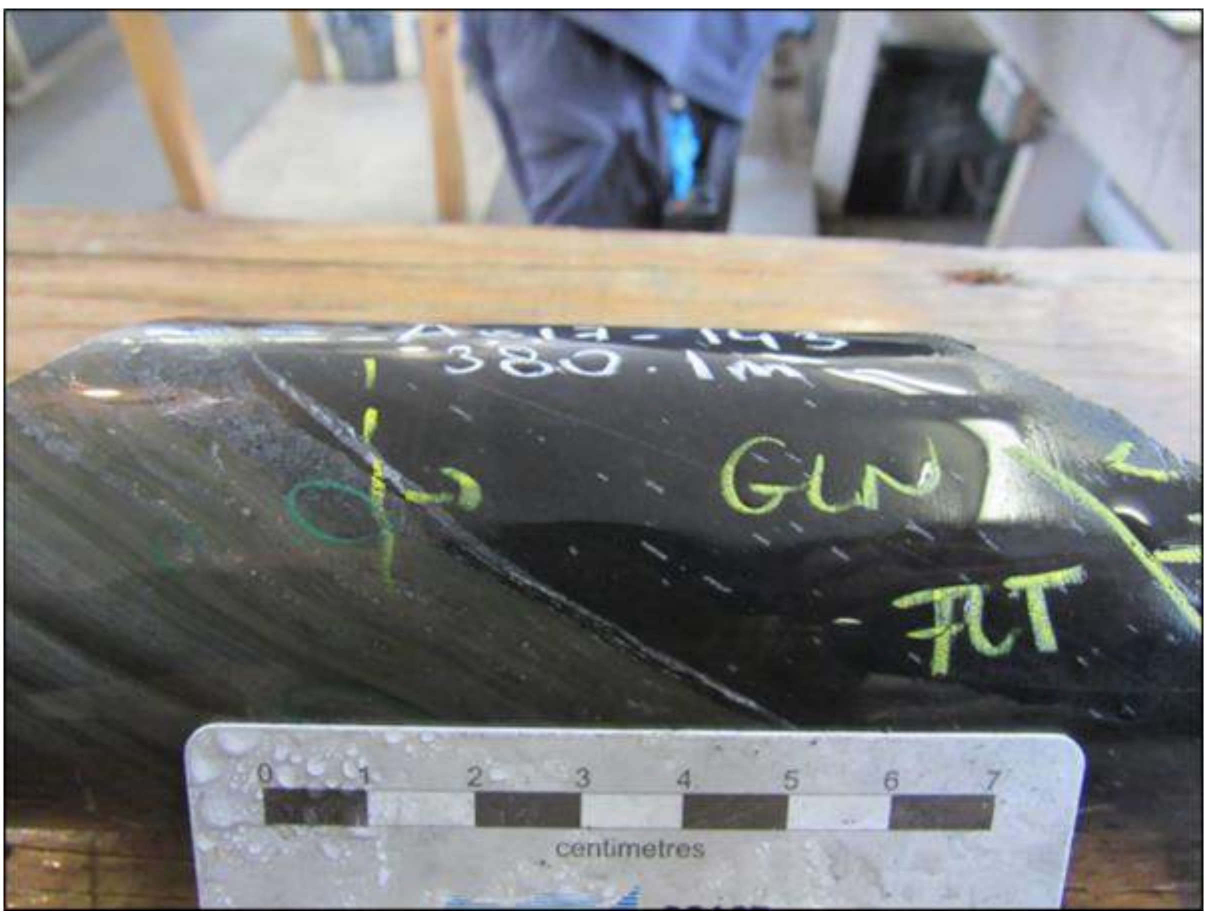
Figure 10-26: "Nick"-style Mineralization in a-17-143 @ 392.84 m

Source: CZM (2017)

Of the 10 occurrences, seven of them occur along the contact between the Kwadacha Limestone and the Silurian Siltstone (Road River Group). The Kwadacha Limestone outcrops in abundance along Silver Creek and to the northwest of the deposit forming prominent cliffs. There are thick sequences of Kwadacha Limestone that outcrop along the eastern edges of the property that is in thrust contact with both the Silurian Siltstone and Earn Group rocks. Prospecting along any exposures of the contact between the limestone and the siltstones of the Silurian Siltstone could possibly identify a surface exposure as well as assess the true thickness of the horizon and viability as an exploration target in future exploration programs.

Another interesting characteristic of the mineralization observed in hole A-17-143 is the presence of clustered galena present within the siliceous black shale interbeds directly associated with the Cardiac Creek Zone mineralization. Fine grains of millimetre sized galena appear to be disseminated within select shale interbeds within the Cardiac Creek Zone. The grains range in size from sub-millimetre to several millimetres in the long direction. Some of the larger grains can be rectangular or lath-like in shape with distinct boundaries while other grains are somewhat rounded into an ellipsoid shape. The long axis of the larger grains tends to be oriented parallel to the primary cleavage observed in the rock and in some cases the grains appear to have strain shadows. It is unclear whether this represents some local remobilization of the galena (Figure 10-27).

Figure 10-27: Remobilized Galena within Gunsteel Formation Shale in A-17-143 @ 380.10 m



Source: CZM (2017)

10.4.4 Conclusions and Recommendations

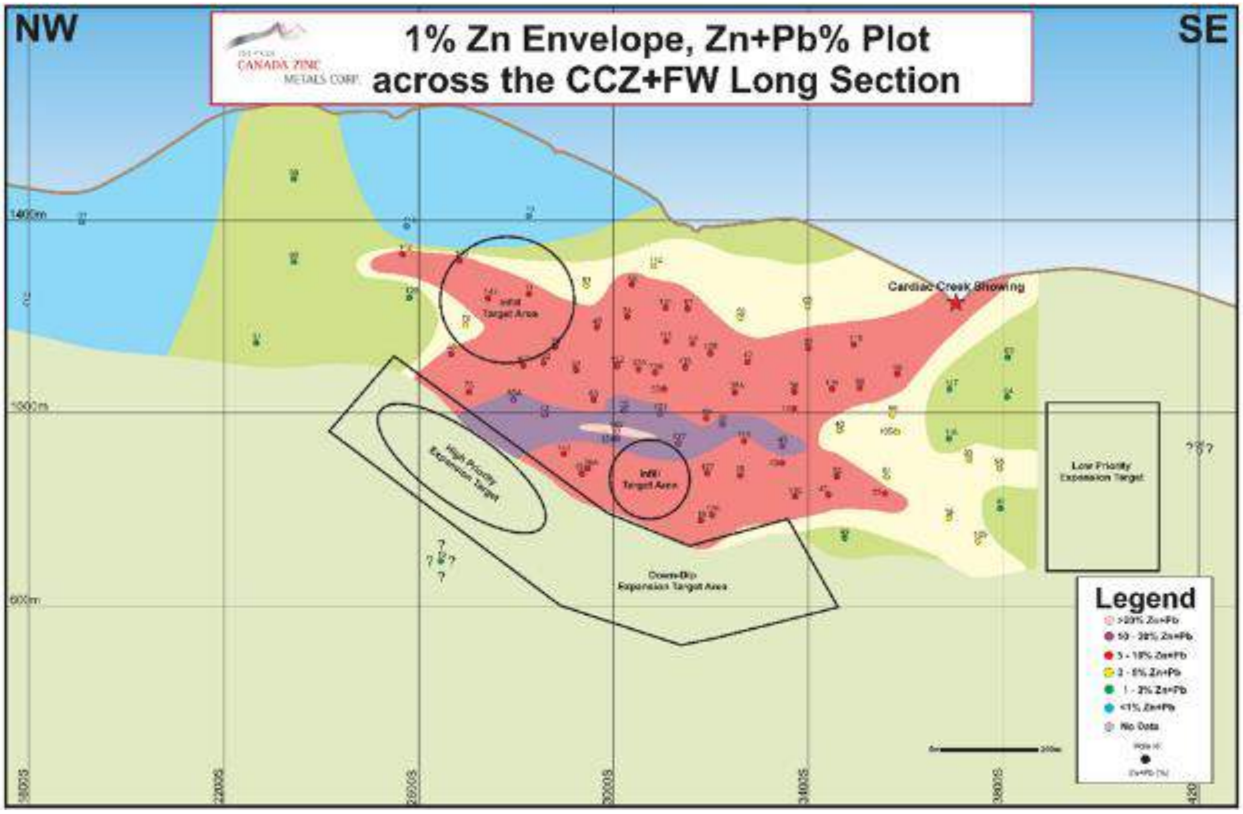
The 2017 Akie drilling program was successful in achieving its objectives. All the planned drill targets were tested returning good results.

1. Infill drilling (A-17-132, A-17-133, A-17-137, A-17-138, and A-17-141) on the Cardiac Creek deposit continues to demonstrate the consistency of the high-grade core in both thickness and grade. The results from A-17-137 represent the highest-grade intersect from the Cardiac Creek Zone ever encountered;
2. Drilling along the boundaries of the indicated resource at the 5% Zn cut-off both down-dip and along-strike to the northwest was successful. Drill hole A-17-140 continued to intersect both the Cardiac Creek and Footwall Zones with similar results compared to the surrounding holes as well as two additional Hanging Wall Zones. Despite the issues with deviation hole A-17-142 returned the best results ever encountered from the Footwall Zone. The final hole A-17-143 returned a thicker and higher-grade interval of the Cardiac Creek Zone than expected pushing the known limits of the high-grade core further northwest; and
3. Drill holes A-17-143 intersected additional “Nick”-style mineralization at the base of the debris flows bringing the total number of known intersections for this style of mineralization spread across the Akie property to ten.

Based on the 2017 drill results a couple of recommendations can be made for future exploration programs on the Cardiac Creek deposit:

1. Select target areas of the deposit are recommended for infill drilling. Given the results from the central area of the deposit, the area down-dip of hole 137 is recommended for drilling. Further infill drilling near hole 143 is also recommended. These target areas are outlined in Figure 10-28;
2. The deposit remains open both along strike and down-dip. Drilling is recommended in these directions to expand the known limits of the deposit despite the depth to targets. The area along strike and down-dip of hole 42 would be given a high priority given the high-grade character of the Footwall Zone area. These target areas are outlined in Figure 10-28; and
3. Future prospecting is recommended along the contact between the Kwadacha Limestone and Silurian Siltstone, if exposed, on the Akie property.

Figure 10-28: Proposed Target Locations on the Cardiac Creek Deposit



Source: ZincX (2018)

10.5 Sampling Method and Approach

Work completed on the Akie property is described in various assessment and internal reports. A review of these reports suggests that rock samples collected from the property were either random grab samples or chip samples over a specific width. With respect to the Inmet Mining drill core, an examination of the core remaining on the property indicates that only the mineralized intervals were split and sampled. These intervals were removed from the property and stored in a Vancouver warehouse. When ZincX acquired Ecstall Mining, any remaining mineralized intervals were returned to the property. An examination of drill logs indicates that the core was sampled in intervals ranging from 0.20 m and 2.50 m. The length of sample intervals appears to have been determined by the amount and type of sulphide present; shorter intervals were taken within the massive sulphide zone.

The following description of the sampling method and approach was provided by Nick Johnson, Project Geologist for ZincX.

From 2005 to 2017, ZincX implemented the following stringent procedures with regards to the sampling methodology and preservation of the sampling record:

1. The drill core is delivered by air in bundles of 8 to 12 core boxes in a steel mesh cage to prevent loss of core during flight;

2. A geo-technician prepares the boxes of core for the geologist by measuring the “from” and “to” down-hole distance of each box marked on the upper left-hand corner and bottom right-hand corner of each box. This information, including the box number and drill hole number are recorded on an aluminum tag and stapled to the left hand-side of each box;
3. The geologist records his or her observations on a predefined Excel worksheet with drill-log headings such as lithology, mineralization, structure, RQD, alteration, sampling, etc;
4. Sampling is at the discretion of the geologist who is instructed to sample all observed exhalative mineralization and any other observed features of interest for exploration purposes. Sample boundaries must conform to lithological boundaries;
5. Sampling is generally restricted to a minimum of 30 cm and a maximum of 1.50 m. The beginning and end of a sample are marked with a lumber crayon. Sample boundaries and sample numbers are marked with a permanent marker on the wooden divider of the core box just above the sample. Sample boundaries are also marked by an aluminum tag stapled to the core box. The sample number, from and to distance, and project name are recorded on a paper sample tag; the sample number is also recorded on an aluminum tag and both are stapled to the core box at the beginning of a sample;
6. QA/QC procedures are in place during sampling of the drill core. A series of standards, blanks and duplicates are inserted in the sample stream every ten samples. Each sample has its own sample number;
7. Once all of the geological observations and sampling have been recorded, the core boxes are then photographed to obtain a visual record of the drill core as well as the samples collected. The photographs are taken before the core is cut;
8. The remaining paper sample stubs are kept and stored for record purposes;
9. A rock saw is used to cut the sampled core perpendicular to the dominant fabric. One half is returned to the core box and the other half is placed in a polyurethane sampling bag;
10. All samples are double-bagged due to the fissile nature of the drill core which produces sharp edges along breaks and fractures. Samples are double-bagged to avoid cross contamination during transport; and
11. The sample tag is placed in the outer bag to maintain legibility and prevent deterioration. Each sample bag is then sealed using a plastic security zap-strap.

In the authors’ opinion, the core handling, logging, sampling and core storage protocols in place at the Akie Project meet or exceed common industry standards, and the authors are not aware of any drilling, sampling or recovery factors that could materially impact the accuracy and reliability of these results.

11 Sample Preparation, Analyses and Security

Assessment reports reviewed by the author indicate that the 1994 to 1996 analytical work was completed at International Plasma Laboratory Ltd. (IPL) in Vancouver, BC. These reports also include copies of the original assay certificates and a description of the analytical procedures used by IPL. The author believes that sample preparation and sample security were done in an appropriate manner, following industry best practices applicable at the time.

IPL is officially registered with and certified by the BC Ministry of Environment, Lands and Parks (BCMOE) and the Canadian Association for Environmental Analytical Laboratories (CAEAL). IPL's analytical procedures comply with the applicable requirements of the BCMOE, Environment Canada, American Society for Testing and Materials (ASTM), American Water Works Association (AWWA) and US Environmental Protection Agency (USEPA).

Standard sample preparation for rock samples involves logging the sample into the laboratory sample tracking system, drying, crushing and pulverizing the entire sample so that greater than 80% passes a 75-micron screen. Trace elements are determined by leaching a sample aliquot in aqua regia with an analysis by inductively coupled plasma (ICP) emission spectrometry and mass spectrometry. IPL maintains an internal quality-control program, including the use of blank, duplicate and standard samples inserted into the sample stream. The author believes that the IPL sample preparation and analytical methods conform to reasonable data verification controls.

Analytical work for the 2005 to 2008, 2010 to 2011, and 2013 drilling programs was completed by Acme Analytical Laboratories (Vancouver) Ltd. (AcmeLabs). In 2012, AcmeLabs was acquired by Bureau Veritas Minerals and, in 2014 it transitioned to Bureau Veritas Commodities Canada Ltd. In March 2014, Bureau Veritas Commodities Canada introduced an integrated coding system for sample preparation and analytical packages. In January 2015, the Vancouver branch of AcmeLabs was rebranded as Bureau Veritas Mineral Laboratory (Bureau Veritas). The analytical work for the 2013 to 2017 drilling programs was completed by Bureau Veritas. Vancouver's Bureau Veritas laboratory is ISO 9001:2008 and ISO/IEC 17025:2005 certified.

Robert Sim visited the Akie property on three occasions; 16 to 17 October 2007, 18 to 20 September 2013, and 8 to 9 August 2017 during which he reviewed the drill core and data recording practices. The area above the Cardiac Creek deposit was observed from a helicopter and several drill pads were inspected and core drilling was observed in Hole A-13-109. The site visits included a detailed review of the data stream from logging to database entry, to section plotting and, finally, a review of the information with respect to the surrounding geologic interpretation. Mr. Sim found the camp and facilities clean and well-organized. Site personnel were found to follow an effective and methodical approach to processing the drill core. Mr. Sim also inspected the core sampling facility and equipment which was found to be clean, organized and in good working condition. He indicated that ZincX activities were, and continue to be, conducted in a professional manner and that the reviewed equipment and practices followed accepted industry standards and practices.

The following subsections describe the chain of custody / security, sample preparation and analytical procedures for the 2005 to 2017 drilling programs.

11.1 Chain of Custody and Security

Before samples are shipped from the exploration camp to the lab, individual samples are laid out in consecutive order. Samples, five to a bag, are placed into rice bags and sealed with a plastic zap-strap and security tag. The laboratory's address and phone number, the expeditor's address and phone number, and the sample sequence and bag number are recorded on the outside of the rice bag. The contents of each bag and the security tag number are recorded on a spreadsheet. The lab submission form documents the submitted samples and the desired analytic packages. Both the lab submission form and sample tracking sheets are placed in the first bag of each shipment. Separate copies are emailed to a lab representative and digital or hard copies of these forms are kept for record-keeping purposes.

Shipments are backhauled via the grocery truck (Kwadacha Nation Gautier Ventures Freight Services) to the project's expeditor located in Mackenzie, BC. In Mackenzie, the samples are placed on a wooden pallet, shrink-wrapped, and held until pickup. The samples are then shipped to Vancouver's Bureau Veritas lab using bonded transport contractors, such as Bandstra Transportation Systems Ltd., and/or Van-Kam Freightways Ltd. The tracking numbers for each shipment are recorded and given to the Akie site personnel for record-keeping purposes. Bureau Veritas records the delivery data which is available for review by ZincX personnel using Bureau Veritas' "Web Access" (an online database containing searchable analyses information). This information includes delivery date, expected date of completion, sample preparation method requested, analyses requested, etc.

All procedures are carefully implemented and meet or exceed industry standards for collection, handling and transport of drill core samples.

11.2 Sample Preparation and Analyses

Upon delivery of the samples to Bureau Veritas' Vancouver lab, the samples are prepared before they are crushed and analyzed. The preparation method is as follows:

1. After receiving the samples by bonded carrier, the shipment is initially inspected for completeness; and
2. Samples are then sorted and inspected for quality of usefulness. This includes determining the quantity and condition of each sample. Pulps samples are inspected for homogeneity and fineness.

Drill core samples are then prepared for analysis using the Bureau Veritas PRP70-250 sample preparation method. Under the newly integrated coding system, this method replaces Acme's code R200-250. The PRP70-250 method is as follows:

1. Each drill core sample is crushed in a jaw crusher to 70% passing 10 mesh (2 mm). Between each routine sample, the crusher is cleaned with a brush and compressed air;
2. Samples are homogenized and split to obtain a 250 g split using a riffle splitter;
3. The 250 g split is then pulverized to 85% passing 200 mesh (75 microns). The crusher and pulverizer are cleaned with a brush and air compressor between each routine sample. A granite-quartz wash is used to scour the equipment following any high-grade samples, between any changes in rock colour, and at the end of each file; and

4. Granite-quartz is crushed and pulverized as the first sample in sequence and is carried through to analysis.

After the samples have been prepared, three separate analyses are then completed: Bureau Veritas AQ270/AQ371 package (previously Acme Group 7AR/7AX), LF301 for Ba package (previously Acme Group 4A-Ba), and the SPG01 package (previously Acme Group 8 SG). Each of these analyses is briefly summarized here:

AQ270/AQ371 Package

The Acme Group 7AR package used in 2005 and 2006 provided assay data for 24 elements including zinc, lead and silver with no known upper detection limits. The Acme Group 7AX package used since 2007 provided assay data for 34 elements at a lower detection limit, including lead, zinc, silver and numerous trace elements. The Acme Group 7AX package has upper detection limits for zinc at 200,000 ppm (20%) and lead at 40,000 ppm (4%). These samples are automatically rerun using the Acme Group 7AR which provides the value in excess of these limits. Under the newly integrated coding system, the Acme Group 7AX/7AR analytical package has become Bureau Veritas AQ270/AQ371. The package has changed in name only.

The methodology remains unchanged and the technique is as follows:

1. Prepared samples (1 g) are digested for one hour in a hot-water bath with a modified aqua regia solution consisting of equal parts concentrated HCl, HNO_3 and H_2O;
2. Samples are made up to volume with dilute HCl in a Class A volumetric flask;
3. Samples are then analyzed using an ICP atomic emission spectrometer and/or ICP mass spectrometer; and
4. Any high-grade samples are reweighed at lower weight to accommodate analysis up to the 100% upper limit.

LF301 for Ba Package

Under the newly integrated coding system, the Acme Group 4A-Ba analytical package has become Bureau Veritas LF301-Ba. This package provides litho-geochemical data for all major oxides and is used to obtain accurate barium values in the drill core samples. The general insolubility of barium renders other analytical techniques ineffective. The package has changed in name only.

The methodology remains unchanged and the technique is as follows:

1. Prepared samples are mixed with a lithium metaborate / tetraborate flux;
2. Crucibles are fused in a furnace;
3. Cooled beads are then dissolved in ACS (American Chemical Society)-grade nitric acid; and
4. Samples are analyzed using an ICP-emission spectrometer.

SPG01 Package

Under newly integrated coding system, the Acme Group 8-SG analytical package has become Bureau Veritas SPG01. This package provides the specific gravity of each drill core sample which is conducted on the pulverized pulps. The package has changed in name only.

The methodology remains unchanged and the technique is as follows:

1. A split of dry pulp is collected from a sample and weighed to a Class A volumetric flask;
2. The flask and pulp are carefully weighed on a top-loading balance;
3. The weights are measured and recorded; and
4. Specific gravity is then calculated for the sample.

11.3 QA/QC of Analytical Data

ZincX maintained a strict QA/QC policy regarding drill core sampling. Standards, blanks and duplicates were inserted into the sample stream at a rate of one in thirty samples and given their own sample number.

During the 2005 drilling program, blank material was obtained from a local outcrop which contained no visible signs of mineralization. In the 2006 to 2008, 2010 to 2011, and 2013 to 2017 drilling programs, blank material was purchased from WCM Minerals, Burnaby BC, Canada. Standard reference material was also purchased from WCM Minerals. The 2017 standard was purchased from CDN Resource Laboratories, Langley, BC, Canada. A total of eleven certified standards for zinc, lead, silver and copper have been used in the drilling completed at the Akie property (i.e., PB109, PB110, PB111, PB112, PB118, PB123, PB129, PB130, PB136, PB145, and CDN ME-1306). Core duplicate samples were obtained by sawing one-quarter core splits from the sampled interval. Due to the variability observed in the core duplicates from 2006 and 2007, it was recommended that pulp and coarse duplicates be taken as a split from the pulp and reject portion of a sample during sample preparation at Acme labs. This recommendation was implemented for all subsequent drilling programs after 2007.

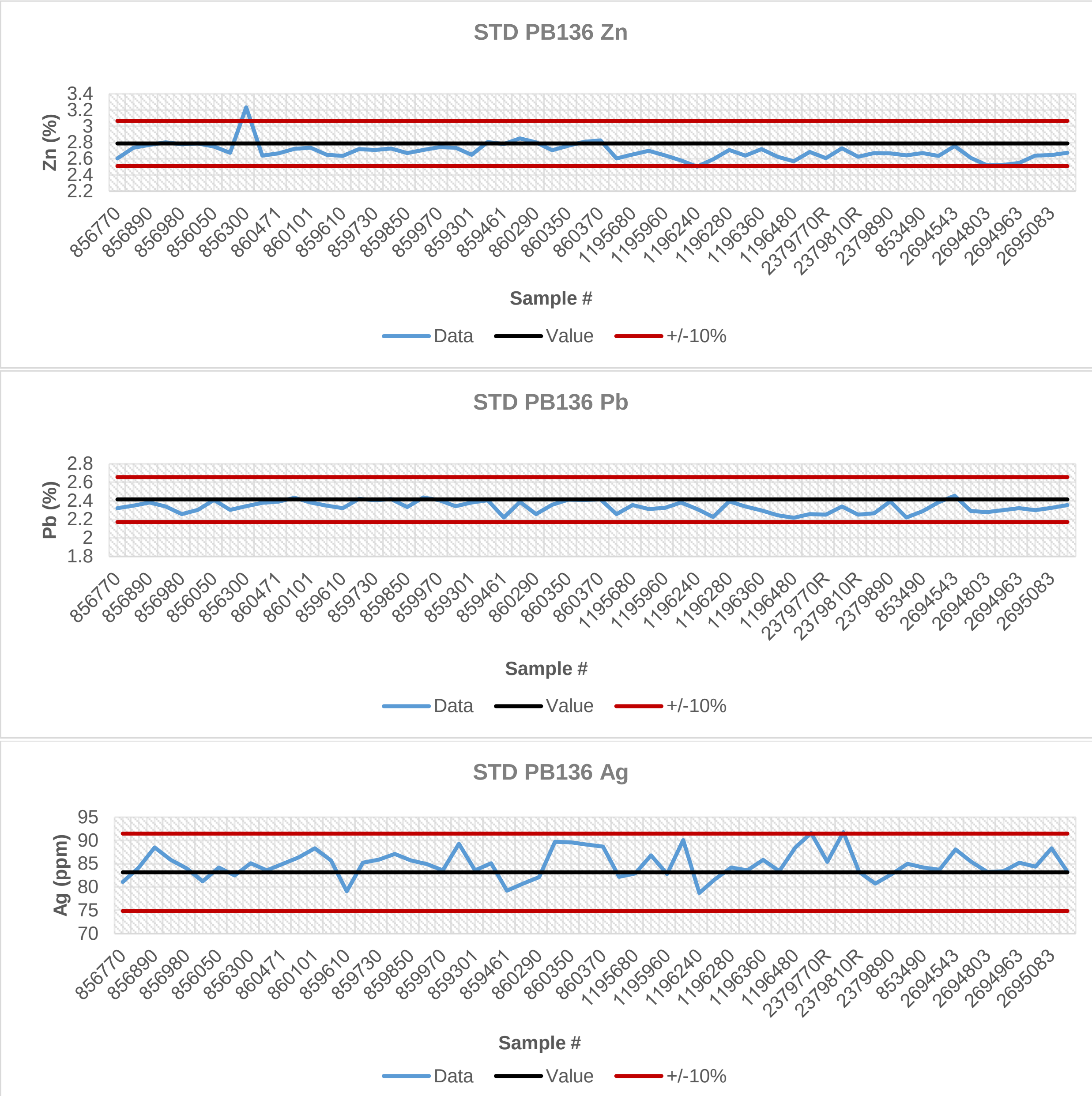
Due to the significant contrast in assay results of the Cardiac Creek Zone between the 2005 Canada Zinc Metals drill program and the historical Inmet Mining drill programs, Canada Zinc Metals had pulp duplicate samples taken from all “significantly mineralized” intervals and had them re-analyzed at Global Discovery labs in Vancouver, BC. Global was a lab run by Teck Cominco Corp. The re-analysis by Global Discovery labs demonstrated the validity of the results in 2005 and 2006, showing similar results for lead, zinc and silver, and this protocol was subsequently discontinued in early 2007. The comparison graphs between the two labs can be seen in the previous technical report by MacIntyre and Sim (2008). In 2013, check assays were re-initiated and approximately 10% of all samples submitted to Bureau Veritas were also sent to ALS’s Vancouver laboratory.

11.3.1 Standard Reference Material (SRM) Performance

The performance of standard reference material (SRM or standards) is evaluated using the following criterion: 90% of the results must fall within ±10% of the accepted value for the assay process to be in control. Results are presented using statistical process control charts. In the control chart, the “accepted” or average value is indicated by a green horizontal line. Control limits at ±10% of the accepted value are indicated by red lines above and below the line showing the accepted value. The assay results for the standard appear on the chart are indicated by a blue line. Examples from standards PB136, PB145 and CDN ME1306 are shown in Figure 11-1 and Figure 11-2, respectively.

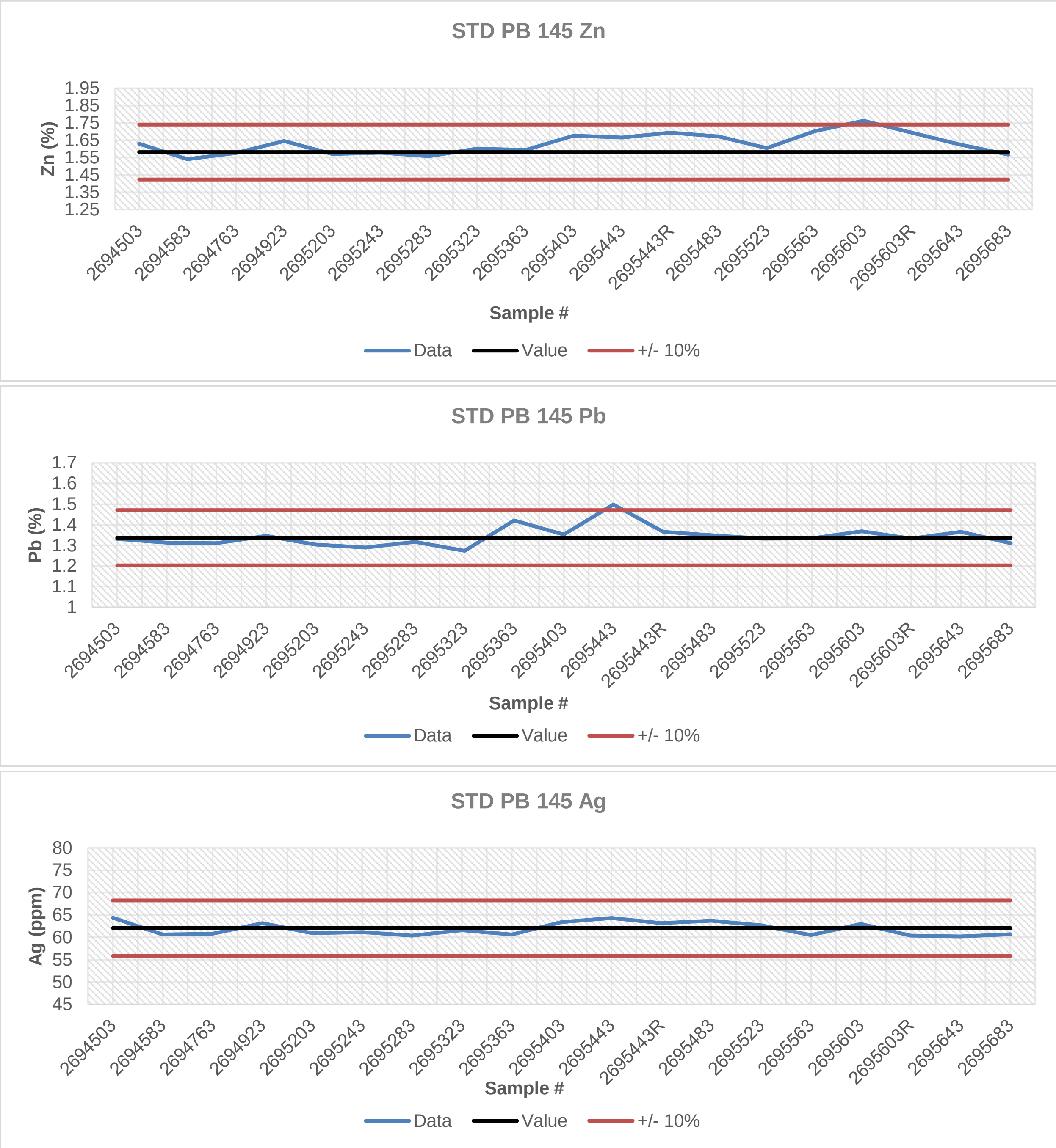
The results for all standards fall within the control limits more frequently than the prescribed rate, showing that no systematic assaying problems exist.

Figure 11-1: SRM Results from STD PB136 for Zinc, Lead and Silver



Source: ZincX (2018)

Figure 11-2: SRM Results from STD PB145 for Zinc, Lead and Silver



Source: ZincX (2018)

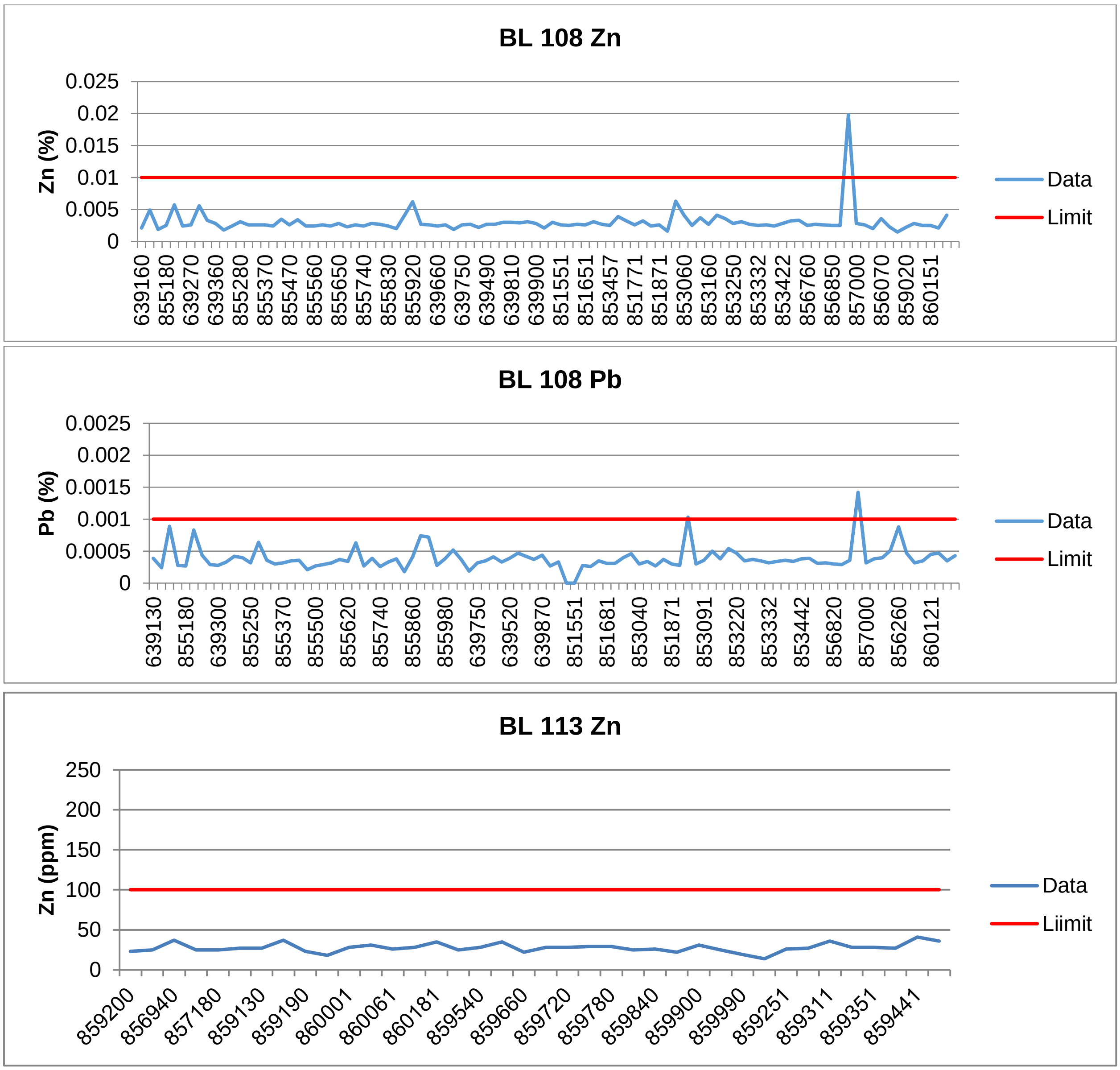
Figure 11-3: SRM Results from CDN ME-1306 for Zinc, Lead and Silver

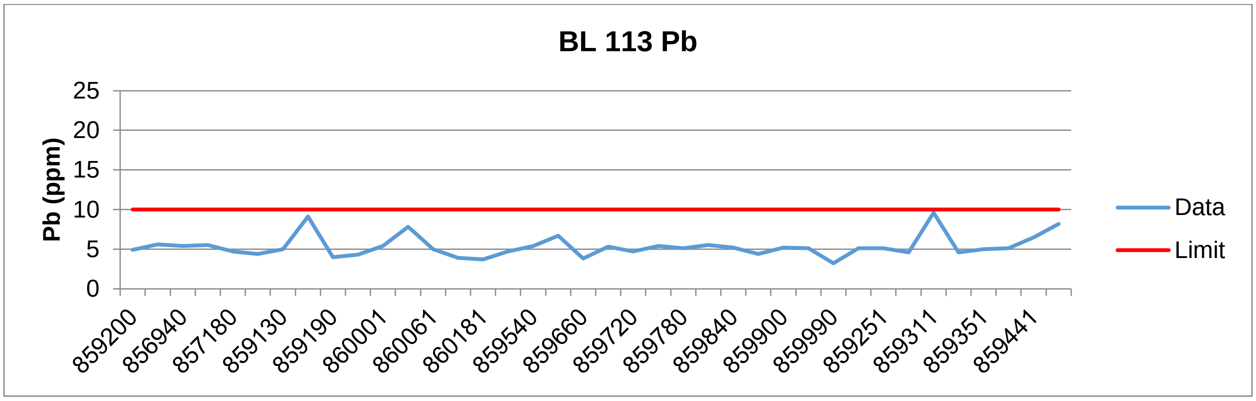
Source: ZincX (2018)

11.3.2 Sample Blank Performance

Control results exceeded the control limit for the blank material assays less than 5% of the time. An example of the blank sample performance is shown in Figure 11-4. During the 2005 drilling program, locally sourced rock was used as blank material but was found to be non-sterile. Since then, a more appropriate blank material has been purchased and used in the QA/QC process.

Figure 11-4: Blank Results from BL 108 and BL 113 for Zinc and Lead



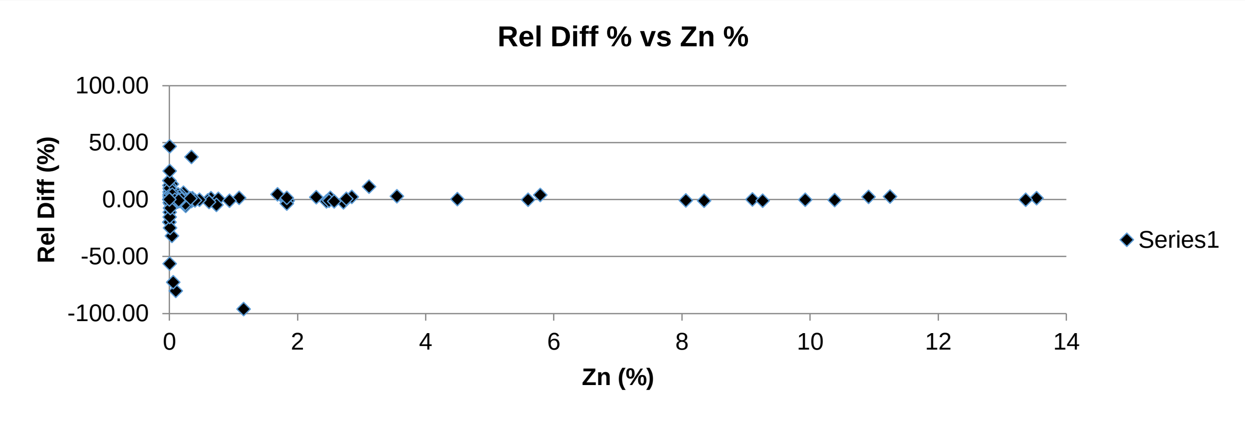


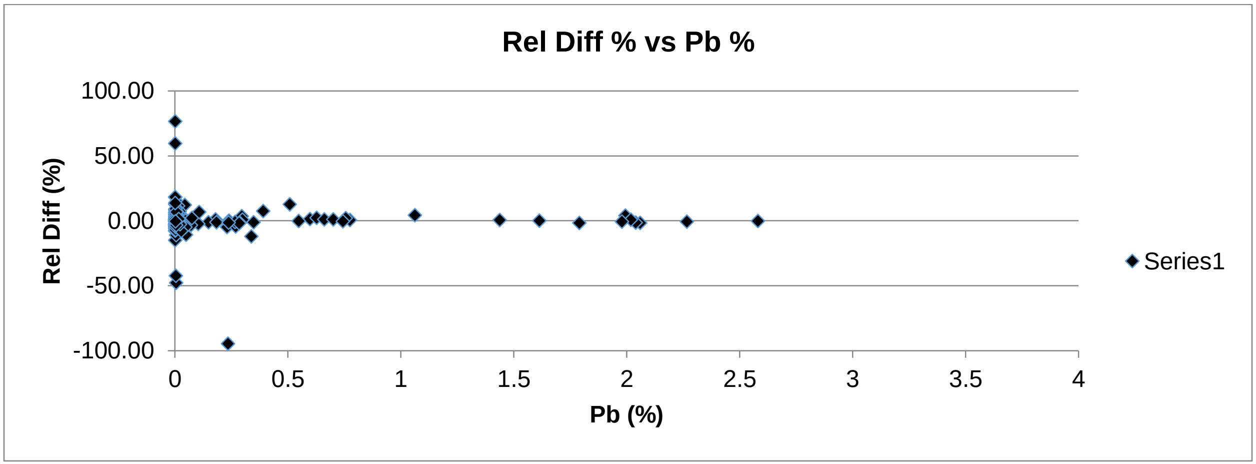
Source: CZM (2017)

11.3.3 Duplicate Sample Performance

The results of the coarse (reject) and pulp duplicates both demonstrate a relatively erratic distribution of variability that diminishes as the grade of samples increases (Figure 11-5 and Figure 11-6). The pulp duplicates showed an average relative difference of 0% Zn, 1% Pb and 1% Ag. The coarse duplicates showed an average relative difference of 0% Zn, 1% Pb and 2% Ag. These results are considered acceptable for both types of sample duplicates.

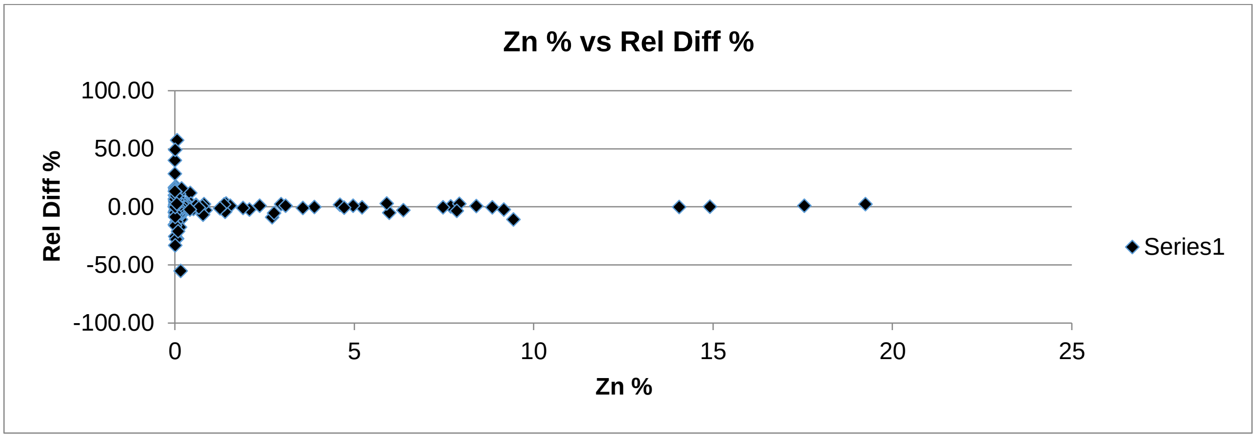
Figure 11-5: Relative Difference of Pulp Duplicate Samples for Zinc and Lead

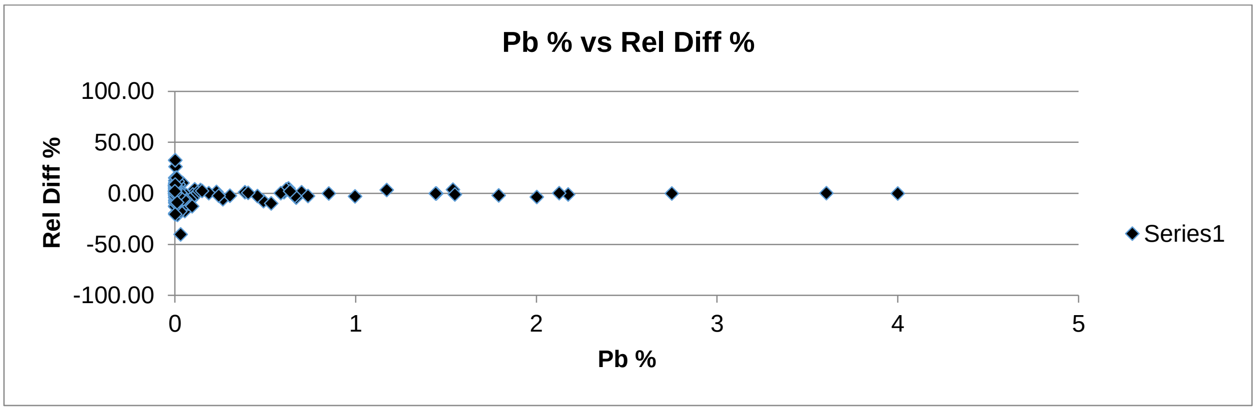




Source: ZincX (2018)

Figure 11-6: Relative Difference of Coarse Duplicate Samples for Zinc and Lead



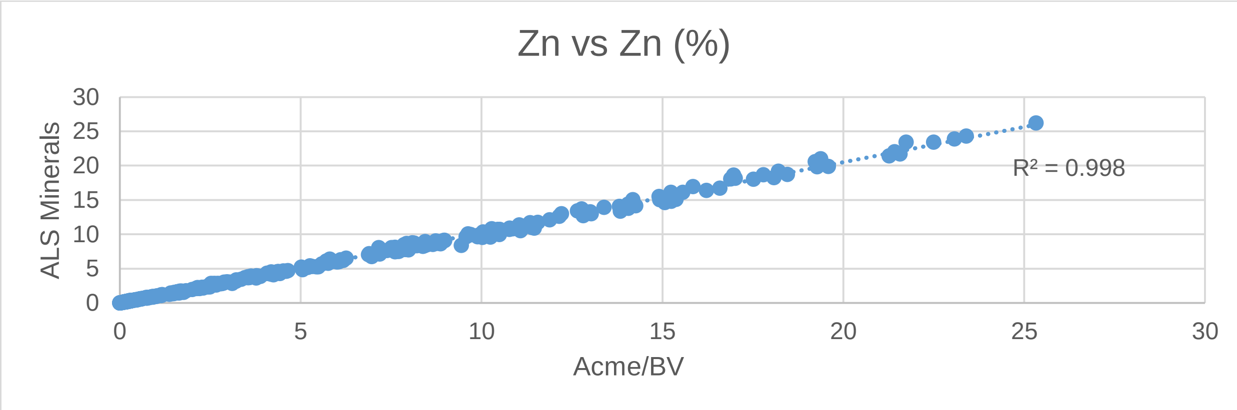


Source: ZincX (2018)

11.3.4 Check Assays

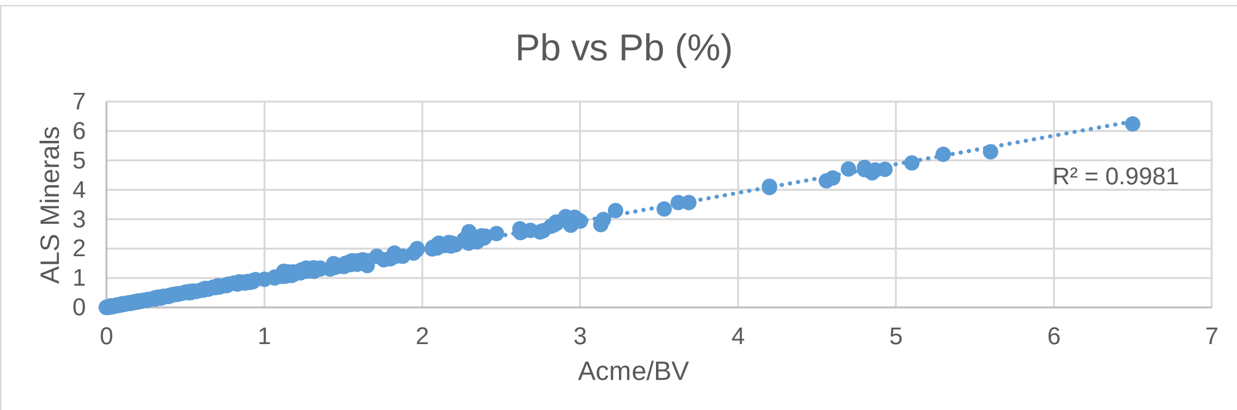
Independent confirmation of the analyses was evaluated through testing of duplicate samples at an outside umpire laboratory. Following the completion of each drilling campaign, approximately 10% of the submitted samples were resubmitted to ALS Canada Ltd. in Vancouver. Pulps of the submitted samples were re-homogenized via light pulverising. Analysis was completed using the ALS ME-OG46 assay package for zinc, lead and silver. A prepared sample of 0.4 grams is digested with concentrated nitric acid for 90 minutes in a graphite heating block. The solution is then diluted with concentrated hydrochloric acid before cooling to room temperature. The samples are then diluted in a volumetric flask with de-mineralized water and analyzed using ICP-ES. The re-analysis conducted by ALS demonstrates the validity of the results between 2013 and 2017, showing similar results for zinc, lead, and silver. The comparison graphs of zinc and lead grades received from the two labs can be seen in Figure 11-7 and Figure 11-8, respectively.

Figure 11-7: Comparison Graph for Zinc



Source: ZincX (2018)

Figure 11-8: Comparison Graph for Lead



Source: ZincX (2018)

11.4 Conclusions

Results from the standard reference material indicate that the zinc, lead and silver assay processes are under sufficient control to produce reliable sample assay data for a resource estimate. Blank results indicate no contamination in the assay process. Coarse reject results confirm that the sample preparation protocol is reliable. Comparisons of inter-lab pulp duplicates show good results with most differences attributed to samples in the low-grade range.

The Akie deposit sampling and assaying program produces sample information that is accurate and reliable and meets industry standards for zinc, lead and silver. The assay results are sufficiently accurate and precise for use in resource estimation.

12 Data Verification

12.1 Geology, Drilling and Assaying

The source of some of the data for the Akie Mineral claims has been the historical work reported by previous operators, which includes geochemical surveys, geophysical surveys and diamond drilling. Examination of the analytical results presented in publicly available assessment reports and in a previous compilation report (Baxter, 1996c) suggests that quality assurance was performed to the best practice standards of the day.

On 26 August 2005, a sample of thin-bedded massive sulphide was collected by D.G. MacIntyre from the Cardiac Creek showing; this was submitted to Acme Analytical Laboratories in Vancouver for analysis. The sample assayed 23.8% Zn, 4.6% Pb and 29 g/t Ag. A copy of the analytical certificate is contained in the previous technical report (MacIntyre, 2005). These values are similar to those reported elsewhere for the Cardiac Creek showing (e.g., by Baxter in 1995 and 1996) and confirm the high-grade nature of the massive sulphide mineralization.

As part of the data validation, approximately 5% of the sample data located in the vicinity of the resource model were randomly selected for manual validation of the assay result back to the original (assay certificate) data source. No significant errors were identified. Manual validation indicates that the assay results in the database are free of errors that could materially impact the estimate of mineral resources.

During the site visits, the QP visually correlated the sphalerite and galena contents of drill core with the reported assay grades for a random selection of drill holes. No discrepancies were noted. The sampling protocols used to develop the ZincX sample database follow accepted industry standards and have been verified through an extensive QA/QC program.

A portion of the database is derived from drilling data generated by Inmet from 1994 to 1996. Although this includes a total of 29 holes in the database, only nine of these holes are located in the vicinity of the resource model. The collar locations of these holes have been verified in the field by ZincX site personnel. The drill core from these holes is stored on the Akie property, and the remaining mineralized intervals have been transported back to the Akie property and stored in the on-site core racks. There are no assay certificates available for the Inmet Mining data. However, the mineralized intervals from Inmet Mining drill holes were visually reviewed and validated in 1996 by Robert Sim who, at that time, was an employee of Inmet Mining at the Vancouver office. All drilling activities conducted by Inmet Mining between 1994 and 1996 were conducted in a professional manner and the resulting data can be considered valid and reliable.

The data verification process indicates that the database is sound and reliable for the purposes of resource estimation.

12.2 Metallurgy

Metallurgical test data was verified through a review of previous studies and testwork reports and an analysis of the new results from the 2017 metallurgical testwork program. Any studies and reports referred to were thoroughly reviewed and align with the PEA metallurgical design and analysis in this report. All metallurgical data was verified and is adequate for this Preliminary Economic Assessment Technical Report as required by NI 43-101 guidelines.

12.3 Mining

All mining data was verified and is adequate for this Preliminary Economic Assessment Technical Report as required by NI 43-101 guidelines.

13 Mineral Processing and Metallurgical Testing

13.1 Introduction

Historical metallurgical testing was performed on Cardiac Creek samples by SGS Mineral Services (SGS) in 2005, 2007 and 2008, and G&T Metallurgical Services Ltd. (G&T) in 2009 and 2010. The most recent test program was completed by Base Metallurgical Laboratories Ltd. (Base Met) and was used as the basis for the process design and recovery method outlined in Section 17. A full breakdown of the results for each test program can be found in the following reports:

- SGS Lakefield Research Limited, "*The Recovery of Lead, Zinc and Silver from the Akie Deposit in British Columbia*", Project No. LR 11068-001, (Issued: 15 September 2005);
- SGS Mineral Services, "*The Recovery of Lead and Zinc from Ore from the Akie Deposit*", Project No. 11068-004, (Issued: 24 January 2007);
- SGS Mineral Services, "*The Metallurgical Response of a Composite from the Akie Deposit*", Project No. 11629-001, (Issued: 2 April 2008);
- G&T Metallurgical Services Limited, "*Preliminary Metallurgical Assessment of Selected Mineralized Samples from the Cardiac Creek Deposit*", Project No. KM2139, (Issued: 21 May 2009);
- G&T Metallurgical Services Limited, "*Preliminary Metallurgical Assessment of the Akie Deposit, British Columbia, Canada*", Project No. KM2530, (Issued: 5 February 2010); and
- Base Metallurgical Services Limited, "*Preliminary Metallurgical Assessment Cardiac Creek*", Project No. BL0148, (Issued: 14 March 2018).

Based on the results from Base Met (2018), a dense media separation (DMS) circuit will be used to pre-concentrate the sulphide minerals, rejecting 25% of the material as waste prior to Pb and Zn sequential flotation. Saleable Pb and Zn concentrates can be produced with a primary grind size of 80% passing (P_{80}) 56 µm, and rougher concentrate regrind sizes of 10 µm for Pb and 15 µm for Zn. For the FF Global Composite, locked cycle flotation test results achieved recoveries of 46.2% Pb and 88.8% Zn at concentrate grades of 45.1% Pb and 52.4% Zn.

13.2 Summary of Historical Test Work

The following section summarizes the main results from the metallurgical test programs conducted by SGS Mineral Services and G&T Metallurgical Services between 2005 and 2010.

13.2.1 SGS Lakefield Research Test Program (2005)

Approximately 30 m of drill core sample, totaling 71.5 kg of material, was tested to evaluate mineralogy, gravity and heavy liquid separation, and Pb, Zn rougher / cleaner flotation. The material had a head grade of 1.3% Pb, 5.7% Zn and 12 g/t Ag. At a P_{80} grind size of 88 µm, the sample showed poor liberation, with the minerals of interest, sphalerite and galena, being attached and locked in composite grains with pyrite and gangue minerals (quartz, barite and mica). Gravity separation was not successful at pre-concentrating the sulphide minerals, while heavy liquid separation achieved 34% waste rejection with metal losses of 5%

Pb, 2% Zn and 4% Ag. Pre-float, rougher and cleaner flotation, at a P_{80} grind size of 50 µm, was completed on the DMS product. Results indicated that sequential Pb, Zn flotation could achieve recoveries of 51% Pb and 63% Zn in concentrate grades of 31% Pb and 49% Zn. Further optimization of reagents and grind size was recommended moving forward.

13.2.2 SGS Mineral Services Test Program (2007)

Core samples from three drill holes were shipped to SGS and compiled into three composites with head grades ranging from 2.18% to 3.81% Pb, 8.69% to 14.3% Zn, and 23.9 g/t to 36.2 g/t Ag. The test program focused on heavy liquid separation and Pb, Zn sequential flotation. At a 5/8" crush size and an SG cut point of 3.00, DMS results for the three composites averaged 21% waste rejection, with metal losses of 4% Pb and 1% Zn. Seventy percent of the carbon in the sample was removed as waste. Comparative flotation testing on fresh and pre-concentrated feed indicated that pre-concentration is potentially required to achieve adequate Pb flotation performance. A locked cycle flotation test on DMS product was completed at a P_{80} grind size of 63 µm with Pb and Zn regrind sizes of 15 µm and 21 µm respectively. The test achieved recoveries of 64.3% Pb and 79.6% Zn at concentrate grades of 50.2% Pb and 50.9% Zn. Pb and Zn concentrates only recovered 36.2% of the Ag. The report hypothesized that the Ag may be associated with pyrite and recoverable with cyanidation.

13.2.3 SGS Mineral Services Test Program (2008)

Representative feed samples from seven drill holes were shipped to SGS and eleven composites were prepared for testing, including seven individual drill hole composites. A main composite was also created using six of the drill holes, while three zonal composites, labelled North, South and Central, were produced by combining material from two or three of the drill holes. The test program assessed heavy liquid separation and how a pre-concentrated product affected grinding and flotation.

All eleven composites were subjected to heavy liquid separation at SG cut points of 2.7, 2.9 and 3.1. An SG between 2.7 and 2.9 was found to produce the best results. Overall, heavy liquid separation rejected 28.5% to 46.4% of the material as waste, while incurring metal losses of 3.8% to 9.2% Pb, 1% to 2.9% Zn and 1.5% to 11.5% Ag.

Bond ball mill work index testing was carried out on both the sinks (product) and floats (waste) to determine how a DMS circuit would impact the grinding circuit. The sinks sample measured 15.9 kWh/t, while the floats sample measured 18.4 kWh/t.

Four flotation tests were completed on the main composite. Primary P_{80} grind sizes between 80 µm and 156 µm were evaluated along with the effect of collector dosage. A P_{80} grind size of 102 µm produced the best Zn flotation results, while a higher collector dosage increased Zn recovery. Test conditions were not optimized before the test program was halted.

13.2.4 G&T Metallurgical Services Test Program (2009)

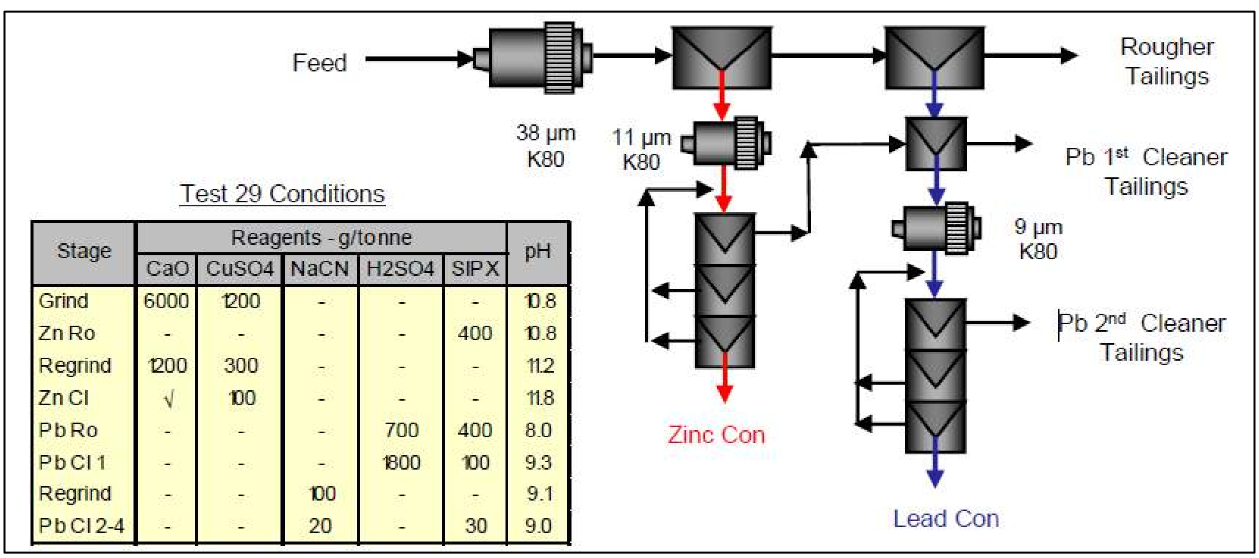
A single composite sample, measuring 1.59% Pb, 8.08% Zn and 12 g/t Ag, was created to further investigate mineralogical characteristics and flotation response. Selective leaching indicated that 40% of the Pb was present in non-sulphide species, making high Pb flotation recovery difficult. The sample also contained 1.4% total organic carbon (TOC), which could interfere with flotation.

Sample was ground to a P_{80} of 100 µm and analyzed for mineral composition and fragmentation. Pyrite was identified as the primary sulphide species, encompassing 20.6% of the sample. The two main minerals of interest, sphalerite and galena, comprised 12.1% and 1.3% of the sample respectively. At the P_{80} grind size of 100 µm, mineral liberation was low, with 17% of the galena and 20% of the sphalerite fully liberated. About 80% of the galena remained in very low-grade multiphase particles, while un-liberated sphalerite was sufficiently rich enough to recover with flotation, including 25% of the sphalerite being associated with pyrite. The results indicated that a fine primary grind would be required to improve Zn recovery, while high Pb recovery would remain a challenge.

Pb, Zn sequential flotation was assessed through rougher and cleaner flotation testing. A pre-flotation stage to remove organic carbon was tested but was found to offer no metallurgical advantage. Rougher test results showed that even at a fine P_{80} grind size of 38 µm, Pb rougher recovery remained low with only 51% of the Pb reporting into a 20% mass pull. In contrast, 85% Zn rougher recovery was observed in a 20% mass pull. At a P_{80} of 72 µm, sequential Pb, Zn cleaner flotation was unable to generate marketable Pb and Zn concentrates. In response, several different flowsheet configurations were investigated, including generating only a Zn concentrate and reversing the flotation process to recover a Zn concentrate first and then a Pb concentrate.

Based on the results from rougher / cleaner flotation testing, locked cycle testing was carried out at a P_{80} grind size of 38 µm. The test flowsheet is presented in Figure 13-1. The reverse Zn, Pb sequential flotation process was able to produce a saleable Zn concentrate, recovering 75% of the Zn at a concentrate grade of 57.3% Zn. The Pb circuit however, remained problematic. A concentrate grade measuring 31.1% Pb only recovered 55% of the Pb. The Ag recovery remained low with 21% being recovered in the two concentrates. The report hypothesized that the Ag responded in a similar fashion to the pyrite and was depressed during flotation.

Figure 13-1: KM2139 Locked Cycle Test Parameters



Stage	Reagents - g/tonne					pH
	CaO	CuSO4	NaCN	H2SO4	SIPX	
Grind	6000	1200	-	-	-	10.8
Zn Ro	-	-	-	-	400	10.8
Regrind	1200	300	-	-	-	11.2
Zn Cl	√	100	-	-	-	11.8
Pb Ro	-	-	-	700	400	8.0
Pb Cl 1	-	-	-	1800	100	9.3
Regrind	-	-	100	-	-	9.1
Pb Cl 2-4	-	-	20	-	30	9.0

Source: G&T (2009)

13.2.5 G&T Metallurgical Services Test Program (2010)

Approximately 60 kg of sample, representing a high-grade zone of the deposit, was shipped to G&T to determine how a higher feed grade would affect metallurgical performance. A high-grade composite, measuring 3.7% Pb, 15.8% Zn, 26 g/t Ag and 1.7% C, was generated to evaluate mineralogy and the performance of Pb, Zn sequential flotation.

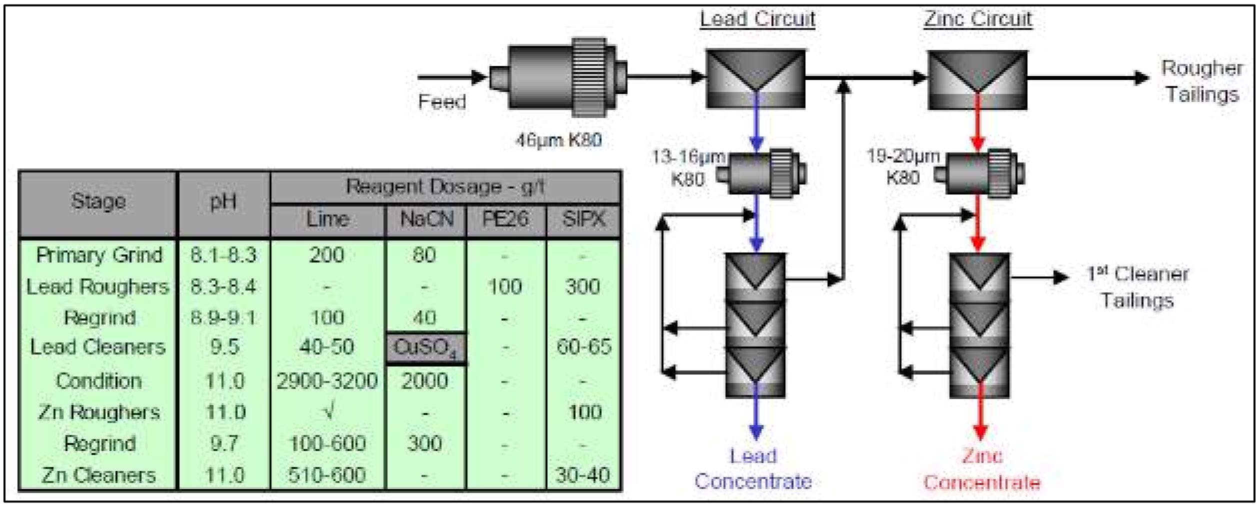
Composite sample was ground to a P_{80} of 61 µm and subjected to mineral composition and fragmentation analysis. The sample contained 24.2% sphalerite and 3.1% galena, with pyrite measuring 20.9%. Using weak acid digestion techniques, it was determined that 25% of the Pb in the sample was associated with non-sulphide species. Fragmentation results were similar to those reported in G&T (2009). At a P_{80} grind size of 61 µm, liberation was still poor. Only 13% of the galena and 26.1% of the sphalerite was fully liberated. The remaining galena was locked in complex multiphase particles, while sphalerite was present in sufficient quantities with pyrite and multiphase particles to be recoverable in a rougher flotation concentrate. In both cases, the rougher concentrates will need to be reground to produce clean saleable concentrates.

Rougher flotation tests were carried out to refine test parameters for cleaner and locked cycle testing. Two strategies were tested to address the presence of naturally hydrophobic organic carbon. A pre-flotation stage was found not to be effective; while the addition of a starch-based depressant (PE26) benefited subsequent Pb flotation performance. Using PE26 allowed recovery of galena while minimizing the rougher concentrate mass pull. Sodium cyanide (NaCN) dosage and primary grind size were also analyzed to improve Pb rougher flotation. Primary grind sizes between 46 µm and 61 µm were tested, with only marginally better Pb rougher recovery observed with decreasing grind size. Zn rougher performance responded favorably at a P_{80} grind size of 61 µm, recovering 89% of Zn in a 45% mass pull.

Cleaner flotation tests, at a P_{80} grind size of 61 µm, were conducted to investigate the effect of PE26 dosage and regrind size on concentrate grade and recovery. Results indicated PE26 was best utilized in the Pb rougher circuit, achieving an overall Pb recovery of 52% at a cleaner concentrate grade of 56% Pb. The corresponding Zn concentrate recovered 85% of the Zn in a concentrate grade measuring 51% Zn. Pb cleaner grade and recovery did not benefit from reducing the P_{80} regrind size from 15 µm to 6 µm, while Zn performance suffered at a coarser Zn regrind size. The most favorable results were observed at Pb and Zn regrind sizes of 12 µm to 15 µm and 21 µm to 22 µm respectively.

Based on the results from rougher / cleaner flotation, locked cycle testing was completed using the test flowsheet displayed in Figure 13-2. From two replicate tests, results indicated that conventional Pb, Zn sequential flotation can generate saleable Pb and Zn concentrates. The Pb concentrate achieved average recoveries of 58% Pb at a grade of 49% Pb; while the Zn concentrate recovered 80% of the Zn in a concentrate grade of 54% Zn. The Ag recovery remained poor with 8% reporting to the Pb concentrate, 30% reporting to the Zn concentrate and the remaining Ag being lost in the tailings. Minor element analysis was completed on the Pb and Zn concentrates, with results showing that both concentrates were relatively free of deleterious elements. It was noted that carbon levels in the Pb concentrate could pose a problem if present in the organic form.

Figure 13-2: KM2530 Locked Cycle Test Parameters



Stage	pH	Reagent Dosage - g/t			
		Lime	NaCN	PE26	SIPX
Primary Grind	8.1-8.3	200	80	-	-
Lead Roughers	8.3-8.4	-	-	100	300
Regrind	8.9-9.1	100	40	-	-
Lead Cleaners	9.5	40-50	$CuSO_4$	-	60-65
Condition	11.0	2900-3200	2000	-	-
Zn Roughers	11.0	√	-	-	100
Regrind	9.7	100-600	300	-	-
Zn Cleaners	11.0	510-600	-	-	30-40

Source: G&T (2010)

13.3 Summary of Recent Test Work – Base Met (2018)

In August 2017, a metallurgical test program was commenced at Base Metallurgical Laboratories in Kamloops, BC (Project No. BL0148). Drill core samples from five drill holes were submitted for metallurgical testing in support of this preliminary economic assessment. The test program focused on mineralogy, comminution, DMS and Pb, Zn sequential flotation.

13.3.1 Sample Selection

Drill core from five drill holes (A-17-132, A-17-133, A-17-137, A-17-138, and A-17-141) was shipped to Base Met where five individual drill core composites were created. Composites were selected from the drill holes based on the assay grade and mineralization intersection. Hanging wall and footwall dilution was included with each composite to represent actual mined material. From these composites, a global composite was generated and used for the majority of the test work. The composition of the Global Composite is summarized in Table 13-1.

Table 13-1: Composition of the Global Composite for Base Met (2018)

Composite	Masses by Hole (kg)				
	A-17-132	A-17-133	A-17-137	A-17-138	A-17-141
Main	45.69	42.15	35.48	29.88	17.63
Footwall	2.99	5.03	0.56	3.32	1.71
Hanging Wall	10.69	8.84	2.84	4.26	1.86
Total	59.37	56.02	38.88	37.46	21.19
Global Composite Total					**212.92**

Source: Base Met (2018)

DMS pre-concentration was applied to 132 kg of Global Composite to generate feed for comminution and flotation testing. Material was screened at 1/4" and the coarse material was separated at a SG cut point of 2.80. The sinks, or sulphide product, was combined with the -1/4" fines to produce the Global Composite Flotation Feed sample (FF Global Composite). DMS rejected 23.8% of the mass with metal losses of 2.6% Pb and 0.8% Zn. The process also rejected 30.5% of the carbon. Measured head assays for the Global Composite and FF Global Composite are shown in Table 13-2.

Table 13-2: Head Assays for Base Met (2018) Global Composites

Composite	Pb (%)	Zn (%)	Fe (%)	S (%)	C (%)	Ag (g/t)
Global Composite	1.34	7.45	8.1	14.9	2.06	-
FF Global Composite	1.88	9.50	10.1	18.2	1.87	17.5

Source: Base Met (2018)

Additional carbon assays were performed, indicating that 53% of the carbon in the FF Global Composite was present as organic carbon. Organic carbon is naturally hydrophobic and will adsorb flotation reagents, as well as contaminate concentrates if it is not adequately controlled.

13.3.2 Dense Media Separation

Prior to producing the bulk FF Global Composite sample, DMS amenability testing was carried out on the Global Composite. The material was coarsely crushed and screened at three sizes; 3/4", 1/2" and 1/4". The metal and sulphur content were nearly equally distributed between the four resulting size fractions. The fine -1/4" material was weighed, assayed and put aside, while the three coarse fractions were subjected to heavy liquid separation at SG cut points of 2.77 and 2.95. The results are summarized in Table 13-3.

Table 13-3: DMS Results for the Base Met (2018) Global Composite

Product	Mass (%)	Assay (%)					Recovery (%)				
		Pb	Zn	Fe	S	C	Pb	Zn	Fe	S	C
2.95 SG Sinks, Fines	72.36	1.74	9.6	8.17	19.5	2.0	95.1	97.8	91.3	93.0	66.3
2.77 SG Sinks, Fines	78.29	1.65	9.0	7.82	18.6	2.0	97.6	99.3	94.6	96.3	73.5
2.77 SG Floats	21.71	0.15	0.24	1.62	2.60	2.6	2.4	0.7	5.4	3.7	26.5
Recalculated Feed	100.0	1.32	7.07	6.47	15.1	2.1	100	100	100	100	100

Source: Base Met (2018)

At an SG cut point of 2.95, 27.6% of the feed was rejected in the floats with metal losses of 4.9% Pb and 2.2% Zn. At a lower SG cut point of 2.77, 21.7% of the feed was rejected in the floats with metal losses of 2.4% Pb and 0.7% Zn. All the size fractions performed well, with separation efficiency increasing as the particle size increased. Based on these results, DMS was incorporated into the process.

13.3.3 Mineralogy

Sample of FF Global Composite was ground to a P_{80} of 56 µm and analyzed using QEMSCAN and PMA routine to determine mineral content and mineral fragmentation. A summary of mineral content is presented in Table 13-4. Approximately 61% of the sample consisted of non-sulphide gangue, including quartz, feldspar and barite. The remaining 39% of the sample was made up of the sulphide minerals pyrite, sphalerite and galena. Quartz and feldspar are innocuous and should not interfere with the process. Barite should be noted as it is a heavy mineral and will be recovered along with the sulphides in a DMS circuit.

Table 13-4: Mineral Content for the Base Met (2018) FF Global Composite

Mineral	Composition (%)
Copper Sulphides	0.01
Galena	2.1
Sphalerite	15.5
Pyrite	22.4
Iron Oxides	0.03
Quartz	26.8
Potassium Feldspar	11.2
Barite	9.28
Ewaldite ($BaSrCaCO_3$)	5.12
Norsethite ($BaMgCO_3$)	0.56
Feldspar Albite	1.54
Muscovite / Illite	3.11
Calcite	1.34
Kaolinite (clay)	0.40
Dolomite / Ankerite	0.24
Rutile / Anatase	0.39
Olekminskite / Strontianite	0.14
Apatite	0.16
Other Minerals	0.48

Source: Base Met (2018)

Mineral fragmentation and liberation results for the sample are summarized in Table 13-5. At a P_{80} of 56 µm, only 27% of the galena was fully liberated. Galena was generally finely disseminated in the samples, most often occurring as complex multiphase particles interlocked with pyrite and non-sulphide gangue. The level of galena liberation in this sample is far too low to expect high recovery into a selective Pb rougher concentrate. Sphalerite liberation was much better at 38%, albeit lower than typically required for efficient recovery into a selective Zn rougher concentrate. Unlike galena, the average particle grade of sphalerite was much higher, which should result in good recovery to a rougher concentrate. Regrinding of the rougher concentrate will be required to achieve a high Zn recovery at a high concentrate grade.

Table 13-5: Mineral Fragmentation Results for the Base Met (2018) FF Global Composite

Mineral Status	2-D Mineral Liberation (%)			
	Galena	Sphalerite	Pyrite	Gangue
Liberated	27.4	38.2	26.1	59.8
Binary - Galena		0.6	0.6	0.5
Binary – Sphalerite	4.9		12.5	2.8
Binary – Pyrite	6.8	16.8		19.0
Binary – Gangue	12.3	8.8	34.0	
Multiphase	48.5	35.5	26.8	18.0
Total	**100**	**100**	**100**	**100**

Source: Base Met (2018)

The release curves for both galena and sphalerite exhibited similar trends to previous test programs. For galena, there was no liberated galena in the sample until about 38 µm. At about 5 µm, only about 60% of the galena was liberated. The release curves and low average grade of the galena particles would indicate that even with a very fine regrind size of less than 10 µm, the efficient recovery of a high grade Pb concentrates will be challenging. A target P_{80} regrind size of 10 µm was selected to balance these concerns.

The release curves for sphalerite were better than galena. At about 38 µm, 10% of the sphalerite was liberated. At about 15 µm, the samples had about 30% liberated sphalerite. The grade of the sphalerite particles was also significantly better. A target P_{80} regrind size of 15 µm should result in relatively good concentrate grades and recoveries.

13.3.4 Comminution

A comparison was carried out between Global Composite and FF Global Composite to determine how DMS pre-concentration affected grinding specific energy requirements. Bond ball mill work index tests were completed on both samples at a sieve size of 106 µm. The results are summarized in Table 13-6. By removing some of the hard gangue material, DMS pre-concentration reduced the specific energy requirements for grinding by 3.4%. Both samples can be classified as moderately hard.

Table 13-6: Bond Ball Mill Work Index Results for Base Met (2018) Global Composites

Composite	Sieve Size (µm)	Feed Size, F_{80} (µm)	Product Size, P_{80} (µm)	Grams per Revolution (g)	Bond Ball Mill Work Index (kWh/t)
Global Composite	106	2,290	77	1.03	**17.5**
FF Global Composite	106	2,312	77	1.09	**16.9**

Source: Base Met (2018)

13.3.5 Flotation

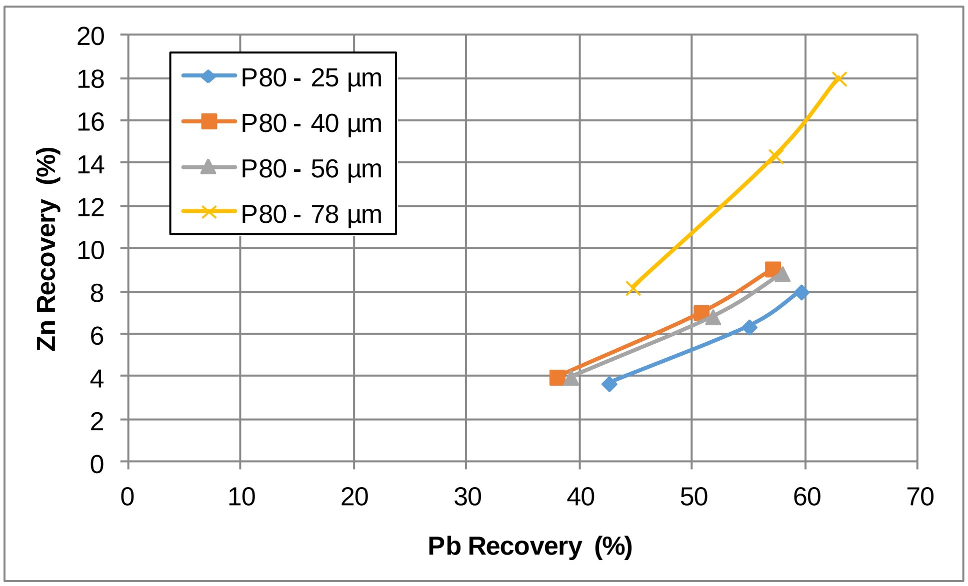
Rougher and cleaner flotation tests were conducted on the FF Global Composite to refine the test conditions for locked cycle testing.

13.3.5.1 Rougher Flotation

The majority of the test work focused on Pb rougher flotation. Due to the fine disseminated texture of galena, primary P_{80} grind size was tested over a size range of 25 µm to 78 µm. Other important parameters tested included collector dosage, the addition of a carbon depressant and the effect of sodium cyanide (NaCN) on Zn depression.

Overall Pb rougher recovery was not sensitive to primary grind size, but Pb, Zn selectivity improved as grind size got finer. A graph depicting Pb, Zn selectivity is shown in Figure 13-3. Rougher recoveries ranged between 57% and 62% with Pb, Zn selectivity improving considerably as the grind size was reduced from 78 µm to 56 µm. Only marginal gains were observed in Pb, Zn selectivity at grind sizes below 56 µm. Based on these results, a primary P_{80} grind size of 56 µm was selected for design.

Figure 13-3: The Effect of Primary Grind Size on Pb, Zn Selectivity in Base Met (2018)



Source: Base Met (2018)

Collector dosage was much higher than what is typically needed for a Pb feed grade of 1.88% Pb. Sodium isopropyl xanthate (SIPX) dosages ranging from 250 g/t to 650 g/t were tested, with the highest Pb rougher recoveries observed at 650 g/t. Pb flotation performance improved once the dosage was raised from 250 g/t to 450 g/t. It is hypothesized that the increased collector dosage rates are as a result of organic carbon. A carbon depressant was explored to address this issue.

Carbon depressant (PE26) was used to suppress organic carbon. Tests isolating the effect of PE26 indicated that carbon to the concentrate was reduced as the dosage was increased. It was also noted that as dosage increased, Pb recovery decreased. A dosage that balances these two trends is recommended moving forward.

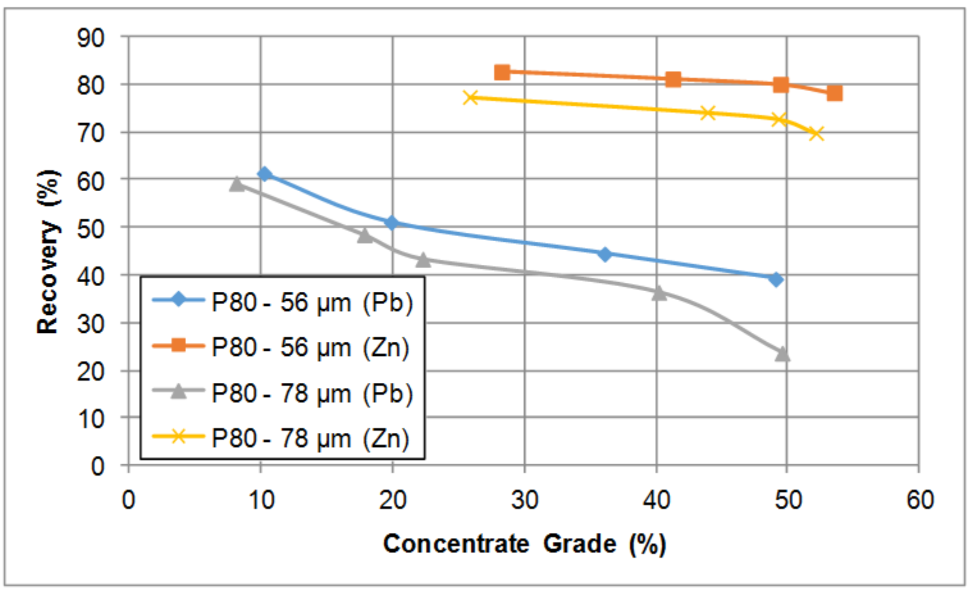
After completing Pb rougher flotation, the pH of the flotation pulp was adjusted up to 11.0 with lime, and copper sulphate was added to activate sphalerite. Initial Zn rougher tests indicated relatively good Zn rougher performance with conventional reagent dosages. Overall, 91% of the Zn in the feed was recovered into the Zn rougher concentrate. Varying the grind size had no measurable effect on Zn recovery.

13.3.5.2 Cleaner Flotation

Two batch cleaner flotation tests were conducted to determine if higher concentrate grades could be achieved while maintaining Pb and Zn recovery. Sequential Pb and Zn flotation at primary P_{80} grind sizes of 78 µm and 56 µm were evaluated, targeting very fine rougher concentrate regrind sizes of 10 µm to 12 µm for the Pb circuit and 15 µm to 20 µm for the Zn circuit. High reagent dosages of 650 g/t SIPX, 200 g/t PE26 and 80 g/t NaCN were maintained in the Pb rougher circuit to maximize Pb recovery. Concentrate grade vs. recovery curves for both tests are presented in Figure 13-4. A finer P_{80} grind size of 56 µm achieved higher recoveries at saleable concentrate grades.

An additional test was completed on the Global Composite to evaluate whether DMS pre-concentration affected flotation performance. Near identical results were obtained, demonstrating that DMS pre-concentration rejected barren shale material and did not reject carbon preferentially.

Figure 13-4: Concentrate Grade vs. Recovery Curves for Base Met (2018) FF Global Composite

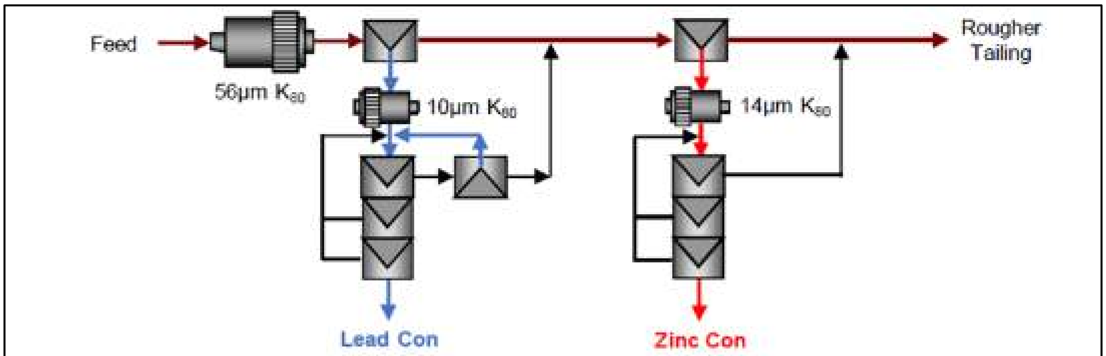


Source: Base Met (2018)

13.3.5.3 Locked Cycle Flotation

Using the optimized conditions developed from rougher flotation testing, a single locked cycle test was completed on FF Global Composite to obtain metallurgical performance data and predict grade and recovery values for economic analysis. An illustration of the flowsheet and test conditions is shown in Figure 13-5. The Pb concentrate recovered 46.2% of the Pb at a concentrate grade of 45.1% Pb; while the Zn concentrate recovered 88.9% of the Zn at a concentrate grade of 52.4% Zn. These values were used in conjunction with average DMS results from Section 13.3.7 to estimate grades and recoveries for economic analysis.

Figure 13-5: Base Met (2018) Locked Cycle Test Parameters



Stage	Reagents - g/tonne				Time Minutes	Electrochemistry	
	Lime	NaCN	PE26	SIPX		pH	Eh-mV
Primary Grind	400	80			25	8.0	50
Pb Rougher	200		200	650	5.5	9.0	45
Pb Regrind					5	9.0	-
Pb Cleaner 1	50	50		60	5	10.1	26
Pb Cleaner 2	√			5	4	9.8	17
Pb Cleaner 3	√				3	10.0	32
Pb Cleaner Scav		-		10	2	9.6	45
		$CuSO_4$					
Zn Condition	1200	2000			5	11.0	19
Zn Rougher 1	-			60	9	11.0	2
Zn Regrind	500	300			10	10.8	59
Zn Cleaner 1	50	-	-	15	9	11.0	35
Zn Cleaner 2	√			-	6	11.0	30
Zn Cleaner 3	√			-	4	11.0	25

Source: Base Met (2018)

13.3.6 Concentrate Quality

The Pb and Zn concentrates from locked cycle testing were analyzed for minor elements. Element specific determinations were performed for mercury (by cold vapour) and for gold, platinum and palladium (by fire assay and inductively coupled plasma (ICP)). Other elements were analyzed using peroxide fusion and ICP or ICP mass spectroscopy. The results are summarized in Table 13-7. Both concentrates had low levels of most minor elements, except in the Zn concentrate, where cadmium was measured at 2,610 g/t.

Table 13-7: Concentrate Quality for Base Met (2018) FF Global Composite

Element	Unit	Pb Concentrate (LCT-21)	Zn Concentrate (LCT-21)
Gold (Au)	g/t	0.536	0.011
Palladium (Pd)	g/t	< 0.005	< 0.005
Platinum (Pt)	g/t	< 0.005	< 0.005
Mercury (Hg)	ppb	795	10,100
Aluminum (Al)	%	0.16	0.12
Arsenic (As)	ppm	48	40
Boron (B)	ppm	< 10	< 10
Barium (Ba)	ppm	6,210	3,750
Beryllium (Be)	ppm	< 3	< 3
Bismuth (Bi)	ppm	< 2	< 2
Calcium (Ca)	%	0.15	0.13
Cadmium (Cd)	ppm	373	2,610
Cerium (Ce)	ppm	4	2.5
Cobalt (Co)	ppm	2.7	1.5
Chromium (Cr)	ppm	< 30	80
Cesium (Cs)	ppm	1	0.8
Copper (Cu)	ppm	137	910
Dysprosium (Dy)	ppm	< 0.3	< 0.3
Erbium (Er)	ppm	0.2	0.1
Europium (Eu)	ppm	0.1	0.1
Iron (Fe)	%	12.7	6.48
Gallium (Ga)	ppm	2.4	12
Gadolinium (Gd)	ppm	0.3	0.2
Germanium (Ge)	ppm	13.6	18.2
Holmium (Ho)	ppm	< 0.2	< 0.2
Hafnium (Hf)	ppm	< 10	< 10
Indium (In)	ppm	0.5	1.9
Potassium (K)	%	< 0.1	< 0.1
Lanthanum (La)	ppm	2.4	1.6
Lithium (Li)	ppm	< 3	< 3
Magnesium (Mg)	%	0.02	0.03
Manganese (Mn)	ppm	196	1,040
Molybdenum (Mo)	ppm	21	14
Niobium (Nb)	ppm	< 2.4	< 2.4
Neodymium (Nd)	ppm	1.8	1.2
Nickel (Ni)	ppm	60	70

Element	Unit	Pb Concentrate (LCT-21)	Zn Concentrate (LCT-21)
Lead (Pb)	ppm	> 5,000	> 5,000
Praseodymium (Pr)	ppm	0.5	0.3
Rubidium (Rb)	ppm	2.6	2.1
Sulphur (S)	%	> 25.0	> 25.0
Antimony (Sb)	ppm	53	11
Selenium (Se)	ppm	141	31.6
Silicon (Si)	%	1.12	1.02
Samarium (Sm)	ppm	0.3	0.2
Tin (Sn)	ppm	1.1	1.4
Strontium (Sr)	ppm	170	211
Tantalum (Ta)	ppm	< 0.2	< 0.2
Terbium (Tb)	ppm	< 0.1	< 0.1
Tellurium (Te)	ppm	< 6	< 6
Thorium (Th)	ppm	0.6	0.3
Titanium (Ti)	%	0.02	0.02
Thallium (Tl)	ppm	300	209
Thulium (Tm)	ppm	< 0.1	< 0.1
Uranium (U)	ppm	1.9	1.1
Vanadium (V)	ppm	41	34
Tungsten (W)	ppm	< 0.7	< 0.7
Yttrium (Y)	ppm	1.2	1
Ytterbium (Yb)	ppm	0.3	0.2
Zinc (Zn)	ppm	> 10,000	> 10,000

Source: Base Met (2018)

13.3.7 Variability Testing

A total of seven variability composites were constructed for testing. Five samples were created from specific drill holes and are identified by the drill hole number. Two additional composites were created. One composite was the same global composite with no hanging or foot wall dilution (UD GC), while the other composite was a low grade hanging wall composite (HW).

Each of the variability composites was screened at 1/4” and the coarse material processed through DMS at a SG cut point of 2.80. Table 13-8 summarizes the results from each test and includes the results from the Global Composite as a comparison. There was considerable variation in mass rejected by composite. The process was able to reject between 17.3% and 31.8% of the material while losing on average 3.7% of the Pb and 1.5% of the Zn. The separation was remarkably efficient from a metallurgical perspective, having very consistent recovery values. Based on these results, an average waste rejection of 25% was used for DMS and plant design.

Table 13-8: DMS Results for Base Met (2018) Variability Testing

Composite	Mass Rejected (%)	Sinks / Fines Head Assay (%)					Sinks / Fines Recovery (%)				
		Pb	Zn	Fe	S	C	Pb	Zn	Fe	S	C
132 VAR	20.7	1.64	9.26	9.27	19.4	2.03	96.2	99.3	95.8	96.8	76.0
133 VAR	26.9	1.26	7.09	9.29	18.7	1.97	94.3	98.4	93.5	94.8	66.7
137 VAR	17.3	2.86	12.6	8.99	20.8	1.90	98.4	99.5	96.4	97.5	76.2
138 VAR	27.4	1.37	7.20	8.52	16.7	2.40	95.9	97.6	92.6	93.5	68.9
141 VAR	30.2	2.30	11.6	8.13	19.6	1.80	98.5	99.1	90.8	93.6	61.9
UD GC VAR	22.1	2.13	11.1	9.26	20.7	1.99	96.8	99.0	95.1	96.3	72.6
HW	31.8	0.29	2.01	8.89	15.1	2.83	92.9	95.5	90.7	92.4	65.5
Global	23.8	1.78	9.42	10.0	18.4	1.83	97.4	99.2	94.3	95.9	69.5
Average	**25.0**	**1.7**	**8.8**	**9.0**	**18.7**	**2.1**	**96.3**	**98.5**	**93.6**	**95.1**	**69.7**

Source: Base Met (2018)

The sinks and fines for each variability composite were then combined and subjected to mineralogy and flotation testing. The chemical and mineral content for each variability flotation feed sample (VAR FF) is shown in Table 13-9.

Table 13-9: Chemical and Mineral Composition for Base Met (2018) Variability Composites

	132 VAR FF	133 VAR FF	137 VAR FF	138 VAR FF	141 VAR FF	UD GC FF
Head Assays						
Pb (%)	1.66	1.27	2.79	1.40	2.28	2.11
Zn (%)	10.0	7.60	13.6	7.75	12.3	12.0
Fe (%)	11.4	11.6	11.3	10.7	10.1	11.9
S (%)	18.5	18.0	21.1	16.2	19.7	20.1
C (%)	1.93	1.84	1.78	2.34	1.75	1.92
TOC (%)	1.1	1.09	1.02	1.49	1.06	1.06
Mineral Content						
Copper Sulphides	< 0.1	< 0.1	< 0.1	< 0.1	< 0.1	< 0.1
Galena	2.0	1.4	4.1	1.4	2.6	2.5
Sphalerite	15.1	11.5	21.5	11.1	19.2	18.4
Pyrite	23.5	22.4	23.2	23.3	21.4	24.2
Quartz	25.9	25.9	22.0	31.7	25.4	25.3
Potassium Feldspar	12.7	11.4	9.3	14.2	9.6	10.2
Barite	5.4	13.6	5.8	3.2	9.8	5.0
Ewaldite ($BaSrCaCO_3$)	6.4	5.4	6.2	5.3	2.4	6.1
Norsethite ($BaMgCO_3$)	0.8	0.7	0.5	0.6	< 0.1	0.7
Feldspar – Albite	2.0	1.5	2.0	1.9	0.1	1.6
Muscovite / Illite	2.9	2.8	2.4	3.3	3.3	2.5
Calcite	1.4	1.2	1.1	1.7	3.2	1.4
Other Minerals	1.9	2.3	1.9	2.1	2.9	2.1

Source: Base Met (2018)

To assess the metallurgical performance of the variability samples, rougher and cleaner tests were performed on each composite. The rougher tests used the standard conditions, as developed in the FF Global Composite flotation test work. These tests were used as scoping tests to determine the appropriate reagent dosages. Two cleaner tests, one open circuit and one closed circuit, were then completed to assess concentrate grade and recovery. The closed-circuit batch cleaner test was performed by adding a cleaner scavenger stage and recycling the Pb cleaner scavenger tailings stream to the Zn rougher feed. This flowsheet change better estimated the Zn circuit metallurgical performance. The results for each test are summarized in Table 13-10.

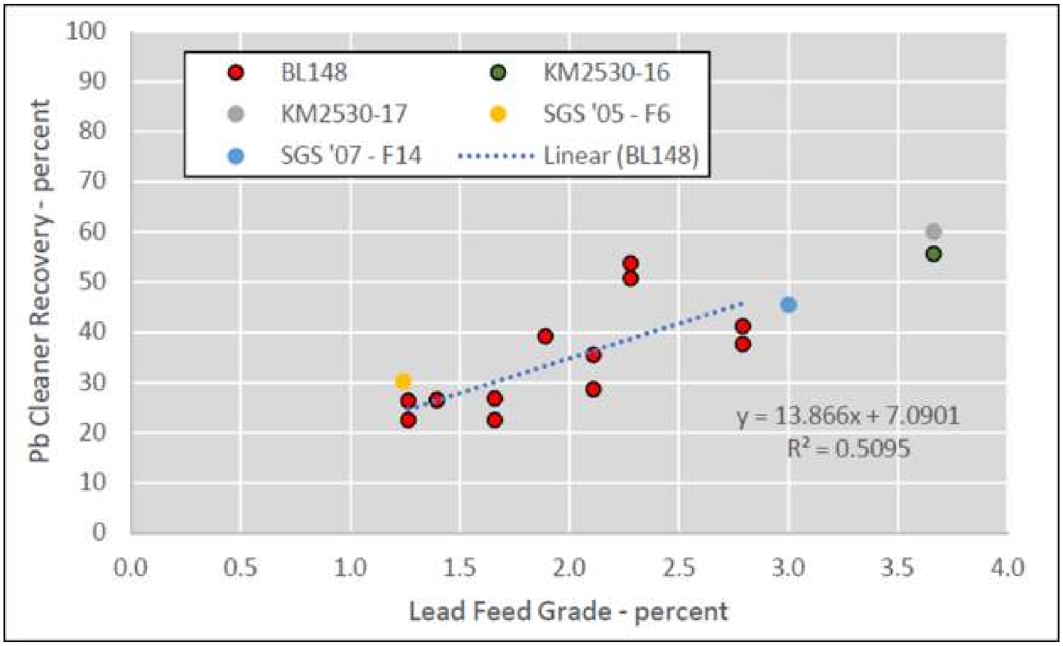
Table 13-10: Batch Cleaner Flotation Test Results for Base Met (2018) Variability Testing

Composite	Circuit Type	Pb Concentrate				Zn Concentrate			
		Grade		Recovery		Grade		Recovery	
		Pb (%)	Zn (%)	Pb (%)	Zn (%)	Pb (%)	Zn (%)	Pb (%)	Zn (%)
132 VAR FF	Open	**51.2**	3.4	**26.7**	0.3	2.0	**54.4**	16.3	**78.3**
132 VAR FF	Closed	**60.0**	6.8	**22.4**	0.5	2.6	**54.7**	20.8	**77.5**
133 VAR FF	Open	**37.2**	4.0	**22.4**	0.4	2.4	**54.2**	15.9	**60.3**
133 VAR FF	Closed	**41.8**	7.2	**26.3**	0.7	2.9	**55.3**	25.0	**76.9**
137 VAR FF	Open	**55.0**	8.4	**41.2**	1.4	2.8	**53.7**	17.1	**72.3**
137 VAR FF	Closed	**56.8**	6.6	**37.6**	0.9	2.9	**56.5**	17.5	**71.4**
138 VAR FF	Open	**48.6**	4.6	**26.5**	0.5	2.3	**55.6**	17.6	**79.6**
138 VAR FF	Closed	**48.6**	5.3	**26.4**	0.8	3.0	**56.0**	22.5	**77.5**
141 VAR FF	Open	**52.5**	3.2	**53.7**	0.6	2.4	**55.5**	16.2	**74.6**
141 VAR FF	Closed	**48.6**	5.3	**50.6**	1.0	2.7	**57.3**	20.8	**81.5**
UD GC VAR FF	Open	**46.1**	6.4	**35.5**	0.9	2.6	**53.2**	21.6	**81.4**
UD GC VAR FF	Closed	**48.6**	5.3	**28.7**	0.6	2.8	**56.2**	22.7	**82.9**
Average	**Open**	48.4	5.0	**34.3**	0.7	2.4	**54.4**	17.4	**74.4**
Average	**Closed**	**50.8**	6.1	**32.0**	0.8	2.8	**56.0**	21.6	**78.0**

Source: Base Met (2018)

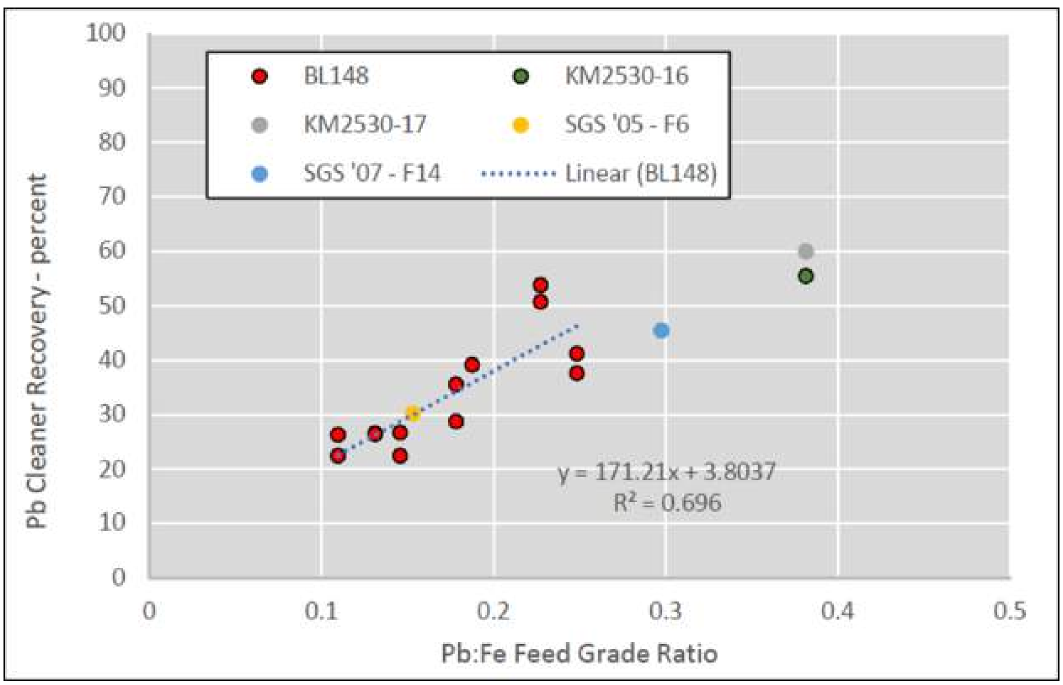
Using the variability results, coupled with results from past metallurgical test programs, two trends were established to help predict Pb circuit performance. Figure 13-6 plots the correlation between Pb feed grade and Pb cleaner flotation recovery; while Figure 13-7 plots the correlation between the Pb / Fe feed ratio and Pb cleaner flotation recovery.

Figure 13-6: Relationship between Pb Feed Grade and Pb Cleaner Flotation Recovery



Source: Base Met (2018)

Figure 13-7: Relationship between Pb/Fe Feed Ratio and Pb Cleaner Flotation Recovery



Source: Base Met (2018)

13.4 Relevant Results

Based on the results from the Base Met (2018) test program, the process flowsheet will include three stages of crushing followed by DMS pre-concentration. Through the test work, DMS has been shown to reduce grinding specific energy requirements, decrease carbon content and increase the concentration of sulphide minerals while losing only a small fraction of the contained Pb and Zn. Based on Base Met (2018) variability testing, 25% of the material will be rejected as waste with metal losses of 3.7% Pb and 1.5% Zn. Heavy sulphide-rich material will then be processed through two stages of ball mill grinding to achieve a P_{80} of 56 µm. The grinding circuit will be designed using a Bond ball mill work index of 16.9 kWh/t.

Cyclone overflow from the secondary grinding circuit will then be subjected to Pb, Zn sequential flotation. To improve concentrate grade, Pb and Zn rougher concentrates will be reground to P_{80} grind sizes of 10 µm and 15 µm respectively, before being cleaned in three stages of cleaner flotation. The test conditions from Base Met (2018) LCT-21 (see Figure 13-5) were used to size the flotation circuit and predict reagent consumable rates. Section 17 provides more detail on each process unit operation.

A preliminary estimate of Pb and Zn recoveries and concentrate grades are summarized in Table 13-11 and provide the basis for the economic analysis presented in Section 22. These projections are a combination of Base Met (2018) DMS variability testing and FF Global Composite locked cycle test results (LCT-21).

Table 13-11: Preliminary Recovery Projections

Description	Concentrate Grade		Recovery	
	Pb (%)	Zn (%)	Pb (%)	Zn (%)
DMS Pre-Concentration			96.3	98.5
Pb, Zn Sequential Flotation	45.1	52.4	46.2	88.8
Overall Recovery Projections	**45.1**	**52.4**	**46.2**	**87.5**

Source: JDS (2018)

14 Mineral Resource Estimate

14.1 Introduction

The mineral resource estimate was prepared under the direction of Robert Sim, P. Geo, with the assistance of Bruce Davis, PhD, FAusIMM. Mr. Sim is the independent Qualified Person (QP) within the meaning of NI 43-101 for the purposes of mineral resource estimates contained in this report.

The mineral resource conforms to the generally accepted CIM Estimation of *Mineral Resources and Mineral Reserves Best Practices Guidelines* (November 2003), and are reported in accordance with the Canadian Securities Administrators (CSA) National Instrument 43-101 (NI 43-101). The previous resource estimate for the Cardiac Creek deposit is described in a technical report dated 28 June 2016, with an effective date of 16 May 2016. A drilling program conducted during the 2017 summer field season provided eight additional intercepts into the deposit. As a result, previously reported Inferred class resources were upgraded into the Indicated category and the extent of Inferred resources were expanded.

Estimations are made from 3D block models based on geostatistical applications using commercial mine planning software (MineSight® v12.0). The project limits are based in the UTM coordinate system using a nominal block size measuring 5 m x 10 m x 5 m; the longer blocks are parallel to the strike of the deposit at Az315°. The primary orientation of the drilling is at Az50° and designed to intersect the steeply dipping deposit (-70° southwest) from the hanging wall side. There are several deep holes drilled from the footwall side of the deposit.

The resource estimate was generated using drill hole sample assay results and the interpretation of a geological model which relates to the spatial distribution of zinc, lead, and silver. Interpolation characteristics were defined based on the geology, drill hole spacing, and geostatistical analysis of the data. The resources were classified according to their proximity to the sample data locations and are reported, as required by NI 43-101, according to the CIM *Definition Standards for Mineral Resources and Mineral Reserves* (May 2014).

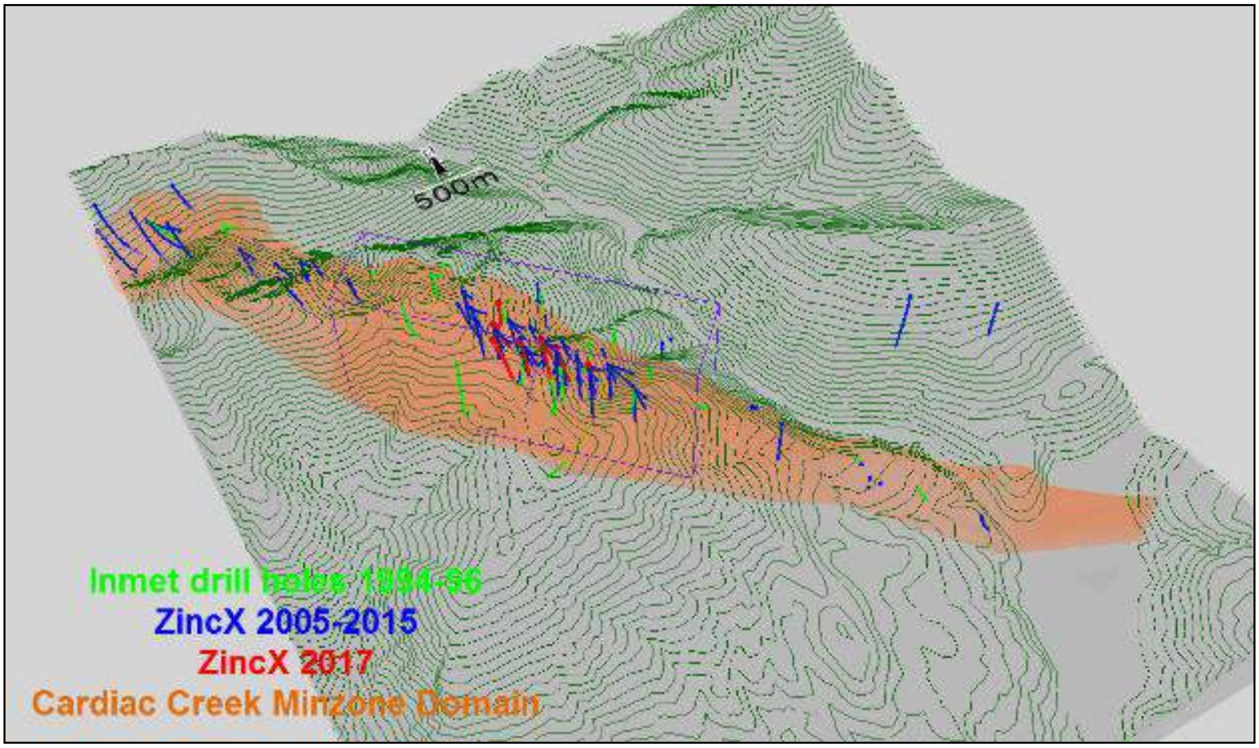
This report includes estimates for mineral resources. No mineral reserves were prepared or reported.

14.2 Available Data

During the 2017 summer field season, ZincX initiated twelve drill holes on the Property, eight of the twelve successfully intersected the Cardiac Creek deposit. The other four holes deviated off plan and were terminated prematurely. There is a total of 151 drill holes on the Property with a total core length of 64,352 m. Of these, 116 drill holes, totaling 51,978 m, are within close enough proximity of the block model to contribute to the estimation of the mineral resources for the Cardiac Creek deposit. The remaining 35 drill holes test the zone over a total strike length of almost 7 km, or these holes test other exploration targets on the Property. The eight holes completed in 2017 have provided additional, more closely-spaced drilling data to define the central and northwestern parts of the deposit.

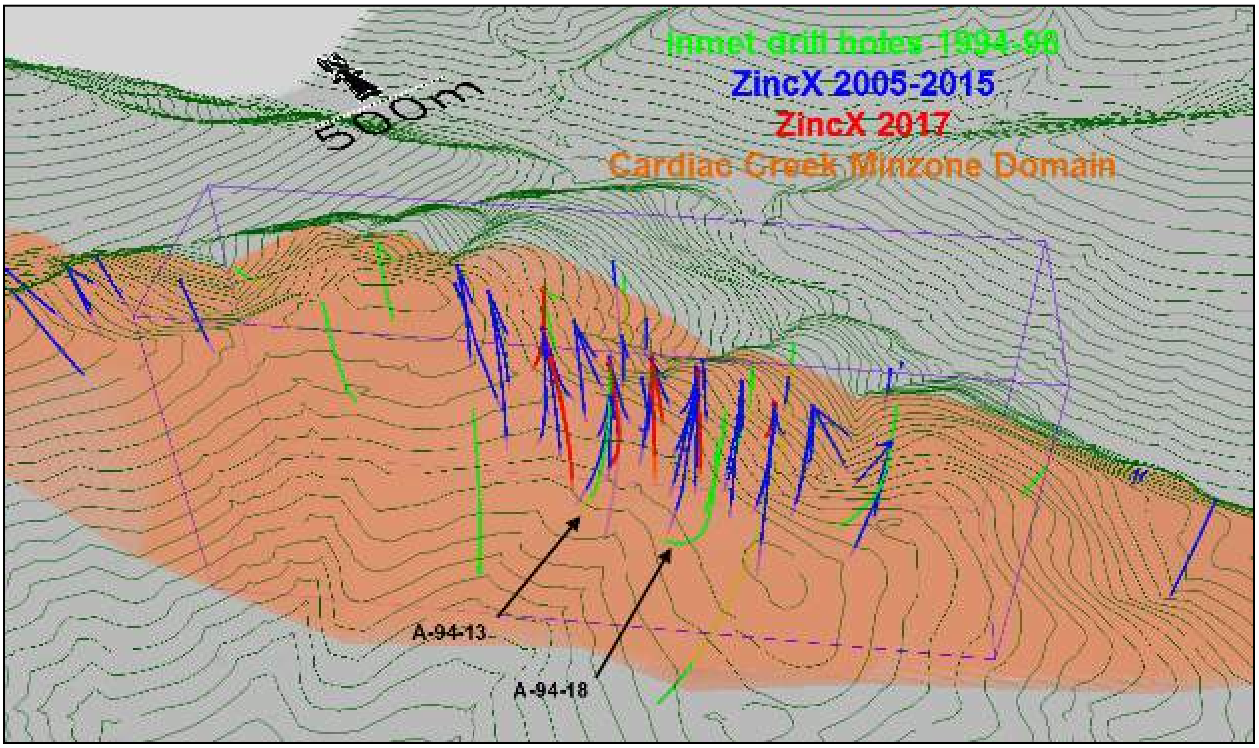
In response to the new drilling information, some minor changes were made to the down-the-hole surveys in two old holes (1994) drilled by Inmet Mining (A-94-13 and A-94-18). As new drilling encroaches on these older drill holes, there exist some minor discrepancies with the correlation of the geologic units. Erratic or irregular survey results, influenced by local acid test data, were locally smoothed, resulting in changes to the locations of these holes by 20 m to 30 m. Although these changes provide better correlation with the local geology, the effect on the estimate of mineral resources is only marginal. The distribution of the new and previous drill holes is shown in Figure 14-1 and Figure 14-2.

Figure 14-1: Isometric View Looking Northeast of the Minzone Domain and Drill Holes by Vintage



Source: Sim (2017)

Figure 14-2: Isometric View Looking West of the Minzone Domain and Drill Holes by Vintage



Source: Sim (2017)

The challenges regarding access to this rugged terrain, combined with often severe drill hole deviations, have resulted in a somewhat variable distribution of drill holes into the Cardiac Creek deposit. The spacing of pierce points into the mineralized zone (Minzone) is highly variable, ranging from 40 m to more than 500 m, with an average of approximately 100 m, in the central part of the resource area.

Select intervals within the drill holes were sampled and analyzed based on a visual observation of sulphide mineralization. A total of 11,559 m of core, in 12,088 individual samples, were analyzed for zinc, lead, and silver (often as part of a 26-element package). Sample intervals, which range from 0.04 m to 3.05 m long, with an average length of 0.96 m, were selected so they do not straddle a geologic boundary; these were also selected to represent intervals of similar sulphide type or content.

The basic statistical properties of the total sample database are shown in Table 14-1. The statistical properties of the data in the vicinity of the resource model, excluding exploration drill holes, are shown in Table 14-2. The distribution of zinc grades in drilling is shown in Figure 14-3.

Table 14-1: Summary of Basic Statistics of Sample Database

Element	# of Samples	Total Sample Length (m)	Min	Max	Mean	Std. Dev.
Zinc (%)	12,088	11,559	0.00	36.73	1.51	3.76
Lead (%)	12,088	11,559	0.00	18.05	0.28	0.80
Silver (g/t)	12,088	11,559	0.00	119.00	3.60	6.25
Density (t/m^3)	11,641	10,980	1.53	4.64	2.77	0.31

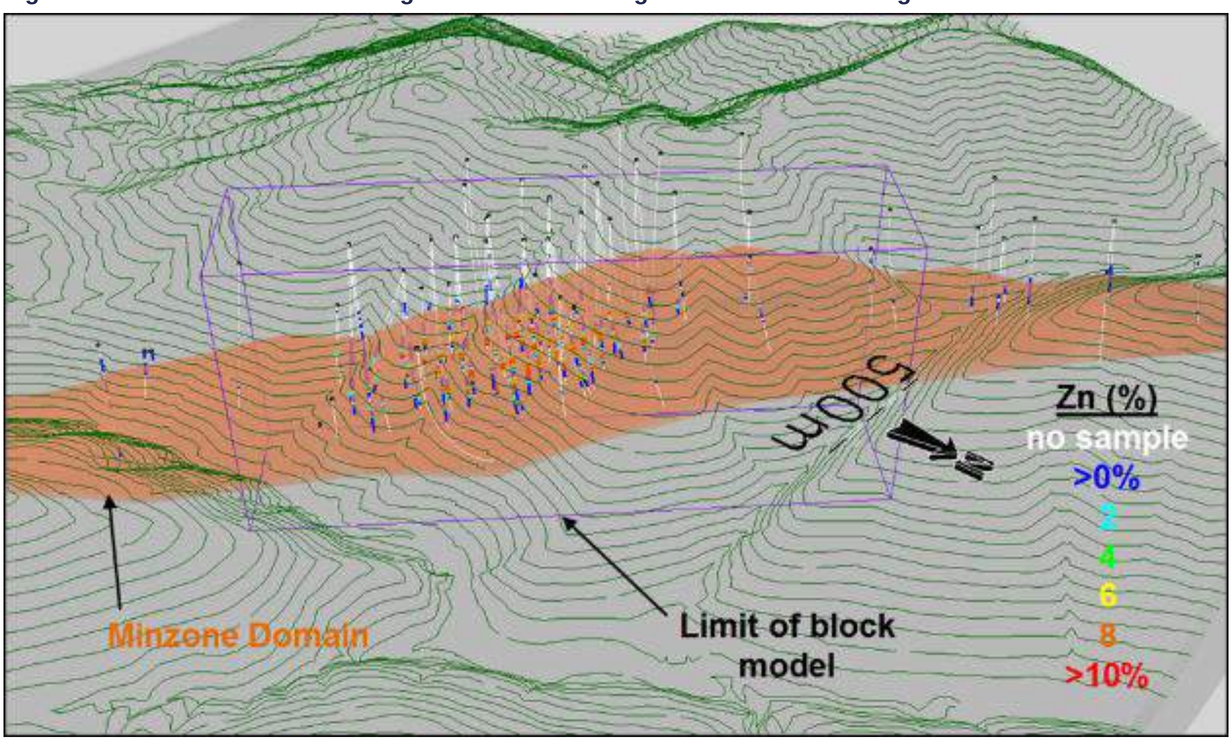
Note: Original sample data weighted by sample length
Source: Sim (2017)

Table 14-2: Summary of Basic Statistics of Data Proximal to the Resource Model

Element	# of Samples	Total Sample Length (m)	Min	Max	Mean	Std. Dev.
Zinc (%)	9,774	9,250	0.00	36.73	1.85	4.13
Lead (%)	9,774	9,250	0.00	18.05	0.35	0.88
Silver (g/t)	9,774	9,250	0.0	119.0	4.2	6.83
Density (t/m^3)	9,409	8,776	1.53	4.64	2.81	0.33

Note: Original sample data weighted by sample length. Exploration drill holes that are too distant to influence the resource model are excluded.
Source: Sim (2017)

Figure 14-3: Isometric View Looking Southwest Showing Zinc Grades in Drilling



Source: Sim (2017)

14.3 Geological Model, Domains and Coding

The Cardiac Creek deposit exhibits properties that are typical of a sedimentary exhalative (SEDEX) deposit, which is common in this area of British Columbia. The deposit occurs as a planar, sheet-like zone of semi-massive to massive sulphides comprised of varying amounts of pyrite, sphalerite, and galena (+/- barite) which has been traced over a strike length of 7 km to a depth of 1,300 m below surface. The mineralized zone ranges from less than 1 m thick to as much as 50 m, with an average of about 20 m (true thickness) in the area of potential economic interest. This Minzone represents the mineralized portion of the Cardiac Creek deposit described earlier in this report.

The Minzone was interpreted from drill hole assay sample data. Points representing the top and bottom of the zones of mineralization, generally above a grade of 1% Zn, were interpreted in all drill holes. The threshold grade of 1% Zn is derived from visual observations of the "natural" increase in the zinc grade in the drill holes and is supported by an inflection in the distribution of zinc sample data on a cumulative probability plot. Several additional points were added to provide projections of the mineralization into areas currently without any drilling activity (i.e., to project the Minzone through to surface or extend the zone at depth). The resulting points are then triangulated into 3D surfaces which are then joined to form a 3D wireframe solid domain. During the interpretation of the Minzone domain, attempts were made to retain its overall planar nature. In doing so, some low-grade intervals, in the range of 0.5% Zn, were included within the domain. Alternatively, some mineralized zones were excluded because they were considered somewhat anomalous, possibly representing only localized fault splays or veins. In general, these deviations from the interpretation of the overall Minzone domain do not represent significant potential resources. The interpreted Minzone domain is shown in Figure 14-1 to Figure 14-3.

Other than some thin surficial oxidation where sulphides occur at surface, there are no indications of significant oxidation of the resource. There is relatively little overburden in the area of the mineral resource, and, as a result, no adjustments have been made to account for overburden in the model.

14.4 Specific Gravity Data

There is a total of 11,641 sample intervals in the drill hole database that have measured values for specific gravity (SG); these determinations were conducted at Acme Analytical Laboratories Ltd. using the weight-in-air versus the weight-in-water method [specific gravity (SG) = weight in air/weight in water]. This represents approximately 96% of the total sample intervals sampled for zinc in the database, and 95% of the intervals contained within the Minzone domain. Overall, SG values range from a minimum of 1.53 to a maximum of 4.64, with a mean of 2.79. When limited to samples within the Minzone domain, the average SG increases to 3.03.

Comparisons between zinc grade and SG density in samples within the Minzone domain show a correlation coefficient of 0.72. Intervals with missing (measured) bulk densities were assigned an SG value using the following regression formula:

$$SG = 2.792 + (Zn\% * 0.049)$$

14.5 Compositing

Compositing of drill hole samples is carried out to standardize the database for further statistical evaluation. This step eliminates any effects related to the sample length which may exist in the data.

Drill hole composites are weighted by both the length and bulk density of the original sample interval and were generated “down-the-hole” which means that composites begin at the top of each hole and are generated at 1 m intervals down the length of the hole. The contacts of the Minzone domain were honoured during compositing of drill holes. Several holes were randomly selected, and the composited values were checked for accuracy. No errors were found.

14.6 Exploratory Data Analysis

Exploratory data analysis (EDA) involves the statistical summarization of the database to better understand the characteristics of the data that may control grade. One of the main purposes of this exercise is to determine if there is evidence of spatial distinctions in grade which may require the separation and isolation of domains during interpolation. The application of separate domains prevents unwanted mixing of data during interpolation and, therefore, the resulting grade model will better reflect the unique properties of the deposit. However, applying domain boundaries in areas where the data is not statistically unique may impose a bias in the distribution of grades in the model.

A domain boundary, which segregates the data during interpolation, is typically applied if the average grade in one domain is significantly different from that of another domain. A boundary may also be applied if there is evidence that a significant change in the grade distribution has occurred across the contact.

14.6.1 Basic Statistics by Domain

The basic statistics for the distribution of zinc, lead and silver inside and surrounding the Minzone domain are shown in Table 14-3 and Table 14-4. Note that this is limited to drill holes that are in the vicinity of the resource model, excluding exploration drill holes. As stated previously, samples are generally selected based on the visual observations of sulphide mineralization. As a result, much of the area surrounding the Minzone domain has not been sampled or analyzed.

Table 14-3: Summary of Basic Statistics of Composited Samples Inside Minzone Domain

Element	# of Samples	Total Sample Length (m)	Min	Max	Mean	Std. Dev.
Zinc (%)	3,448	3,448	0.00	35.15	4.86	5.22
Lead (%)	3,448	3,448	0.00	13.41	0.93	1.17
Silver (g/t)	3,448	3,448	0.3	73.9	9.1	8.54
Density (t/m^3)	3,448	3,448	2.15	4.61	3.03	0.34

Note: 1 m composited sample data weighted by sample length. Limited to drill holes in the vicinity of the resource model.
Source: Sim (2017)

Table 14-4: Summary of Basic Statistics of Composited Samples Outside Minzone Domain

Element	# of Samples	Total Sample Length (m)	Min	Max	Mean	Std. Dev.
Zinc (%)	6,034	6,034	0.00	6.37	0.13	0.28
Lead (%)	6,034	6,034	0.00	3.89	0.02	0.07
Silver (g/t)	6,034	6,034	0.3	22.2	1.4	1.63
Density (t/m^3)	5,636	5,636	1.99	4.46	2.68	0.19

Note: 1 m composited sample data weighted by sample length. Limited to drill holes in the vicinity of the resource model.
Source: Sim (2017)

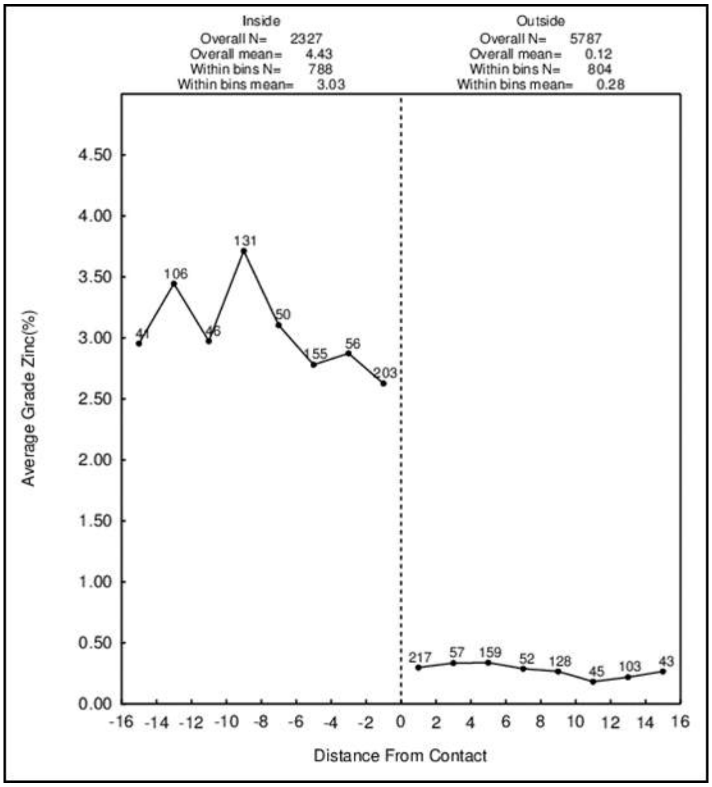
The results in Table 14-3 and Table 14-4 show that although there are several rare mineralized intervals outside of the Minzone domain, the mean grades differ significantly between these datasets.

14.6.2 Contract Profiles

Contact profiles evaluate the nature of grade trends between two domains: they graphically display the average grades at increasing distances from the contact boundary. Those contact profiles that show a marked difference in grade across a domain boundary indicate that the two datasets should be isolated during interpolation. Conversely, if a more gradual change in grade occurs across a contact, the introduction of a hard boundary (e.g., segregation during interpolation) may result in a much different trend in the grade model; in this case, the change in grade between domains in the model is often more abrupt than the trends seen in the raw data. Finally, a flat contact profile indicates no grade changes across the boundary; in the case, hard or soft domain boundaries will produce similar results in the model.

A series of contact profiles were generated to evaluate the nature of zinc, lead, and silver grades across the Minzone domain boundary. Abrupt changes in all grades occur across this contact. An example showing the change in zinc grade at the Minzone domain contact is shown in Figure 14-4.

Figure 14-4: Contact Profile for Zinc Inside vs. Outside Minzone Domain



Source: Sim (2017)

14.6.3 Conclusions and Modeling Implications

The results of the EDA indicate that the zinc, lead, and silver grades within the Minzone domain are significantly different than those in the surrounding area, and that the Minzone domain should be treated as a distinct or hard domain during block grade estimations.

Due to the overall low grades in the area surrounding the Minzone domain, grade estimates were not conducted for this portion of the model.

14.7 Evaluation of Outlier Grades

Histograms and probability plots for the distribution of zinc, lead, and silver were reviewed to identify the presence of anomalous outlier grades in the composite (1 m drilling) database. Following a review of the physical location of potentially erratic samples in relation to the surrounding sample data, it was decided that these would be controlled during block grade interpolations using an outlier limitation. An outlier limitation controls the distance of influence of samples above a defined grade threshold. During grade interpolations, samples above the outlier thresholds are limited to a maximum distance-of-influence of 35 m. The grade thresholds for zinc, lead, and silver, and the resulting effect on the model, are shown in Table 14-5. These measures are considered appropriate for a deposit with this distribution of delineation drilling.

Table 14-5: Outlier Grade Analysis Inside Minzone Domain

Element	O/L Limit	Distance of Influence (m)	# of Samples Affected	Metal Lost (%)
Zinc (%)	35.15	25	6	-0.3
Lead (%)	13.41	6	13	-0.7
Silver (g/t)	73.9	40	24	-0.5

Note: 1 m composited drill hole data.
Source: Sim (2017)

14.8 Variography

The degree of spatial variability in a mineral deposit depends on both the distance and direction between points of comparison. Typically, the variability between samples increases as the distance between those samples increases. If the degree of variability is related to the direction of comparison, then the deposit is said to exhibit anisotropic tendencies which can be summarized with the search ellipse. The semi-variogram is a common function used to measure the spatial variability within a deposit.

The components of the variogram include the nugget, the sill and the range. Often samples compared over very short distances, even samples compared from the same location, show some degree of variability. As a result, the curve of the variogram often begins at some point on the y-axis above the origin: this point is called the nugget. The nugget is a measure of not only the natural variability of the data over very short distances but also a measure of the variability which can be introduced due to errors during sample collection, preparation, and the assay process.

The amount of variability between samples typically increases as the distance between the samples increases. Eventually, the degree of variability between samples reaches a constant, maximum value: this is called the sill, and the distance between samples at which this occurs is called the range.

In this report, the spatial evaluation of the data was conducted using a correlogram rather than the traditional variogram. The correlogram is normalized to the variance of the data and is less sensitive to outlier values, generally giving better results.

Variograms were generated using the commercial software package Sage 2001© developed by Isaaks & Co. Multidirectional variograms were generated for zinc, lead, and silver in the Minzone domain; the results are summarized in Table 14-6.

Table 14-6: Variogram Parameters

Element	Nugget	Sill 1	Sill 2	1st Structure			2nd Structure		
				Range (ft)	Azimuth (º)	Dip	Range (ft)	Azimuth (º)	Dip
Zinc	0.241	0.250	0.510	78	70	-75	618	144	-3
	Spherical			11	239	43	342	1	135
				7	1	32	73	15	41
Lead	0.268	0.266	0.466	63	249	77	505	146	-1
	Spherical			7	3	29	101	11	185
				4	219	27	10	7	47
Silver	0.277	0.231	0.492	75	71	-61	585	135	-7
	Spherical			6	262	38	37	24	168
				5	156	23	134	14	56

Note: Correlograms conducted on 1 m composite sample data
Source: Sim (2017)

14.9 Model Setup and Limits

A block model was initialized in MineSight® and the dimensions are defined in Table 14-7. The selection of a nominal block size measuring 5 m x 10 m x 5 m is considered appropriate with respect to the current drill hole spacing as well as the selective mining unit (SMU) size typical of an operation of this type and scale. The block model is horizontally rotated so that the Y-axis is parallel to the strike of the Minzone at 315°. The origin of the rotation in UTM coordinates is 389150E, 6359450N. The block model limits are represented by the purple rectangle in Figure 14-1 through Figure 14-3.

Table 14-7: Block Model Limits

Direction	Minimum	Maximum	Block Size (m)	# of Blocks
X (Az45º)	0	600	5	120
Y (Az315º)	0	2,400	10	240
Z (elevation)	500	1,600	5	220

Note: -45° rotation about origin at 389150E, 6359450N
Source: Sim (2017)

Blocks in the model were assigned a code number depending on whether they were located wholly or partially within the Minzone domain. Partial block values (i.e., percentage of block inside Minzone domain) were also determined; these were used as weighting items when determining resources.

14.10 Interpolation Parameters

The block model grades for zinc, lead, and silver were estimated using Ordinary Kriging (OK). The results of the OK estimation were compared with the Hermitian Polynomial Change of Support model (also referred to as the Discrete Gaussian Correction). This method is described in more detail in Section 14.11.

The Cardiac Creek OK model was generated with a relatively limited number of samples to match the change of support or Herco (Hermitian Correction) grade distribution. This approach reduces the amount of smoothing or averaging in the model, and, while there may be some uncertainty on a localized scale, this approach produces reliable estimates of the recoverable grade and tonnage for the overall deposit.

The estimation parameters for the various elements in the resource block model are shown in Table 14-8.

In the block model, bulk density estimates were calculated using the inverse-distance (ID) weighted (i.e., ID to the power of two) interpolation method. The parameters used in specific gravity (SG) estimates are also shown in Table 14-8.

All grade estimations use length-weighted composite drill hole sample data.

During grade estimations, the search orientations were designed to follow the general interpreted trend of mineralization. A temporary elevation item is assigned to all composited drill hole samples and model blocks which is "relative" to this trend surface. This approach incorporates a dynamic anisotropy during block grade interpolation that replicates the banded nature of mineralization, seen in drilling, in the resource block model.

The interpolation parameters for zinc, lead, and silver are summarized in Table 14-8.

Table 14-8: Interpolation Parameters

Element	Search Ellipse[1] Range (m)			# of Composites[2]			Other
	X	Y	Z[3]	Min/block	Max/block	Max/hole	
Zinc	500	500	10	5	21	7	1 DH per octant
Lead	500	500	12	5	27	9	1 DH per quadrant
Silver	500	500	10	5	21	7	1 DH per quadrant
SG[4]	250	250	70	3	15	5	1 DH per quadrant

[1] Ellipse orientation parallel to Minzone at Az315°, Dip -70° SW.
[2] 1 m composite length.
[3] *Z* search based on values relative to "trend" plane (centre of Minzone domain).
[4] SG estimated using ID^2 method.
Note: DH = drill hole.
Source: Sim (2017)

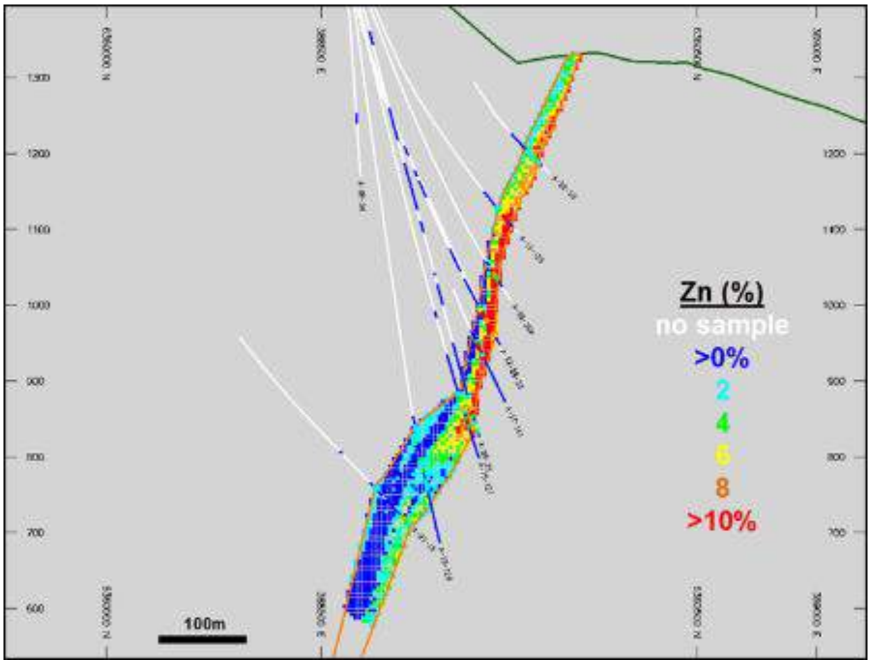
14.11 Validation

The results of the modeling process were validated using several methods. These include a thorough visual review of the model grades in relation to the underlying drill hole sample grades, comparisons with the change of support model, comparisons with other estimation methods and grade distribution comparisons using swath plots.

14.11.1 Visual Inspection

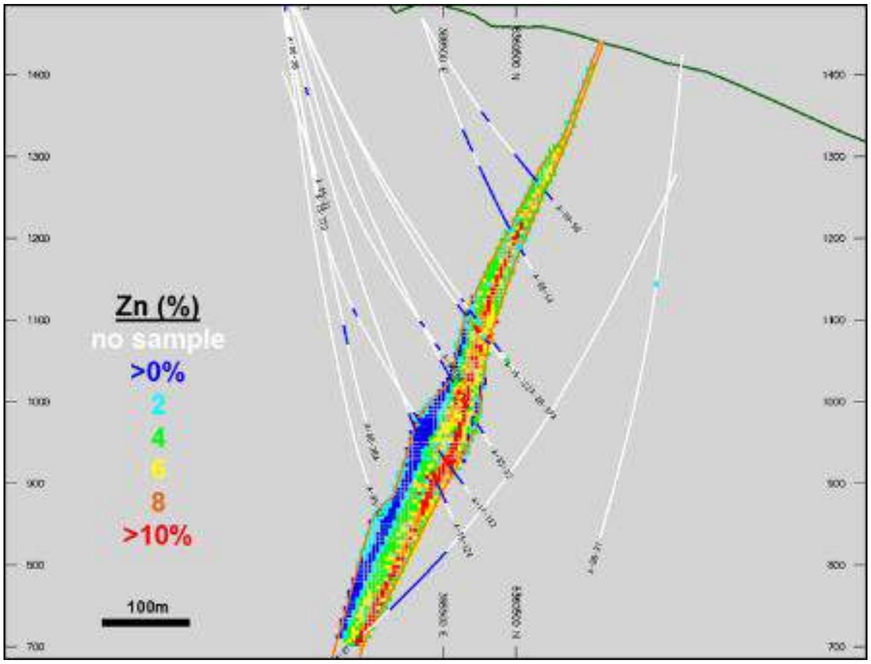
A detailed visual inspection of the block model was conducted in both section and plan to ensure the desired results following interpolation. This includes confirmation of the proper coding of blocks within the Minzone domain. The zinc, lead, and silver grades in the model appear to be a valid representation of the underlying drill hole sample data. Examples of vertical cross sections through the deposit are shown in Figure 14-5 and Figure 14-6.

Figure 14-5: Vertical Cross Section at Azimuth 50 Degrees Showing Zinc Grade in Model Blocks



Source: Sim (2017)

Figure 14-6: Vertical Cross Section at Azimuth 50 Degrees Showing Zinc Grade in Model Blocks



Source: Sim (2017)

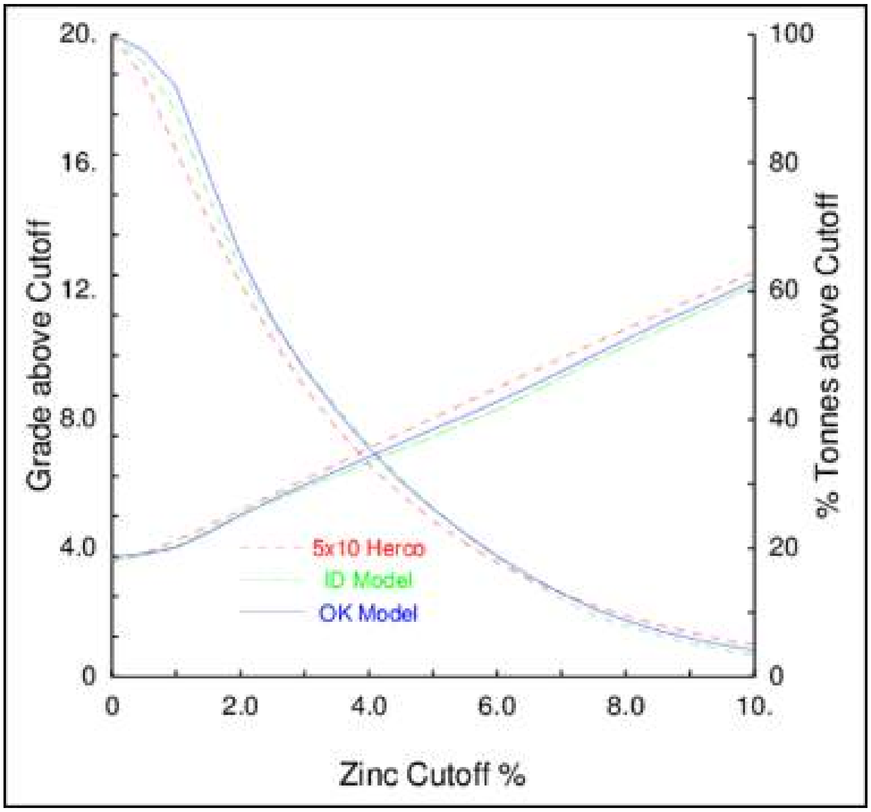
14.11.2 Model Checks for Change of Support

The relative degree of smoothing in the block model estimates were evaluated using the Discrete Gaussian of Hermitian Polynomial Change of Support method (described by Journel and Huijbregts, Mining Geostatistics, 1978). With this method, the distribution of the hypothetical block grades can be directly compared to the estimated (OK) model through the use of pseudo-grade / tonnage curves. Adjustments are made to the block model interpolation parameters until an acceptable match is made with the Herco distribution. In general, the estimated model should be slightly higher in tonnage and slightly lower in grade when compared to the Herco distribution at the projected cut-off grade. These differences account for selectivity and other potential ore-handling issues which commonly occur during mining.

The Herco (Hermitian correction) distribution is derived from the de-clustered composite grades which have been adjusted to account for the change in support, going from smaller drill hole composite samples to the large blocks in the model. The transformation results in a less skewed distribution but with the same mean as the original de-clustered samples.

The Herco analysis was conducted on the distribution of zinc, lead and silver in the block model. An example showing the distribution of zinc models is shown in Figure 14-7.

Figure 14-7: Herco Grade / Tonnage Plot for Zinc Models



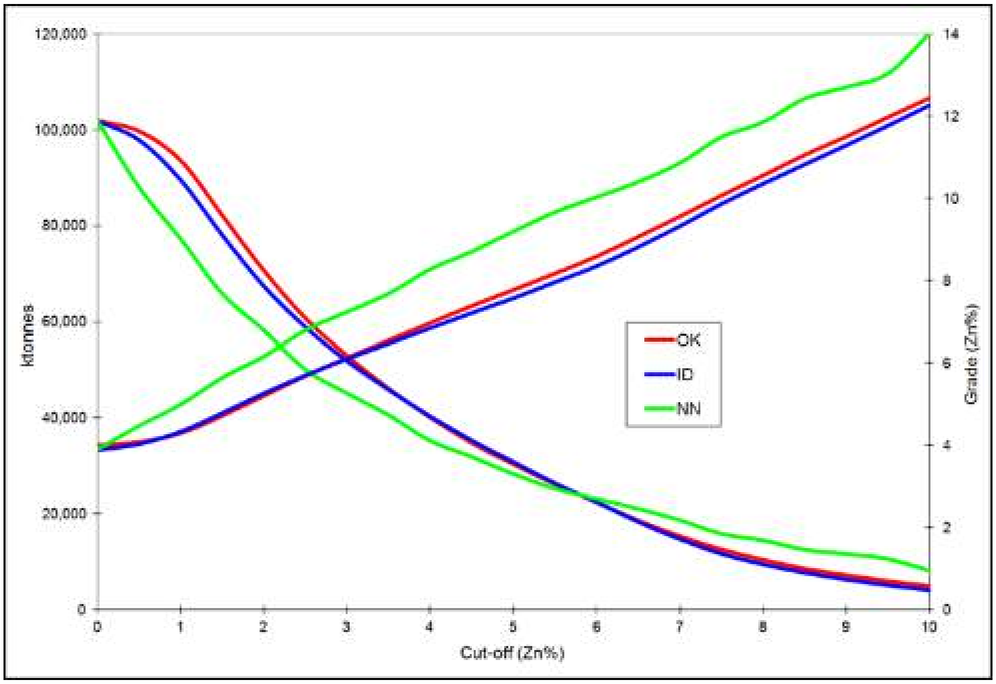
Source: Sim (2017)

14.11.3 Comparison of Interpolation Methods

For comparison purposes, additional models for zinc, lead and silver were generated using both the inverse distance weighted (IDW) and nearest neighbour (NN) interpolation methods (the NN model was made using data composited to 5 m intervals).

Comparisons are made between these models on grade / tonnage curves. An example of the grade / tonnage curves for zinc is shown in Figure 14-8. There is good correlation between the OK and ID models throughout the range of cut-off grades. The NN distribution, generally showing less tonnage and higher grade, is the result of the absence of smoothing in this modeling approach. Reproduction of the model using different methods tends to increase the confidence in the overall resource.

Figure 14-8: Grade / Tonnage Comparison of Zinc Models



Source: Sim (2017)

14.11.4 Swath Plots (Drift Analysis)

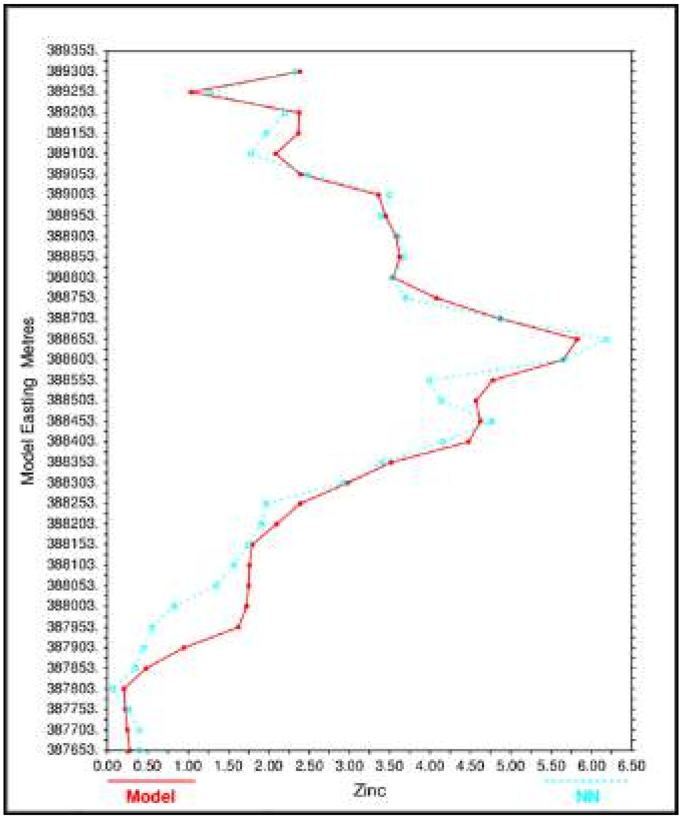
A swath plot is a graphical display of the grade distribution derived from a series of bands, or swaths, generated in several directions through the deposit. Grade variations from the OK model are compared using the swath plot to the distribution derived from the de-clustered (NN) grade model.

On a local scale, the NN model does not provide reliable estimations of grade, but, on a much larger scale, it represents an unbiased estimation of the grade distribution based on the underlying data. Therefore, if the OK model is unbiased, the grade trends may show local fluctuations on a swath plot, but the overall trend should be similar to the NN distribution of grade.

Swath plots have been generated in three orthogonal directions for all models. An example showing the zinc distribution in north-south swaths is shown in Figure 14-9.

There is good correspondence between the models in most areas. The degree of smoothing in the OK model is evident in the peaks and valleys shown in the swath plots. Areas where there are large differences between the models tend to be the result of "edge" effects, where there is less available data to support a comparison. The validation results indicate that the OK model is a reasonable reflection of the underlying sample data.

Figure 14-9: Swath Plot of Zinc Models by Easting



Source: Sim (2017)

14.12 Resource Classification

A common method used in the classification of mineral resources involves geostatistical methods which define categories based on confidence limits. Measured resources are defined as material in which the predicted grade is within ±15% on a quarterly basis, at a 90% confidence limit. In other words, there is a 90% chance that the recovered grade for a quarter-year of production will be within ±15% of the achieved production grades. Similarly, Indicated resources are defined as material in which the predicted grade is within ±15% on an annual basis at a 90% confidence limit.

The method of estimating confidence intervals is an approximate method that has been shown to perform well when the volume being predicted from samples is sufficiently large (Davis, 1997). In this case, the smallest volume where the method would most likely be appropriate is the production from one annual quarter. Using these guidelines, an idealized block configured to approximate the volume produced in one month is estimated by ordinary kriging using a series of idealized sample grids. Relative variograms for zinc grade are used in the estimation of the block. Relative variograms are used rather than ordinary variograms because the standard deviations from the kriging variances are expressed directly in terms of a relative percentage.

The kriging variances from the ideal blocks and grids are divided by twelve (assuming approximate independence in the production from month to month) to get a variance for yearly ore output. The square root of this kriging variance is then used to construct confidence limits under the assumption of normally distributed errors of estimation.

The classification is based on the distribution of zinc because zinc is the main metal contributing to the potential revenue of the deposit. Based on preliminary analysis of available data, annual production forecasts, within ±15% accuracy at 90% confidence limits, can be achieved with drill holes spaced on a nominal grid pattern of approximately 100 m.

As a result, the following criteria were used to determine resource classification in the Indicated and Inferred categories. At this stage of project evaluation, there are no resources included in the Measured category.

14.12.1 Indicated Resources

Resources in this category are delineated from multiple drill holes located on a nominal 100 m grid pattern. Indicated resources must exhibit a high degree of continuity between drill holes.

14.12.2 Inferred Resources

Resources in this category include blocks in the Minzone domain within a maximum distance of 150 m from a drill hole.

14.13 Mineral Resources

CIM *Definition Standards for Mineral Resources and Mineral Reserves* (May 2014) define a mineral resource as:

"[A] concentration or occurrence of solid material of economic interest, in or on the Earth's crust in such form, grade or quality and quantity, that there are reasonable prospects for eventual economic extraction. The location, quantity, grade or quality, continuity and other geological characteristics of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge, including sampling."

The "reasonable prospects for eventual economic extraction" requirement generally imply that quantity and grade estimates meet certain economic thresholds and that mineral resources are reported at an appropriate cut-off grade taking into account extraction scenarios and processing recovery.

The "base case" cut-off grade of 5% Zn is considered reasonable based on assumptions derived from operations with similar characteristics, scale, and location. The distribution of Indicated and Inferred mineral resources, above a cut-off grade of 5% Zn, occurs as a continuous zone which is favourable with respect to selectivity and other factors when considering possible mining options. The current resource extends to a maximum depth of 850 m below surface. The true thickness of the base case resource typically ranges between 8 m and 50 m, with an average of about 20 m. The shape and location of the deposit indicates that it is potentially amenable to underground mining methods, or a combination of surface and underground methods, and, as a result, the stated resource is considered to exhibit reasonable prospects for eventual economic extraction. It is important to note that this is not a mineral reserve because the actual economic viability has not been demonstrated.

The estimate of mineral resources for the Cardiac Creek deposit is presented in Table 14-9. The location of the mineral resource is shown in Figure 14-10.

There are no known factors related to environmental, permitting, legal, title, taxation, socio-economic, marketing, or political issues which could materially affect the mineral resource. Resources in the Inferred category have a lower level of confidence than that applying to Indicated resources and, although there is sufficient evidence to imply geologic grade and continuity, these characteristics cannot be verified based on the current data. It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.

Table 14-9: Estimate of Mineral Resources (5% Zinc cut-off)

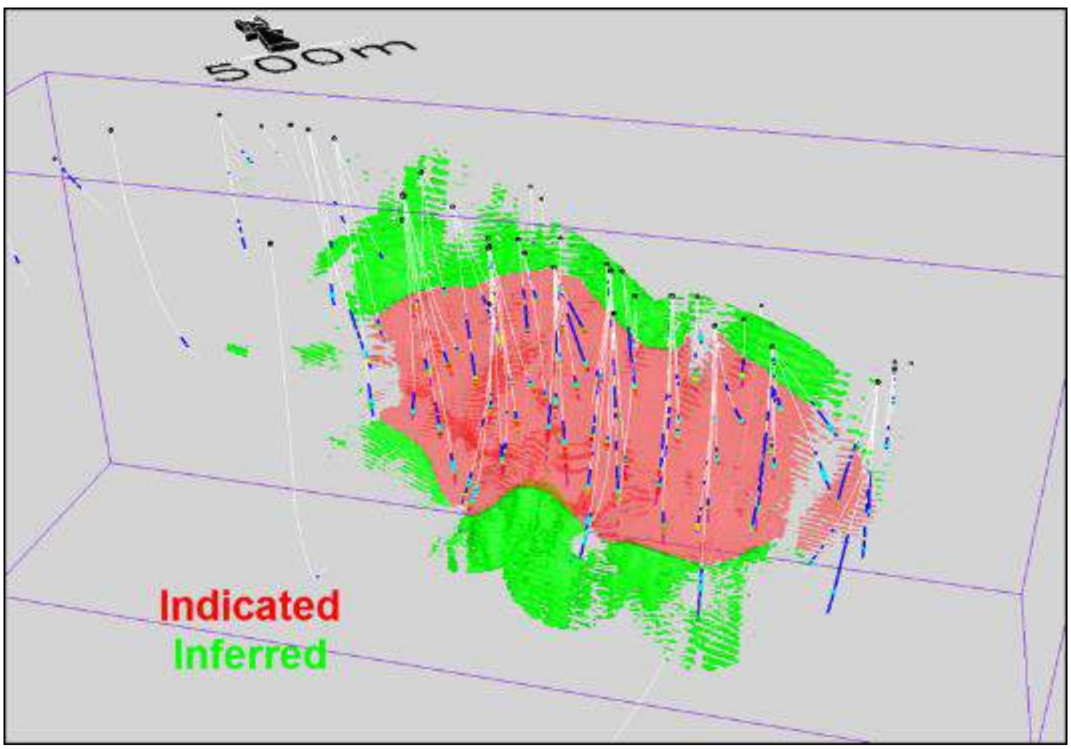
		Average Gade:			Contained Metal:		
Category	Tonnes (million)	Zn (%)	Pb (%)	Ag (g/t)	Zn (Mlbs)	Pb (Mlbs)	Ag (Moz)
Indicated	22.7	8.32	1.61	14.1	4,162	804	10.3
Inferred	7.5	7.04	1.24	12.0	1,169	205	2.9

Notes:

1. Mineral resources are not mineral reserves because the economic viability has not been demonstrated.
2. The effective date of the mineral resource estimate is November 2017.

Source: Sim (2017)

Figure 14-10: Distribution of Mineral Resources by Class



Source: Sim (2017)

14.14 Sensitivity of Mineral Resources

The sensitivity of mineral resources is demonstrated by listing resources at a series of cut-off thresholds as shown in Table 14-10.

Table 14-10: Sensitivity of Mineral Resources

		Average Grade:			Contained Metal:		
Cut-off Grade (Zn %)	Tonnes (million)	Zn (%)	Pb (%)	Ag (g/t)	Zn (Mlbs)	Pb (Mlbs)	Ag (Moz)
Indicated							
2	41.5	6.08	1.16	10.7	5,563	1,062	14.3
3	34.1	6.86	1.32	11.9	5,161	994	13.0
4	28.1	7.58	1.46	13.0	4,700	908	11.7
5 (base case)	**22.7**	**8.32**	**1.61**	**14.1**	**4,162**	**804**	**10.3**
6	17.9	9.08	1.75	15.2	3,584	691	8.7
7	13.5	9.93	1.91	16.4	2,949	567	7.1
Inferred							
2	30.0	4.11	0.69	7.5	2,715	455	7.3
3	18.5	5.15	0.89	9.1	2,098	361	5.4
4	11.8	6.11	1.07	10.5	1,591	278	4.0
5 (base case)	**7.5**	**7.04**	**1.24**	**12.0**	**1,169**	**205**	**2.9**
6	4.8	7.97	1.40	13.6	835	147	2.1
7	2.8	8.99	1.59	15.4	561	99	1.4

Note: Mineral resources are not mineral reserves because the economic viability has not been demonstrated.
Source: Sim (2017)

14.15 Comparison with the Previous Resource Estimate

In Table 14-11, the current mineral resource estimate (November 2017) is compared to the previous mineral resource estimate (effective date 16 May 2016, presented in a technical report dated 28 June 2016).

Delineation drilling now comprises a continuous zone of 100 m-spaced drill holes over an area measuring roughly 1,200 m along strike by about 500 m along the dip plane of the deposit, resulting in the estimation of resources in the Indicated category. In general, the new (2017) drill holes intersected similar or slightly thicker intervals of mineralization with higher grades than the previous 2016 results. Indicated resources have also increased by more than 3 Mt, with marginal increases in the average grades of zinc, lead, and silver. Inferred class resources have only decreased by about one half million tonnes because additional resources were encountered in the northwestern parts of the deposit.

Table 14-11: Comparison of November 2017 and May 2016 Mineral Resources (5% Zn cut-off)

Class	November 2017				May 2016			
	Tonnes (million)	Zn (%)	Pb (%)	Ag (g/t)	Tonnes (million)	Zn (%)	Pb (%)	Ag (g/t)
Indicated	22.7	8.32	1.61	14.1	19.6	8.17	1.58	13.6
Inferred	7.5	7.04	1.24	12.0	8.1	6.81	1.16	11.2

Source: Sim (2017)

14.16 Summary and Conclusions

The drill holes completed in 2017 encountered similar or slightly thicker intervals of mineralization, with slightly higher zinc, lead, and silver grades compared to the previous (proximal) drill results. The 2017 drilling increases the area that is delineated with 100 m spaced drill holes, expanding the extent of resources in the Indicated category in the northwest area of the deposit. Indicated resources have increased by about 3 Mt (+15%) compared to the previous estimate with minor increases in the average grades of zinc, lead, and silver. After previously reported Inferred class resources were upgraded to the Indicated category, the 2017 drilling also added about 2.5 Mt of new resources in the Inferred category, primarily in the northwestern part of the deposit.

The current distribution of resources has 75% in the Indicated category and 25% in the Inferred category. The previous mineral resource estimate was approximately 70% Indicated and 30% Inferred.

15 Mineral Reserve Estimate

15.1 Mineral Reserve Non-Compliance

No Mineral Reserve has been established at the Akie Project to date.

Mineral resources are not mineral reserves and have no demonstrated economic viability. **This preliminary economic assessment does not support an estimate of mineral reserves, since a pre-feasibility or feasibility study is required for reporting of mineral reserve estimates.** This report is based on mine plan tonnage (mine plan tonnes and/or mill feed).

Mine plan tonnes were derived from the resource model described in the previous section. Measured, indicated and inferred mineral resources were used to establish mine plan tonnes.

Inferred mineral resources are considered too speculative geologically to have economic considerations applied to them that will enable them to be categorized as mineral reserves, and there is no certainty that all or any part of the mineral resources or mineral resources within the PEA mine plan will be converted into mineral reserves.

16 Mining Methods

16.1 Summary

Mining of the Akie deposit will be conducted using bulk underground mining methods. The mine will be accessed using a primary decline ramp which will connect Portal One at 1055 mASL to the 920 level and serve as both as the primary production haulage route as well as a fresh air source. Additionally, a second portal will be constructed up-slope at the 1220 level to serve as the primary fresh air intake and to provide secondary egress. Levels will be located throughout the mine at 20 m vertical increments from (580 to 1320 levels), which will be connected by a primary spiral ramp, sized at 5.5 mW x 6 mH and located in the footwall of the deposit.

The primary stoping method for the Akie deposit will be longitudinal long-hole with paste backfill replacement in the mined-out voids. Thinner portions of the orebody will be mined using longitudinal long-hole methods but employ permanent pillars to avoid the requirement for cemented self-standing backfill.

The mined rock will be extracted from the mine at a rate of approximately 4,000 t/d which will be crushed and fed to a Dense Media Separator (DMS) prior to grinding and flotation. Approximately 25% of mine yield will be floated in the DMS plant, resulting in a milling rate of 3,000 t/d.

Underground haul trucks will take the broken rock to surface and dump it on the portal pad. The mineralized material will then be loaded into surface trucks and transported to the mill, a distance of approximately 2.6 km. A production pass chute will be located on the 920 level for truck loading. All mine tonnes above the 920 level will be fed to chutes on each level that connect to this production pass. Mine tonnes below the 920 level will be loaded directly into trucks for haulage out of the mine.

Once a mining panel has been exhausted, the space will be backfilled using either cemented paste or conventional cemented rock fill (CRF). Stopes less than 10.0 m wide, and all stopes not requiring self-supported fill walls will be filled with loose rock fill. Paste backfill will utilize 73% of the process tailings over the life of mine.

Additionally, 100% of the potentially acid generating (PAG) rock generated from development activities will be used as CRF or loose rock fill underground. Non-potentially acid generating (NPAG) rock will be stored on surface and where possible, used in the construction of site infrastructure. DMS reject (the float rock) will also be stored on surface.

16.2 Geotechnical Analysis and Recommendations

16.2.1 Geotechnical Data

Geotechnical specific drilling and testing programs have not yet been carried out for the underground mine area. To estimate geotechnical design parameters for the PEA, JDS has relied primarily upon rock quality designation (RQD) and core recovery data collected during the resource core logging program as well as core photographs from select drill holes.

High-level estimates of rock mass quality were made according to the Barton Q' rock mass rating system (Barton, 2002). The estimates were made using the average RQD values for each zone and applying reasonably conservative estimates of the number of joint sets and joint condition parameters, Jr and Ja based on the core photographs and experience in similar geologic environments.

The following data sources were used as the basis estimating Q' values and development PEA level geotechnical mine design parameters:

- Core photographs, RQD and core recovery data for intervals from six drill holes spread across the deposit; (A-06-35, A-06-41, A-08-56, A-11-96 and A-15-124); and
- Geotechnical logs prepared by Michael Cullen Geotechnical Ltd. (CGL) for three short drill hole intervals (A-06-39A, A-07-53 and A-08-58) in the deposit footwall and one drill hole (A-11-82) at the proposed portal collar location.

16.2.2 Anticipated Ground Conditions

The mineralization is hosted within siliceous, carbonaceous black shales of the Gunsteel Formation with bedding oriented sub-parallel to the deposit hanging wall. Drill core commonly breaks along bedding planes indicating that bedding will form planes of weakness in the rock mass. It is anticipated that bedding planes will control stope hanging wall stability, limiting the maximum area of that can be open at any one time, prior to backfilling.

The mineralized horizons and HW are generally of 'Fair' rock quality with Q' values estimated to range between 4 and 10, according to the Barton Q rock mass classification system. RQD values typically range from 70% to 80% with localized areas of more heavily fractured rock. The average core recovery for the HW and mineralized zones is 98%.

Footwall development will be in the Road River Group which consists of more massive siltstones. The footwall rock mass is typically of 'Good' rock quality with Q' values estimated between approximately 10 and 20. Footwall RQD values typically range between 80% and 90% with 98% average core recovery.

Based on existing RQD data the upper portion of the Gunsteel Formation, approximately 150 m above the mineralized zone, is of poor to very poor rock mass quality with RQD values typically less than 20%. This zone however is not anticipated to impact the underground mine as currently designed.

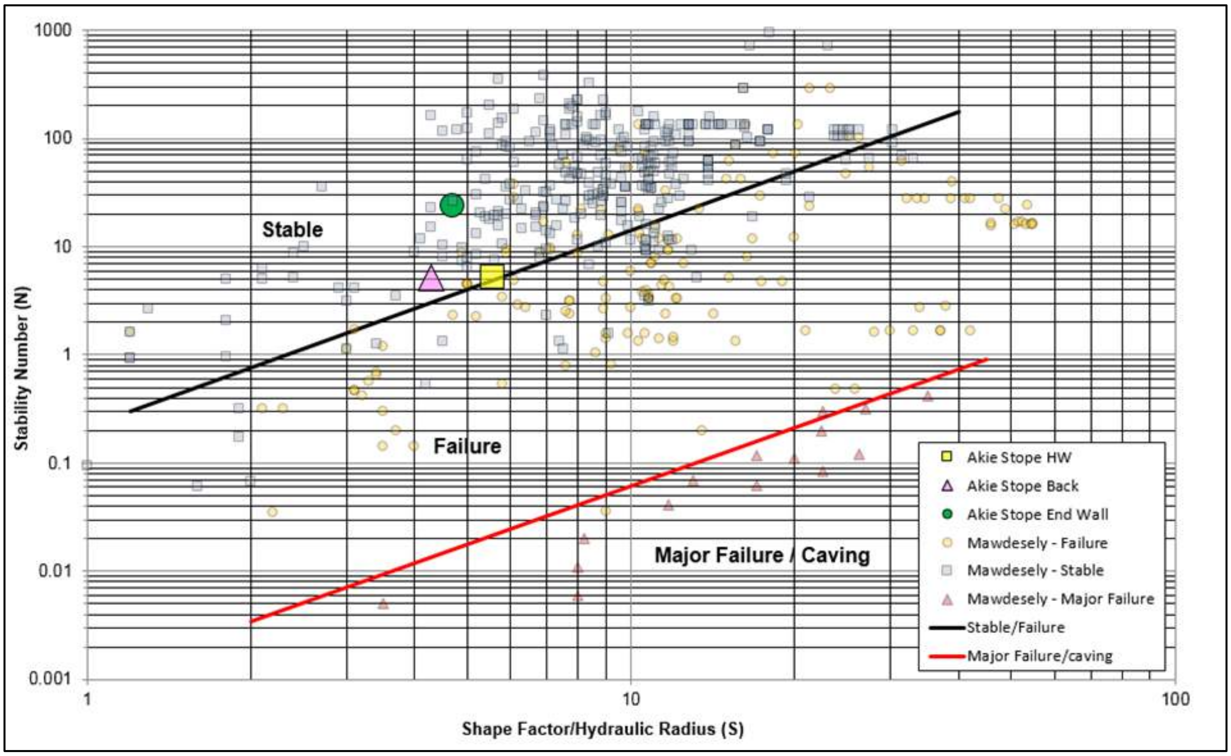
Review of the CGL geotechnical log for drillhole A-11-82 indicates that approximately 14 m of overburden soil and completely weathered rock exist at the proposed portal location. Below a depth of 14 m the rock is weathered and heavily fractured to a depth of approximately 28 m.

16.2.3 Stope Dimensions

Empirical stope design analyses are based on a series of stability graphs where the Stability Number (N') is plotted on the vertical axis against the hydraulic radius (wall area divided by wall perimeter) of the particular stope face being evaluated on the horizontal axis. The stability number is calculated based on the Barton (2002) Q' rock mass rating system, the face dip, geologic structure and induced stress conditions.

Limiting stope dimensions were estimated using the Potvin (2001) and Trueman (2003) empirical stope design methods assuming the average estimated rock mass conditions. Based on a 20 m level spacing, maximum unsupported stope lengths were estimated from the charts for the various vein widths. Figure 16-1 contains the Trueman (2003) empirical stability chart with the respective stability numbers plotted against the stope hanging wall, back and end hydraulic radii. The hydraulic radii shown represent a 25 m high (20 m stope plus 5 m top cut) by 20 m long and 15 m wide stope.

Figure 16-1: Empirical Stope Stability Plot



Source: Trueman & Mawdesley (2003)

16.2.4 Ground Support

Based on the anticipated rock quality (Q' values) as well as the size and expected life and use of the various mine openings, ground support requirements were initially assessed according to the Barton (2002) criteria. The Q-system also takes into account the life and use of the opening (ex. man-entry or equipment only) with the excavation support ratio (ESR) parameter. The ESR is used to adjust the design span which in effect imposes a higher factor of safety on critical structures with long life (ex. an underground nuclear power station with an ESR of 0.5 to 0.8) than on temporary tunnels (ex. temporary mine workings with an ESR rating of 2 to 5).

Cable bolt spacing and lengths for stope backs greater than 15 m wide were estimated using empirical methods developed by Hutchinson and Diederichs (1996).

The ground support recommendations include the following:

- Temporary and permanent waste development (5 mW to 6 mW x 5 mH):
  - 2.4 m long #7 resin bolts on 1.5 m ring spacing and 1.5 m within the ring with 6 gauge welded wire mesh in back to within 1.5 m of floor; and

- Assume 5% of the total permanent waste development will require 5 cm of shotcrete in addition to bolting. No shotcrete required for temporary waste development.

- Temporary ore development (5 m x 5 m):
  - 2.4 m long #7 resin bolts on 1.5 m ring spacing and 1.5 m within the ring with 6 gauge welded wire mesh in back to floor;
  - No shotcrete required in ore development; and
  - 6 m long twin strand cables on 2 m x 2 m spacing for stopes wider than 15 m.

16.3 Mine Access and Development

16.3.1 Portals

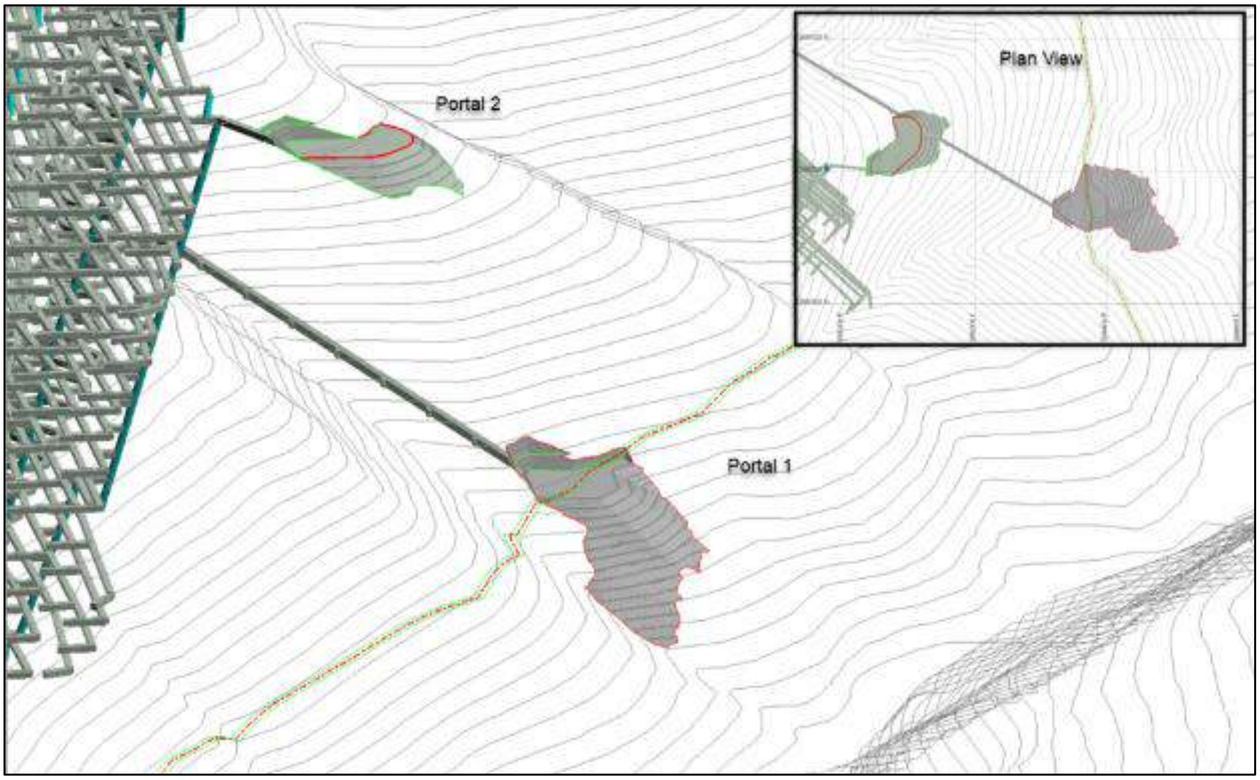
The Mine will require two portals during development and operations.

Portal One, located at 1055 mASL, will be the primary haulage portal and will be provided with a laydown pad for run of mine rock. Due a limited surface footprint of just 4,000 m^2, no other materials will be stored on this pad. Portal One will be equipped with fans and heaters to act as a fresh air feed.

Portal Two will be collared up-slope at 1220 mASL to provide secondary access and egress for the mine at 1220 L and will also be equipped with fans and heaters to act as a secondary fresh air feed. The high flow requirement in this heading will prohibit its use for regular vehicular access in and out of the mine.

Both portal excavations have been designed with a rock cut of 4:1 V:H with a 5.0 m catch bench at 10.0 m in height. Due to the relatively steep terrain, rock fill slopes have been set to 1:1.5 V:H. A general arrangement of the portals is shown in Figure 16-2.

Figure 16-2: Akie Portal Locations

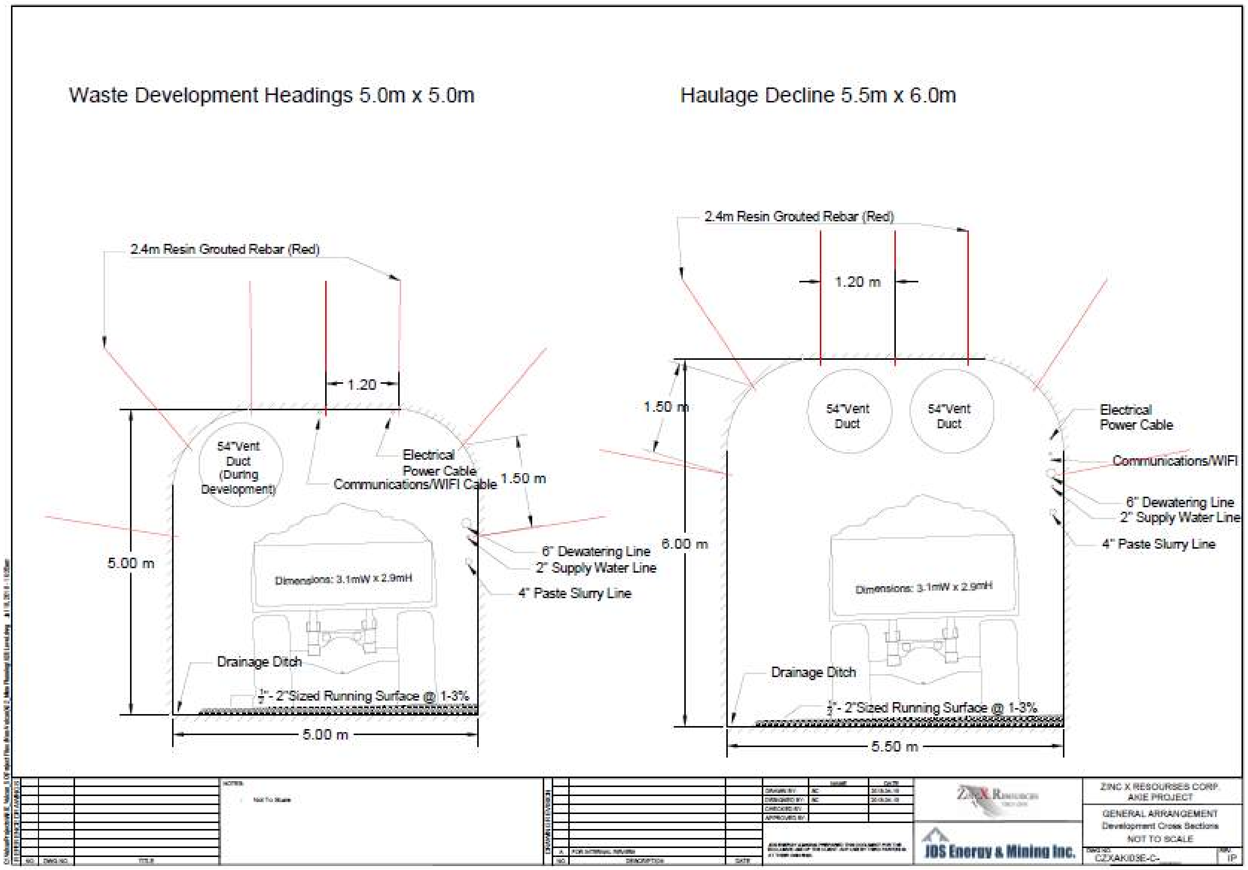


Source: JDS (2018)

16.3.2 Lateral Development

The primary decline will be driven from Portal One, at 1055 mASL, to the 920 L, a linear distance of 900 m. It will be driven at 5.5 mW x 6.0 mH, sized to accommodate the necessary ventilation ducting and services. The primary decline will be used for all haulage from the mine. It will also act as a fresh air feed into the mine, with a primary fan and heater located at Portal One. Pull outs have been designed every 100 m and the decline has been designed at a maximum gradient of 15%. A general cross section of the Mine development headings is shown in Figure 16-3.

Figure 16-3: Development Heading General Cross Sections



Source: JDS (2018)

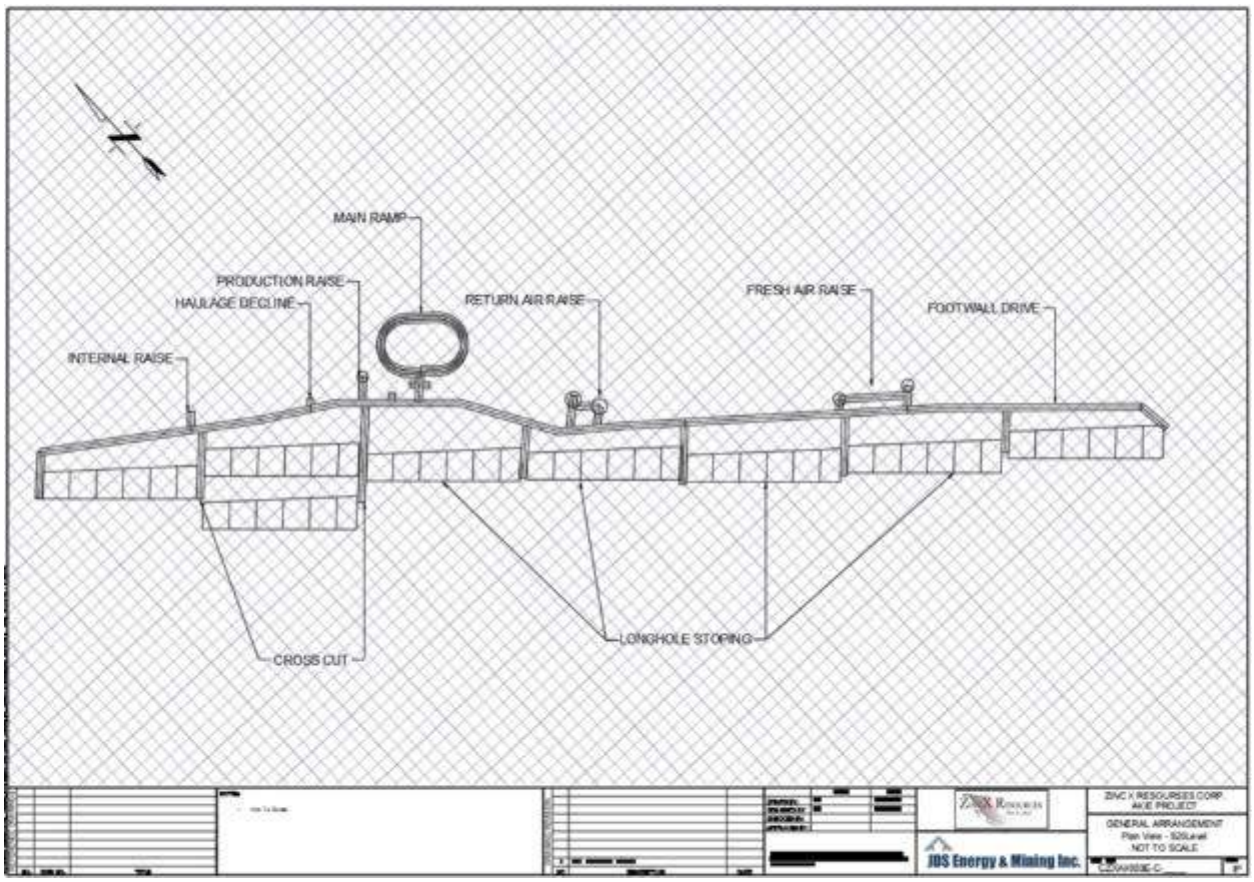
Each working level of the mine will be connected using a 5 mW x 5 mH spiral ramp located in the footwall of the deposit. The spiral ramp will be used for mineralized rock and waste haulage, fresh air ventilation, and will connect all 37 levels of the mine, from 580 mASL to 1320 mASL. The ramp has been designed at a relatively shallow gradient of 12.5% to accommodate the relatively close level spacing of 20 m and a minimum turning radius of 20 m.

Each working level of the mine will be connected using a 5 mW x 5 mH spiral ramp located in the footwall of the deposit. The spiral ramp will be used for mineralized rock and waste haulage, fresh air ventilation, and will connect all 37 levels of the mine, from 580 mASL to 1320 mASL. The ramp has been designed at a relatively shallow gradient of 12.5% to accommodate the relatively close level spacing of 20 m and a minimum turning radius of 20 m.

Each mining level will have a 5.0 mW x 5.0 mH footwall drive located at a minimum offset of 20 m from the deposit in the footwall. Crosscuts will be located at 120 m intervals along the footwall drifts, connecting them to the stoping blocks. These will be driven through the orebody to the hanging wall of the deposit. The footwall drives will house the majority of services including remuck bays, ancillary bays, ventilation raise accesses and production pass access. Every third footwall drift will also house an electrical sub-station and a refuge bay. A plan view of a typical level is shown in Figure 16-4.

Stope sill drifts will be driven at 5.0 mW x 5.0 mH on each level for longhole drilling, mineral extraction, and backfill placement. For stopes less than 16 m thick, only one stope sill drift will be driven, located near the centerline of the stopes. Stope sin the thicker portions of the orebody will have two stope sill drifts to provide multiple drawpoints for mucking.

Figure 16-4: Typical Level Layout (920 Level Shown)



Source: JDS (2018)

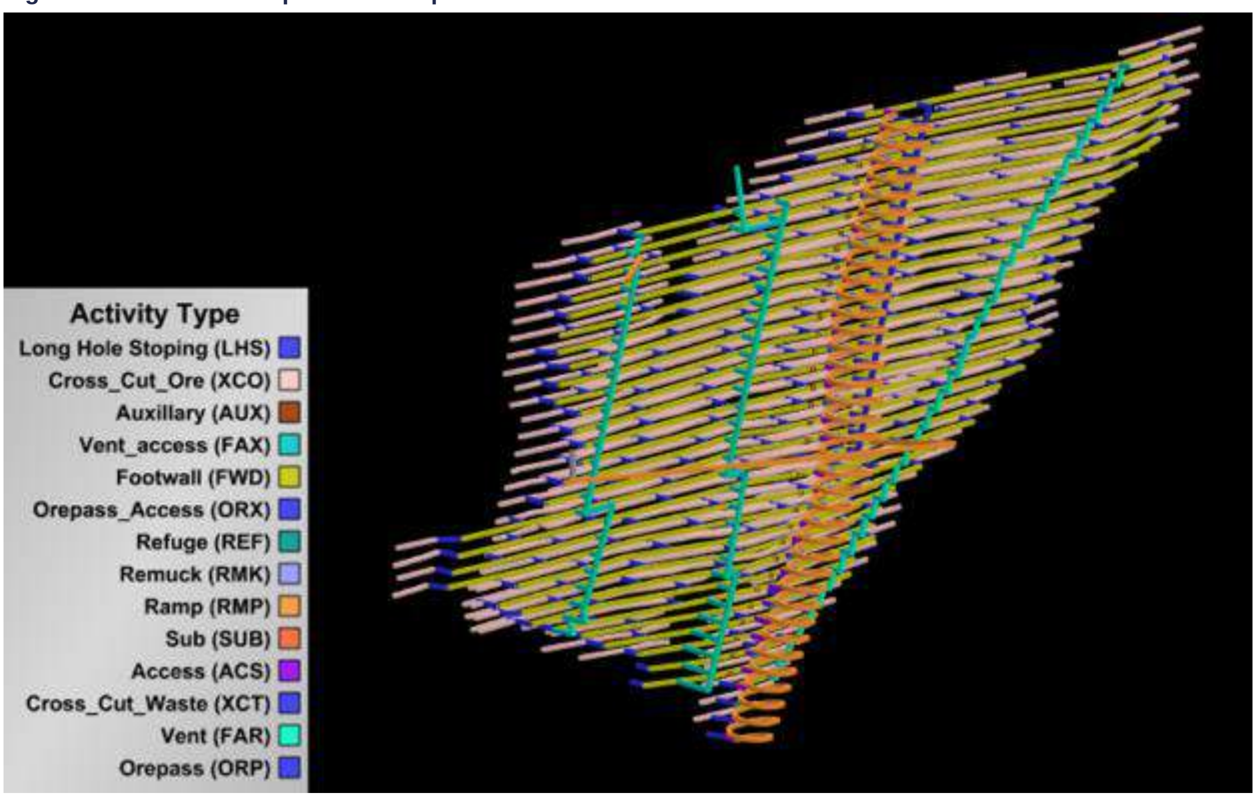
The lateral development requirement for the LOM is shown in Table 16-1 and Figure 16-5.

Table 16-1: Lateral Development Summary

Items	Units	Width	Height	Type	Total Planned
Meters - Ramp	m	5.5	6.0	CAPEX	7,368
Meters - Footwall Drive	m	5.0	5.0	CAPEX	22,072
Meters - Access	m	5.0	5.0	CAPEX	931
Meters - Remuck	m	5.0	5.0	CAPEX	243
Meters - Auxiliary	m	5.0	5.0	CAPEX	160
Meters - Vent Drive	m	5.0	5.0	CAPEX	2,259
Meters – Production pass Drive	m	5.0	5.0	CAPEX	581
Meters - Refuge	m	5.0	5.0	CAPEX	90
Meters - Sump	m	5.0	5.0	CAPEX	90
Meters - Waste Crosscut	m	5.0	5.0	OPEX	8,578
Meters – Sub-Station	m	5.0	5.0	CAPEX	285
Meters - Mineralization Drive	m	5.0	5.0	OPEX	41,693
Meters - Total Waste Lateral	**m**				**42,176**

Source: JDS (2018)

Figure 16-5: Akie Development - Oblique View



Source: JDS (2018)

16.3.3 Vertical Development

Vertical development will be used to provide one production pass and three ventilation raise systems.

A 4 m diameter production pass will be driven from 920L to 132L using a raise boring machine.

Fresh air raise 1 (FAR1) will be driven from 1220L to 760L by raise bore and will act as a primary fresh air raise and escape way.

Return air raise 1 (RAR1) will be driven from Surface to 640L by raise bore and will act as an exhaust raise.

Return air raise 2 (RAR2) will be a connected series of conventionally driven drop raises from 1320L to 640L acting as a bleed return air raise between levels.

All fresh air raise sections will be equipped with manway installations to act as secondary egress.

A summary of all vertical development is shown in Table 16-2 as well as in Figure 16-9 of the ventilation section.

Table 16-2: Vertical Development Summary

Development Item	Total Length (m)	Inclination (deg)	Number of Sections	Method	Diameter (m)
Fresh Air Raise 1 (FAR 1)	503	75	3	Raise Bore	4.9
Return Air Raise 1 (RAR1)	661	75	3	Raise Bore	5.8
Internal Vent Raise (RAR2)	714	70	34	Conventional Raise	3.0
Production Raise (ORP)	413	75	1	Raise Bore	4.0

Source: JDS (2018)

16.3.4 Underground Infrastructure

There will be two primary installations underground:

- A maintenance shop, and
- A paste plant.

16.4 Mining Method

The sole mining method for the mine will be mechanized longitudinal long-hole mining with a combination of paste, rockfill (RF) and CRF.

Stope sills will be driven at 5.0 mW x 5.0 mH at 20 m vertical increments. Stopes will in general be a maximum of 20 m along strike, making a typical maximum exposed hanging wall and footwall of 20 mL x 25 mH. Where the orebody is greater than 16 m in width, two parallel sill drives will be used to ensure adequate drill coverage and to provide multiple extraction points for mucking.

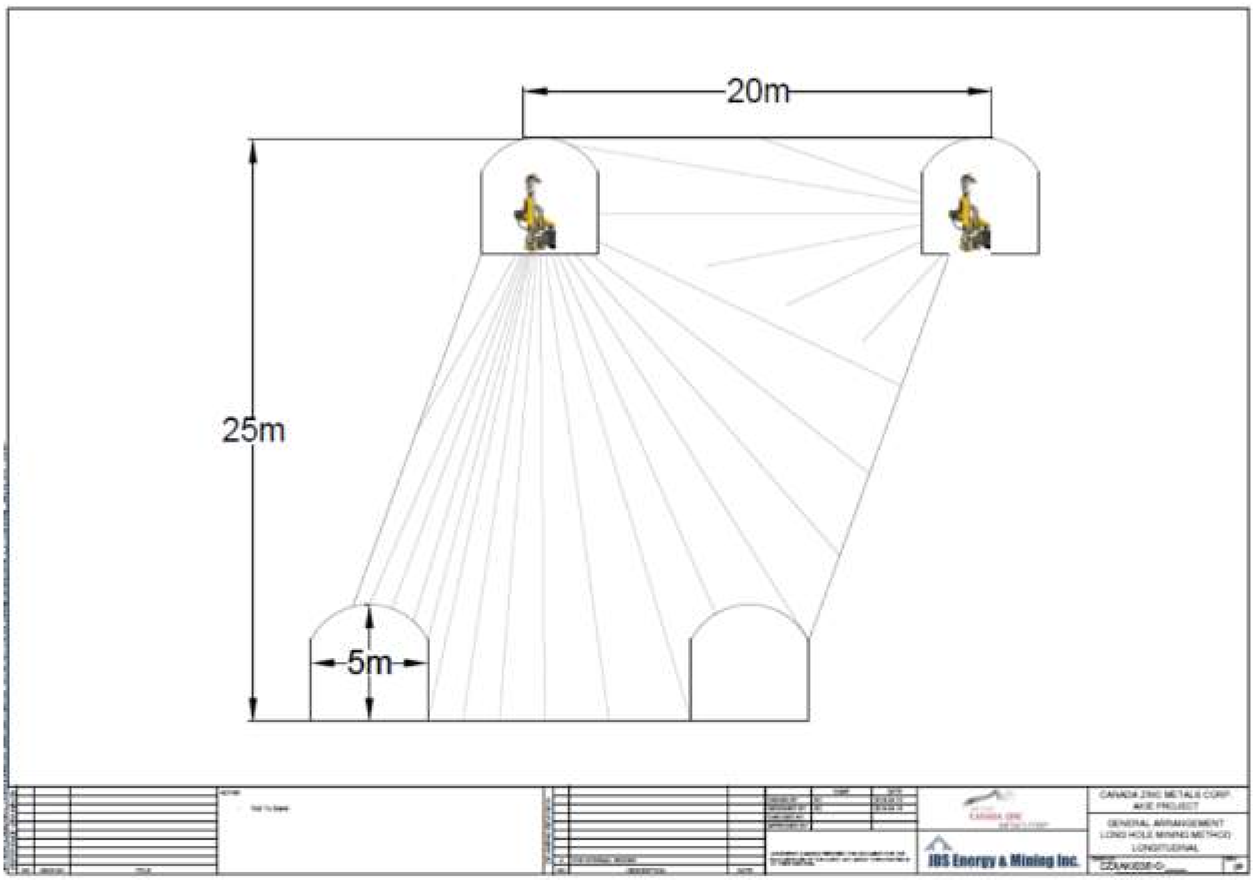
Production drilling will be done using 64 mm downholes for most of the stopes, using drop raises, slots, and drill rings for extraction. A general long hole cross section and typical ring pattern for the longitudinal long-hole method is shown as Figure 16-5.

In situations where the final uppermost stopes of the panel form a crown pillar beneath backfill, stopes will be extracted using uphole drilling and inverse raises.

A general long hole cross section of the longitudinal long-hole method as applied to the mine is shown in Figure 16-5.

After a stope has been extracted, it will be filled with self-standing paste or CRF backfill, allowing the mining of the next, adjacent stope.

Figure 16-6: Long-hole General Cross Section



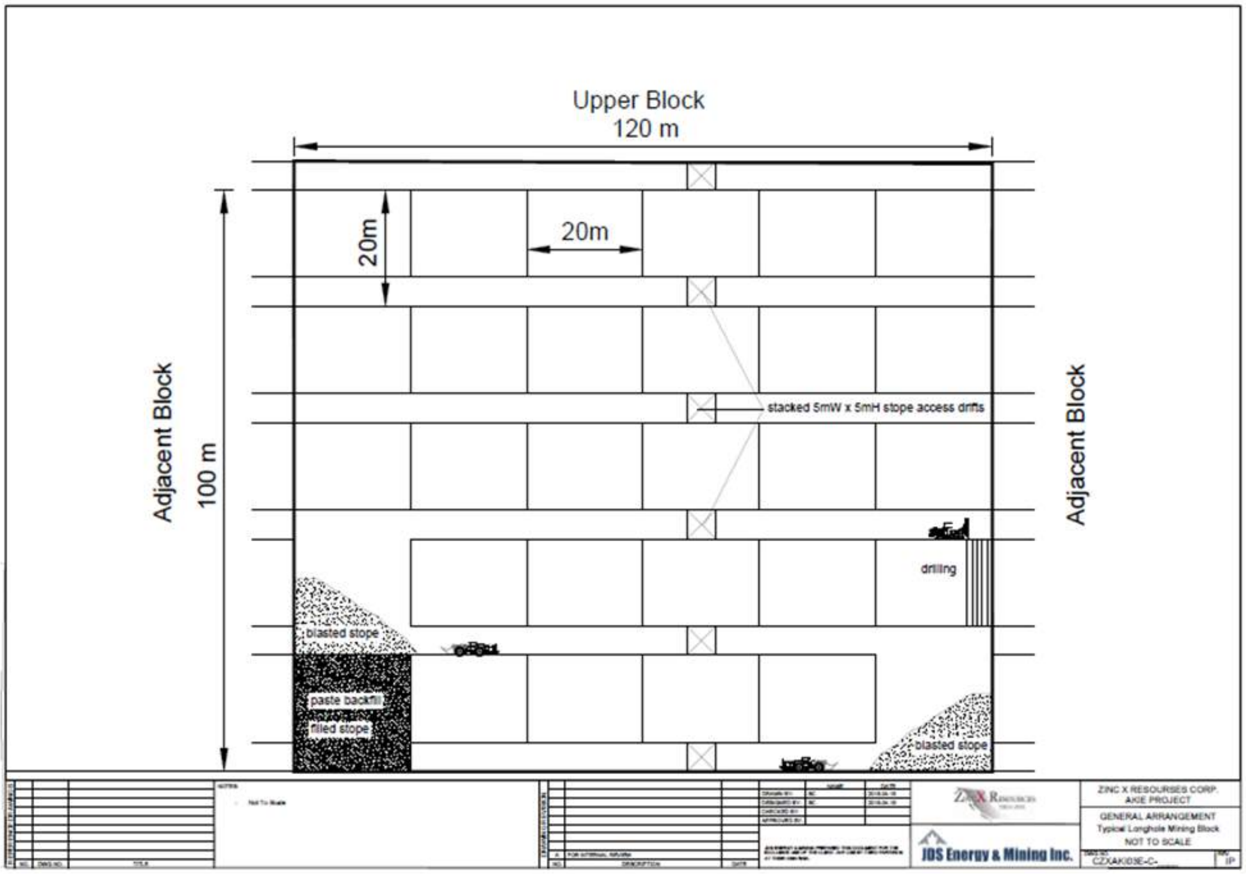
Source: JDS (2018)

Nominally, panels will be comprised of 30 individual stopes; six stopes along strike by five stopes high. Thus, the typical panel will have a length of 120 m along the strike and a height of 105 m, spanning five mining levels. A typical mining block is shown as Figure 16-7.

In situations where the final uppermost stopes of the panel form a crown pillar beneath backfill, stopes will be extracted using uphole drilling and inverse raises.

After a stope has been extracted, it will be filled with self-standing paste or CRF backfill, allowing the mining of the next, adjacent stope.

Figure 16-7: Long Section showing Mining Sequence for a Typical Block of Stopes



Source: JDS (2018)

16.5 Mineral Inventory

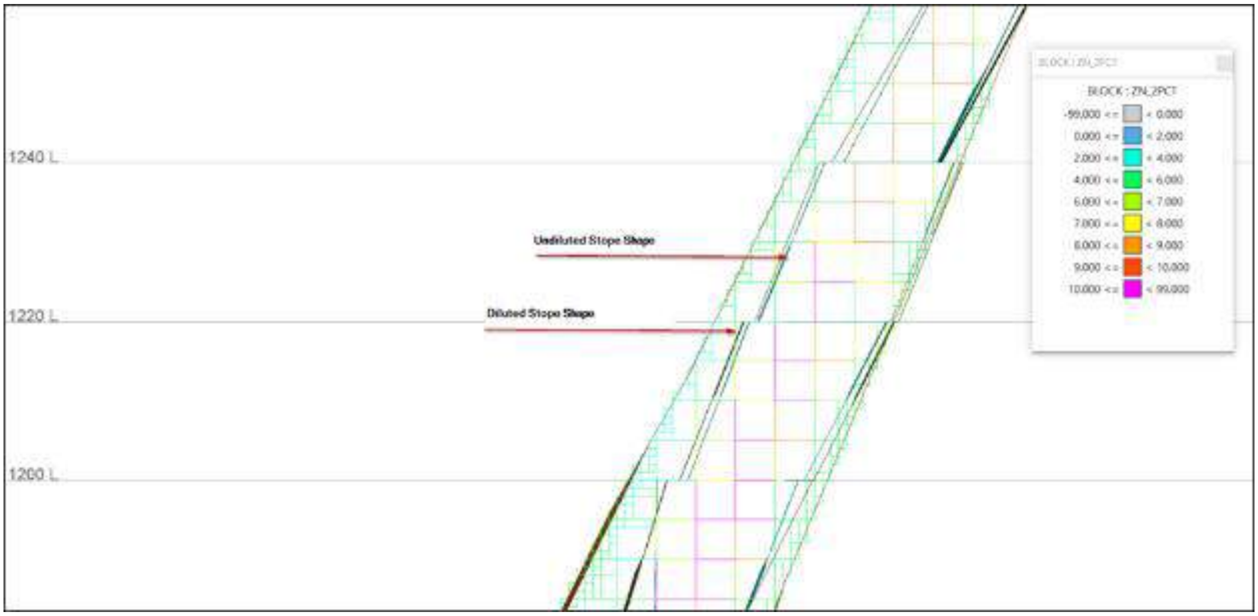
Mining factors were applied for the selected method and stope geometries to represent the anticipated dilution and recoveries throughout the mine.

16.5.1 Dilution

Hanging wall and footwall dilution were modelled within the Vulcan® software on the assumption of 1.0 m over break along both planes, as shown in Figure 16-8.

In areas where dilution was still within the ore body, dilution values were modelled to include the grades found within these 1.0 m skins. Globally the mine plan includes 13% hanging wall and footwall dilution by tonnage with an average zinc grade of 4.7%.

Figure 16-8: Vulcan® Hanging Wall and Footwall Dilution Modelling



Source: JDS (2018)

Where stopes included internal waste blocks, these tonnes were included at zero grade. Internal waste dilution was negligible, accounting for only 0.1% of the total mine production by tonnage.

Backfill dilution was modelled based on stope width and an assumption of 0.5 m dilution from all backfill planes (walls and floors) during mucking. A global factor of 3% was applied to all stopes greater than 10 m in width.

16.5.2 Recovery

Mining recoveries were applied as follows:

- 95% - Stopes greater than 10 m wide; and
- 85% - Stopes less than 10 m wide.

16.5.3 Mine Yield

The total resource contained in the mine is summarized in Table 16-3. These results are based upon preliminary mineable stope designs and incorporate the factors for recovery and dilution noted in Section 16.6.1. This does constitute a mining reserve, as the mining factors and geometries have been applied to inferred resources which are not considered to be sufficiently proven geologically for reliance in an economic model.

Table 16-3: Akie Mine Plan by Resource Classification

Zone	Tonnes (kt)	Zn Eq (%)	Zn (%)	Pb (%)	Ag (g/t)	NSR ($CAD)
Indicated	20,739	9.0	8.3	1.6	14.1	129.9
Inferred	5,061	7.8	7.2	1.4	13.0	112.8
Total Mine Plan	**25,800**	**8.8**	**7.6**	**1.5**	**13.1**	**126.5**

Notes:

1. Mineral Resources are estimated at a cut-off of 5.5% ZnEq. (ZnEq = Pct Zn + {0.45*Pct Pb})
2. Metal prices used for this estimate were: Zinc 1.17US$/lb; Lead 1.00US$/lb; Silver 16.95US$/oz
3. Mine planning tonnes include an additional 27.5kt of internal dilution at zero grade, which is neither inferred nor indicated.

Source: JDS (2018)

16.6 Material Handling

16.6.1 Mineralized Tonnes

Mucking will be carried out using 14 t scoop trams with remote tramming capabilities. Mineralized material will be trammed only as far as the cross-cut intersection where muck will be loaded into 45 t haul trucks. Material mined above the 920 L will be hauled to and dumped into the internal production pass. A chute will be located on the 920 L for truck loading. The loaded trucks will haul to surface up the primary decline. Material mined below the 920 L will be hauled up the spiral ramp and out the primary decline. Once on surface at Portal One, material will be stockpiled. A fleet of 40 t articulated surface haul trucks, loaded by a 5.6 m^3 front end loader, will transport this material from the portal to the mill.

16.6.2 Waste Tonnes

Development waste rock will be classified as one of two types:

- Non-potentially acid generating (NPAG); and
- Potentially acid generation (PAG).

Waste rock will be mucked using 14 t scoop trams and will also be loaded, at cross cuts, into 45 t haul trucks.

NAG waste rock will be hauled to surface and stockpiled at Portal One. This material will be re-handled by the surface fleet primarily to construct the tailings facility.

PAG waste rock will be stockpiled and used in cemented or un-cemented rock fill (CRF and RF). No PAG rock is planned to be stored on surface for any period of time.

16.7 Backfill

16.7.1 Paste Backfill

Paste backfill has been chosen as the primary backfill method, accounting for 94% of the placed backfill.

Due to the limited surface space available at the two portals, the paste plant will be located underground on 1220 L adjacent to Portal Two. Sulfide tailings will be pumped from the process plant to the plant via

Portal Two, and binder will be stockpiled at Portal Two. Paste will be distributed throughout the mine using overhead steel piping with thicknesses and strengths matched to pressure requirements. HDPE piping will be used as the final sacrificed pipe into the stope. The paste plant has been sized for a maximum batch production rate of 2,000 t/d.

The paste recipe will consist of 75% sulfide tailings, 5% binder and 20% water.

Paste will be delivered to empty stopes using distribution boreholes, one for every six stopes. To contain the paste, a bulkhead will be required for each stope sill drift. Engineered bulkheads will be constructed using a combination of steel piping, fabrene filter fabric, wire mesh and shotcrete.

16.7.2 CRF and RF Backfill

CRF and RF backfill will be used as a supplementary backfill system to the paste, primarily to dispose of PAG development rock. CRF backfill will contain between 3% and 5% cement with an overall average of 4% and will be self-standing. In applications where fill will not be exposed by future mining, RF will be used. CRF and RF will utilize only PAG development waste rock. CRF will be mixed in remuck bays and placed using scoop trams. All PAG development waste rock will be retained within the mine as fill.

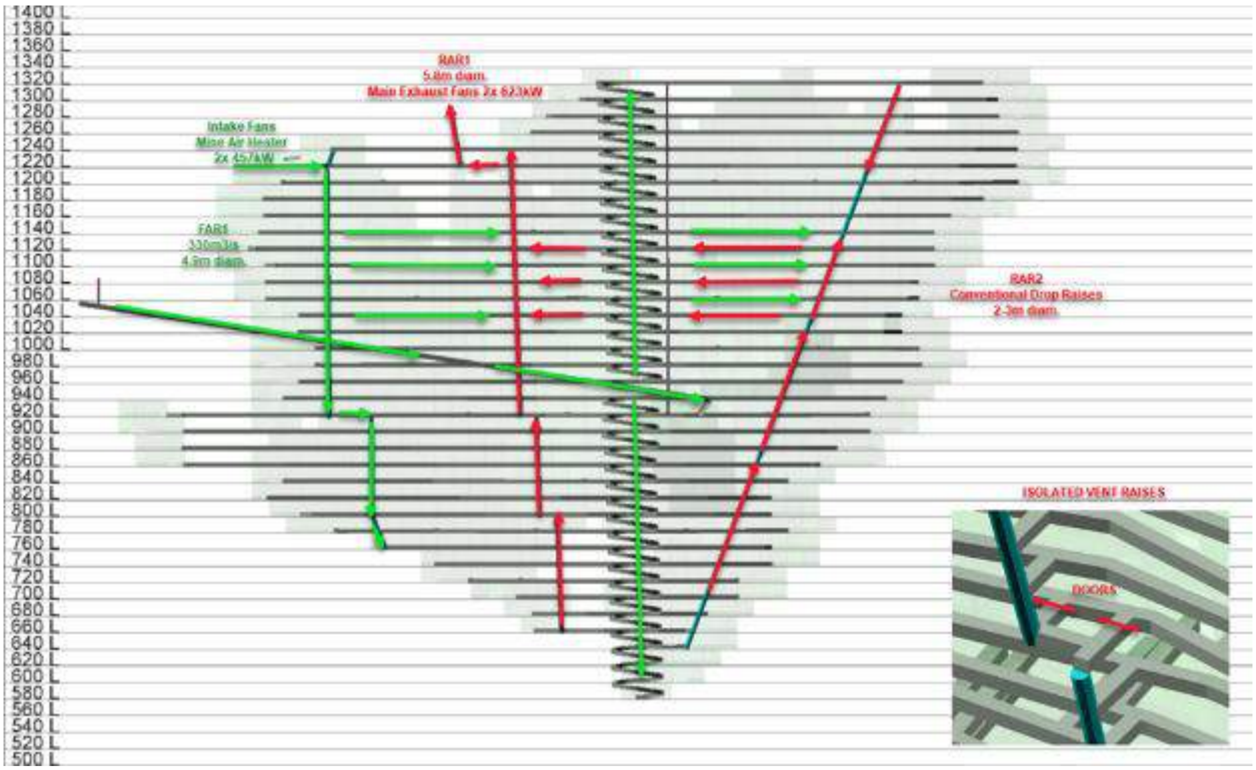
16.8 Mine Services

16.8.1 Ventilation

Ventilation for the Mine will be managed through a series of raises, two surface portals and one surface collar. The fan locations and duty points are shown on Table 16-4. Fresh air will be forced into both portals and exhaust air will be drawn out the surface raise. Air flow will be provided by a series of co-axial fans installed at both portals and the return air raise collar.

The main fans are sized to accommodate a peak airflow of 450 m^3/s. The FAR1 and main decline will provide fresh air to all the active levels of the mine. The return airflow from each level will be exhausted through the return air raise (RAR1), which will act as the only exhaust for the mine. Fresh air on the levels on the east end of the mine will flow across the footwall drift to RAR1. The ventilation schematic during production is shown in Figure 16-9.

Figure 16-9: Akie Mine Ventilation Section



Source: JDS (2018)

FAR1 will be raise bored to a final diameter of 4.9 m. RAR1 is comprised of three legs, the top leg will be raise bored with a final dimeter if 4.9 m, whereas the middle and the bottom legs are raise bored at 4.3 m diameter. RAR2 will be comprised of a series of short drop raises driven from level to level at 2.5 m diameter capable of handling 100 m^3/s, which will be developed as the mine progresses.

A minimum airflow of 25.0 m^3/s will be provided to each level through a regulator installed in the FAR1 access drift. Smaller 22 kW production fans will be employed to provide fresh air to the face of each stope, providing 18.0 m^3/s of flow through 1.4 m collapsible ducts.

Table 16-4: Fan Locations and Duty Points

Fan	Location	Airflow (Q)	Pressure (kPa)	Fan Power* (kW)	Fan Power Adjusted (kW)
Main Intake fans (x2)	FAR1 Collar	165.0	2.0	457	609
Main Exhaust fans (x2)	RAR1 Collar	225.0	2.0	623	779
Decline Development fan	Ventilation Portal and Haulage Decline Portal	23.6	2.2	83	93
Level Development fan	Each Level	23.6	2.2	83	93
Stope fan	Stope	18.0	1.0	19	22
Main Intake fans (x2)	FAR Collar	190.0	2.6	684	745

* Fan power is calculated at an air density of 1.2 kg/m^3
* Fan efficiency is assumed to be 70%
Source: JDS (2018)

16.8.2 Mine Dewatering

Dewatering activity at the mine will consist of a series of strategically placed primary sumps collecting water from active working faces. The majority of underground water will be pumped and discharged to surface from the collection sumps to be treated by the site water treatment system.

The underground dewatering system has been sized based on estimated groundwater inflow information. Maximum projected inflow during mine operations is modelled at 113.0 L/s or 1,785 USGPM ("*Cardiac Creek Deposit – Summary of Hydrogeology and Preliminary Estimate of Mine Inflows for Preliminary Economic Assessment*", Lorax 2018).

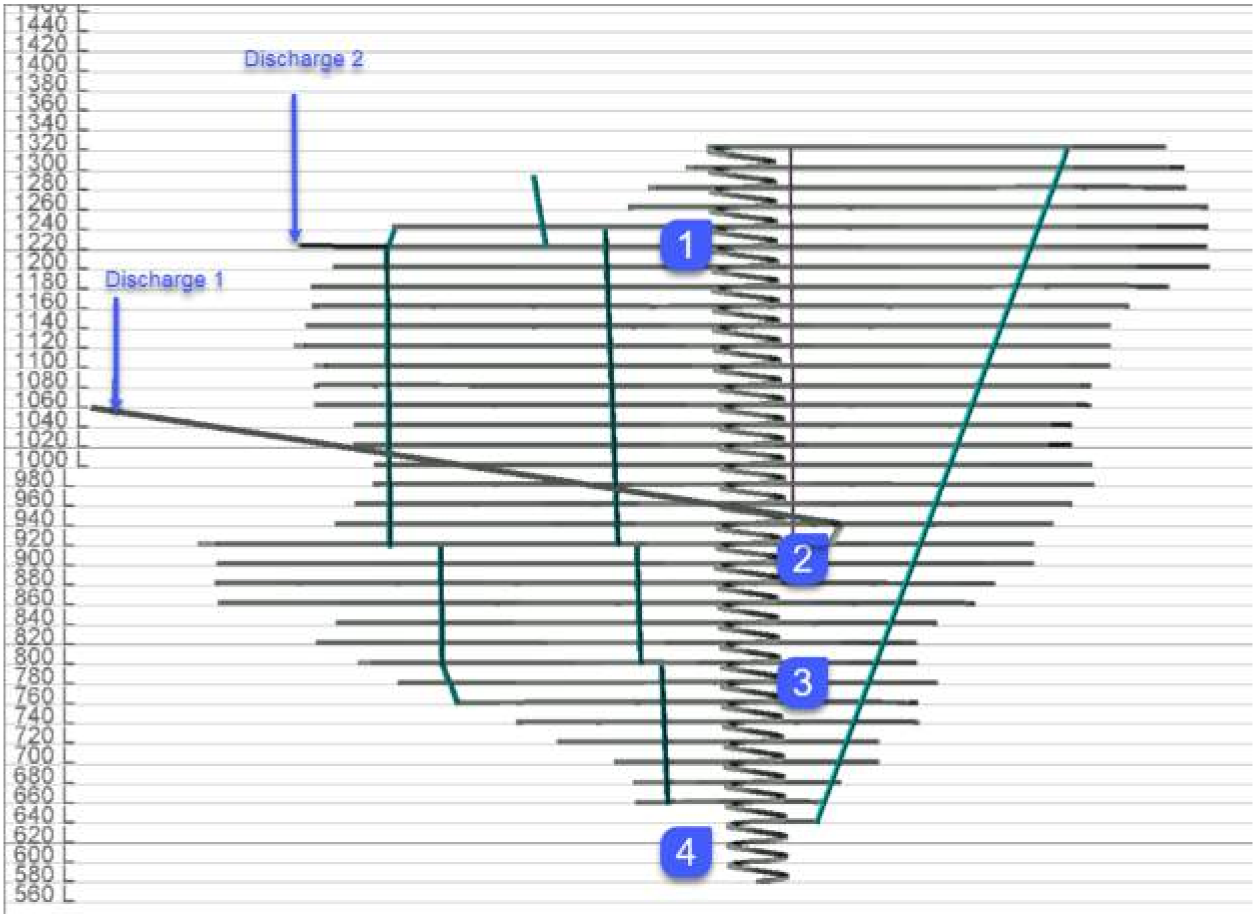
Primary pumping will be conducted using a series of high volume electric submersible pumps. The size, quantity and power requirements of these pumps are summarized in Table 16-5. The pump locations are shown in Figure 16-10.

Table 16-5: Akie Mine Pumping Summary

Primary Pumping Summary	Level	Equivalent Pump	Discharge (inches)	# of Pumps	Power (kW)
Primary Collection Sump 1	1220	LH845	8.0	1	56
Primary Collection Sump 2	920	LH8110	8.0	3	388
Primary Collection Sump 3	820	LH8110	8.0	2	276
Primary Collection Sump 4	580	LH8110	8.0	5	650

Source: JDS (2018)

Figure 16-10: Akie Property Pumping Locations



Source: JDS (2018)

In addition to the primary dewatering pumps, tertiary pumping will be required in various working faces to direct water to the main dewatering sumps. Tertiary dewatering will be conducted using a series of easily moved pneumatic and electric submersible pumps that connect to service water lines. Discharge locations for these pumps will be internal to the mine, with the majority of the water being directed to the spiral ramp. A summary of tertiary pumping requirements is presented in Table 16-6.

Table 16-6: Additional Pumping Requirements

Tertiary Pumping Estimate	# of Pumps
Ramp Face Pumps - 15hp	5
Level Pumps - 30hp	10
Wilden Pneumatic	10

Source: JDS (2018)

16.8.3 Compressed Air and Water Supply

Compressed air will be supplied throughout the mine. Pressure will be provided from a surface compressor located at Portal Two. To maintain consistent pressures throughout the operation, underground surge tanks will be positioned in ancillary bays at strategic positions throughout the mine.

All water used for underground operations such as drilling will be drawn from collected mine inflow. Operations water will be cleaned and filtered and re-distributed to the working faces through 6-inch overhead water lines.

16.8.4 Electrical Distribution

Power will be supplied from the site power station at 13.8 kV via overhead lines to a step-down transformer at a laydown pad adjacent to the Portal One pad. Power will be distributed from this transformer underground using 4160 V overhead service lines and bore-holes between levels. Each working level will include a primary sub-station and power panel off the spiral ramp where power will be further stepped down to 1 kV and distributed to the working faces. Total average installed load during production is estimated to be 7.5 MW with an average load of 5.6 MW.

16.9 Mine Personnel

The mine will require a full-time work force of mining, maintenance, services, technical and administrative personnel. Mine operations will be run 365 d/a, 22 h/d, primarily through two – 11 hour shifts, allowing one hour for smoke clearing between shifts. Mine operations will consist of personnel working two different rotations;

- Two week on / two week off (2x2): Mine and Mill Operations, Maintenance, Construction Labor, Site Services (12 hour shifts); and
- Four days on / three days off (4x3): Engineering, Administrative, Management (day shift only).

During full production the mine will require 147 people on site, including those on 4x3 rotations, and a total payroll of 276 workers.

Staffing will be ramped up to full production requirements during the first year of operations. Certain production related positions are not expected to be necessary during preliminary mine development and construction, such as ore control geologist, some non-critical engineering and administrative roles and apprentice / trainee roles.

Similarly, some positions, such as training, have been reduced or eliminated during the winding down period of the final years of operations.

A break-down of the on-site requirement of personnel in the ramp-up, production and ramp-up years is presented in Table 16-7.

Table 16-7: On-site Personnel, Mine Operations

On-Site Personnel	Year -1	Peak Production	Year 19
Mining Management	9	9	6
Operations	26	50	32
Services	30	44	2
Mine Maintenance	9	23	9
Technical Services	8	21	10
Grand Total	**82**	**147**	**59**

Source: JDS (2018)

16.10 Mine Equipment

16.10.1 Mobile Equipment

Diesel and electric over diesel equipment will employed throughout the mine. The primary haulage fleet will consist of low profile 45 t articulated haul trucks and 5.4 m^3 scoop trams. Development drilling will be conducted using two-boom jumbos and long-hole drilling will be conducted using Simba type long-hole drills.

Surface haulage of the mineralized rock to the plant site, and NAG waste to the tailings facility will be performed by 40 t articulated haul trucks loaded by 5.6 m^3 front end loaders (FEL).

Equipment requirements were developed from first principles, based on the maximum annual duty hours for an individual piece of equipment, modified for mechanical availability and projected utilization.

A list of the underground production and support equipment and respective factors used in the mine plan are shown in Table 16-8.

Table 16-8: Mine Mobile Equipment Summary

Mobile Equipment	Max # of Units	Mech. Availability (%)	Average LOM Utilization (%)
Underground Haul Truck (45t/21.3m^3)	6	85%	89%
LHD (14t/5.4m^3)	4	80%	89%
Jumbo - 2 Boom	4	65%	49%
Bolter	2	70%	54%
Longhole Drill	4	70%	77%
Large Explosives Truck	2	80%	66%
Scissor Lift	2	85%	39%
Shotcrete + Transmixer	2	70%	44%
Jackleg / Stoper	4	90%	2%
Grout Pump	2	70%	1%
Personnel Carrier	4	85%	89%
Fuel / Lube Truck	2	85%	71%
Boom Truck	2	85%	56%
Electrician Truck	2	85%	69%
Grader	2	85%	69%
Utility Vehicle	4	85%	42%
Backhoe	2	85%	28%
Telehandler	2	85%	42%
Mechanics Truck	2	85%	85%
Pickup Truck	6	85%	49%
FEL (5.6 m^3, WA500)	1	80%	79%
Surface Haul Truck (40t)	4	85%	40%

Source: JDS (2018)

16.10.2 Fixed Plant Equipment

Fixed plant equipment in the mine plan includes a ventilation fans, dewatering pumps, a paste backfill plant and the electrical distribution system, is summarized in Table 16-9.

Table 16-9: Mine Fixed Plant Equipment Summary

Fixed Plant Equipment	Max # of Units
Ventilation	
1,190 hp axial fan	2
1,730 hp axial fan	2
320 hp axial fan	1
mine air heater system	1
Primary Dewatering Pumps	
150hp	22
60hp	1
Backfill	
underground paste plant	1
CRF plant	1
Services	
13.8-4.2kV transformer (surface)	1
4.2kV underground substation	15
surface compressor	1

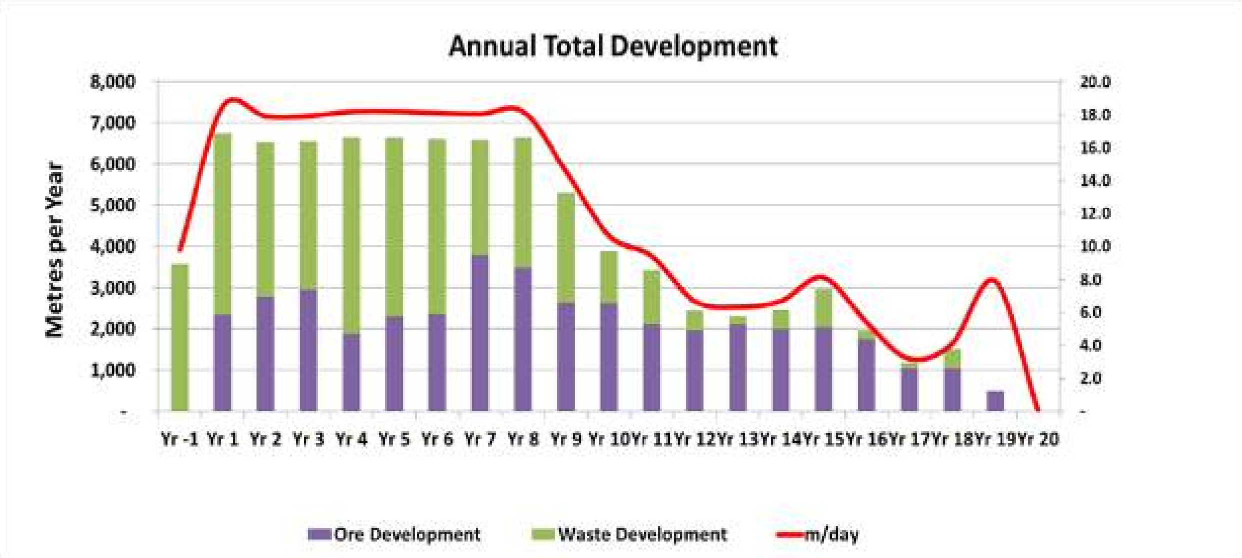
Source: JDS (2018)

16.11 Mine Schedules

16.11.1 Mine Development Schedule

The mine development schedule is based on accessing and developing the highest grade production levels of the mine first. The schedule has also been designed to provide secondary egress and positive ventilation flow throughout the mine prior to production. The development schedule is summarized in Figure 16-11.

Figure 16-11: Mine Development Schedule



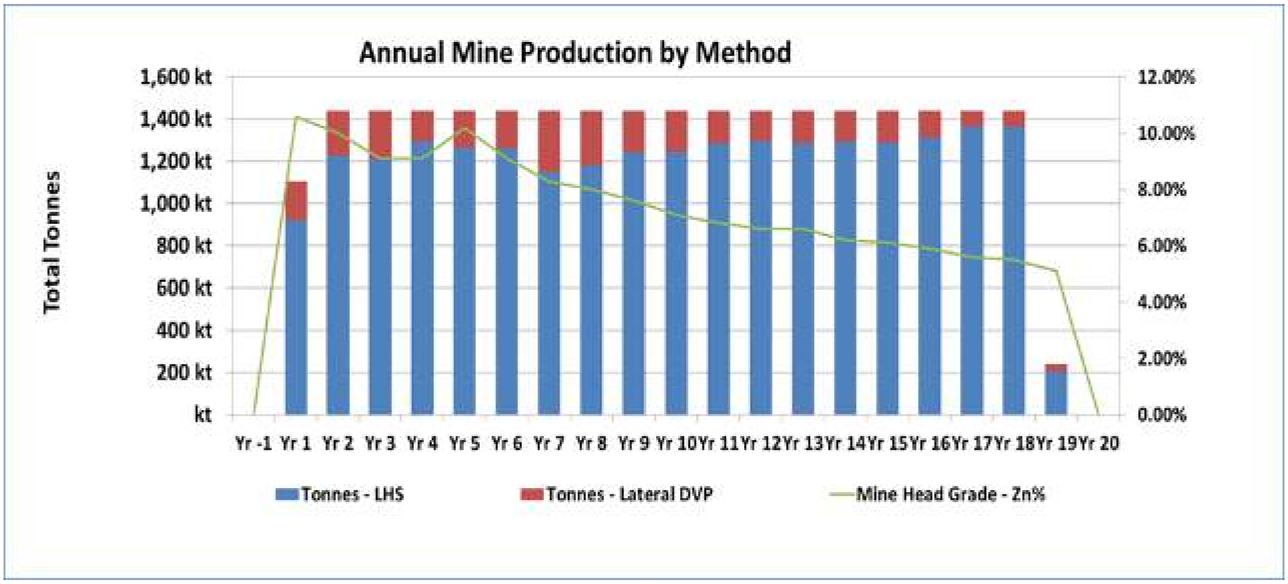
Source: JDS (2018)

16.11.2 Mine Production Schedule

Mine production is expected to commence in year one, at 1.1 Mt mined, approximately 80% of the steady-state production rate. The mine is expected to produce at a full production rate of 1.4 Mt/a for 17 years (Years 2 to 18) with production ending in Q1 of Year 19. A summary of the mine production schedule is presented in Figure 16-12, and summarized as follows:

- Years 1 to 4: mining will begin on the 920 L, accessing the thicker and higher grade portions of the orebody and proceeding upward level by level;
- Years 5 to 8: with mining complete for the highest grade portions of levels 820 to 920 L, mining will commence at 820 L to recover the high grade portion of mine tonnes between 820 L and 920 L; and
- Years 9 to 19: with the high-grade core extracted, mining will continue laterally to the maximum extents on each level and upward to 1320 L, the highest in the mine and the spiral ramp is extended downward and the lowest portion of the deposit is extracted.

Figure 16-12: Mine Production Schedule

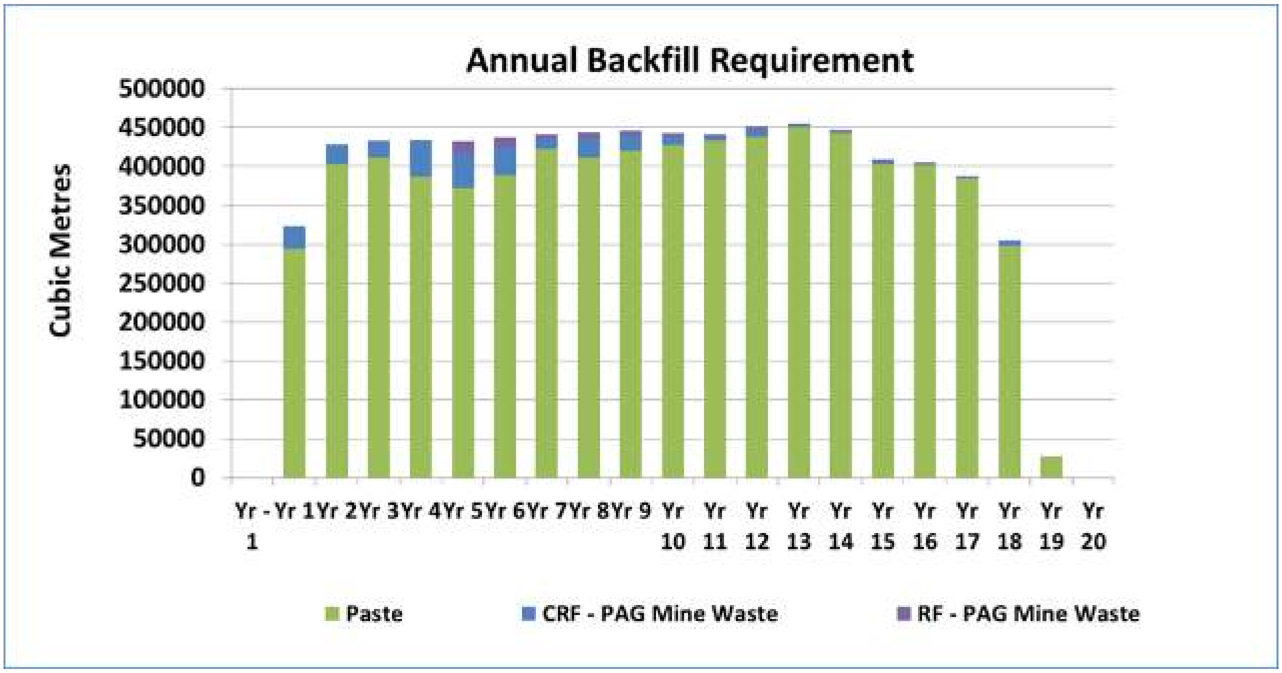


Source: JDS (2018)

16.11.3 Mine Backfill and Material Movement Schedule

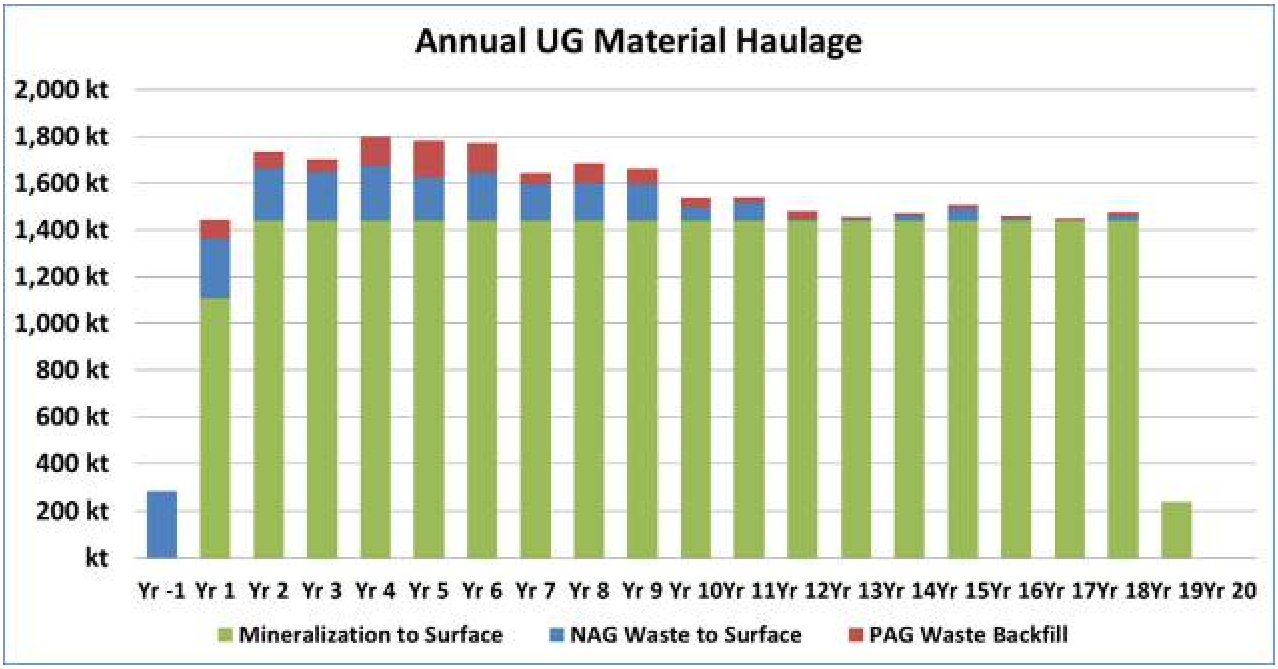
Mine backfill and material movement are based on the mining sequence of both development waste and production tonnages. The schedule includes the distribution of paste backfill, as well as the movement of PAG and NAG waste rock for both backfill and construction activities. The backfill and material movement schedule is summarized in Figure 16-12 and Figure 16-13.

Figure 16-13: Mine Backfill Schedule



Source: JDS (2018)

Figure 16-14: Mine Haulage Schedule



Source: JDS (2018)

17 Recovery Methods

The ZincX's Akie Project focuses on developing the Cardiac Creek Pb/Zn/Ag deposit. The recent metallurgical test program completed at Base Metallurgical Labs in Kamloops, BC (BL0148), summarized in Section 13, has demonstrated that standard Pb and Zn sequential flotation, with pre-concentration using dense media separation (DMS), can yield an overall Pb recovery of 46.2%, at a concentrate grade of 45.1% Pb, and a Zn recovery of 88.9%, at a concentrate grade of 52.4% Zn (BL0148-LCT21). Results from this test program were used to develop the corresponding process design criteria, mechanical equipment list, flowsheets and operating costs.

The process plant will include:

- Three stages of crushing;
- Dense media separation;
- Two stages of ball mill grinding in reverse closed-circuit with cyclones;
- Sequential Pb and Zn flotation circuits, each incorporating three cleaning stages;
- Concentrate dewatering circuits using thickeners and pressure filters;
- Concentrate storage and load-out facilities; and
- Dewatering, filtering and storage of dry stack tailings.

The crushing plant will have a throughput of 4,000 t/d with average life of mine (LOM) head grades of 1.48% Pb, 13 g/t Ag and 7.63% Zn. The circuit will operate at an availability of 75%, resulting in an hourly throughput of 222 t/h. The DMS, milling and flotation circuits will operate 24 h/d, 365 d/a with an estimated availability of 92%. DMS pre-concentration will reject 25% of the plant feed as waste; the milling, flotation, and dewatering circuits are designed for a throughput of 3,000 t/d.

The three-stage crushing circuit will reduce the material to a product size of 80% passing (P_{80}) 8.8 mm. The DMS circuit will then reject 25% of the feed material while maintaining metal recoveries of 96.3% Pb and 98.5% Zn. The subsequent two stage grinding circuit will target a P_{80} grind size of 56 µm, before Pb and Zn are recovered into concentrates using sequential flotation. Zn rougher and Zn 1st cleaner tailings, designated as final tailings, will be thickened and pressure filtered to a moisture content of 15% and transferred to a dry stack tailings facility.

17.1 Introduction

The recovery method will consist of the following unit operations:

- **Primary Crushing** – A vibrating grizzly feeder and jaw crusher in open circuit, producing a final product P_{80} of 100 mm;
- **Secondary / Tertiary Crushing** – Two stages of cone crushing in closed circuit with a double deck vibrating screen, producing a final product P_{80} of 8.8 mm;

- **Crushed Material Stockpile and Reclaim** – A 24 h live capacity stockpile (4,000 t) with two reclaim belt feeders feeding the DMS circuit;
- **Dense Media Separation** – Dense media cyclones with a cut SG of 2.80 to pre-concentrate sulphide minerals and reject waste material;
- **Primary Grinding** – A ball mill in open circuit, producing a T80 transfer size of approximately 250 µm;
- **Secondary Grinding** – A ball mill in reverse closed circuit with a cluster of hydro-cyclones, producing a final product P_{80} of 56 µm;
- **Pb Flotation** – Rougher and cleaner flotation to produce a saleable Pb concentrate;
- **Pb Rougher Concentrate Regrind** – A stirred regrind mill in open circuit, reducing Pb rougher concentrate to a P_{80} of 10 µm;
- **Pb Concentrate Dewatering** – A 4 m diameter high-rate thickener to achieve an underflow solids density of 55%, and a pressure filter to reduce the concentrate to a final moisture content of 8%;
- **Zn Flotation** – Rougher and cleaner flotation to produce a saleable Zn concentrate;
- **Zn Rougher Concentrate Regrind** – A stirred regrind mill in open circuit, reducing Zn rougher concentrate to a P_{80} of 15 µm;
- **Zn Concentrate Dewatering** – A 12 m diameter high-rate thickener to achieve an underflow solids density of 55%, and a pressure filter to reduce the concentrate to a final moisture content of 8%; and
- **Final Tailings Dewatering** – A filter plant to reduce final tailings to a moisture content of 15% for dry stacking.

17.2 Plant Design Criteria

17.2.1 Process Design Criteria

The Process Design Criteria and Mass Balance detail the annual production capabilities, major mass flows and capacities, and availability for the process plant. Consumption rates for major operating and maintenance consumables can be found in the operating cost estimate described in Section 22. Key process design criteria from Section 13 are summarized in Table 17-1.

Table 17-1: Process Design Criteria

Criteria	Unit	Nominal Value	Source
General			
Crushing Plant Throughput	t/d	**4,000**	2018 mine plan
Process Plant Throughput	t/d	**3,000**	Engineering Calculation - 25% waste rejection
Process Plant Availability	%	**92**	Industry Standard
Process Plant Throughput	t/h	**136**	Engineering Calculation
LOM Average Pb Head Grade	%	**1.48**	2018 mine plan
LOM Average Zn Head Grade	%	**7.63**	2018 mine plan
LOM Average Ag Head Grade	g/t	**13**	2018 mine plan
Overall Pb Recovery	%	**46.2**	Base Met (2018): BL0148 LCT-21 Post DMS pre-concentration
Pb Concentrate Grade	% Pb	**45.1**	Base Met (2018) LCT-21
Overall Zn Recovery	%	**88.9**	Base Met (2018): BL0148 LCT-21 Post DMS pre-concentration
Zn Concentrate Grade	% Zn	**52.4**	Base Met (2018) LCT-21
Crushing			
Availability/Utilization	%	**75**	Industry Standard
Crushing Plant Throughput	t/h	**222**	Engineering Calculation
Number of Crushing Stages	-	**3**	Vendor Recommended – three stage crushing plant
Crushing System Product Size (P_{80})	mm	**8.8**	Vendor Simulation - estimated based on a final product aperture screen size of 12 mm
Crushed Material Stockpile			
Stockpile Capacity (live)	t	**4,000**	Design Consideration
Stockpile Capacity (live)	h	**24**	Engineering Calculation
Dense Media Separation			
Equipment Type	-	**Dense Media Cyclones**	Design Consideration – based on top size of material
Dense Media Type	-	**Ferrosilicon**	Industry Standard
Operating Specific Gravity	SG	**2.80**	Base Met (2018): BL0148
Pb Stage Recovery	% Pb	**96.3**	Base Met (2018): BL0148 Average of DMS variability testing
Zn Stage Recovery	% Zn	**98.5**	Base Met (2018): BL0148 Average of DMS variability testing
Mass Rejection	%	**25**	Base Met (2018): BL0148 Average of DMS variability testing
Grinding			
Bond Ball Mill Work Index (overall)	kWh/t	**16.9**	Base Met (2018): BL0148

Criteria	Unit	Nominal Value	Source
			Global Composite
Bond Abrasion Index	g	**0.225**	Estimated based on results from similar projects
Primary Grinding Mill Type	-	**Ball Mill**	Industry Standard for primary grinding to target transfer size
Mill Diameter	m	**4.3**	Vendor Recommended
Mill Length	m	**7.3**	Vendor Recommended
Installed Power	kW	**2,238**	Vendor Recommended
Circuit Configuration	-	**Open**	Design Consideration
Primary Grinding Transfer Size (T_{80})	µm	**250**	Design Consideration
Secondary Grinding Mill Type	-	**Ball Mill**	Selected to achieve target product size
Mill Diameter	m	**4.3**	Vendor Recommended
Mill Length	m	**7.3**	Vendor Recommended
Installed Power	kW	**2,238**	Vendor Recommended
Circuit Configuration	-	**Reverse Closed**	Industry Standard
Circulating Load	%	**300**	Industry Standard
Final Product Size (P_{80})	µm	**56**	Base Met (2018): BL0148 Global Composite
Flotation			
Rougher Flotation Time Scale-up	-	**2.5**	Industry Standard
Cleaner Flotation Time Scale-up	-	**4.0**	Industry Standard
Pb Rougher Flotation			
Laboratory Retention Time	min	**5.5**	Base Met (2018): BL0148 LCT-21
Design Retention Time	min	**13.8**	Engineering Calculation based on 2.5x scale-up factor
Number of Rougher Flotation Cells	#	**6**	Designed to achieve retention time
Rougher Flotation Cell Size	m^3	**20**	Designed to achieve retention time
Installed Retention Time	min	**20**	Engineering Calculation
Pb Regrind Circuit			
Rougher Concentrate Mass Pull	%	**8.3**	Base Met (2018): BL0148 LCT-21
Regrind Mill Type	-	**Stirred Mill**	Industry Standard
Final Product Size (P_{80})	µm	**10**	Base Met (2018): BL0148 LCT-21
Pb Cleaner Flotation			
Number of Stages	#	**3**	Base Met (2018): BL0148 LCT-21
Laboratory Retention Time	min	**7 / 4 / 3**	Base Met (2018): BL0148 LCT-21
Design Retention Time	min	**28 / 16 / 12**	Engineering Calculation based on 4.0x scale-up factor

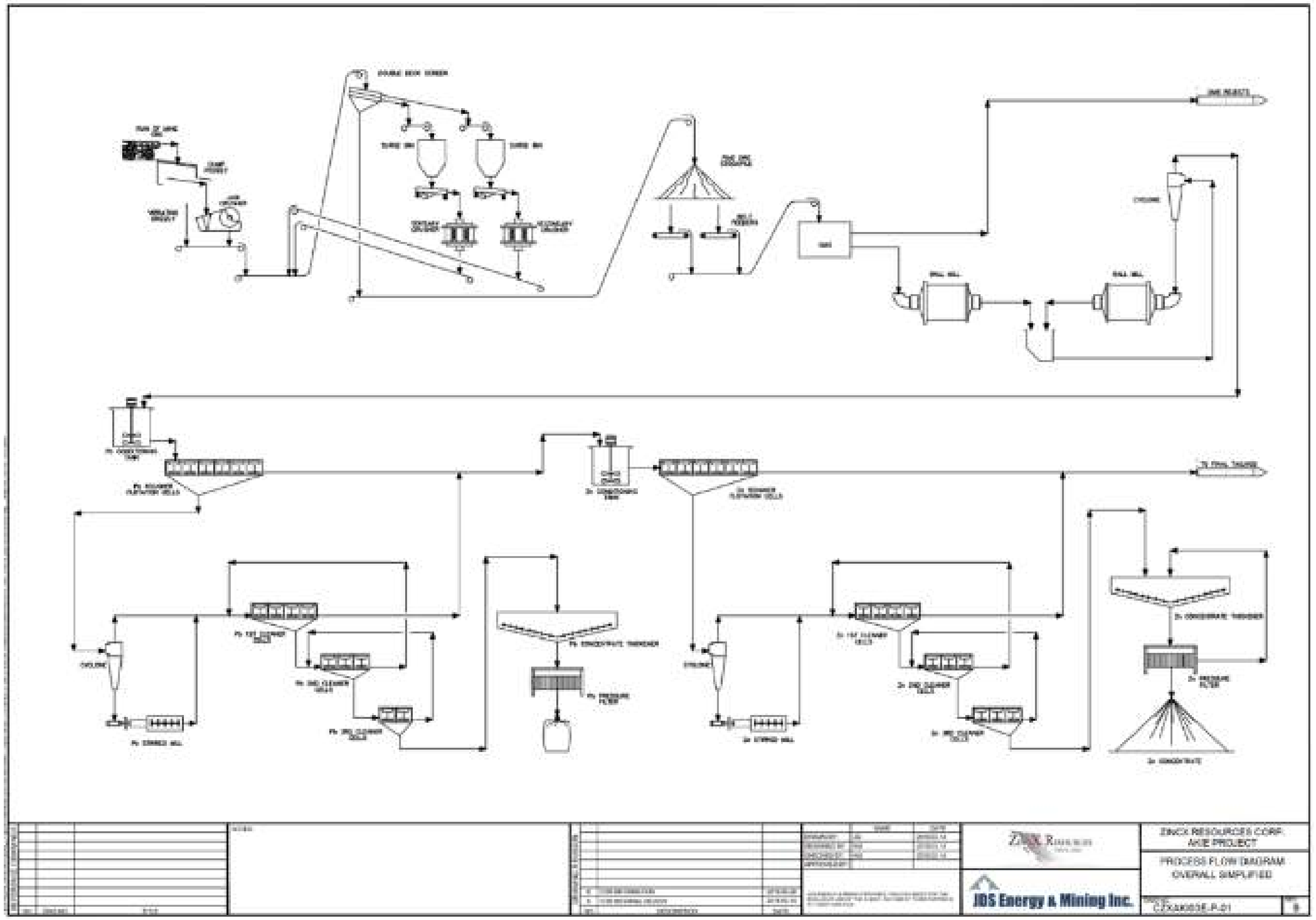
Criteria	Unit	Nominal Value	Source
Number of Cleaner Flotation Cells	#	**6 / 3 / 3**	Designed to achieve retention time
Cleaner Flotation Cell Sizes	m^3	**5 / 3 / 1.5**	Designed to achieve retention time
Installed Retention Time	min	**36 / 17 / 17**	Engineering Calculation
Pb Stage Recovery	%	**46.2**	Base Met (2018): BL0148 LCT-21
Zn Rougher Flotation			
Laboratory Retention Time	min	**9**	Base Met (2018): BL0148 LCT-21
Design Retention Time	min	**22.5**	Engineering Calculation based on 2.5x scale-up factor
Number of Rougher Flotation Cells	#	**6**	Designed to achieve retention time
Rougher Flotation Cell Size	m^3	**20**	Designed to achieve retention time
Installed Retention Time	min	**20**	Engineering Calculation
Zn Regrind Circuit			
Rougher Concentrate Mass Pull	%	**27.6**	Base Met (2018): BL0148 LCT-21
Regrind Mill Type	-	**Stirred Mill**	Industry Standard
Final Product Size (P_{80})	µm	**15**	Base Met (2018): BL0148 LCT-21
Zn Cleaner Flotation			
Number of Stages	#	**3**	Base Met (2018): BL0148 LCT-21
Laboratory Retention Time	min	**9 / 6 / 4**	Base Met (2018): BL0148 LCT-21
Design Retention Time	min	**36 / 24 / 16**	Engineering Calculation based on 4.0x scale-up factor
Number of Cleaner Flotation Cells	#	**8 / 6 / 3**	Designed to achieve retention time
Cleaner Flotation Cell Size	m^3	**10**	Designed to achieve retention time
Installed Retention Time	min	**35 / 30 / 22**	Engineering Calculation
Zn Stage Recovery	%	**88.8**	Base Met (2018): BL0148 LCT-21
Concentrate Dewatering			
Thickener Type	-	**High Rate**	Industry Standard
Pb Thickener Loading Rate	$t/h/m^2$	**0.26**	Design Consideration
Pb Thickener Diameter	m	**4**	Vendor Recommended
Zn Thickener Loading Rate	$t/h/m^2$	**0.27**	Design Consideration
Zn Thickener Diameter	m	**12**	Vendor Recommended
Filtration Type	-	**Pressure**	Industry Standard
Final Concentrate Moisture Content	%	**8**	Design Consideration

Source: JDS (2018)

17.3 Plant Description

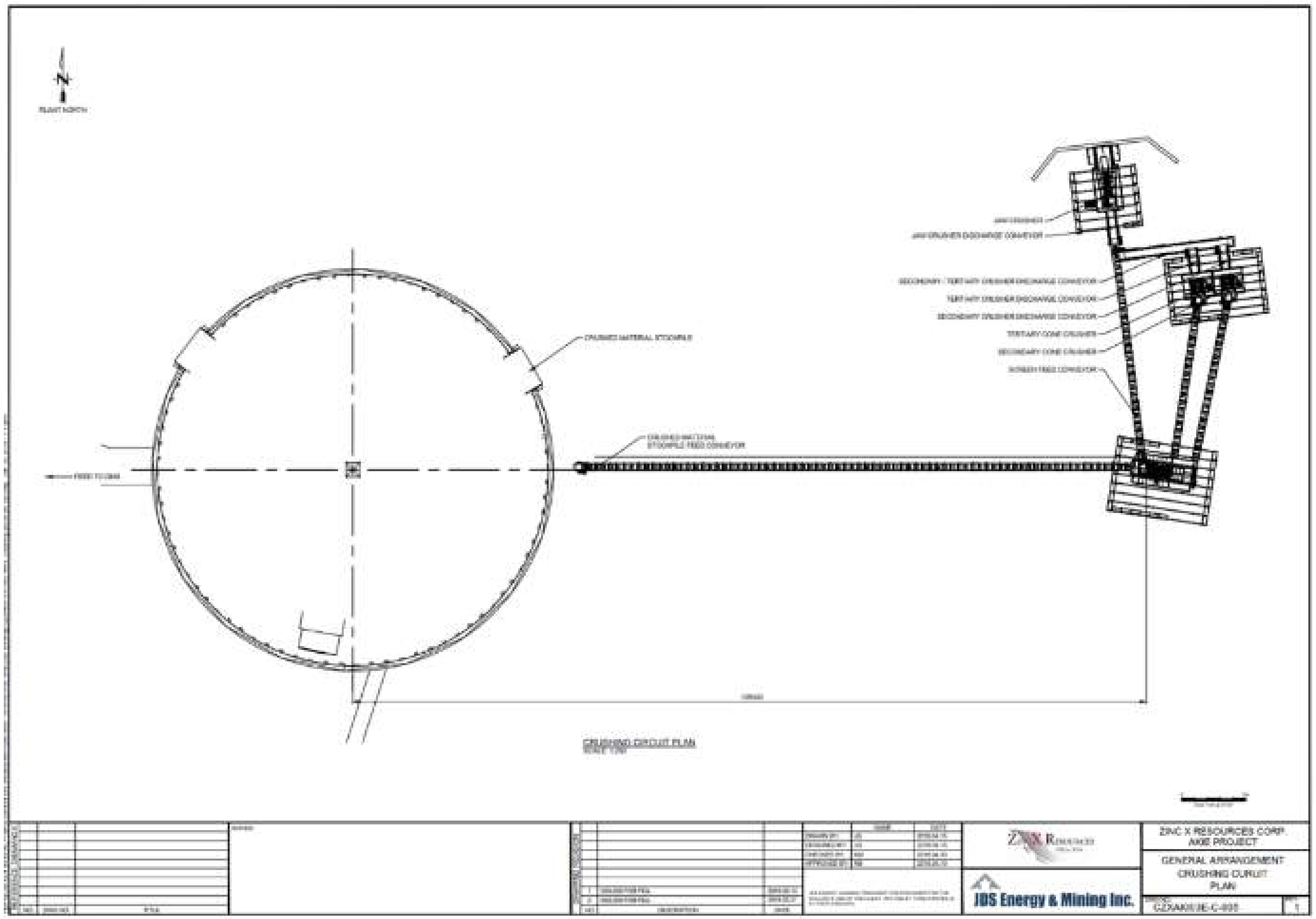
A summary of the process flowsheet is presented in Figure 17-1. Models of the crushing and process facilities are displayed in Figure 17-2 and Figure 17-3, respectively.

Figure 17-1: Overall Process Flowsheet



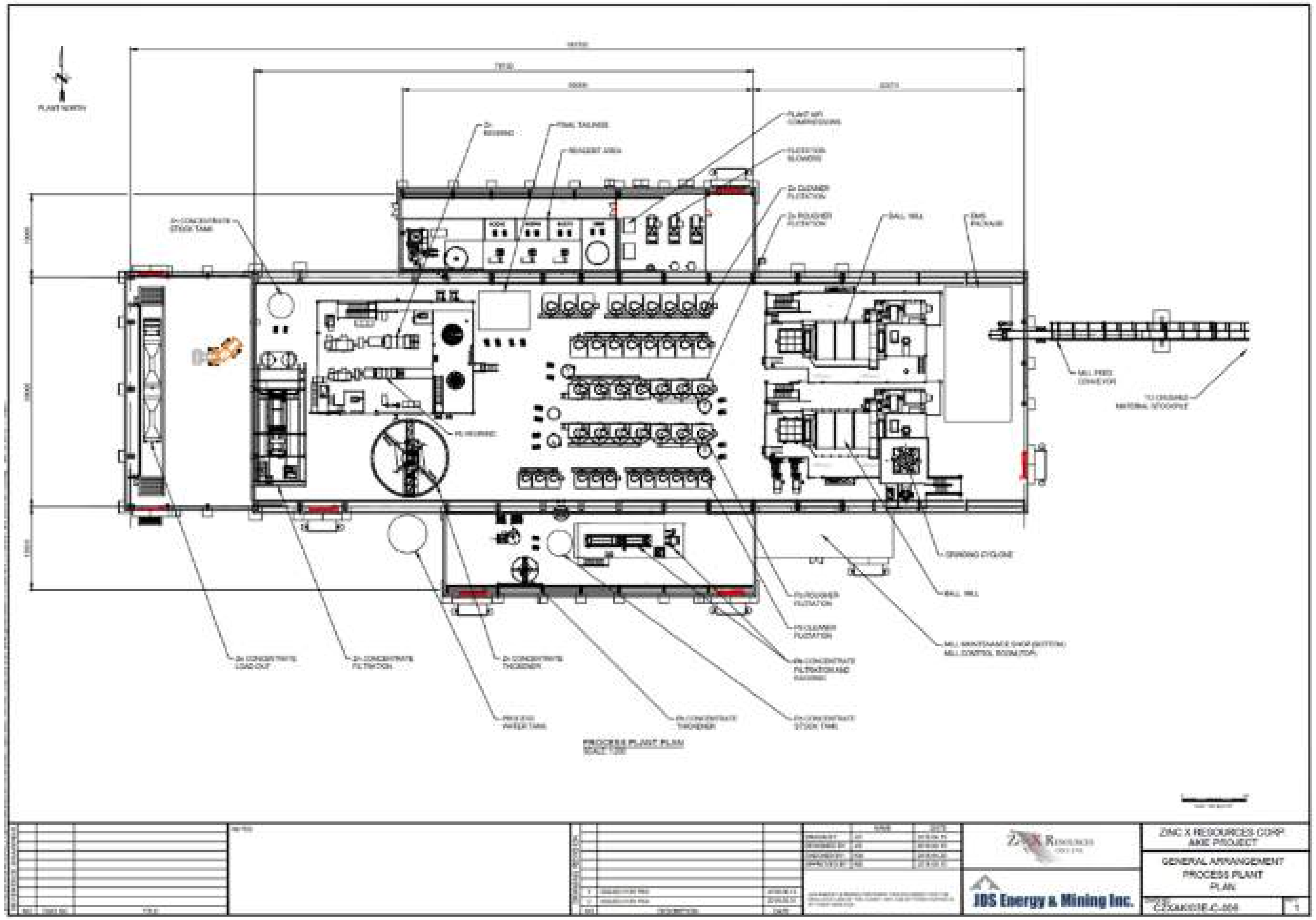
Source: JDS (2018)

Figure 17-2: Crushing and Mineralized Material Storage Layout



Source: JDS (2018)

Figure 17-3: Process Plant Layout



Source: JDS (2018)

17.4 Process Plant Description

17.4.1 Crushing

Material from the underground mine will feed a crushing plant that consists of three stages of crushing. The plant will process 222 t/h of material, operate 18 h/d and produce a final product P_{80} of 8.8 mm.

17.4.1.1 Primary Crushing

Material will be stockpiled near the jaw crusher or direct dumped through an 800 mm static grizzly into a dump pocket. Stockpiled ROM material will be re-handled by a front-end loader and fed into the crusher. The material will discharge through the static grizzly into a 40 t live feed hopper. Oversize material from the static grizzly will be removed for later size reduction using a rock breaker.

A vibrating grizzly feeder will draw material from the feed hopper at a rate of 222 t/h. The vibrating grizzly oversized material will feed directly into a 762 mm x 1,067 mm (30” x 42”) jaw crusher with an installed power of 110 kW. The undersized -75 mm material will bypass the crusher and feed directly onto the screen feed conveyor. The primary crushing stage will produce a product P_{80} of approximately 100 mm at a crusher closed side setting (CSS) of 89 mm.

The screen feed conveyor will collect crushed product from all three stages of crushing and feed a 2,438 mm x 6,096 mm (8’ x 20’) double-deck vibrating screen. The top deck will have an aperture size of 35 mm, and the +35 mm material will be conveyed to the secondary crusher. The bottom deck will have an aperture size of 12 mm, and the -35 mm, +12 mm material will be conveyed to the tertiary crusher. The -12 mm final product, at an estimated P_{80} of 8.8 mm, will discharge onto the Stockpile Feed Conveyor and be transferred to the Crushed Material Stockpile.

17.4.1.2 Secondary Crushing

Material from the secondary crusher feed conveyor will discharge into a cone crusher with an installed power of 132 kW. The secondary crusher will reduce the material to a nominal product P_{80} of approximately 25 mm using a CSS of 25.4 mm. Crushed product will be transferred to the screen feed conveyor and be circulated back to the double-deck screen.

17.4.1.3 Tertiary Crushing

Material from the Tertiary Crusher Feed Conveyor will discharge into a cone crusher with an installed power of 160 kW. The tertiary crusher will reduce the material to a nominal product P_{80} of 12 mm with a CSS of 12.7 mm. Crushed product will be transferred to the Screen Feed Conveyor and be circulated back to the double-deck screen.

17.4.2 Crushed Material Stockpile

The double-deck screen undersize, with a final P_{80} product size of 8.8 mm, will be conveyed to the Crushed Material Stockpile. The stockpile will provide 4,000 t, or twenty-four hours, of live storage capacity. Two belt feeders, located in a corrugated tunnel under the stockpile, will be installed with variable frequency drives (VFD) to control the reclaim rate feeding the DMS circuit. Each belt feeder will be capable of providing the total throughput of 181 t/h.

17.4.3 Dense Media Separation

The DMS circuit will be used to pre-concentrate crushed material prior to grinding. The DMS circuit will use dense media cyclones and a ferrosilicon dense media to separate the material based on an SG cut point of 2.80. The lighter material, or floats, contains waste and will be rejected from the process as tailings. The heavier material, or sinks, contains the sulphides and will be sent to the grinding circuit for further processing. The DMS circuit is designed to process 181 t/h (4,000 t/d) of crushed material and reject 25% of the feed as waste, resulting in 136 t/h (3,000 t/d) of product feeding the grinding circuit.

Crushed material will be fed onto a feed preparation screen. At an aperture size of 1.4 mm, the material will be wet screened to remove any fines. The fine -1.4 mm material will be pumped to the grinding circuit, while the coarse +1.4 mm material will discharge into a mixing vessel and combine with the circulating dense media (ferrosilicon). This mixture will then be pumped to a cluster of two 510 mm dense media cyclones.

The cyclone overflow, containing the lighter floats, will flow by gravity onto a drain and rinse screen. The ferrosilicon dense media will be removed by spray water and recovered in the screen undersize, while the screen oversize will discharge onto a conveyor and be transferred to a stockpile. This waste material will eventually be trucked to a waste dump or used as aggregate for road construction.

The cyclone underflow, containing the heavier sinks, will flow by gravity onto a drain and rinse screen. The ferrosilicon dense media will be removed by spray water and recovered in the screen undersize, while the screen oversize will discharge onto a conveyor and be sent to the grinding circuit.

The undiluted media from both drain and rinse screens, comprised of the first one-third of the screen undersize, will be pumped to an agitated dense media storage tank and circulated back into the circuit. The diluted media, comprised of the final two-thirds of the screen undersize, will be pumped to a magnetic separator where the ferrosilicon will be recovered and transported to the dense media storage tank. Water and fresh ferrosilicon will periodically be added to the dense media storage tank to maintain the SG cut point of 2.80.

The DMS reject will be co-disposed of in the Tailings Management Facility (TMF) along with the filtered tailings.

17.4.4 Grinding

The grinding circuit will consist of a primary ball mill followed by a secondary ball mill. The primary ball mill will operate in open circuit, while the secondary ball mill will operate in reverse closed circuit with a cluster of hydro-cyclones. The grinding circuit will be able to process a nominal throughput of 136 t/h (fresh feed) and produce a final product P_{80} of 56 µm.

Sinks and fines from the DMS circuit will feed a 4.3 m diameter x 7.3 m long overflow ball mill via the ball mill feed conveyor. The mill will be installed with a 2,238 kW induction motor. A belt-scale on the feed conveyor will monitor feed rate. Water will be added to the ball mill to maintain the slurry charge in the mill at a constant density of 70%. Slurry will overflow from the ball mill onto a trommel screen attached to the discharge end of the mill. The trommel screen oversize will discharge into a trash bin for removal from the system.

Product from the primary ball mill, at an approximate T80 transfer size of 250 µm, will flow into the cyclone feed pump box and combine with the secondary ball mill discharge before being pumped up to a cluster of ten (eight operating / two standby) 375 mm hydro-cyclones for size classification. The coarse underflow will flow by gravity to the secondary ball mill for additional grinding, while the fine overflow, at a final product P_{80} of 56 µm, will report to the Pb conditioning tank. The hydro-cyclones have been designed for a 300% circulating load.

Cyclone underflow will feed a 4.3 m diameter x 7.3 m long overflow ball mill with an installed power of 2,238 kW. Ground slurry will overflow from the ball mill onto a trommel screen attached to the discharge end of the mill. The trommel screen oversize will discharge into a trash bin for removal from the system, while the undersize will flow into the cyclone feed pump box.

Both ball mills are the same size to allow for common spares.

17.4.5 Lead Flotation

Cyclone overflow will flow by gravity to a 13 m^3 Pb conditioning tank, which will provide 2 minutes of conditioning time prior to Pb flotation. Frother methyl isobutyl carbinol (MIBC), sulphide collector sodium isopropyl xanthate (SIPX), Zn depressant sodium cyanide (NaCN), pH modifier lime and carbon depressant PE26 will be added to the conditioning tank. The slurry will then gravitate to the rougher flotation circuit, which consists of six 20 m^3 flotation tanks cells operating in series.

Rougher concentrate will be collected in a common launder and fed to a conical pump box. The de-aerated concentrate will then be pumped to the Pb regrind circuit for further mineral liberation, while the Pb rougher tailings will be pumped to the Zn conditioning tank.

Pb rougher concentrate will be pumped to a cluster of three (two operating / one spare) 150 mm densifying cyclones to achieve a cyclone underflow density of 50%. The cyclone underflow will then flow by gravity to a pump box where density control water will be added to ensure an adequate feed density to the regrind mill. The slurry will then be pumped to an 800 kW stirred mill where high-intensity grinding with 2 mm ceramic grinding media will reduce the bulk concentrate to a P_{80} of 10 µm. The product will then combine with the cyclone overflow and be transported to the Pb first cleaner flotation circuit.

Product from the Pb regrind circuit will combine with the Pb second cleaner tailings and flow into the first of six 5 m^3 Pb first cleaner flotation tank cells. The Pb first cleaner concentrate will be collected in a common launder and fed to the Pb second cleaner flotation circuit, while the Pb first cleaner tailings will combine with the Pb rougher tailings and be pumped to the Zn conditioning tank.

The Pb first cleaner concentrate will combine with the Pb third cleaner tailings and flow into the first of three 3 m^3 Pb second cleaner flotation tank cells. The Pb second cleaner concentrate will be collected in a common launder and fed to the Pb third cleaner flotation circuit, while the Pb second cleaner tailings will flow back to the Pb first cleaner flotation feed box.

The Pb second cleaner concentrate will flow into the first of three 1.5 m^3 Pb third cleaner flotation tank cells. The Pb third cleaner concentrate will be collected in a common launder and pumped to the Pb concentrate thickener, while the Pb third cleaner tailings will flow back to the Pb second cleaner flotation feed box.

Pb concentrate from the third cleaners will report to a 4 m diameter high-rate thickener. The thickener overflow will be sent to the process water tank, while thickened Pb concentrate will be pumped to an 8-hour stock tank that feeds a pressure filter for further dewatering. Pb final concentrate, at approximately 8% moisture, will be bagged and loaded onto trucks for transportation to Trail, BC.

17.4.6 Zinc Flotation

Tailings from the Pb flotation circuit will feed a 34 m^3 Zn conditioning tank, which will provide 5 minutes of conditioning time prior to Zn flotation. Frother Polyfroth H57, SIPX, Zn activator copper sulphate ($CuSO_4$) and lime will be added to the conditioning tank. The slurry will then gravitate to the rougher flotation circuit, which consists of six 20 m^3 flotation tanks cells operating in series.

Rougher concentrate will be collected in a common launder and fed to a conical pump box. The de-aerated concentrate will then be pumped to the Zn regrind circuit for further mineral liberation, while the Zn rougher tailings will flow into the final tailings pump box.

Zn rougher concentrate will be pumped to a cluster of five (four operating / one spare) 150 mm densifying cyclones to achieve a cyclone underflow density of 50%. The cyclone underflow will then flow by gravity to a pump box where density control water will be added to ensure an adequate feed density to the regrind mill. The slurry will then be pumped to a 1,120 kW stirred mill where high-intensity grinding with 2 mm ceramic grinding media will reduce the bulk concentrate to a P_{80} of 15 µm. The product will then combine with the cyclone overflow and be transported to the Zn first cleaner flotation circuit.

Product from the Zn regrind circuit will combine with the Zn second cleaner tailings and flow into the first of eight 10 m^3 Zn first cleaner flotation tank cells. The Zn first cleaner concentrate will be collected in a common launder and fed to the Zn second cleaner flotation circuit, while the Zn first cleaner tailings will combine with the Zn rougher tailings in the final tailings pump box.

The Zn first cleaner concentrate will combine with the Zn third cleaner tailings and flow into the first of six 10 m^3 Zn second cleaner flotation tank cells. The Zn second cleaner concentrate will be collected in a common launder and fed to the Zn third cleaner flotation circuit, while the Zn second cleaner tailings will flow back to the Zn first cleaner flotation feed box.

The Zn second cleaner concentrate will flow into the first of three 10 m^3 Zn third cleaner flotation tank cells. The Zn third cleaner concentrate will be collected in a common launder and pumped to the Zn concentrate thickener, while the Zn third cleaner tailings will flow back to the Zn second cleaner flotation feed box.

Zn concentrate from the third cleaners will report to a 12 m diameter high-rate thickener. The thickener overflow will be sent to the process water tank, while thickened Zn concentrate will be pumped to an 8-hour stock tank that feeds a pressure filter for further dewatering. Zn final concentrate, at approximately 8% moisture, will be loaded onto bulk trucks and rail cars for transportation to Trail, BC.

17.4.7 Tailings Management

Zn rougher tailings and Zn first cleaner tailings will combine in the final tailings pump box and be pumped to a filter plant where the material will be dewatered to a moisture content of approximately 15%. The tailings will then be co-disposed of in the Tailings Management Facility (TMF) as filtered tailings with the DMS reject. The water recovered in the dewatering process will be circulated back to the process water tank.

17.4.8 Reagents Handlings and Storage

Reagents consumed within the plant will be prepared on-site and distributed via the reagent handling systems. These reagents include: sodium isopropyl xanthate (SIPX), sodium cyanide (NaCN), hydrated lime, PE26, methyl isobutyl carbinol (MIBC), Polyfroth H57, copper sulphate ($CuSO_4$), ferrosilicon, flocculant and antiscalant. All reagent areas will be bermed with sump pumps which transfer spills to the final tailings pump box, with the exception of the flocculant. Flocculant spills will be returned to the storage tank. The reagents will be mixed, stored and then delivered to the DMS, lead flotation, zinc flotation and dewatering circuits. Dosages will be controlled by flow meters and manual control valves. The capacity of the storage tanks will be sized to handle one day of production.

Table 17-2 summarizes the reagents used in the plant and their estimated daily consumption rates. The table also includes other major process consumables.

Table 17-2: Reagents and Process Consumables

Description	Delivered Form	Daily Usage
SIPX	1 tonne bags (dry)	2.4 t/d
NaCN	1 tonne bags (dry)	390 kg/d
Lime	2 tonne bags (dry)	7.2 t/d
PE26	500 kg bags (dry)	600 kg/d
$CuSO_4$	1 tonne bags (dry)	6.9 t/d
MIBC	1 tonne totes (liquid)	270 kg/d
Polyfroth H57	1 tonne totes (liquid)	405 kg/d
Antiscalant	1 tonne tote (liquid) or 50 kg barrels	216 kg/d
Flocculant	25 kg bags (dry)	9 kg/d
Ferrosilicon	1 tonne bags (dry)	800 kg/t
Ball Mill Grinding Media – 75 mm chrome steel	1 tonne bags	1.9 t/d
Ball Mill Grinding Media – 50 mm chrome steel	1 tonne bags	1.9 t/d
Pb Stirred Mill Grinding Media – 2 mm ceramic	500 kg bags	92 kg/d
Zn Stirred Mill Grinding Media – 2 mm ceramic	500 kg bags	169 kg/d

Source: JDS (2018)

17.4.9 Air Supply

An instrument and plant air system, with air compressors and associated dryers, filters, and receivers, will be located in a compressor room inside the plant building. Air blowers will be used to provide air to the Pb and Zn flotation circuits.

17.4.10 Water Supply and Consumption

The following water types will be used in the process plant:

Process Water – Overflow water from the Pb and Zn concentrate thickeners will be used as process water. This water will be used predominantly in the grinding circuit to dilute slurry to the required densities.

Fresh Water – Fresh water for the process plant will be pumped from a fresh water supply, such as the local water course or an impoundment which may potentially be located adjacent to the process plant. Fresh water will be used as reagent make-up water, gland water and process make-up water. The estimated fresh water consumption in the process plant will be 17 m^3/h, and approximately 117 m^3/h for potable water.

Reclaim Water – Water reclaimed from the tailings filter plant will be used as process water in the grinding and flotation circuits. Based on the water balance and a dry stack moisture content of 85%, 322 m^3/h of water will be reclaimed from the tailings filter plant.

18 Project Infrastructure and Services

The project envisions the upgrading or construction of the following key infrastructure items:

- Crushing circuit and crushed ore stockpile;
- Process plant and DMS facilities;
- LNG power plant and LNG storage facilities;
- On-site power distribution with overhead power lines;
- TMF;
- Permanent camp (established for the construction stage);
- Administration and mine dry buildings;
- Truck shop and warehouse;
- 150,000 L of on-site fuel storage and distribution;
- Industrial waste management facilities such as the incinerator; and
- Site water management facilities.

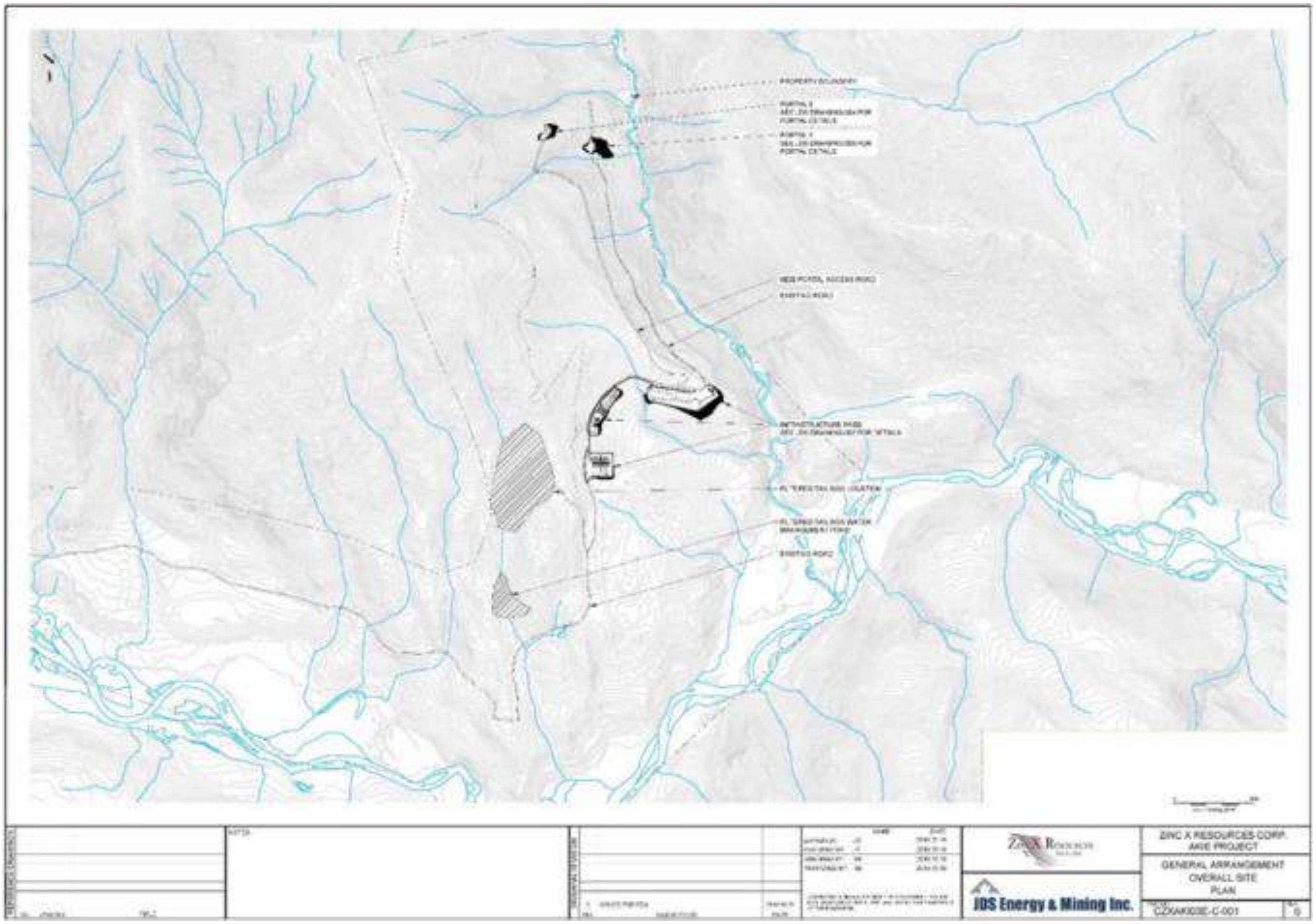
18.1 General Site Layout

The overall project site layout site is shown in Figure 18-1.

The proposed site layout has been configured for optimal construction access and operational efficiency. Primary buildings have been located to allow easy access from the site access road and utilize existing topography to minimize bulk earthworks volumes. The primary crusher has been located as close as safely possible to the portal and at an elevation that facilitates mill feed conveying. The existing site roads will be used with additional roads to Portal 2 and the tailings storage facility being added. The proposed tailings location minimizes its construction earthwork volume and containment area, while maximizing its storage capacity.

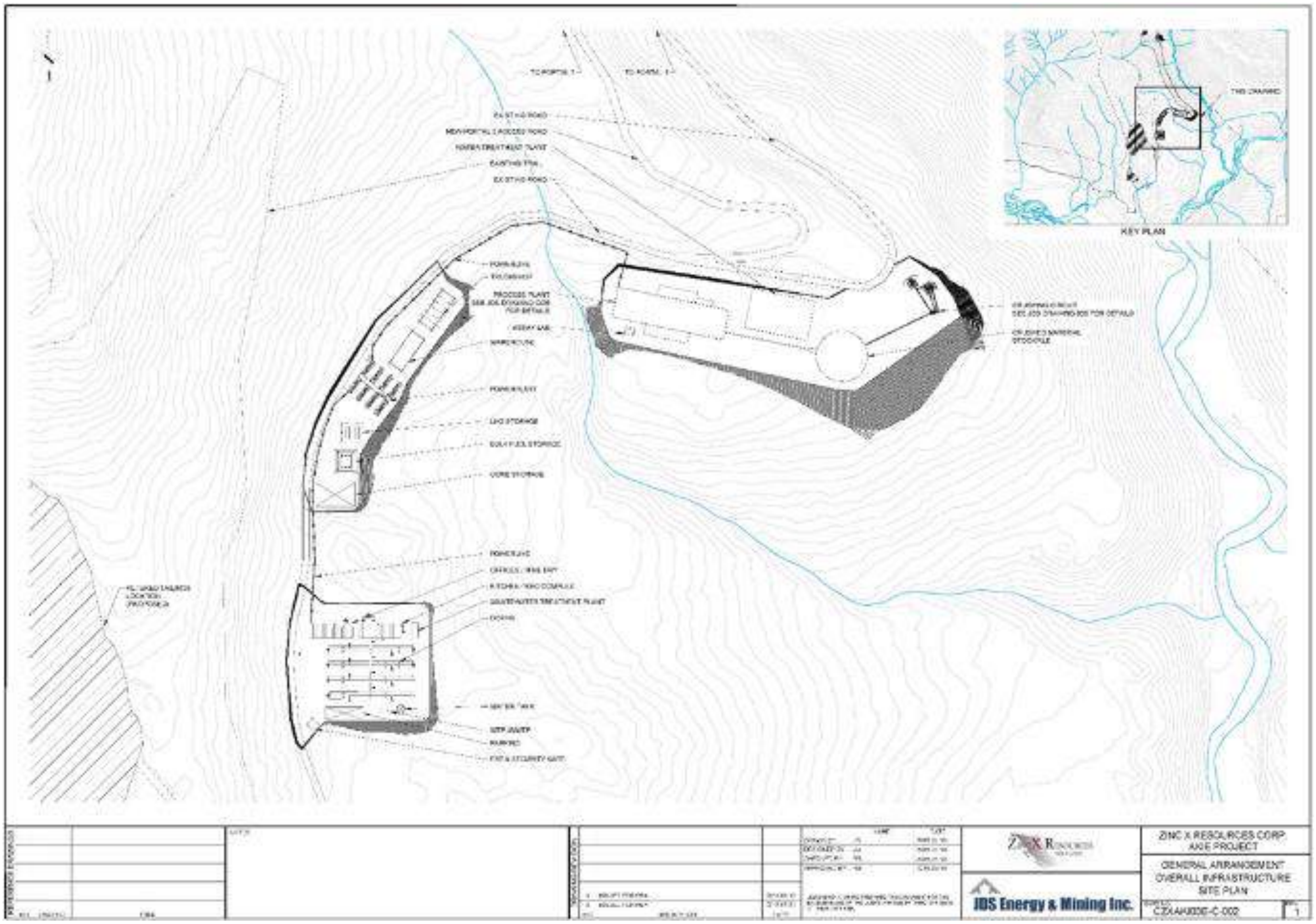
The site infrastructure layout and plant location are shown in Figure 18-2.

Figure 18-1: Overall Site Layout



Source: JDS (2018)

Figure 18-2: Site Infrastructure Layout

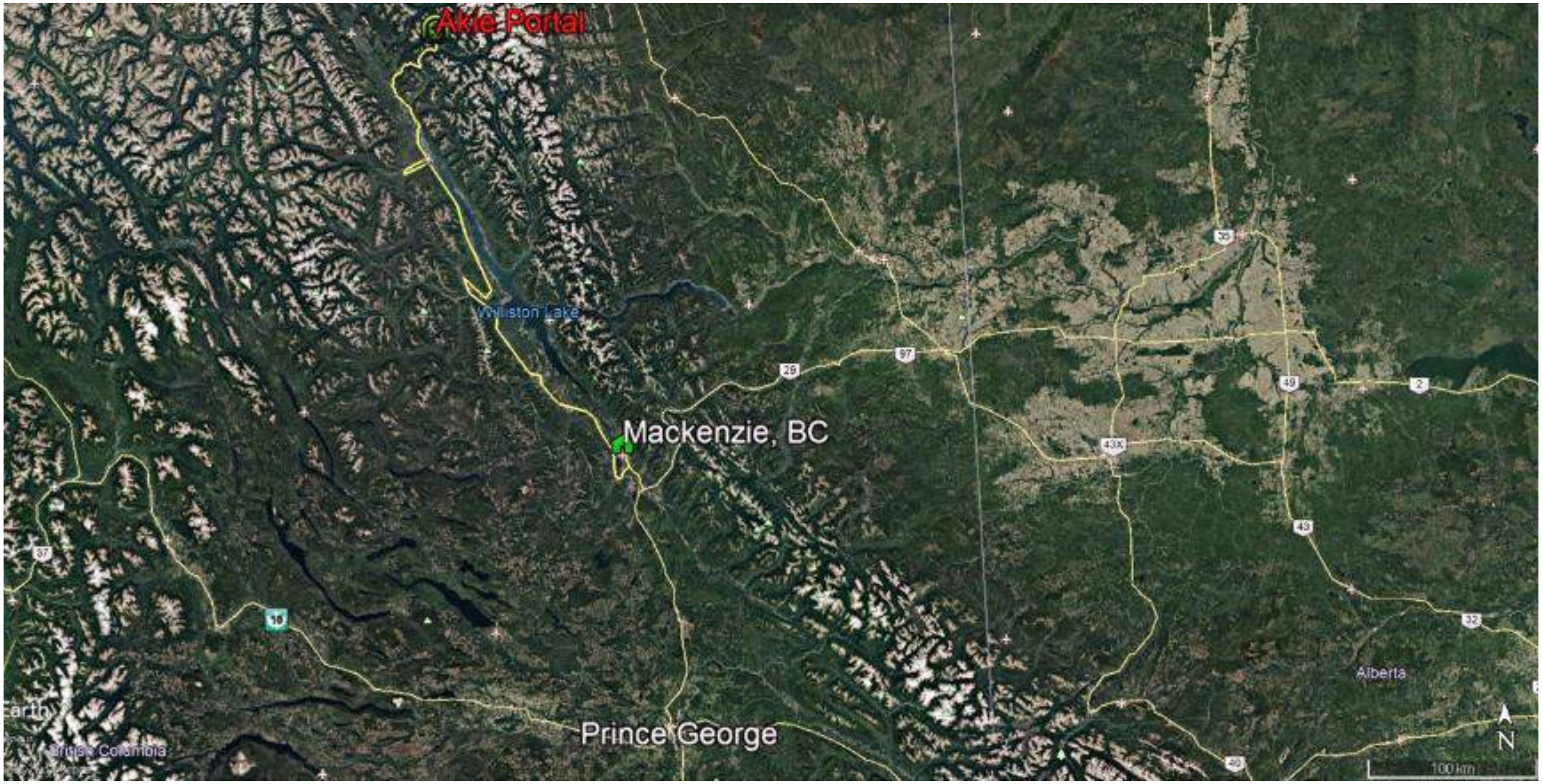


Source: JDS (2018)

18.2 Site Access Road

The Akie property is accessible year-round by a network of all-weather logging roads leading north from Mackenzie, BC. Travel on the forest service roads is approximately 430 km to site from there. It is expected that the Company will share in road maintenance expenses with other resource users including local forestry licensees and mining companies. Mackenzie is connected to the BC provincial highway network via Highway 39 that branches off Highway 97. No road or bridge upgrades are anticipated, and road maintenance costs are factored into the concentrate trucking costs from site to Mackenzie. The route to site is shown in Figure 18-3.

Figure 18-3: Site Access Route



Source: Google Maps & JDS (2018)

18.3 Power Supply and Distribution

Power necessary to support the Akie Project operation will be supplied by on-site generator sets. A single power plant set up comprising seven natural gas-fired reciprocating engine generator sets (gensets) in a N+2 (5+2) arrangement will provide electricity to operate the mine, processing plant and site infrastructure. Each genset will be driven by a 2,500 kW cat engine G3520H (or equivalent) operating at 1,500 rpm, and generating power at 13.8 kV. The plant will be initially set up with six gensets to begin operation, with an additional genset added in year 1.

To maximize the overall efficiency, this power plant will operate as a combined heat and power plant (CHP Plant), providing heat to the process plant and site infrastructure buildings at the project site.

The power plant will be modular with all gensets interconnected. Each genset will be packaged in a walk-in, sound-attenuated enclosure that is constructed, assembled and tested prior to shipment to site.

A LNG storage facility with sufficient capacity for five to seven days of operation, with vaporizer and a bermed containment area will provide fuel for the power plant.

18.4 Process Plant

A three-stage crushing plant will be set up in three covered structures. One for the primary crusher and crusher control room, one for the secondary and tertiary crushers, and one for the vibrating double deck screen.

The DMS and process plant will be located in a pre-engineered structural steel building with dimensions of 121.5 m long by 36 m wide. Additional lean to areas will house the re-agent area, lead concentrate filtering and load out, and the control room and the plant maintenance shop. Overhead cranes will be provided for equipment maintenance. The building will be heated by glycol air handlers and unit heaters.

18.5 Ancillary Facilities

18.5.1 Camp

The camp will comprise single-occupancy rooms with central washrooms. It will be used during the construction stage and throughout the operations stage. There will be six dormitory wings, each capable of housing 42 people for a total of 252 beds.

The kitchen / dining / recreation complex will include the following:

- Kitchen complete with cooking, preparation and baking areas, dry food storage and walk-in freezer / cooler. The kitchen will be provided with appropriate specialized fire detection and suppression systems;
- Dining room with serving and lunch preparation areas;
- First aid room;
- Mudroom complete with coat and boot racks, benches and male-female washrooms;
- Housekeeping facilities;

- Reception desk and lobby; and
- Recreation area.

The camp will be constructed from modular units manufactured off-site in compliance with highway transportation size restrictions. Camp modules will rest on wood cribbing. The camp will comply with all building and fire code requirements and be provided with sprinklers throughout. Arctic corridors will connect the main camp complex and dormitory wings.

18.5.2 Truck Shop and Warehouse

A truck shop will be located on the infrastructure pad near the process plant location. The truck shop will be a 48 m long by 18 m wide structural steel, pre-engineered building designed to accommodate facilities for repair and maintenance of mining equipment and light vehicles. It will also provide warehouse storage space for the mine vehicle maintenance.

A warehouse will be located next to the truck shop, consisting of an insulated sprung structure with overhead doors. Covered cold storage will also be provided at each portal with 40 ft sea containers.

18.5.3 Mine Dry and Office Complex

The main site office complex and the mine dry will be located at the camp site. It will be constructed from modular units manufactured off-site and in compliance with highway transportation size restrictions. Modules will rest on wood cribbing. The complex will comply with all building and fire code requirements and be provided with sprinklers throughout.

The mine dry facility will service construction and operations staff during the life of the project. It will contain the following:

- Male and female clean and dirty lockers; and
- Showers and washroom facilities with separate male and female sections.

The site office facility will contain the following items:

- Private offices;
- Main boardroom; and
- Mine operations line-up area.

18.5.4 Fuel Storage

On-site diesel fuel storage is designed with a one-week supply capacity. Two 75,000 L tanks will be installed within a lined containment berm. Fuel dispensing equipment for mining, plant services, and freight vehicles will be located adjacent to the fuel tank bund and the fueling area will drain into the bund. A fuel transfer module will provide fuel to the power plant day tank and diesel consumers in the process plant.

LNG and diesel will be transported by contractor to the project site daily via the main access road.

18.5.5 Off-Site Airstrip

An existing 1,503 m air strip at Tsay Keh Dene, BC, will be upgraded to serve personnel transport for the construction and operating periods. The strip will be upgraded with navigation aids and full lighting for year-round use. The strip is capable of handling 18 passenger aircraft, such as a Beechcraft B1900D, and the project will be served by charter aircraft flying out of Prince George, BC.

18.6 Waste Rock and DMS reject Management

To the maximum extent possible, all non-potentially acid generating (NPAG) rock from underground mine development approximately 1.6 Mt (JDS material Balance Rev2) will be used in construction of the TMF embankment and water management pond (WMP) dam. All potentially acid-generating (PAG) rock from development approximately 0.6 Mt (JDS material Balance Rev2) will be stored underground as backfill to mitigate ARD/ML generating conditions on surface. A dense media separation (DMS) circuit will remove PAG float rock from the ore prior to processing. This DMS reject material will be co-disposed in the tailings management facility (TMF) with filtered tailings. Approximately 6.2 Mt of DMS reject (3.6 Mm^3) will be stored in the TMF.

18.7 Tailings Management

18.7.1 Tailings Management Best Available Technology (BAT) Alternatives Assessment

The TMF location and tailings technology selected for the PEA was identified in a Mine Waste Disposal Alternatives Assessment. This assessment was based on the understanding of the geochemical properties of the various mine waste materials, and the recognition that the Best Management Practice (BMP) for the PAG waste rock was storage in the underground mine workings, along with as much of the tailings as possible in the form of paste backfill.

All remaining waste materials (the balance of tailings, all DMS reject, and all NPAG waste rock) will be stored on surface. The alternatives assessment was completed to identify the best combination of location and storage technology for all mine waste materials being stored on surface.

Candidates were characterized from a high-level perspective to identify the most technically and economically suitable candidate for mine waste disposal at the current design stage. Environmental and socio-economic factors were not included at this time and will be addressed in future design stages.

The majority of tailings (72%) will be used for paste tailings backfill under this plan. To simplify operations, it is assumed that all tailings will be processed at the paste plant and the balance of tailings stored on surface (28%) will be handled as either paste tailings or filtered tailings.

Potential surface storage locations were evaluated in addition to assessing tailings technology. Nine candidate locations were identified for paste tailings disposal, and eight for filtered tailings disposal. For paste tailings candidates it was assumed that the DMS reject would require storage in a separate, standalone facility because combining the DMS reject and paste tailings in an impoundment with a large confining embankment(s) is more challenging at the site. For filtered tailings candidates, the DMS reject and filtered tailings can be co-mingled into a single facility due to the stackable nature of the filtered tailings, with the DMS reject placed at the downstream side of the stack as the coarser of the two materials.

A pre-screening assessment was conducted to identify candidates with obvious deficiencies or “fatal flaw” characteristics. This eliminated four paste tailings candidates and three filtered tailings candidates from the assessment. The preferred filtered tailings candidate (Candidate 4F) and the preferred paste tailings candidate (7P) were then assessed using a highlevel multiple accounts ledger.

The two preferred candidates (7P and 4F) were compared to identify the overall preferred candidate for mine waste disposal. Both candidates ranked similarly from an economic perspective, however Candidate 4F was identified as the preferred candidate from a technical perspective due to the following advantages:

- Enhanced physical stability provided by filtered tailings;
- Co-mingling waste products into a single facility minimizes the project footprint, reducing the environmental impact of the project;
- Simpler construction and operations with only one facility;
- A smaller catchment and only one facility results in simpler water management; and
- Less earthworks and material required from borrow sources.

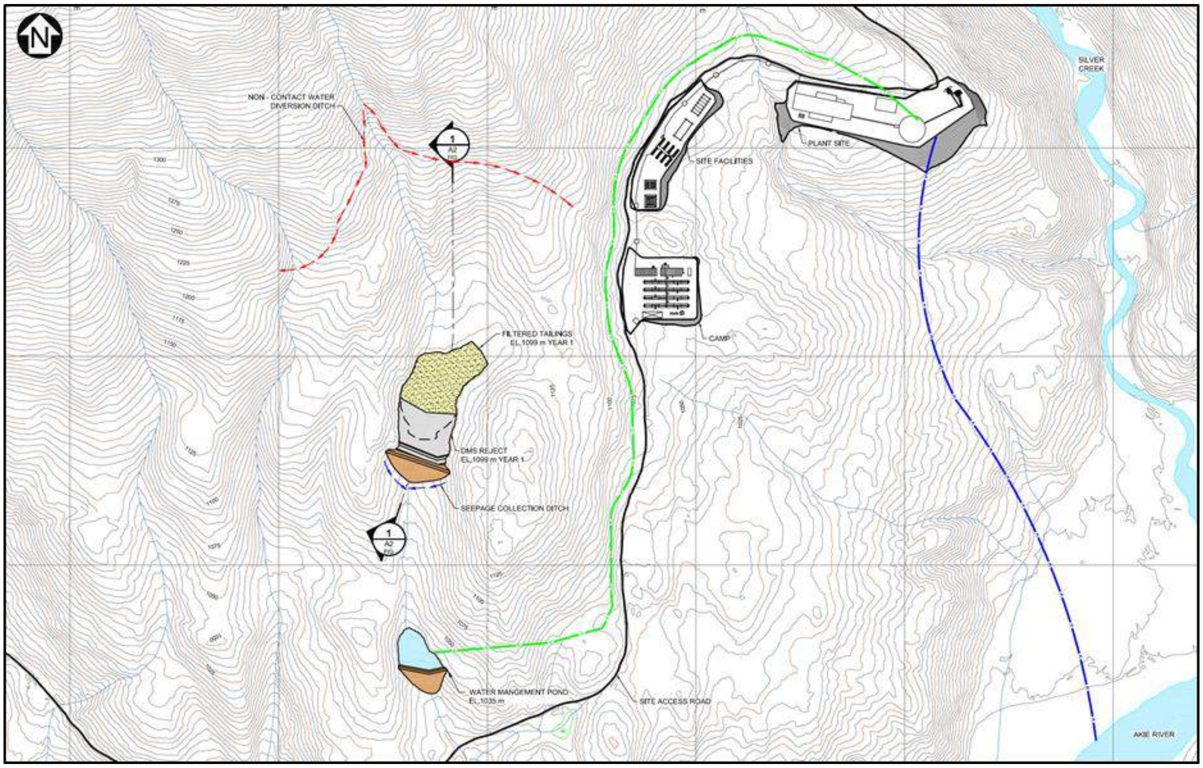
18.7.2 Tailings Management Facility Staging and Filling Schedule

The TMF was designed to store 4.0 Mt of filtered tailings and 6.2 Mt of DMS reject. The TMF site has capacity for future expansion potential if required. The TMF embankment will be constructed in Year -1 with an additional saddle embankment constructed in later years.

The design includes development of the TMF as a filtered tailings stack, with construction of the stack in 10 m high benches. An embankment will be constructed to contain 1 year of paste tailings, DMS reject and associated water management as a contingency measure in the event of delays with the tailings filters. This embankment will not be raised over the life of mine; ongoing construction will involve placement of filtered tailings and DMS reject in the TMF stack, with foundation preparation as the stack footprint expands.

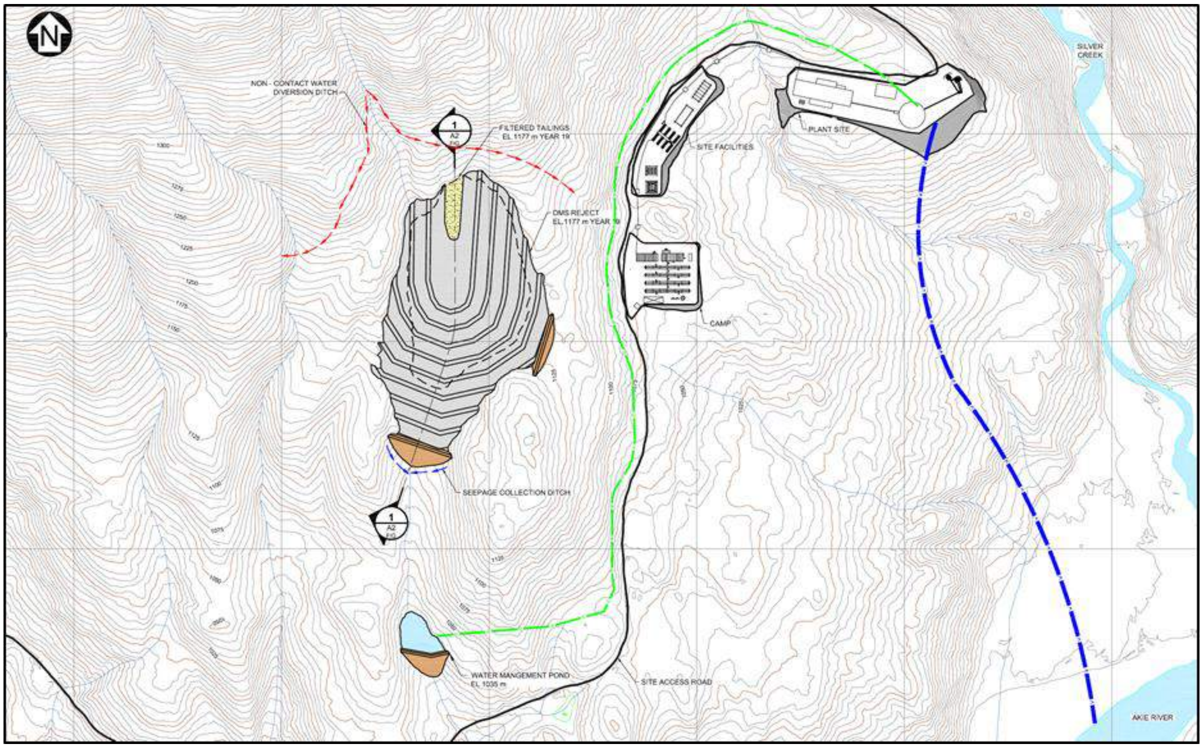
The Year 1 general arrangement is shown on Figure 18-4 and the final (Year 19) general arrangement is shown on Figure 18-5.

Figure 18-4: TMF General Arrangement – Year 1



Source: KP (2018)

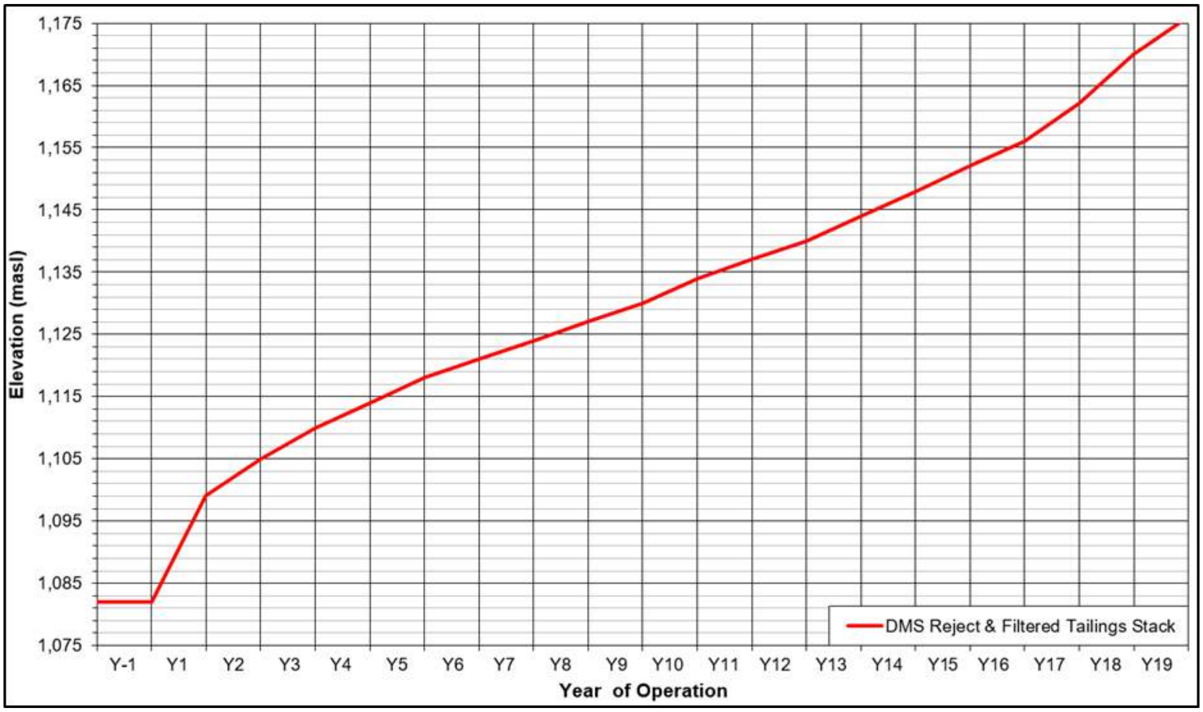
Figure 18-5: TMF General Arrangement – Year 19



Source: KP (2018)

The filling schedule for the stack is shown on Figure 18-6. The average annual rate-of-rise of the stack, after the first year of material placement, is approx. 4 m/year. The actual filling rate may vary depending on a variety of operating factors.

Figure 18-6: TMF Stack Filling



Notes:

1. Tailings tonnages and ramp-up schedule provided by JDS (10 May 2018)
2. Average dry density assumed to be 1.7 t/m^3 for both filtered tailings and DMS reject

Source: KP (2018)

18.7.3 Tailings Management Facility Design

The principal design objectives for mine waste disposal are to provide safe and secure storage of mill tailings and DMS reject while protecting groundwater and surface waterbodies during operations and in the long-term (i.e. post-closure), and to achieve effective reclamation at mine closure. The design of the TMF has taken into account the following requirements:

- Permanent, secure and total confinement of all solid waste materials within an engineered disposal facility;
- Control, collection and removal of free draining liquids from the surface of the TMF during operations;
- Minimize the amount of fresh water that comes into contact with mine facilities and active construction areas by diverting upslope runoff to the maximum practical extent; and
- The inclusion of monitoring features for all aspects of the facility to ensure performance goals are achieved and design criteria and assumptions are met.

The TMF will be operated as a filtered tailings facility, with the filtered tailings and DMS reject co-disposed in the same stack. The total stack volume is approx. 6.0 Mm^3 (approx. 2.4 Mm^3 of tailings and 3.6 Mm^3 of

DMS reject at an average dry density of 1.7 t/m^3 for both materials). The TMF has the following specific features for tailings and water management:

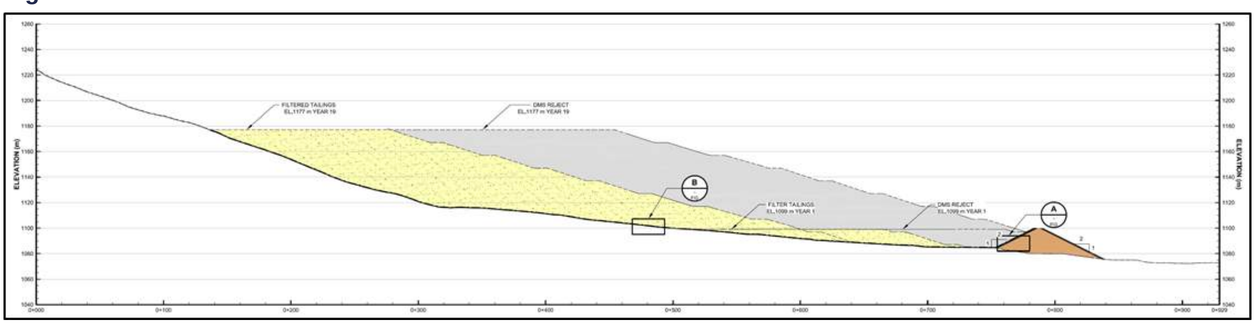
- TMF embankment constructed with NPAG waste rock from underground mine development;
- Fully lined pad to minimize seepage;
- Filtered tailings and DMS Reject constructed in a stack, with DMS Reject material used as the outer shell of the stack;
- Basin underdrain system;
- Water management Pond; and
- Non-contact water diversion ditches.

The TMF has one cross-valley embankment, approximately 23 m high, which will be constructed using NPAG waste rock from underground mining. The embankment will be a rockfill embankment with 2H:1V side slopes. The minimum embankment crest width will be 6 m. A layer of liner bedding material, 0.5 m thick, will be placed on the upstream face of the embankment to facilitate placement of a HDPE geomembrane liner. A transition zone layer, 0.5 m thick, will underlie the liner bedding layer to prevent the migration of fines through the embankment.

The filtered tailings and DMS reject will be placed and compacted in thin lifts with the DMS reject forming the downstream shell of the stack.

A cross-section of the TMF is shown on Figure 18-7.

Figure 18-7: TMF Embankment Section



Source: KP (2018)

The majority of fill for the TMF embankment will be general NPAG waste rock from underground mine development. The upstream face of the embankment will include a layer of filter sand, which will function as a geomembrane liner bedding. The geomembrane liner will be installed on the filter sand material. Instrumentation will be included for ongoing monitoring of the performance of the TMF embankment. This will include vibrating wire piezometers installed in the foundation and embankment fill, in addition to inclinometers and survey monuments.

18.7.4 Seepage Control Measures

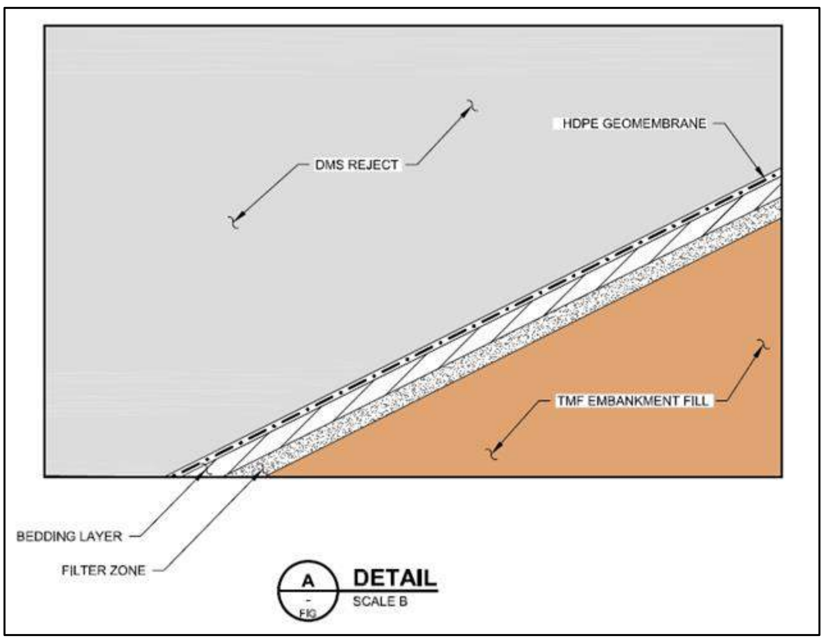
Potential seepage from the TMF will be controlled by incorporating a geomembrane liner and a basin underdrain.

Geomembrane Liner

The entire footprint of the TMF stack, including the upstream face of the TMF embankment, will be lined with HDPE geomembrane. The liner system will include a layer of non-woven geotextile above and below the liner, for protection from the adjacent materials. The liner system will also incorporate a 0.5 m thick prepared subgrade, comprised of processed bedding material.

The geomembrane will be effectively impermeable, with seepage occurring mainly through defects that may occur during fabrication and/or installation. The geomembrane liner detail is shown on Figure 18-8 below.

Figure 18-8: Embankment Geomembrane Liner Detail



Source: KP (2018)

Basin Underdrain

A basin underdrain will be installed above the geomembrane on the basin floor to allow for dewatering of the tailings after precipitation events, and to minimize seepage. The underdrain will be connected to an internal wet well sump and recycle pump system. Collected water will be pumped to a WMP downstream of the TMF.

The basin underdrain will be constructed using processed material from local borrow, or selective DMS reject material. The underdrain will include perforated drain pipes with a free draining material surrounding the pipes. A blanket layer of filter sand, 300 mm thick, will be placed on the basin floor above the liner and surrounding the drain pipes, to assist in providing drainage, to prevent tailings migration, and to protect the geomembrane liner. The system will drain towards the wet well sump, located at the topographical low point of the TMF.

Water Management Pond (WMP)

Seepage from the TMF will be controlled and minimized by the HDPE geomembrane liner and basin underdrain system. The WMP, located downstream of the TMF, will collect seepage and runoff from the surface of the TMF and contact water surrounding the TMF.

The WMP has been sized to store 1 month of total seepage, as well as contact water and runoff from a 1-in-200 year, 24-hour precipitation event for the catchment that reports to the WMP. Collection ditches, constructed downstream of the TMF, will collect and convey seepage and contact water to the WMP.

Collected water will be pumped to the Process Plant for treatment, if required, and subsequently discharged to the environment.

18.7.5 Tailings Management Facility (TMF) Operations

Filtered tailings and DMS reject will be delivered to the TMF from the filter press at the paste plant, and the crushing laydown area, respectively. The materials will be placed in the TMF and compacted in thin lifts (max. 300 mm lift thickness) using vibratory compactors.

The DMS reject is float rock collected after the three-stage crushing process, before grinding in the mill. Tailings will be produced as a slurry from the mill and pumped to a paste plant where they will be filtered, and a large fraction subsequently processed into paste tailings for paste backfill of the underground mine.

The material will be managed in a stack, constructed at an overall 4H:1V slope, with a 3H:1V inter-bench slope angle, 10 m bench height, and 10 m bench width.

The DMS reject will be placed as the downstream outer shell of the stack for enhanced stability due to the larger particle size.

18.8 Water Management

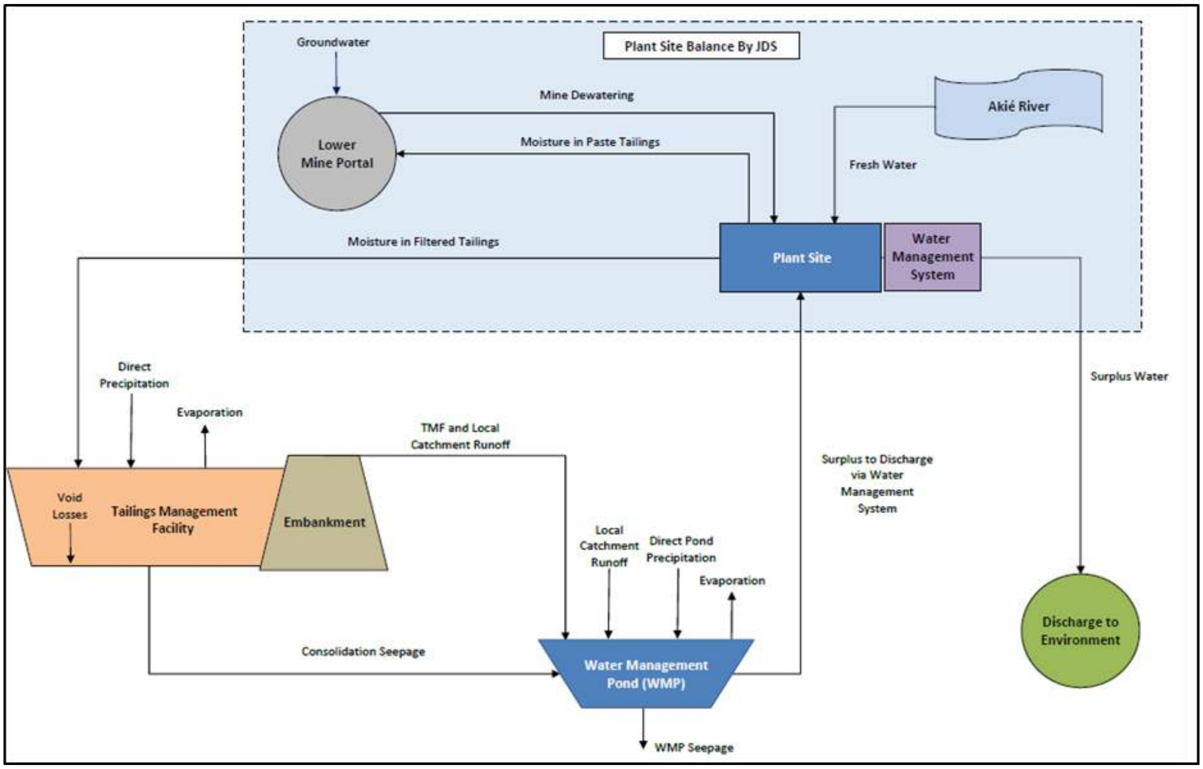
The water management plan assumes that non-contact will be diverted around mine facilities to the downstream waterways wherever possible. A diversion channel will therefore direct runoff from the catchment upslope of the TMF, away from project facilities. Runoff from the TMF itself and surrounding the local catchment area will be directed to the WMP downstream of the TMF. The WMP will contain runoff from the local catchment, seepage from the TMF underdrain system, and precipitation directly on the pond itself.

Surplus water from the WMP and Portal One dewatering will be directed to a water management system at the Process Plant prior to treatment, if required, and discharge to the environment.

A preliminary water balance model was prepared to estimate the magnitude of the annual surplus or deficit conditions of the WMP and to provide a summary of surplus water directed to the water management system. The model was developed using average monthly inputs. The schematic for the water balance model is presented on Figure 18-9.

The preliminary water balance indicates that the WMP will be in a surplus condition with an estimated annual volume of approximately 176,000 m^3 directed to the water management system for discharge. Additionally, Portal One dewatering will generate an annual volume of approximately 3.5 Mm^3, which will result in a total of 3.7 Mm^3 of surplus to be discharged annually.

Figure 18-9: Site Wide Water Balance Flow Schematic



Source: KP (2018)

19 Market Studies and Contracts

19.1 Market Studies

No market studies have been completed for the project at this time, but the concentrates are very clean and likely to be attractive to smelters.

19.2 Contracts

No contractual arrangements for smelting exist at this time. Furthermore, no contractual arrangements have been made for the sale of zinc or lead concentrate at this time.

19.3 Royalties

The project is not subject to any royalties.

19.4 Metal Prices

The precious metal markets are highly liquid and benefit from terminal markets around the world (London, New York, Tokyo, and Hong Kong). Historical lead, zinc and silver prices are shown in Figure 19-1, Figure 19-2 and Figure 19-3. Historical exchange rate trends are plotted in Figure 19-4.

Figure 19-1: Historical Lead Price

Source: London Metals Exchange (2018)

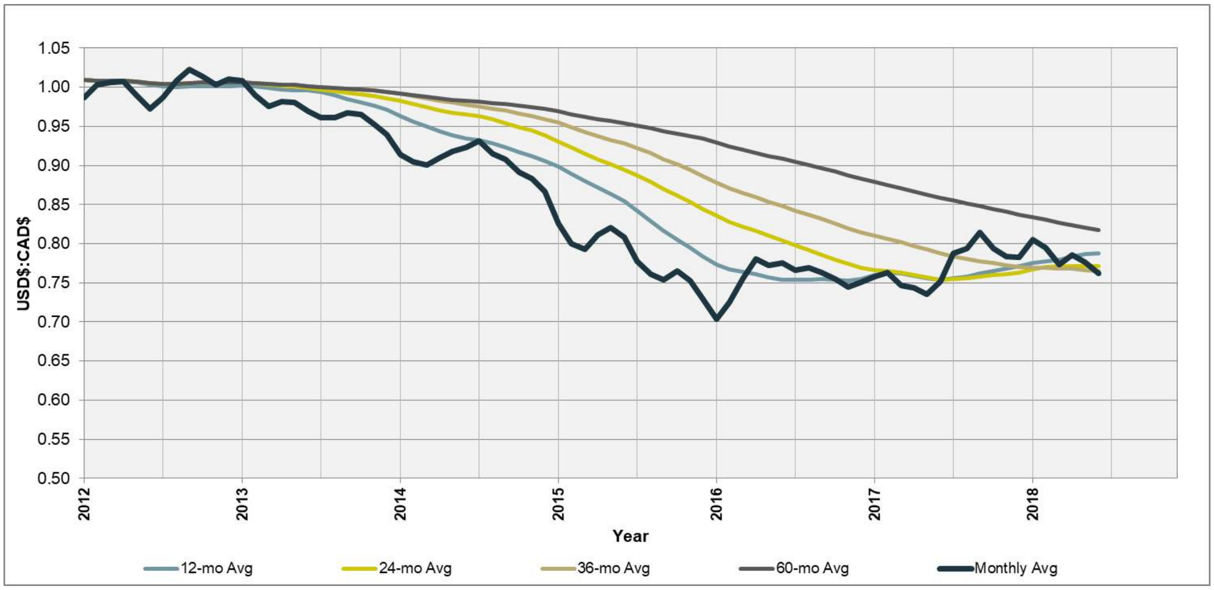
Figure 19-2: Historical Zinc Price

Source: London Metals Exchange (2018)

Figure 19-3: Historical Silver Price

Source: London Metals Exchange (2018)

Figure 19-4: Historical US$:C$ F/X Rates



Source: London Metals Exchange (2018)

The lead price, zinc price, and silver price used in this PEA study were selected based on the average of three years past and projected two years forward by analysis of London Metal Exchange futures as of 30 April 2018. These parameters are in line with other recently released comparable Technical Reports.

A sensitivity analysis on metal prices and exchange rates was completed as part of the overall economic analysis. The results of this are discussed in Section 23. Table 19-1 outlines the metal prices used in the PEA economic analysis.

It must be noted that metal prices are highly variable and are driven by complex market forces and are extremely difficult to predict.

Table 19-1: Metal Price and Exchange Rate

Parameter	Unit	Value
Lead Price	US$/lb	1.00
Zinc Price	US$/lb	1.21
Silver Price	US$/oz	16.95
Exchange Rate	USC	0.77

Source: JDS (2018)

20 Environmental Studies, Permitting and Social or Community Impacts

20.1 Baseline Environmental Studies

Environmental studies and monitoring programs in support of the Akie Project have been conducted over the past 20 years, with the majority of programs conducted in 2007 and 2008. Studies included meteorology, surface water quality, hydrogeology, hydrology, geochemistry, fish and wildlife, and terrain and soils.

A summary outlining the current baseline understanding for the site can be found in Section 20.1.1. A list of anticipated complementary baseline studies can be found in Section 20.1.2. A detailed gap analysis to develop complementary baseline studies to meet current regulatory expectations will be established at the next stage of development.

20.1.1 Current Environmental Baseline Studies

Hallam Knight Piésold prepared an environmental baseline data report in 1997, summarizing results of surface water quality and wildlife observation programs. Environmental Dynamics Inc. (EDI) conducted baseline studies on fisheries, wildlife, birds, water quality, sediment quality, benthic taxonomy, periphyton and vegetation within a local study area (LSA) and a larger regional study area (RSA) in 2008. Investigations were primarily conducted in the LSA, with wildlife studies primarily in a focal area south of Cardiac Creek, in the area where mine infrastructure was planned. A total of 11 wildlife species listed in the Species at Risk Act and/or the BC Identified Wildlife Management Strategy were identified as potentially occurring in the Akie Project area. The recommendations of the 2008 study included:

- Establish the level of grizzly bear use in the surrounding habitats using wildlife transects;
- Initiate dedicated surveys for American marten, fisher and wolverine using winter tracking transect surveys in habitats most likely to be used;
- Continue ungulate pellet plot surveys in late spring, expand the transect surveys for moose, elk, and woodland caribou, and conduct a winter aerial survey for stone’s sheep and mountain goat; and
- Initiate observation trials at mineral lick sites.

EDI also conducted a baseline fisheries investigation in the summer and fall of 2007 to provide a preliminary indication of the status of pre-development fish populations and aquatic biophysical habitat in the LSA. The LSA encompasses portions of the Akie River mainstem, the lower reaches of Silver Creek, and a number of lower-order tributaries draining into these systems. Bull trout, mountain whitefish, and slimy and spoonhead sculpin were captured, primarily within the Akie River and Silver Creek mainstems. Three samples of slimy sculpin tissue were also collected for analysis of heavy metal accumulation in tissue.

Madrone Environmental Services conducted a terrain stability assessment and evaluated soils suitability for reclamation surveys in specific areas within the development footprint of the exploration project in 2008. The purpose of the program was to provide recommendations regarding road location and construction, locations for future site development, and areas suitable for soil reclamation and waste rock storage. Recommendations for future monitoring were to conduct a detailed terrain stability and hazard assessment and detailed soil survey and land capability assessment.

Levelton Consultants Ltd. collected meteorological data in the Akie claim from November 2007 to September 2015. Temperature, wind speed, wind direction, and standard deviation of wind direction were collected starting in November of 2007; net radiation and precipitation instruments were added to the monitoring program in July 2008. The purpose of the meteorological program was to collect all necessary representative data required for air quality dispersion modelling. Temperature data are available from January 2008 through August 2015; the data for the remainder of the parameters are available for the period July 2008 to August 2015.

As is typical for site climate stations, some data gaps were noted, as a result of staff not being on-site to maintain the instruments in the winter months, equipment malfunction, and damage from wildlife. Winter precipitation data was collected in 2013/2014.

Lorax prepared annual Environmental Monitoring Reports for Effluent Permit 106429 for 2012 through 2016. Monitoring was comprised of:

- Collection of surface water quality samples from two creeks and a sediment pond;
- Discharge measurement at two stream sites and a sediment pond; and
- Collection of water quality samples from groundwater wells.

In addition to the annual reports listed above, weekly and monthly reports were occasionally prepared summarizing total suspended solids in creeks near the deposit area.

Surface water sampling was conducted at 17 surface water quality sites in 2012 on Silver Creek, Avalanche Creek, Cardiac Creek, “T” Creek, the Akie River and a sedimentation pond. The 2013 surface water monitoring program involved collection of samples from five sites monthly between May and September and at two additional sites in September; samples were only collected when the Exploration Camp was staffed. The monitoring program in 2014 was again restricted to the periods when the Exploration Camp was staffed; hence, water quality sampling for a full suite of parameters was conducted monthly at two sites (June and July) during 2014. Surface water samples were collected in June, July and September in 2015 and in June 2016.

Three hydrometric monitoring stations were established in 2012 (at the WRD Sedimentation Pond, “T” Creek downstream and Silver Creek upstream). Spot flow measurements were taken during freshet (in May) and in September on “T” Creek and Silver Creek in 2013. A stage-discharge (rating) curve was developed for each of Silver Creek and “T” Creek based on a total of four points collected in 2012 and 2013. Discharge measurements were taken in June and July 2014. Hydrology monitoring was also conducted in June and July 2015.

The Silver Creek rating curve was updated with one additional measurement point, but the hydrometric station at “T” Creek was buried by a debris slide and the station had to be re-established at a new location approximately 150 m downstream from the initial location. As a result of this, a new rating curve was developed. Discharge was measured in June, July and September in 2015. It was noted that additional high flow measurements were required to validate the rating curve for Silver Creek, and that the rating curve for “T” Creek developed based on the points collected since the debris slide in 2014 required additional moderate and high flow measurements to establish a robust curve. In 2016, discharge measurements were taken in June at the Silver Creek and “T” Creek sites: flows were almost twice as high as the previously recorded high flows. In contrast, flows in the Akie River during the same period in June were close to the 20th percentile flows.

Lorax prepared a Phase I Hydrogeology Baseline Evaluation in 2007 and recommended installation of multiple groundwater wells, water quality sampling (initial and quarterly sampling thereafter) and conducting hydraulic conductivity tests. Additional recommendations were made regarding the hydrogeology program in memos prepared by Lorax in 2009 and 2010. A hydrogeology monitoring program was established by Lorax in the 2010 with the results summarized in the Lorax 2012 hydrogeology final report.

Baseline groundwater water quality monitoring was conducted in the Project area starting in 2010 at five sites in the vicinity of the waste rock dump (WRD), the haul road and the Portal pad. From two to seven samples were collected from each of the wells in 2012. One round of groundwater sampling was conducted once per year between 2013 and 2016 at a number of the wells.

In 2011 Michael Cullen Geotechnical Ltd. completed a geotechnical assessment of the waste rock storage facility, access road and portal area. Designs for the waste rock area pond, portal access road and the portal pad and pond were finalized in 2011.Construction of the waste rock storage facility and portal access road were initiated September 2011.

An application for an effluent permit was submitted to the regulatory authority in 2012. To support the application a water quality model was prepared by Lorax (2012) alongside a technical assessment of the advanced underground exploration project (Lorax, 2012). The baseline surface water quality and groundwater quality were described by Lorax (2013), hydrometric monitoring of Silver Creek and T Creek are ongoing. Annual water quality reports have been prepared and filed by Lorax for 2012 through 2017. Akie River is instrumented by a public agency with extended historical records publicly available.

A geochemical characterization study was completed by Lorax in 2010 focusing on the geochemical characterization of waste and overburden.

Historic environmental reporting is summarized in Table 20-1.

Table 20-1: Summary of Historic Environmental Reporting

Discipline	Report Title	Year	Author
Surficial Hydrology, Hydrogeology and Water Quality	Environmental Baseline Report	1997	Hallam Knight Piésold Ltd
	Akie Mine Site Phase 2 Summary – Reconnaissance Baseline Studies Phase II Surface Hydrology Baseline Studies DOSSIER 07.0152	2008	Madrone Environmental Services
	Akie Mine Site Phase 3 Summary – Reconnaissance Baseline Studies Phase 3 Surface Hydrology Baseline Studies. DOSSIER 07.0152	2008	Madrone Environmental Services
	Status of Lower Cardiac Creek Extension Road Exploration Area. DOSSIER 09.0074	2010	Madrone Environmental Services
	Summary of Madrone 2009 Report (as of Dec 16/09)	2009	Madrone Environmental Services
	Akie Exploration - Monthly Monitoring Report Submission MOE Authorization # 105788 Project No.11-P-0219	2012	EDI Environmental Dynamics Inc.
	Akie Property Reach Break Analysis: Cardiac Creek, Avalanche Creek, and T Creek Project No. 11-P-0219	2012	EDI Environmental Dynamics Inc.
	Annual review of TSS within T Creek and Cardiac Creek as per BC's Ministry of Environment Approval # 105788 Project No. 11-P-0219	2013	EDI Environmental Dynamics Inc.
	Surface Erosion and Sediment Control Plan Underground Portal and Access Road Upgrades - Akie Exploration Project. Project No. 11-P-0219	2011	EDI Environmental Dynamics Inc.
	Surface Erosion Protection and Sediment Control Plan Underground Portal and Access Road Upgrades - Akie Exploration Project. EDI PROJECT NO.: 11-P-0219	2011	EDI Environmental Dynamics Inc.
Surficial Hydrology, Hydrogeology and Water Quality	Phase I Hydrogeology Baseline Evaluation Project No. 831-1	2007	Lorax Environmental
	Akie Portal – Hydrogeology Investigation Recommendations Project No. 831-3 Akie	2009	Lorax Environmental
	2010 Akie Hydrogeology Characterization Program - Scope Project No. 831-3 Akie	2010	Lorax Environmental
	Technical Note: AKIE Monitoring Well Development and Sampling Guidance	2010	Lorax Environmental
	Akie Exploration Project Water Quality Effects Model	2011	Lorax Environmental
	AKIE 2012 Environmental Program Project No. 831-4	2012	Lorax Environmental
	AKIE 2012 Environmental Program Project No.831-4	2012	Lorax Environmental
	Hydrology of the Cardiac Creek Project Area	2018	Lorax Environmental
	Cardiac Creek Deposit - Summary of Hydrogeology and Preliminary Estimate of Mine Inflows for Preliminary Economic Assessment	2018	Lorax Environmental

Discipline	Report Title	Year	Author
Meteorology, Climate and Air Quality	Akie Zinc Lead Exploration Environmental Baseline Meteorological Monitoring Station Data Summary Report. File: EE07-0187-00.	2008	Levelton Consultants Ltd.
	Levelton Akie Baseline Programs end 2009 As of 1245 pst Mon Jan 04,2010	2010	Levelton Consultants Ltd.
	Akie Zinc Lead Exploration Environmental Baseline Meteorological Monitoring Station Data Summary Report. File: EE07-0187-00.	2012	Levelton Consultants Ltd.
	Akie Zinc Lead Exploration Environmental Baseline Meteorological Monitoring Station Data Summary Report. File: EE07-0187-00	2013	Levelton Consultants Ltd.
	Akie Zinc Lead Exploration Environmental Baseline Meteorological Monitoring Station Data Summary Report. File: EE07-0187-00	2015	Levelton Consultants Ltd.
Fish and Fish Habitat, Wildlife	Akie Project: 2007 Environmental Baseline Program Report. Report No.07-BC-0030	2007	EDI Environmental Dynamics Inc.
	Akie Mine Development Project Baseline Fisheries Investigation	2007	EDI Environmental Dynamics Inc.
	Akie Project Environmental Baseline Analysis: State of Baseline	2007	EDI Environmental Dynamics Inc.
	Akie Project Environmental Baseline Analysis: State of Baseline. EDI project # 07-BC-0030	2007	EDI Environmental Dynamics Inc.
	Akie Project wildlife Reconnaissance. EDI project # 07-BC-0086	2007	EDI Environmental Dynamics Inc.
	Akie Baseline Investigations: Ecological Resource Component 2007/2008. EDI Project No.: 08-BC-0052	2008	EDI Environmental Dynamics Inc.
	Appendix 1: Summary of All Mammal Species/Sign Encountered During 2007/08 Mammal Investigations	2008	EDI Environmental Dynamics Inc.
	Akie Mineral Exploration 2007, 2008 (and 2009) EDI Baseline Acquisition Summary	2010	EDI Environmental Dynamics Inc.
	EDI Akie Baseline Programs As of 1445 pst Jan 04.2010	2010	EDI Environmental Dynamics Inc.
Aquatic Sediments	Akie Exploration Project Supporting Information for Technical Assessment Report	2011	Lorax Environmental
	Akie Exploration Project Water Quality Effects Model Overview of Findings	2011	Lorax Environmental
	Water Quality Effects Model	2011	Lorax Environmental
Geology, Geotechnical and Terrain Hazards	Akie Mine Site Phase 2 Summary - Reconnaissance Baseline Studies. Phase II 2007 Activity Summary and Phase III 2008 Recommendations - Terrain, Soils, and Hydrology for Phase 2 Reconnaissance Baseline Studies	2007	Madrone Environmental Services

Discipline	Report Title	Year	Author
	AKIE STUDY AREA - BASELINE STUDIES Phase III - Detailed Soils and Terrain Hazard Mapping for Proposed Development. DOSSIER 09.0074	2009	Madrone Environmental Services
	Terrain Stability Assessment: Akie Exploration Portal Access Road (Sta. 10+465 to 12+270) DOSSIER 10.0182	2011	Madrone Environmental Services
	Preliminary Slope Stability Assessment, Akie Access Road Sta 3+700 km Slide. File No. 2010-10	2011	Michael Cullen Geotechnical Ltd.
	Report On Slope Stability Assessment, Slide At Sta 3+700 km Akie Access Road	2011	Michael Cullen Geotechnical Ltd.
	Preliminary Avalanche Assessment of the Proposed Akie River Project	2007	Bear Enterprises Ltd.
	Avalanche Control of the Road Construction Site at the Akie River Project	2011	Bear Enterprises Ltd.
	Avalanche Assessment of the Road Construction Site at the Akie River Project	2011	Bear Enterprises Ltd.
Geochemistry	Akie Field Weathering Bin Construction and Sampling Protocol Project No.831-2	2008	Lorax Environmental
	Overview of 2008 ARD/ML Characterization Program. Project No. 831-2	2008	Lorax Environmental
	2008 Akie Geochemical Characterization Program.	2010	Lorax Environmental
	Akie Kinetic Test Report	2011	Lorax Environmental
	Summary of solid phase results for overburden and footwall material that will be extracted and stockpiled during decline development at the Akie Property Project No. 831-2	2011	Lorax Environmental
	Geochemical Characterization of Akie Ore and Tailings	2018	Lorax Environmental
	Summary of Akie Project Geochemical Characterization Studies Project #: A480-1	2018	Lorax Environmental

Source: KP (2018)

20.1.2 Proposed Environmental Baseline Studies

Based on a high-level gap analysis of all previously available baseline studies, additional programs will need to be implemented to update the characterization of existing environmental conditions and meet current regulatory requirements. The primary guidance document for project developers is the “Water and Air Baseline Monitoring Guidance Document for Mine Proponents and Operators” developed by the provincial Ministry of Environment, now Ministry of Environment and Climate Change Strategy (ENV), in June 2016. A comprehensive understanding of the baseline environment, along with project design information, will assist in identifying potential impacts and developing mitigation and monitoring measures to minimize risks. Anticipated environmental studies are summarized in Table 20-2.

Table 20-2: Anticipated Baseline Environmental Studies

Discipline	Additional Information
Meteorology, Climate, and Air Quality	Air Quality • Air Quality monitoring programs developed in consultation with ENV, including parameters measured, associated instruments, frequency of measurements, and spatial distribution of instruments across a site Meteorology and Climate • Continued monitoring all parameters at the site, especially winter precipitation data, for calibration of the air quality dispersion model and in the water management plan • Meteorological instruments in conformance with standards used by ENV
Surficial hydrology	• Additional discharge measurements taken over a range of flows to collect validated points for the establishment of a robust rating curve for each system • Obtain low flow measurements during the winter months • Potential expansion of hydrometric network design, taking into consideration hydrologic subzones, topographic variability, and the project footprint
Hydrogeology	• Continued groundwater quality and quantity monitoring to obtain one year of quarterly data to assess seasonal variations.
Water Quality	• Continued surface water sampling to obtain one year of monthly data with additional weekly sampling (i.e., 5 samples in 30 days) during periods of maximum hydrograph fluctuation
Aquatic Sediments	Stream sediment sampling program: • Co-locate sites with the surface water quality and benthic invertebrate monitoring sites • Sample analysis for grain size, organic carbon, moisture, sulfur, total metals • Once per year through the baseline and operational phases during late summer low flow periods.
Tissue Residue	• Continued fish tissue metals loading program • Tissue samples from sites upstream, adjacent to, and downstream of mine influence • Eight replicates collected per site to describe statistically both within-site and between-site variability
Aquatic Life	• Collection of periphyton and benthic macroinvertebrates from surface water monitoring sites • Minimum one-year survey, preferable two or more consecutive years • Analyze for community composition
Fish and Fish Habitat	• Expansion of fisheries baseline program: document species presence and distribution in the larger RSA, with sampling conducted in different seasons, to document life stages, habitat use, ecological flow needs.

Source: KP (2018)

20.2 Geochemical Characterization

20.2.1 Current Geochemical Site Characterization

Geochemical characterization studies have been undertaken for the Project since 2010. The most recent report (Lorax, 2018a) compiled a summary of the geochemical characteristics of the Akie overburden, waste rock, tailings and ore.

The acid generating potential of the primary geologic units and ore processing by-products were designated as follows:

- Gunsteel Formation shale – potentially acid generating (PAG),
- Cardiac Creek Zone massive sulphide – PAG,
- Paul River Formation carbonate breccia – PAG,
- Road River Group siltstone, shale and calcareous shale – non-PAG (NPAG),
- Overburden from the waste dump area – NPAG based on the neutral paste pH values and the low total sulphur (S) content,
- Ore – PAG with elevated S content,
- Dense media separation (DMS) tailings – PAG, and
- Flotation tailings – PAG.

Leaching potential for project materials was identified for several parameters including SO_4, Al, Cd, Co, Cu, Fe, Pb, Mn, Tl, Se, U, and Zn. However, baseline water quality monitoring in Silver Creek indicated there were naturally elevated concentrations of several metals including Al, Cd, Cu, Fe, Se and Zn (Lorax, 2018b).

Based on an initial desktop analysis, the geological units along the proposed access road were given a low-risk rating risk rating for ARD, except for the Gunsteel shale unit that is exposed within the project site, which was identified as being PAG (Lorax, 2011a).

20.2.2 Proposed Geochemical Characterization Studies

The geochemical characterization studies completed to date provide a solid understanding of the geochemical risks associated with the Project. However, additional testing will be required to support environmental assessment and feasibility design. Anticipated geochemical studies are listed below in Table 20-3.

Table 20-3: Anticipated Geochemical Studies

Discipline	Additional Information
Geochemistry	• Expand laboratory-based kinetic testing to include representative samples of flotation and DMS tailings. • Expand kinetic laboratory-based testing to evaluate the variation of metal leaching potential associated with each geologic unit. • Additional laboratory-based static testing to improve spatial coverage through the proposed mine workings • Develop a set of geochemical source terms that estimate the range of chemistry for runoff and seepage from each mine component that contains material with ML/ARD potential.

Source: Lorax (2018)

20.3 Social and Community

ZincX and its predecessors have engaged the Kwadacha First Nation and the Tsay Keh Dene First Nation since April 2006. The Akie property lies within an area of overlap between the respective traditional territories of the Tsay Keh Dene and Kwadacha First Nations, the two communities closest to the Akie property. The company has consulted with both communities providing economic benefits through community funding, employment and direct engagement of contractors.

A brief summary of this engagement includes a number of activities:

- Consultation with Band Chief, Band Councillors and community elders;
- Annual information meetings;
- Community and consultation engagement funding;
- Direct engagement of First Nation suppliers and workers as required for exploration;
- Donations for fund raising and annual community endeavors;
- Participation in trade and community career fairs;
- Site visits by community members;
- Consultation on drill permit applications and the Underground Exploration Permit;
- Participation in archaeological assessments; and
- Participation in formal Implementation Committee Meetings; since 2013 and held typically on a quarterly basis.

The community of Tsay Keh Dene is located at the north end of Williston Reservoir approximately 215 km north of Mackenzie, BC. The village lies just south of where the Finlay River flows into the north end of Williston Reservoir, in the Rocky Mountain Trench. The community is located approximately 50 km southwest of the Akie property. The population of the Tsay Keh Dene Nation is about 450 persons.

The Kwadacha First Nation is located at Fort Ware, approximately 280 km north of Mackenzie, BC. The village lies at the confluence of the Fox, the Kwadacha, and Finlay rivers in the Rocky Mountain Trench.

The community is located about 45 km northwest of the Akie property. The population of the Kwadacha First Nation is over 500 persons.

In 2013 ZincX's predecessor, Canada Zinc Metals, signed a formal tripartite Exploration Cooperation and Benefit Agreement with the Kwadacha First Nation and the Tsay Keh Dene First Nation. The agreement covers all exploration and related activities on the shared territory. The general purpose of the agreement is to enhance understanding and cooperation between the First Nations and the Company regarding exploration programs and contribute to the programs' overall success. The agreement is also designed to mitigate any effects of exploration programs on the traditional lands of the First Nations and foster a relationship based on mutual respect, trust, mutual benefit and certainty for all parties.

The agreement ensures that the company will continue to provide both communities with opportunities to give meaningful input into such aspects as exploration permitting and environmental studies, with the goal to ensure exploration activities minimize impacts to First Nations' environmental values, heritage values, and traditional activities.

The agreement sets out a framework for employment and training, contracting and business opportunities for members of the two First Nations, and funding for community development and participation and engagement. In return, the company is afforded a greater measure of certainty with respect to ongoing exploration on the Akie Property and the support of both communities as the project advances. Both First Nations have provided many letters of support for drill permit applications and the underground exploration permit and have greatly enhanced the provincial consultation process which has helped assist in the successful issuance of permits.

There are a number of additional indigenous communities within the Treaty 8 lands that lie east of the Akie Project that ZincX may need to include in future engagement plans as the project develops. Until recently the area of interest was located solely within the overlap between two traditional territories of Kwadacha and Tsay Key Dene First Nations. However, with the recent 2017 B.C. Supreme Court ruling (West Moberly First Nations v. British Columbia, 2017 BCSC 1700), the Province of B.C. is now legally required to engage and consult with three additional First Nations from Treaty 8 territory. This decision declared that the western boundary of Treaty 8 is the height of land along the continental divide between the Arctic and Pacific watersheds. The province of British Columbia and the Kaska Dena Council had argued that the boundary runs along the Rocky Mountains, well east of the Arctic-Pacific Divide. The decision effectively shifts the western boundary of Treaty 8 further westward to an area that includes the Akie Project.

The three new First Nations are Doig River First Nation, Halfway River First Nation and West Moberly First Nation located in northeast B.C. The Crown did successfully engage these three First Nations in October 2017 when the company sought a date extension to its underground exploration permit. There were no stated objections from the three Treaty 8 groups, and the date extension was granted. It remains to be seen if the Treaty 8 court ruling will stand as it is expected to be appealed by the province and by the Kaska Dena. The Kwadacha First Nation is a member band of the Kaska Dena Council. The Tsay Keh Dene First Nation has advised the company they will become intervenors in any appeal.

20.4 Environmental Assessment and Permitting

The Akie Project will need to undergo a Provincial and Federal Environmental Assessment, as well as obtain a number of Provincial and Federal Permits and Authorizations. A list of key provincial and federal authorizations is listed in Table 20-4 and Table 20-5 below. Various municipal or regional permits may also be required for operation of any camps or potable water supplies.

Table 20-4: Key Provincial Authorizations

Agency	Permit	Legislation
BC Environmental Assessment Office	Environmental Assessment Certificate	Environmental Assessment Act
Ministry of Energy, Mines and Petroleum Resources	Mines Act Permit	Mines Act
	Operating Permit	
	Mining Right of Way Permit	Mining Right of Way Act
Ministry of Environment and Climate Change Strategy	Waste Discharge Permits (effluent, air, solid wastes)	Environmental Management Act
Ministry of Forests, Lands and Natural Resource Operations	Licensing or approving water use (surface and groundwater)	Water Sustainability Act
	Change Approvals (works in and about a stream)	
	Wildlife Act Authorizations	Wildlife Act
	License of Occupation	Land Act
	License to Cut	Forestry Act

Source: KP (2018)

Table 20-5: Key Federal Authorizations

Agency	Permit	Legislation
Canadian Environmental Assessment Agency	Environmental Assessment Certificate	Canadian Environmental Assessment Act, 2012.
Fisheries and Oceans Canada	Serious Harm to Fish or Fish Habitat Authorization	Fisheries Act
Natural Resources Canada	Explosives Permit	Explosives Act

Source: KP (2018)

20.5 Mine Closure

The conceptual reclamation and closure plan for the Akie Project will involve an active closure period and a post-closure period, in which all mine components will be prepared for permanent closure. Closure will be completed in a manner that will satisfy physical, chemical and biological stability, as well as follow the applicable regulatory framework. The primary objective of the closure and reclamation initiatives will be to return the surface facilities (plant site, laydowns, roads and TMF) to a self-sustaining condition with pre-mining usage and capabilities as much as is practicable.

Management strategies of waste materials (waste rock, tailings and DMS reject) have been developed to meet a key closure objective, which is to achieve good long-term water quality from the surface and underground mine facilities. This will be achieved by preferentially placing excavated materials with ARD/ML potential underground. PAG waste rock, along with paste tailings, will be placed in the mined-out underground workings during the life of mine, with NPAG waste rock, DMS reject and the balance of tailings (as a filtered product) placed in the TMF. The underground workings will be allowed to flood, and the portal will be sealed to prevent release of water in the post-closure phase of the mine.

Roads not required in the post-closure phase will be decommissioned, with culverts removed and growth medium applied to assist with revegetation. Structures, explosives, magazines, fuel tanks and other ancillary facilities not required in the post-closure phase will be dismantle and removed from site during the active closure phase. Concrete footings will be cut down to grade and concreate slabs will be covered with a suitable growth medium to facilitate revegetation.

During the process of removing structures, a sampling program will be undertaken to determine if remediation of surficial materials is required. If so, contaminated rock, soil and/or overburden will be disposed of in an approved manner, either by remediating in an approved on-site facility, or hauling to a designated off-site facility.

The closure of the TMF embankment and stack will be carried out progressively during the operations phase, and at the end of economically viable mining. Specific measures will be taken to ensure that:

- Dust is not emitted from the facility as a result of moisture loss from the TMF surface;
- Runoff does not affect surface or groundwater quality; and
- The TMF embankment, DMS reject, and stored filtered tailings remain physically and chemically stable.

The reclaimed TMF will be required to maintain long-term geochemical and physical stability, protect the downstream environment, and shed surface water. Activities that will be carried out during operations and at closure to achieve these objectives are:

- Grading of the TMF stack to facilitate the shedding of surface water post-closure;
- Closure capping of the TMF stack with a HDPE geomembrane liner and construction of a rock cover to shed runoff from the stack, which may be possible to conduct progressively throughout operations as the stack is constructed and NPAG waste rock becomes available from the underground mine to construct the closure cover;
- Establishment of a spillway through the TMF embankment to facilitate the shedding of runoff and ensure no ponded water exists in the TMF post-closure;
- Removal of the seepage and runoff collection ditches, WMP, and reclaim system at such time that suitable water quality for direct release is achieved; and
- Long-term stabilization and vegetation of all exposed erodible materials.

21 Capital Cost Estimate

21.1 Capital Cost Summary

All capital costs are in Canadian dollars.

LOM project capital costs total $617.9 M, consisting of the following distinct phases:

- Pre-production Capital Costs – includes all costs to develop the property to a 4,000 t/d mining production, and 3,000 t/d milling operation. Initial capital costs total $302.3 M and are expended over a 24-month pre-production construction and commissioning period;
- Sustaining Capital Costs – includes all costs related to the development, acquisition, replacement, or major overhaul of assets during the mine life required to sustain operations. Sustaining capital costs total $305.9 M and are expended in operating years 1 through 19; and
- Closure Costs – includes all costs related to the closure, reclamation, and ongoing monitoring of the mine post operations. Closure costs total $9.7 M (net of equipment salvage values) and are incurred in Year 20.

The capital cost estimate was compiled using a combination of quotations, database costs, and database factors. Once compiled, the overall cost estimate was benchmarked against similar operations.

Table 21-1 presents the capital estimate summary for initial, sustaining, and closure capital costs in Q2 2018 dollars with no escalation.

Table 21-1: Capital Cost Summary

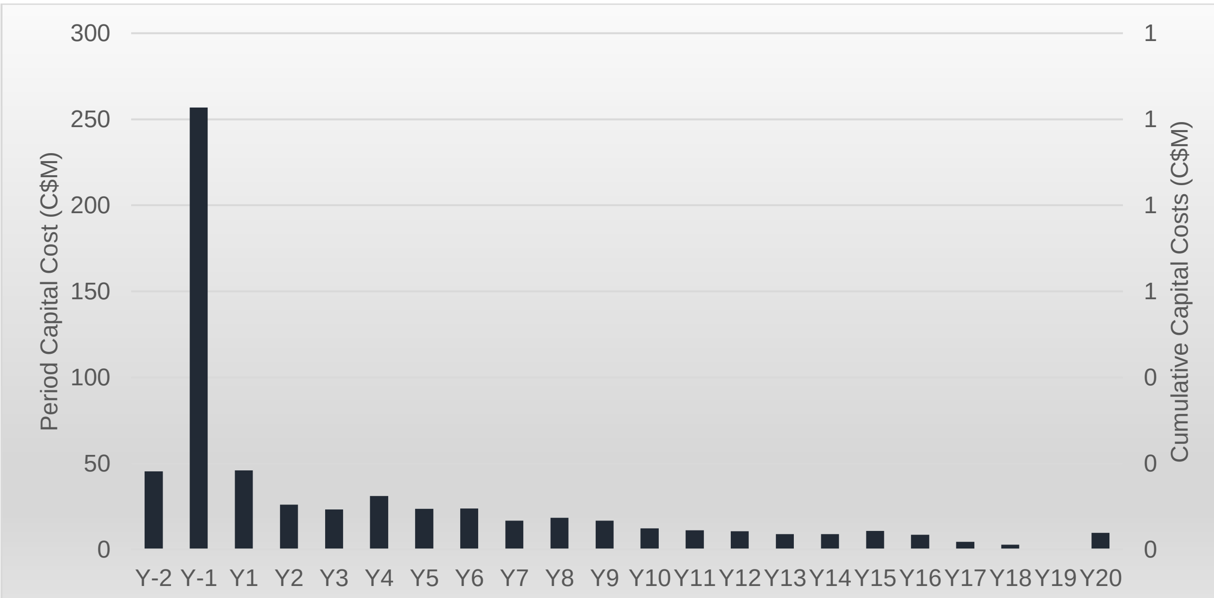
Area	Pre-Production (M$)	Sustaining (M$)	Closure (M$)	Total (M$)
Mining	58.2	260.0	-	318.2
Site Development	7.5	0.7	-	8.2
Mineral Processing	78.8	11.8	-	90.6
Tailings Management	5.0	8.3	-	13.3
On-Site Infrastructure	55.1	6.3		61.4
Off-Site Infrastructure	1.0	0.2	-	1.2
Indirect Costs	28.0	5.1	-	33.2
EPCM	17.4	1.5		18.8
Owners Costs	5.6	-	-	5.6
Closure Costs	-	-	8.9	8.9
Subtotal Pre-Contingency	256.7	293.8	**8.9**	559.4
Contingency	45.7	12.1	0.8	58.5
Total Capital Costs	**302.3**	**305.9**	**9.7**	**617.9**

Source: JDS (2018)

21.2 Capital Cost Profile

All capital costs for the Project have been distributed against the development schedule in order to support the economic cash flow model. Figure 21-1 presents an annual life of mine (LOM) capital cost profile.

Figure 21-1: Capital Cost Distribution



Source: JDS (2018)

21.3 Key Assumptions

The following key assumptions were made during development of the capital estimate:

- Underground mine development activities will be performed by the Owners forces; and
- All surface construction (civil, structural, architectural, mechanical, piping, electrical, and instrumentation) will be performed by contractors.

21.4 Key Estimate Parameters

- Estimate Class: The capital cost estimates are considered Class 4 estimates (-20% / +30%). The overall Project definition is estimated to be 10%;
- Estimate Base Date: The base date of the estimate is February 2018. No escalation has been applied to the capital cost estimate for costs occurring in the future;
- Units of Measure: The International System of Units (SI) is used throughout the capital estimate; and
- Currency: All capital costs are expressed in Canadian Dollars (C$). Portions of the estimate were estimated in US Dollars (US$) and converted to Canadian Dollars at a rate of C$1.00: US$0.77

21.5 Basis of Estimate

21.5.1 Underground Mine CAPEX

Capital cost estimates are based on a combination of budgetary quotes from equipment suppliers, in-house cost databases and similar mines in western Canada. Table 21-2 summarizes the underground mine capital cost estimate.

Table 21-2: Mining CAPEX Summary

Description	Unit	Initial	Sustaining	Total
UG Mobile Equipment Purchases	$M	2.7	10.3	**13.0**
UG Mobile Equipment Lease	$M	5.8	100.0	**105.8**
UG Mobile Equipment Rebuilds	$M	-	3.8	**3.8**
UG Infrastructure	$M	16.3	21.6	**37.9**
Capital Lateral Development	$M	21.6	111.2	**132.8**
Capital Vertical Development	$M	2.4	9.1	**11.5**
Capital Period Opex	$M	5.5	-	**5.5**
Total	**$M**	**54.3**	**256.0**	**310.3**

Source: JDS (2018)

21.5.1.1 Mobile Equipment Purchase and Replacement

Underground mining equipment quantities and costs were determined through buildup of mine plan quantities and associated equipment utilization requirements. Budgetary quotes were received and applied to the required quantities. Mobile equipment for the mine will be purchased under a lease agreement to distribute and defer capital costs. The totals include the total purchase and replacement value of the fleet over time as well as the actual down payment and lease payment spread between the initial and sustaining capital periods.

21.5.1.2 Underground Infrastructure

Design requirements for underground infrastructure were determined from design calculations for ventilation, dewatering, and material handling.

Budgetary quotations or database costs were used for major infrastructure components. Allowances have been made for miscellaneous items, such as initial PPE, radios, water supply, refuge stations, and geotechnical investigations. Acquisition of underground infrastructure is timed to support the mine plan requirements.

21.5.1.3 Lateral and Vertical Capital Waste Development

The majority of lateral development in waste rock for the mine has been capitalized. Underground infrastructure, with the exception of waste cross cuts into mineralization, have been considered capital projects. These items account for 80% of all lateral development in the mine. All costs associated with waste crosscuts, the other 20%, are captured in OPEX.

Additionally, 100% of the vertical development and associated costs are considered CAPEX.

21.5.1.4 Capitalized Operating Costs

Capitalized production costs are defined as mine operating expenses (operating development, mineralized material extraction, mine maintenance, and mine general costs) incurred prior to and during commissioning and ceasing at commencement of commercial operations and generation of project revenues. They are included as a pre-production capital cost. Once plant feed is processed, these costs transition to operating expenses.

The basis of these costs is described in Section 22, Operating Costs, as they are estimated in the same manner. Capitalized production costs are included in the asset value of the mine development and are depreciated over the mine life within the financial model.

21.5.2 Surface Construction Costs

Surface construction costs include site development, crushing plant, mineral processing plant, tailings management facility, on-site and off-site infrastructure. These cost estimates are primarily based on database or recently quoted costs, with factors applied for minor cost elements. Table 21-3 presents a summary basis of estimate for the various commodity types within the surface construction estimates.

Table 21-3: Surface Construction Basis of Estimate

Commodity	Basis
Contractor Labour Rates	Database values based on similar northern Canadian projects
Bulk Earthworks, Including On-Site Roads	Estimate volumes from preliminary site layout model Database unit rates for bulk excavation and fill, grading and surfacing Allowances for surface drainage and site water management
Concrete	Quantities developed based on building sizes outlined in general arrangements and cross checked against similar projects Database unit rates in BC from recent local contractor's quotations in the region
Structural Steel	Quantities developed based on equipment sizes and cross checked against similar projects Database unit rates in Canada
Pre-Engineered Buildings	Database unit rates ($/m^2) applied against the building sizes outlined in the general arrangements Database allowances for lighting, small power, electrical/control rooms, and fire detection
Modular Buildings & Warehouses	Database costs from similar northern projects for the mine dry, administration offices, mine maintenance building, mine warehouse, and camp structures
Mechanical Equipment	A combination of quoted costs and database costs from recent quotations on similar projects A combination of actual install hours based on equipment size and database factors applied against mechanical equipment costs for installation
Piping	Database factors applied against mechanical equipment costs
Electrical and Instrumentation	Database factors applied against mechanical equipment costs
On-site Power Transmission Lines	Database costs from similar projects Quantities developed based on general arrangements and site layouts

Source: JDS (2018)

21.5.2.1 Surface Construction Sustaining Capital

Sustaining capital costs are included in the estimate for continued construction of the TMF. The balance of the facility is expanded yearly throughout the LOM.

The sustaining capital cost estimate also include an additional generator in year 1 as UG mining power demand increases.

Allowances are provided for the processing plant, on-site infrastructure and the off-site airstrip for major equipment overhauls, minor capital projects and upgrades.

21.5.3 Indirect Costs

Indirect costs are those that not directly accountable to a specific cost object. Table 21-4 presents the subjects and basis for the indirect costs within the capital estimate.

Table 21-4: Indirect Costs Basis of Estimate

Commodity	Basis
Heavy Equipment	Factor (1.5%) of on-site direct costs for heavy equipment rental (i.e. 100 t + crane), and factor (1%) of off-site infrastructure direct costs
Contractor Field Indirect Costs	Factor (6.0%) for the following items: • Time based cost allowance for general construction site services (temporary power, heating & hoarding, contractor support, etc.) applied against the surface construction schedule • Construction offices and ablution facilities • Combination of diesel and transmission line construction power • Contractor mobilization
Freight & Logistics	Factor (8%) for freight and logistics related to the materials and equipment required for the crushing plant, mineral processing plant, on-site and off-site infrastructure. Factor excludes mining equipment as prices are FOB site
Vendor Representatives	Factor (1.5%) of direct costs for the provision of vendor services for commissioning equipment
Capital Spares	Factor (5%) of direct costs for spare parts
Start-up and Commissioning	Includes plant staffing for 3 months, 2 months of power, maintenance and wear parts, and 1 month supply of re-agents for first fills
Detailed Engineering & Procurement	Factor (7%) applied against direct and indirect hours for engineering management, detailed design, drawings, and major equipment procurement
Project & Construction Management	Staffing plan built up against the development schedule for project management, health and safety, construction management, field engineering, Project controls, contract administration and the start-up and commissioning in year 1.
	Database unit (hourly) rates

Source: JDS (2018)

21.5.4 Owners Costs

Owner's costs are items that are included within the operating costs during production. These items are included in the initial capital costs during the construction phase and capitalized. The cost elements described below are described in more detail within Section 22.

- Pre-production General & Administration: Costs of the Owner's labour and expenses (safety, finance, security, purchasing, management, etc.) incurred prior to commercial production; and
- Surface Support: Costs of the Owner's surface support labour, maintenance, and equipment usage costs for contract water supply and waste removal prior to commercial production.

21.5.5 Closure Cost Estimate

Closure costs have been estimated based on the typical closure, reclamation, and monitoring activities for a surface mine in northern Canada. Activities include:

- Removal of all surface infrastructure and buildings;
- Closure and capping of the TMF; and
- Re-vegetation and seeding allowances.

The majority of closure costs are incurred immediately following completion of operations (Year 20).

21.5.6 Cost Contingency

An overall contingency of 15% was applied to the initial capital costs of the project. LOM project contingency amounts to $58.5 M, or approximately 10% of LOM capital costs. The overall contingency is a blend of separate factors that were applied different areas as follows:

- Mobile mining equipment and capital development – 0%;
- Underground infrastructure – 20%;
- Process Plant, Site Infrastructure and Project Indirect Costs – 20%;
- Civil Works and Tailings Management – 35%; and
- Indirect and Owners Costs – 20%.

21.6 Processing Capital Costs

The process plant capital costs consist of the equipment, structural steel, concrete foundations, electrical equipment, instruments, controls, labour and all piping and wiring materials necessary for installation to an operational readiness level. The costs per area are provided in Table 21-5.

Table 21-5: Process Plant CAPEX

Description	Unit	Initial	Sustaining	Total
Crushing & Ore Handling	$M	5.9	0.9	**6.7**
Crushed Material Storage & Reclaim	$M	2.5	0.4	**2.9**
DMS	$M	5.2	0.8	**6.0**
Grinding	$M	13.4	2.0	**15.4**
Lead Circuit				
Pb Rougher Flotation	$M	2.3	0.3	**2.6**
Pb Regrind	$M	4.8	0.7	**5.5**
Pb Cleaner	$M	2.2	0.3	**2.5**
Pb Dewatering - Concentrate	$M	2.2	0.3	**2.5**
Zinc Circuit				
Zn Rougher Flotation	$M	2.4	0.4	**2.8**
Zn Regrind	$M	4.8	0.7	**5.5**
Zn Cleaner	$M	5.0	0.8	**5.8**
Zn Dewatering - Concentrate	$M	2.6	0.4	**3.0**
Tailings	$M	1.0	0.1	**1.1**
Reagents	$M	1.6	0.2	**1.8**
Plant Utilities, Building, & General				
Plant Building	$M	19.9	3.0	**22.9**
Plant Water Systems	$M	0.5	0.1	**0.6**
Plant Air Systems	$M	0.7	0.1	**0.8**
Assay Lab	$M	1.8	0.3	**2.1**
TOTAL	**$M**	**78.8**	**11.8**	**90.6**

Source: JDS (2018)

21.7 Infrastructure Capital Costs

The infrastructure capital costs include the direct costs to supply and construct the tailings management facility, on-site infrastructure and off-site infrastructure associated with the project. The infrastructure costs are provided in Table 21-6.

Table 21-6: Infrastructure CAPEX

Description	Unit	Initial	Sustaining	Total
Tailings Management Facility	$M	5.0	8.3	13.3
Camp Complex and Accommodations	$M	8.9	0.7	9.6
Power Supply & Distribution				
LNG Generators and Fuel Storage	$M	24.2	5.1	29.3
On-Site Power Distribution	$M	1.5	0.1	1.6
Water Supply, Distribution & Management	$M	10.9	0.0	10.9
Waste Management	$M	1.3	0.1	1.4
Ancillary Buildings				
Mine Dry	$M	0.4	0.0	0.4
Mine Office	$M	0.3	0.0	0.3
Mine Maintenance Shop / Truck Shop	$M	2.1	0.2	2.3
Mine / Plant Warehouse	$M	0.5	0.0	0.5
Emergency Response Facility	$M	0.1	0.0	0.1
Surface Mobile Equipment	$M	3.8	0.0	3.8
Bulk Fuel Storage & Distribution	$M	0.5	0.0	0.5
IT & Communications	$M	0.6	0.0	0.6
Off-Site Airstrip Upgrades	$M	1.0	0.2	1.2
TOTAL	**$M**	**61.1**	**14.7**	**75.8**

Source: JDS (2018)

21.8 Capital Estimate Exclusions

The following items have been excluded from the capital cost estimate:

- Working capital (included in the financial model);
- Financing costs;
- Currency fluctuations;
- Lost time due to severe weather conditions beyond those expected in the region;
- Lost time due to force majeure;
- Additional costs for accelerated or decelerated deliveries of equipment, materials or services resultant from a change in Project schedule;
- Warehouse inventories, other than those supplied in initial fills, capital spares, or commissioning spares;
- Any Project sunk costs (studies, exploration programs, etc.);
- Provincial sales tax;

- Closure bonding; and
- Escalation cost.

22 Operating Cost Estimate

22.1 Operating Cost Summary

The operating cost estimate (OPEX) is based on a combination of experiential judgment, reference projects, budgetary quotes and factors as appropriate with a PEA study.

Preparation of the OPEX is based on the JDS philosophy that emphasizes accuracy over contingency and utilizes defined and proven Project execution strategies.

All operating costs are in Canadian dollars.

Total LOM operating costs amount to $2,014.1 M or an average unit cost of $102.38 /t processed. The LOM costs are summarized in Table 22-1. UG mining costs average $38.13 /t mined ($50.05 / t processed).

Table 22-1: LOM Total Operating Cost Estimate

Description	Total Estimate ($M)	Average Unit Cost ($/t processed)
UG Mining	984.7	50.05
Processing	651.7	33.13
Tailings & DMS rejects	56.5	2.87
G&A	321.3	16.33
Total Operating Costs	**2,014.1**	**102.38**

Source: JDS (2018)

22.2 Mine Operating Costs

22.2.1 Underground Mine Operating Costs

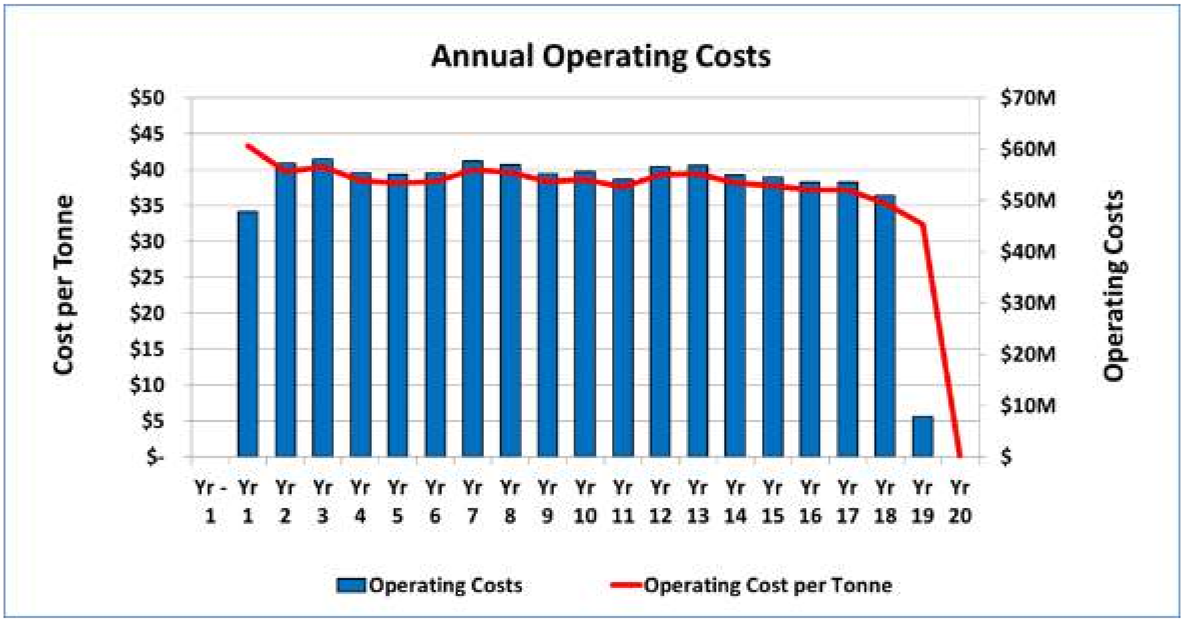
The total mine operating costs per tonne mined are broken out by cost center in Table 22-2 and shown graphically by year in Figure 22-1.

Table 22-2: Overall Mining OPEX

Description	Total ($M)	Average Unit Cost ($/t mined)
Lateral Waste Development	23.7	0.94
Production	520.5	20.41
Backfill	209.9	8.29
Mine Maintenance	72.0	2.86
Mine General	158.6	6.31
Total	**984.7**	**38.81**

Source: JDS (2018)

Figure 22-1: Annual Operating Costs



Source: JDS (2018)

22.2.1.1 Development

Development operating costs are for non-capitalized development.

A total of 8,729 m of lateral waste is classified as operating costs over the LOM, costs associated with these meters are shown in Table 22-3.

Table 22-3: Waste Development OPEX

Description	Total ($M)	Average Unit Cost ($/t mined)
Labour	10.5	0.41
Fuel	0.3	0.01
Equipment	1.9	0.07
Power	2.8	0.13
Consumables	4.6	0.18
Explosives	3.6	0.14
Total	**23.7**	**0.94**

Source: JDS (2018)

22.2.1.2 Production

Production operating costs are those costs which are directly associated with the extraction of the mineable resource, including lateral development through mineralization and long-hole. These costs are summarized in Table 22-4.

Table 22-4: Mine Production OPEX

Description	Total ($M)	Average Unit Cost ($/t mined)
Labour	226.6	8.87
Fuel	41.4	1.66
Equipment	104.7	4.15
Power	41.0	1.59
Consumables	74.1	2.87
Explosives	32.7	1.27
Total	**520.5**	**20.41**

Source: JDS (2018)

22.2.1.3 Backfill

Backfill operating costs are associated with the manufacturing, distribution and placement of paste, CRF and RF at the mine. These costs include all consumable materials including cement and binder required to manufacture the product, as summarized in Table 22-5:

Table 22-5: Backfill OPEX

Description	Total ($M)	Average Unit Cost ($/t mined)
Labour	18.4	0.71
Fuel	2.1	0.08
Equipment	2.2	0.08
Power	26.5	1.19
Cement	146.1	5.66
Parts, Other Consumables, Bulkheads	12.2	0.47
Waste Crushing & Screening	2.5	0.10
Total	**209.9**	**8.29**

Source: JDS (2018)

22.2.1.4 Mine Maintenance

Mine maintenance OPEX includes all costs associated with labour and general shop consumables required to maintain the mobile fleet, as summarized in Table 22-6. Mine maintenance costs do not include mobile equipment consumable parts or major overhaul costs.

Table 22-6: Mine Maintenance OPEX

Description	Total ($M)	Average Unit Cost ($/t mined)
Labour	69.3	2.72
Shop Consumables	2.7	0.14
Total	**72.0**	**2.86**

Source: JDS (2018)

22.2.1.5 General Mine OPEX

General mine expenses include pumping, ventilation, compressed air, definition drilling, and supervisory and technical support, as summarized in Table 22-7:

Table 22-7: General Mine OPEX

Description	Total ($M)	Average Unit Cost ($/t mined)
Power	5.2	0.20
Fuel	17.3	0.73
Equipment	17.0	0.70
Definition Drilling	9.1	0.35
Mine Air Heating	31.2	1.25
Technical Services Labour	71.3	2.80
Technical Services Supplies	2.5	0.11
Misc Supplies/PPE	4.4	0.17
Total	**158.6**	**6.31**

Source: JDS (2018)

22.3 Process Operating Costs

Process operating costs include all lead and zinc recovery steps required to produce saleable concentrates. The crushing and DMS plants will process 4,000 t/d and the process plant will process 3,000 t/d with assumed availabilities of 75% and 92%, respectively. Labour rates and benefit packages were based on industry information compiled by JDS. Power costs were calculated from the total installed power assuming $0.147 /kWh. Liner pricing and Vendor recommended spare parts for one year of operation were used to estimate mill and crusher wear costs. Costs for media were determined using engineering calculations based on mill power draw, estimated abrasion index and vendor quotes for media as a cost per tonne. Reagent costs were developed using the metallurgical test results and pricing supplied by Vendors. Equipment maintenance was calculated by applying a factor of 4% to major process equipment cost. A breakdown of the process operating costs is summarized in Table 22-8.

Table 22-8: Breakdown of Process Operating Costs

Description	$M/a	$/t processed
Labour	6.6	6.04
Power	9.2	8.41
Maintenance, Consumables & Tailings Facility	20.4	18.67
Total Processing OPEX	**36.3**	**33.13**

Source: JDS (2018)

22.4 General and Administration Costs

General and administrative costs comprise the following categories:

- Administration, site services and power plant labour;
- On-site items as such camp catering, health and safety, environmental, human resources, legal, external consulting, communications and office supplies, site service equipment operation and maintenance; and
- Employee travel via air charter from Prince George, BC.

The total G&A unit operating cost is summarized in Table 22-9.

Table 22-9: G&A OPEX Estimate by Area

Parameter	Total ($M)	$/t processed
G&A Labour	112.8	5.65
G&A Items - On-site	160.0	7.97
Employee Travel	54.2	2.72
Total Operating Cost – G&A	**327.0**	**16.33**

Source: JDS (2018)

23 Economic Analysis

This PEA is preliminary in nature and includes the use of inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the results of this PEA will be realized.

An engineering economic model was developed to estimate annual cash flows and sensitivities of the project. Pre-tax estimates of project values were prepared for comparative purposes, while after-tax estimates were developed and are likely to approximate the true investment value. It must be noted, however, that tax estimates involve many complex variables that can only be accurately calculated during operations and, as such, the after-tax results are only approximations.

Sensitivity analyses were performed for variations in metal prices, US$:C$ exchange rates, operating costs, capital costs, and discount rates to determine their relative importance as project value drivers.

This technical report contains forward-looking information regarding projected mine production rates, construction schedules and forecasts of resulting cash flows as part of this study. The mill head grades are based on sufficient sampling that is reasonably expected to be representative of the realized grades from actual mining operations. Factors such as the ability to obtain permits to construct and operate a mine, or to obtain major equipment or skilled labour on a timely basis, to achieve the assumed mine production rates at the assumed grades, may cause actual results to differ materially from those presented in this economic analysis.

The estimates of capital and operating costs have been developed specifically for this project and are summarized in Section 21 and Section 22 of this report (presented in 2018 Canadian dollars). The economic analysis has been run with no inflation (constant dollar basis).

23.1 Assumptions

The model excludes all pre-development and sunk costs up to the start of detailed engineering (i.e. exploration and resource definition costs, engineering fieldwork and studies costs, environmental baseline studies costs, financing costs, etc.).

Table 23-1 outlines the metal prices and exchange rate assumptions used in the economic analysis. The base case metal prices were selected based on the average of three years past and projected two years forward by analysis of London Metal Exchange futures as of 30 April 2018. The spot prices presented are at close of London Metal Exchange on 15 June 2018. These parameters are in line with other recently released comparable Technical Reports.

The reader is cautioned that the metal prices and exchange rates used in this study are only estimates based on recent historical performance and there is absolutely no guarantee that they will be realized if the project is taken into production. The metal prices are based on many complex factors and there are no reliable long-term predictive tools.

Table 23-1: Metal Price and Exchange Rates Used in Economic Analysis

Parameter	Unit	Base Price Value	Spot Price Value
Lead Price	US$/lb	1.00	1.08
Zinc Price	US$/lb	1.21	1.42
Silver Price	US$/oz	16.95	16.95
Exchange Rate	US$:C$	0.77	0.77

Source: JDS (2018)

Other economic factors include the following:

- Discount rate of 7%;
- Closure cost of $8.9 M (pre-contingency);
- Nominal 2018 dollars;
- Revenues, costs, taxes are calculated for each period in which they occur rather than actual outgoing / incoming payment;
- Working capital calculated as two months of operating costs (mining, processing, tailings & DMS rejects, and G&A) in Year 1;
- Results are presented on 100% ownership; and
- No management fees or financing costs (equity fund-raising was assumed).

23.2 Processing and Concentrate Terms

Mine revenue is derived from the sale of zinc concentrate and lead concentrate into the international marketplace. No contractual arrangements for refining exist at this time. Details regarding the terms used for the economic analysis can be found in the market studies (Section 19) of this report. The concentrate terms for the Akie PEA assumed shipping to the Trail Smelter. An additional scenario was run for an overseas option, and economic results for this scenario are presented in Section 23.4.1.

Table 23-2 outlines the recoveries, payable terms, treatment charges and transportation costs used in the economic analysis.

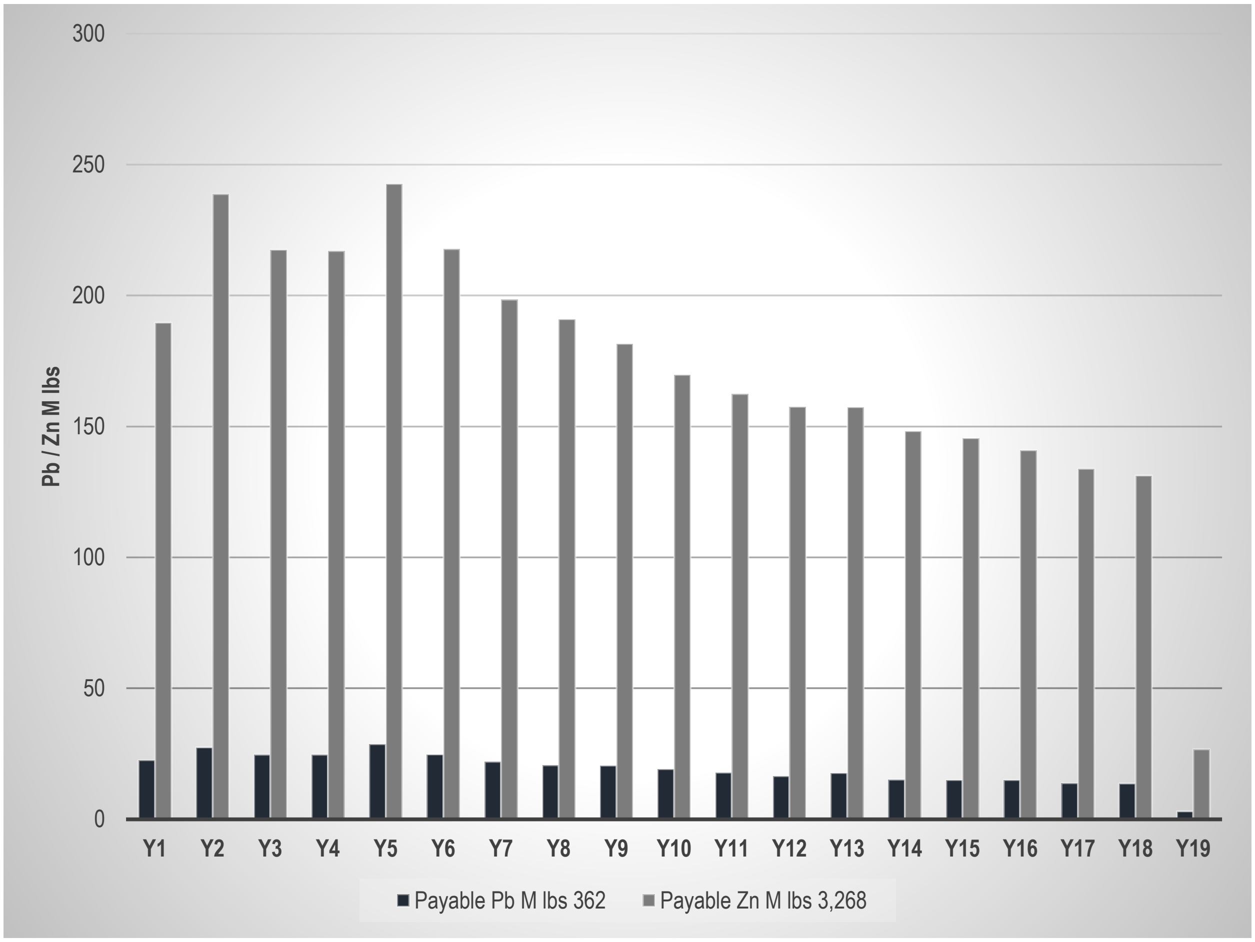
Table 23-2: Concentrate Terms

Assumptions & Inputs	Unit	Value
Lead Concentrate		
Metal Recovery to Concentrate	% Pb	46.2
	% Zn	1.3
	% Ag	4.8
Pb Concentrate Grade Produced	% Pb	45.1
Minimum Deduction	% Pb/t	3
	g/t Ag	50
Metal Payable	% Pb	95
	% Ag	95
Pb Treatment Charge	US$/dmt conc.	140
Ag Refining Charge	US$/oz	1.50
Moisture Content	%	8
Pb Concentrate Transportation Cost to Trail, BC	C$/wmt	231
Zinc Concentrate		
Metal Recovery to Concentrate	% Pb	0.0
	% Zn	88.8
	% Ag	18.3
Zn Concentrate Grade Produced	% Zn	52.4
Minimum Deduction	% Pb/t	0
	%Zn/t	8
	g/t Ag	93.31
Metal Payable	% Pb	0
	% Zn	85
	% Ag	85
Zn Treatment Charge	US$/dmt conc.	190
Ag Refining Charge	US$/oz	0.50
Moisture Content	%	8
Zn Concentrate Transportation Cost to Trail, BC	C$/wmt	180

Source: JDS (2018)

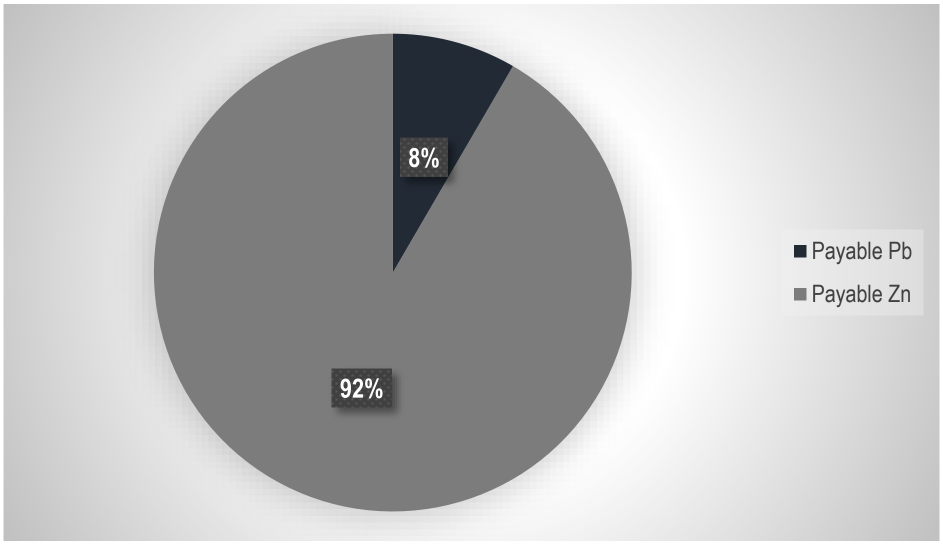
Figure 23-1 shows a breakdown of the payable lead and zinc recovered during the mine life. A total of 362 Mlbs of lead and 3,268 Mlbs of zinc are projected to be produced during the mine life. Zinc accounts for about 92% of project revenues and lead for about 8% as illustrated in Figure 23-2.

Figure 23-1: Payable Metal Production by Year



Source: JDS (2018)

Figure 23-2: Revenue Distribution



Source: JDS (2018)

23.3 Taxes

The Project has been evaluated on an after-tax basis to provide a more indicative, but still approximate, value of the potential project economics. A tax model was prepared by a tax consultant with applicable British Columbia mineral tax regime experience. Current tax pools were used in the analysis. The tax model contains the following assumptions:

- 15% federal income tax rate; and
- BC Mineral Taxes
  - 2% Net Current Proceeds Tax
  - 13% Net Revenue Tax

Total taxes for the project amount to $458.0 M over the LOM.

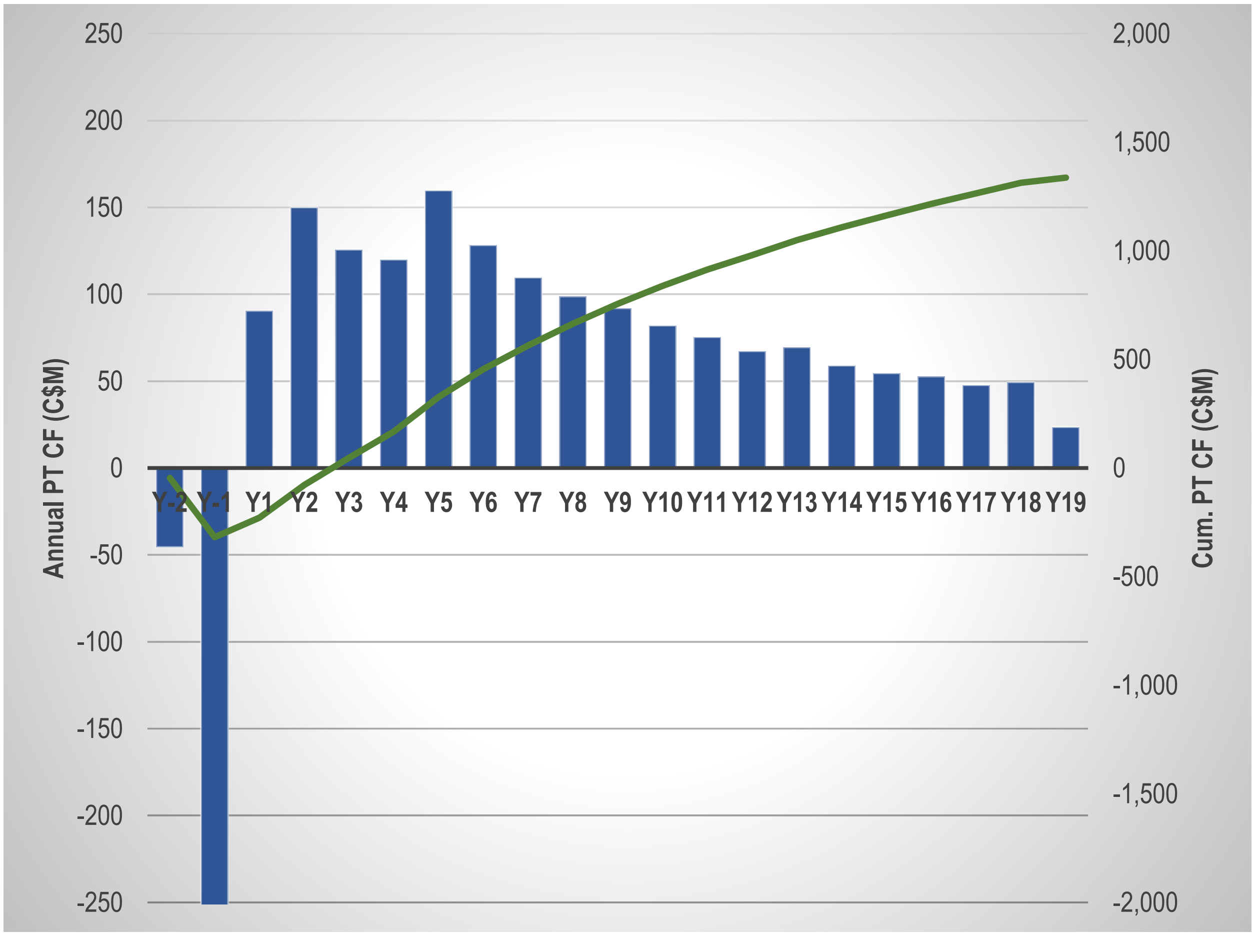
23.4 Economic Results

At this preliminary stage, the project has an after-tax IRR of 27% and a net present value using a 7% discount rate ($NPV_{7\%}$) of $401 M using the metal prices described in Section 19.

Figure 23-3 shows the projected pre-tax cash flows, and Table 23-3 summarizes the economic results of the Akie Project.

The pre-tax break-even zinc price for the project is approximately US$0.90/lb, based on the LOM plan presented herein, a lead price of US$1.00/lb, silver price of US$16.95/oz, and an FX rate of 0.77 US$:C$.

Figure 23-3: Annual Pre-Tax Cash Flow



Source: JDS (2018)

Table 23-3: Summary of Results

Parameter	Unit	Base Price Value	Spot Price Value
Capital Costs			
Pre-Production Capital	C$M	256.7	256.7
Pre-Production Contingency	C$M	45.7	45.7
Total Pre-Production Capital	**C$M**	**302.3**	**302.3**
Sustaining & Closure Capital	C$M	302.7	302.7
Sustaining & Closure Contingency	C$M	12.9	12.9
Total Sustaining & Closure Capital	C$M	315.6	315.6
Total Capital Costs Incl. Contingency	**C$M**	**617.9**	**617.9**
Cash Flows			
Working Capital	C$M	15.8	15.8
Pre-Tax Cash Flow	**LOM C$M**	**1,327.7**	**2,256.5**
	C$M/a	**72**	**123**
Taxes	LOM C$M	458.0	797.5
After-Tax Cash Flow	**LOM C$M**	**869.6**	**1,459.0**
	C$M/a	**47**	**80**
Economic Results			
Pre-Tax NPV$_{7\%}$	**C$M**	**649**	**1,160.3**
Pre-Tax IRR	**%**	**35.0**	**52.2**
Pre-Tax Payback	**Years**	**2.6**	**1.8**
After-Tax NPV$_{7\%}$	**C$M**	**401**	**727.2**
After-Tax IRR	**%**	**27.0**	**40.0**
After-Tax Payback	**Years**	**3.2**	**2.2**

Source: JDS (2018)

23.4.1 Overseas Smelter Scenario

An additional scenario was investigated assume an overseas smelter and transportation costs to the Port of Prince Rupert, BC. Estimated transportation costs for zinc and lead concentrate to Prince Rupert were C$227/wmt and C$289/wmt respectively. Treatment charges for zinc and lead concentrate were estimated at US$190/dmt and US140/dmt respectively. All other concentrate terms list is Table 23-2 were utilized.

At this preliminary stage, the overseas smelter scenario has an after-tax IRR of 24% and a net present value using a 7% discount rate (NPV$_{7\%}$) of $330 M using the metal prices described in Section 19.

23.5 Sensitivities

A univariate sensitivity analysis was performed to examine which factors most affect the project economics when acting independently of all other cost and revenue factors. Each variable evaluated was tested using the same percentage range of variation, from -15% to +15%, although some variables may experience significantly larger or smaller percentage fluctuations over the LOM. For instance, the metal prices were evaluated at a +/- 10% range to the base case, while the mill feed grade and all other variables remained constant. This may not be truly representative of market scenarios, as metal prices may not fluctuate in a similar trend. The variables examined in this analysis are those commonly considered in similar studies – their selection for examination does not reflect any particular uncertainty.

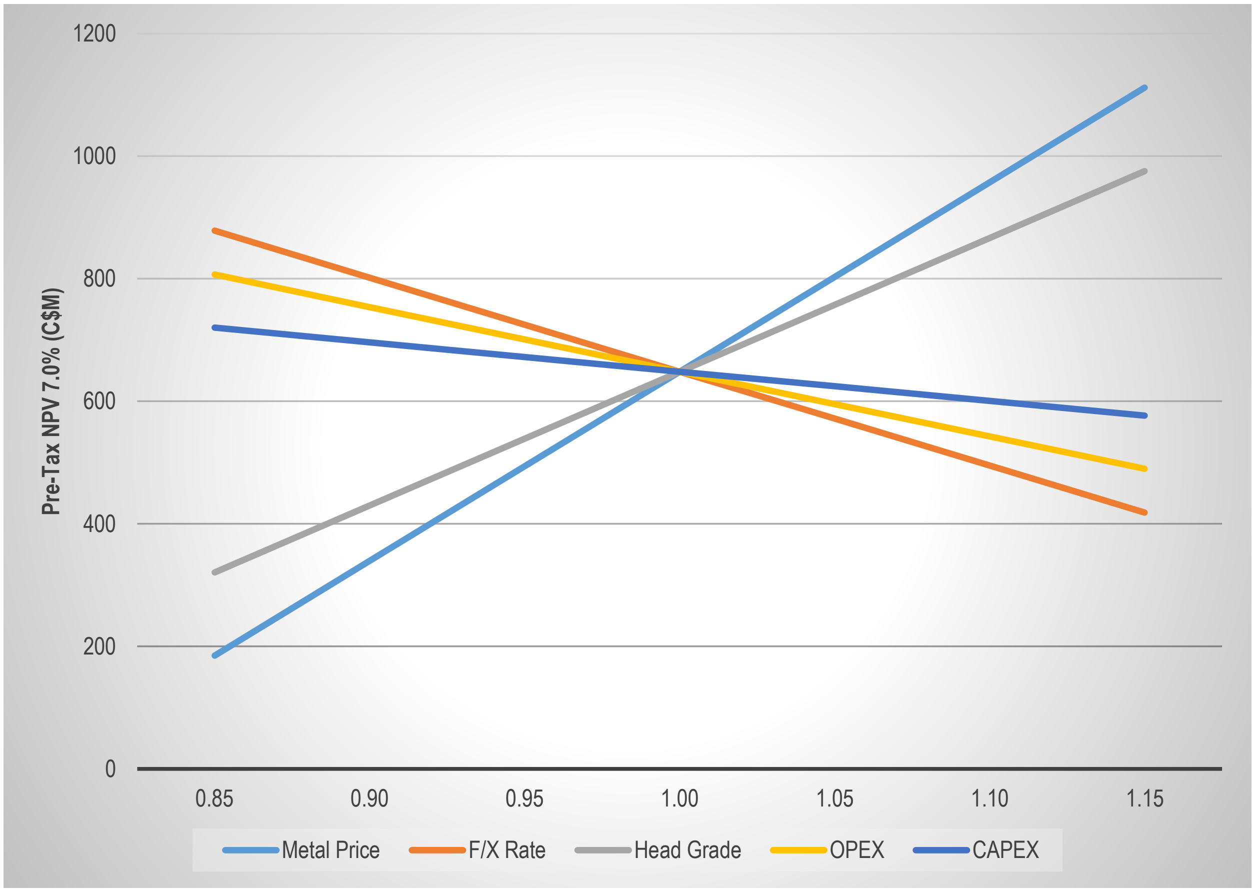
Notwithstanding the above noted limitations to the sensitivity analysis, which are common to studies of this sort, the analysis revealed that the project is most sensitive to metal prices, followed by mill feed grade, exchange rate, and operating costs. The Project showed the least sensitivity to capital costs. Table 23-4 and Figure 23-4 show the results of the sensitivity tests.

Table 23-4: Sensitivity Results on Base Case Pre-Tax $NPV_{7\%}$

Parameter	-15%	-10%	-5%	Base	+5%	+10%	+15%
Metal Price	185	340	494	649	803	958	1112
C$:US$ FX	879	802	725	649	572	495	418
Mill Feed Grade	321	430	539	649	758	867	976
OPEX	807	754	701	649	596	543	490
CAPEX	720	696	672	649	625	601	577

Source: JDS (2018)

Figure 23-4: Sensitivity, Pre-Tax NPV @ 7% Discount Rate



Source: JDS (2018)

Sensitivities were also completed specific to Zinc price and FX rate, while keeping all other metal prices constant. See Table 23-5 and Table 23-6 for results. The economic cash flow model for the project is illustrated in Table 23-7.

Table 23-5: Sensitivity, Pre-Tax NPV 7%, Zn Price

Parameter	US$1.01/lb	US$1.11/lb	Base Case (US$1.21/lb)	US$1.31/lb	US$1.41/lb
Pre-Tax NPV 7%	181	415	649	862	1,116

Source: JDS (2018)

Table 23-6: Sensitivity, Pre-Tax NPV 7%, FX Rate

Parameter	0.73	0.75	Base Case (0.77)	0.79	0.81
Pre-Tax NPV 7%	728	688	649	607	569

Source: JDS (2018)

Table 23-7: Economic Cash Flow Model

ZincX Resources Corp.
Akie Project, British Columbia, Canada
PEA Economic Model

LEGEND: link | input cell | calc

	Unit	LOM Total	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14	Y15	Y16	Y17	Y18	Y19	Y20
METAL PRICES & F/X RATE																								
Ag	US$/oz	16.95	16.95	16.95	16.95	16.95	16.95	16.95	16.95	16.95	16.95	16.95	16.95	16.95	16.95	16.95	16.95	16.95	16.95	16.95	16.95	16.95	16.95	16.95
Pb	US$/lb	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00
Zn	US$/lb	1.21	1.21	1.21	1.21	1.21	1.21	1.21	1.21	1.21	1.21	1.21	1.21	1.21	1.21	1.21	1.21	1.21	1.21	1.21	1.21	1.21	1.21	1.21
F/X	USD:CAD	0.77	0.77	0.77	0.77	0.77	0.77	0.77	0.77	0.77	0.77	0.77	0.77	0.77	0.77	0.77	0.77	0.77	0.77	0.77	0.77	0.77	0.77	0.77
PRODUCTION SCHEDULE																								
Ore Mined	**ktonnes**	**25,827**	-	-	1,106	1,440	1,440	1,440	1,440	1,440	1,440	1,440	1,440	1,440	1,440	1,440	1,440	1,440	1,440	1,440	1,440	1,440	242	-
Production Rate	tpd	3,895	-	-	3,029	3,945	3,945	3,945	3,945	3,945	3,945	3,945	3,945	3,945	3,945	3,945	3,945	3,945	3,945	3,945	3,945	3,945	3,900	-
Ag	g/t	13.08	-	-	20.03	17.64	15.15	15.60	19.03	16.19	13.59	13.26	12.93	12.40	11.32	11.15	11.37	10.29	10.38	9.54	8.84	9.10	8.43	-
Pb	%	1.5%	-	-	2.2%	2.0%	1.8%	1.8%	2.1%	1.8%	1.6%	1.5%	1.5%	1.4%	1.3%	1.2%	1.3%	1.1%	1.1%	1.1%	1.0%	1.0%	1.0%	-
Zn	%	7.6%	-	-	10.6%	10.0%	9.1%	9.1%	10.2%	9.1%	8.3%	8.0%	7.6%	7.1%	6.8%	6.6%	6.6%	6.2%	6.1%	5.9%	5.6%	5.5%	5.1%	-
Mining Rate	*ktpd*	3.9	-	-	3.0	3.9	3.9	3.9	3.9	3.9	3.9	3.9	3.9	3.9	3.9	3.9	3.9	3.9	3.9	3.9	3.9	3.9	0.7	-
Contained Metal																								
Ag	kg	337,793	-	-	22,147	25,402	21,822	22,464	27,405	23,315	19,572	19,097	18,613	17,862	16,307	16,049	16,366	14,812	14,941	13,743	12,734	13,105	2,037	-
Pb	ktonnes	381	-	-	24	29	26	26	30	26	23	22	22	20	19	17	19	16	16	16	14	14	2	-
Zn	ktonnes	1,970	-	-	117	144	131	131	147	131	120	115	109	102	98	95	95	89	88	85	81	79	12	-
Waste Mined	ktonnes	3,041	-	3,038	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	-
TOTAL MINED (Ore+Waste)	*ktonnes*	*28,868*	-	*3,038*	*1,106*	*1,440*	*1,440*	*1,440*	*1,440*	*1,440*	*1,440*	*1,440*	*1,440*	*1,440*	*1,440*	*1,440*	*1,440*	*1,440*	*1,440*	*1,440*	*1,440*	*1,440*	*242*	-
MILL SCHEDULE																								
Ore Milled	**ktonnes**	**19,672**	-	-	821	1,095	1,095	1,095	1,095	1,095	1,095	1,095	1,095	1,095	1,095	1,095	1,095	1,095	1,095	1,095	1,095	1,095	236	-
Milling Rate	tpd	3,003	-	-	2,250	3,000	3,000	3,000	3,000	3,000	3,000	3,000	3,000	3,000	3,000	3,000	3,000	3,000	3,000	3,000	3,000	3,000	3,812	-
Ag	g/t	17.17	-	-	26.30	23.22	19.96	20.47	24.88	21.35	17.94	17.42	16.98	16.31	14.92	14.64	14.91	13.56	13.62	12.58	11.65	11.93	11.26	-
Pb	%	1.9%	-	-	2.9%	2.6%	2.4%	2.4%	2.7%	2.4%	2.1%	2.0%	2.0%	1.8%	1.7%	1.6%	1.7%	1.5%	1.4%	1.4%	1.3%	1.3%	1.3%	-
Zn	%	10.0%	-	-	13.9%	13.1%	12.0%	11.9%	13.4%	12.0%	10.9%	10.5%	10.0%	9.3%	8.9%	8.7%	8.7%	8.2%	8.0%	7.8%	7.4%	7.2%	6.8%	-
Mill Throughput	*ktpd*	2.9	-	-	2.3	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	0.6	-
Contained Metal																								
Ag	kg	337,793	-	-	21,597	25,425	21,859	22,412	27,240	23,381	19,642	19,079	18,597	17,856	16,332	16,031	16,326	14,848	14,911	13,771	12,757	13,066	2,660	-
Pb	ktonnes	381	-	-	24	29	26	26	30	26	23	22	22	20	19	17	19	16	16	16	14	14	3	-
Zn	ktonnes	1,970	-	-	114	144	131	131	146	131	120	115	109	102	98	95	95	89	88	85	81	79	16	-
Pb CONCENTRATE																								
Recovery to Pb Concentrate	% Pb	46.2%	46.2%	46.2%	46.2%	46.2%	46.2%	46.2%	46.2%	46.2%	46.2%	46.2%	46.2%	46.2%	46.2%	46.2%	46.2%	46.2%	46.2%	46.2%	46.2%	46.2%	46.2%	46.2%
	% Zn	1.3%	1.3%	1.3%	1.3%	1.3%	1.3%	1.3%	1.3%	1.3%	1.3%	1.3%	1.3%	1.3%	1.3%	1.3%	1.3%	1.3%	1.3%	1.3%	1.3%	1.3%	1.3%	1.3%
	% Ag	4.8%	4.8%	4.8%	4.8%	4.8%	4.8%	4.8%	4.8%	4.8%	4.8%	4.8%	4.8%	4.8%	4.8%	4.8%	4.8%	4.8%	4.8%	4.8%	4.8%	4.8%	4.8%	4.8%
Metal in Pb Concentrate	Pb ktonnes	176	-	-	11	13	12	12	14	12	11	10	10	9	9	8	9	7	7	7	7	7	1	-
	Pb Mlbs	388.0	-	-	24.2	29.3	26.4	26.4	30.6	26.5	23.5	22.0	22.0	20.5	19.1	17.6	19.0	16.2	16.1	16.1	14.7	14.6	3.2	-
	Ag kg	16,214	-	-	1,037	1,220	1,049	1,076	1,308	1,122	943	916	893	857	784	769	784	713	716	661	612	627	128	-
	Ag koz	521	-	-	33	39	34	35	42	36	30	29	29	28	25	25	25	23	23	21	20	20	4	-
Pull Factor		54	-	-	34	37	41	41	36	41	46	49	50	53	57	62	57	67	68	68	74	74	74	-
Pb Concentrate Grade	%	45.1%	45%	45%	45%	45%	45%	45%	45%	45%	45%	45%	45%	45%	45%	45%	45%	45%	45%	45%	45%	45%	45%	45%
	g/t Ag	41.62	-	-	42.66	41.36	39.49	40.58	42.43	42.16	39.88	41.38	40.41	41.48	40.84	43.41	41.05	43.68	44.18	40.81	41.40	42.59	40.19	-
	Ag koz	1.34	-	-	1.37	1.33	1.27	1.30	1.36	1.36	1.28	1.33	1.30	1.33	1.31	1.40	1.32	1.40	1.42	1.31	1.33	1.37	1.29	-
Pb Concentrate Produced	dmt	390,273	-	-	24,299	29,509	26,569	26,508	30,819	26,622	23,643	22,133	22,090	20,665	19,195	17,725	19,089	16,315	16,199	16,199	14,792	14,727	3,177	-
	wmt	424,210	-	-	26,412	32,075	28,879	28,813	33,499	28,937	25,699	24,057	24,011	22,462	20,864	19,266	20,749	17,733	17,608	17,608	16,078	16,007	3,453	-
Moisture Content	%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%
Minimum Deduction	%Pb/tonne	3%	3%	3%	3%	3%	3%	3%	3%	3%	3%	3%	3%	3%	3%	3%	3%	3%	3%	3%	3%	3%	3%	3%
Pb Payable	% Payable	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%
Pb Payable based on Min. deduction	%	42%	42%	42%	42%	42%	42%	42%	42%	42%	42%	42%	42%	42%	42%	42%	42%	42%	42%	42%	42%	42%	42%	42%
Pb Payable based on %	%	43%	43%	43%	43%	43%	43%	43%	43%	43%	43%	43%	43%	43%	43%	43%	43%	43%	43%	43%	43%	43%	43%	43%
Payable Pb in Pb Concentrate	ktonnes	164	-	-	10	12	11	11	13	11	10	9	9	9	8	7	8	7	7	7	6	6	1	-
	Mlbs	362.2	-	-	22.6	27.4	24.7	24.6	28.6	24.7	21.9	20.5	20.5	19.2	17.8	16.5	17.7	15.1	15.0	15.0	13.7	13.7	2.9	-
Revenues Pb in Pb Concentrate	**US$M**	**362.2**	-	-	**22.6**	**27.4**	**24.7**	**24.6**	**28.6**	**24.7**	**21.9**	**20.5**	**20.5**	**19.2**	**17.8**	**16.5**	**17.7**	**15.1**	**15.0**	**15.0**	**13.7**	**13.7**	**2.9**	-
	C$M	**470.4**	-	-	**29.3**	**35.6**	**32.0**	**32.0**	**37.1**	**32.1**	**28.5**	**26.7**	**26.6**	**24.9**	**23.1**	**21.4**	**23.0**	**19.7**	**19.5**	**19.5**	**17.8**	**17.8**	**3.8**	-
Min. Deduction Ag in Pb Conc	g/t Ag	50.0	50.00	50.00	50.00	50.00	50.00	50.00	50.00	50.00	50.00	50.00	50.00	50.00	50.00	50.00	50.00	50.00	50.00	50.00	50.00	50.00	50.00	50.00
Ag Payable	% Payable	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%
Payable Ag in Pb Concentrate	Ag koz	0.0	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Revenues Ag in Pb Concentrate	**US$M**	**0.0**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	C$M	**0.0**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Pb Treatment Charge	US$/dmt conc	140.00	140.00	140.00	140.00	140.00	140.00	140.00	140.00	140.00	140.00	140.00	140.00	140.00	140.00	140.00	140.00	140.00	140.00	140.00	140.00	140.00	140.00	140.00
	US$M	54.6	-	-	3.4	4.1	3.7	3.7	4.3	3.7	3.3	3.1	3.1	2.9	2.7	2.5	2.7	2.3	2.3	2.3	2.1	2.1	0.4	-
	C$M	71.0	-	-	4.4	5.4	4.8	4.8	5.6	4.8	4.3	4.0	4.0	3.8	3.5	3.2	3.5	3.0	2.9	2.9	2.7	2.7	0.6	-
Ag Refining Charge	US$/Ag oz	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50
	US$M	0.0	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	C$M	0.0	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Total Treatment + Refining Charges	**US$M**	**54.6**	-	-	**3.4**	**4.1**	**3.7**	**3.7**	**4.3**	**3.7**	**3.3**	**3.1**	**3.1**	**2.9**	**2.7**	**2.5**	**2.7**	**2.3**	**2.3**	**2.3**	**2.1**	**2.1**	**0.4**	-
	C$M	**71.0**	-	-	**4.4**	**5.4**	**4.8**	**4.8**	**5.6**	**4.8**	**4.3**	**4.0**	**4.0**	**3.8**	**3.5**	**3.2**	**3.5**	**3.0**	**2.9**	**2.9**	**2.7**	**2.7**	**0.6**	-
Pb Concentrate Transport Cost	US$/wmt conc	178.21	178.21	178.21	178.21	178.21	178.21	178.21	178.21	178.21	178.21	178.21	178.21	178.21	178.21	178.21	178.21	178.21	178.21	178.21	178.21	178.21	178.21	178.21
	US$M	75.6	-	-	4.7	5.7	5.1	5.1	6.0	5.2	4.6	4.3	4.3	4.0	3.7	3.4	3.7	3.2	3.1	3.1	2.9	2.9	0.6	-
	C$M	98.2	-	-	6.1	7.4	6.7	6.7	7.8	6.7	5.9	5.6	5.6	5.2	4.8	4.5	4.8	4.1	4.1	4.1	3.7	3.7	0.8	-
Penalties	US$/dmt conc	0.00	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	US$M	0.0	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	C$M	0.0	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
NSR Pb in Pb Concentrate	US$M	232.0	-	-	14.4	17.5	15.8	15.8	18.3	15.8	14.1	13.2	13.1	12.3	11.4	10.5	11.3	9.7	9.6	9.6	8.8	8.8	1.9	-
	C$M	301.3	-	-	18.8	22.8	20.5	20.5	23.8	20.6	18.3	17.1	17.1	16.0	14.8	13.7	14.7	12.6	12.5	12.5	11.4	11.4	2.5	-
NSR Ag in Pb Concentrate	US$M	0.0	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	C$M	0.0	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Total Pb Concentrate NSR	**US$M**	**232.0**	-	-	**14.4**	**17.5**	**15.8**	**15.8**	**18.3**	**15.8**	**14.1**	**13.2**	**13.1**	**12.3**	**11.4**	**10.5**	**11.3**	**9.7**	**9.6**	**9.6**	**8.8**	**8.8**	**1.9**	-
	C$M	**301.3**	-	-	**18.8**	**22.8**	**20.5**	**20.5**	**23.8**	**20.6**	**18.3**	**17.1**	**17.1**	**16.0**	**14.8**	**13.7**	**14.7**	**12.6**	**12.5**	**12.5**	**11.4**	**11.4**	**2.5**	-
Zn CONCENTRATE																								
Recovery to Zn Concentrate	% Pb	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%
	% Zn	88.8%	88.8%	88.8%	88.8%	88.8%	88.8%	88.8%	88.8%	88.8%	88.8%	88.8%	88.8%	88.8%	88.8%	88.8%	88.8%	88.8%	88.8%	88.8%	88.8%	88.8%	88.8%	88.8%
	% Ag	18.3%	18.3%	18.3%	18.3%	18.3%	18.3%	18.3%	18.3%	18.3%	18.3%	18.3%	18.3%	18.3%	18.3%	18.3%	18.3%	18.3%	18.3%	18.3%	18.3%	18.3%	18.3%	18.3%
Metal in Zn Concentrate	Zn ktonnes	1,749	-	-	101	128	116	116	130	117	106	102	97	91	87	84	84	79	78	75	72	70	14	-
	Zn Mlbs	3,856.3	-	-	223.7	281.8	256.6	256.1	286.3	256.9	234.2	225.4	214.2	200.3	191.7	185.9	185.7	174.9	171.8	166.3	158.0	154.9	31.5	-
	Ag kg	61,816	-	-	3,952	4,653	4,000	4,101	4,985	4,279	3,595	3,491	3,403	3,268	2,989	2,934	2,988	2,717	2,729	2,520	2,335	2,391	487	-
	Ag koz	1,987	-	-	127.1	149.6	128.6	131.9	160.3	137.6	115.6	112.3	109.4	105.1	96.1	94.3	96.1	87.4	87.7	81.0	75.1	76.9	15.7	-
Pull Factor		6	-	-	4	4	5	5	4	5	5	6	6	6	7	7	7	7	7	8	8	8	9	-

ZincX Resources Corp.
Akie Project, British Columbia, Canada
PEA Economic Model

LEGEND
link
input cell
calc

			Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14	Y15	Y16	Y17	Y18	Y19	Y20
Zn Concentrate Grade	%	52.4%	52%	52%	52%	52%	52%	52%	52%	52%	52%	52%	52%	52%	52%	52%	52%	52%	52%	52%	52%	52%	52%	52%
	g/t Ag	18.39	-	-	20.41	19.08	18.01	18.50	20.11	19.24	17.73	17.89	18.35	18.85	18.01	18.23	18.58	17.94	18.35	17.51	17.07	17.83	17.85	-
	Ag oz	0.59	-	-	0.66	0.61	0.58	0.59	0.65	0.62	0.57	0.58	0.59	0.61	0.58	0.59	0.60	0.58	0.59	0.56	0.55	0.57	0.57	-
Zn Concentrate Produced	dmt	3,338,208	-	-	193,679	243,901	222,151	221,696	247,848	222,387	202,741	195,110	185,453	173,367	165,913	160,965	160,790	151,428	148,709	143,944	136,752	134,104	27,269	-
	wmt	3,628,486	-	-	210,521	265,109	241,468	240,974	269,400	241,725	220,371	212,077	201,579	188,443	180,340	174,962	174,772	164,595	161,640	156,461	148,643	145,765	29,641	-
Moisture Content	%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%
Minimum Deduction	%Zn/tonne	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%
Zn Payable	% Payable	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%
Zn Payable based on Min. deduction	%	44%	44%	44%	44%	44%	44%	44%	44%	44%	44%	44%	44%	44%	44%	44%	44%	44%	44%	44%	44%	44%	44%	44%
Zn Payable based on %	%	45%	45%	45%	45%	45%	45%	45%	45%	45%	45%	45%	45%	45%	45%	45%	45%	45%	45%	45%	45%	45%	45%	45%
Payable Zn in Zn Concentrate	ktonnes	1,482	-	-	86	108	99	98	110	99	90	87	82	77	74	71	71	67	66	64	61	60	12	-
	Mlbs	3,267.6	-	-	189.6	238.7	217.5	217.0	242.6	217.7	198.5	191.0	181.5	169.7	162.4	157.6	157.4	148.2	145.6	140.9	133.9	131.3	26.7	-
Revenues Zn in Zn Concentrate	**US$M**	3,953.8	-	-	**229.4**	**288.9**	**263.1**	**262.6**	**293.6**	**263.4**	**240.1**	**231.1**	**219.7**	**205.3**	**196.5**	**190.6**	**190.4**	**179.4**	**176.1**	**170.5**	**162.0**	**158.8**	**32.3**	-
	C$M	5,134.8	-	-	**297.9**	**375.2**	**341.7**	**341.0**	**381.2**	**342.1**	**311.9**	**300.1**	**285.3**	**266.7**	**255.2**	**247.6**	**247.3**	**232.9**	**228.7**	**221.4**	**210.3**	**206.3**	**41.9**	-
Min. Deduction Ag in Zn Conc	g/t Ag	93.31	93.31	93.31	93.31	93.31	93.31	93.31	93.31	93.31	93.31	93.31	93.31	93.31	93.31	93.31	93.31	93.31	93.31	93.31	93.31	93.31	93.31	93.31
Ag Payable	% Payable	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%
Payable Ag in Zn Concentrate	Ag koz	0	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Revenues Ag in Zn Concentrate	**US$M**	0.0	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	C$M	0.0	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Zn Treatment Charge	US$/dmt conc	190.00	190.00	190.00	190.00	190.00	190.00	190.00	190.00	190.00	190.00	190.00	190.00	190.00	190.00	190.00	190.00	190.00	190.00	190.00	190.00	190.00	190.00	190.00
	US$M	634.3	-	-	36.8	46.3	42.2	42.1	47.1	42.3	38.5	37.1	35.2	32.9	31.5	30.6	30.6	28.8	28.3	27.3	26.0	25.5	5.2	-
	C$M	823.7	-	-	47.8	60.2	54.8	54.7	61.2	54.9	50.0	48.1	45.8	42.8	40.9	39.7	39.7	37.4	36.7	35.5	33.7	33.1	6.7	-
Ag Refining Charge	US$/Ag oz	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50
	US$M	0.0	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	C$M	0.0	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Total Treatment + Refining Charges	**US$M**	634.3	-	-	**36.8**	**46.3**	**42.2**	**42.1**	**47.1**	**42.3**	**38.5**	**37.1**	**35.2**	**32.9**	**31.5**	**30.6**	**30.6**	**28.8**	**28.3**	**27.3**	**26.0**	**25.5**	**5.2**	-
	C$M	823.7	-	-	**47.8**	**60.2**	**54.8**	**54.7**	**61.2**	**54.9**	**50.0**	**48.1**	**45.8**	**42.8**	**40.9**	**39.7**	**39.7**	**37.4**	**36.7**	**35.5**	**33.7**	**33.1**	**6.7**	-
Zn Concentrate Transport Cost	US$/wmt conc	138.50	138.50	138.50	138.50	138.50	138.50	138.50	138.50	138.50	138.50	138.50	138.50	138.50	138.50	138.50	138.50	138.50	138.50	138.50	138.50	138.50	138.50	138.50
	US$M	502.5	-	-	29.2	36.7	33.4	33.4	37.3	33.5	30.5	29.4	27.9	26.1	25.0	24.2	24.2	22.8	22.4	21.7	20.6	20.2	4.1	-
	C$M	652.7	-	-	37.9	47.7	43.4	43.3	48.5	43.5	39.6	38.1	36.3	33.9	32.4	31.5	31.4	29.6	29.1	28.1	26.7	26.2	5.3	-
Penalties	US$/dmt conc	0.00	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	US$M	0.0	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	C$M	0.0	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Zn Price Participation	US$/dmt conc	0.00	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	US$M	0.0	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	C$M	0.0	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
NSR Zn in Zn Concentrate	US$M	2,817.0	-	-	163.4	205.8	187.5	187.1	209.1	187.7	171.1	164.6	156.5	146.3	140.0	135.8	135.7	127.8	125.5	121.5	115.4	113.2	23.0	-
	C$M	3,658.4	-	-	212.3	267.3	243.5	243.0	271.6	243.7	222.2	213.8	203.2	190.0	181.8	176.4	176.2	166.0	163.0	157.8	149.9	147.0	29.9	-
NSR Ag in Zn Concentrate	US$M	0.0	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	C$M	0.0	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Total Zn Concentrate NSR	**US$M**	2,817.0	-	-	**163.4**	**205.8**	**187.5**	**187.1**	**209.1**	**187.7**	**171.1**	**164.6**	**156.5**	**146.3**	**140.0**	**135.8**	**135.7**	**127.8**	**125.5**	**121.5**	**115.4**	**113.2**	**23.0**	-
	C$M	3,658.4	-	-	**212.3**	**267.3**	**243.5**	**243.0**	**271.6**	**243.7**	**222.2**	**213.8**	**203.2**	**190.0**	**181.8**	**176.4**	**176.2**	**166.0**	**163.0**	**157.8**	**149.9**	**147.0**	**29.9**	-
NSR Royalty	US$M	0.0	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	C$M	0.0	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Total NSR	**US$M**	3,049.0	-	-	177.9	223.4	203.3	202.8	227.5	203.5	185.1	177.8	169.6	158.6	151.4	146.4	147.0	137.5	135.1	131.1	124.2	121.9	24.9	-
	C$M	3,959.7	-	-	231.0	290.1	264.0	263.4	295.4	264.3	240.4	230.9	220.3	205.9	196.6	190.1	190.9	178.5	175.5	170.3	161.3	158.3	32.3	-
	C$/tonne	201.28	-	-	281.30	264.91	241.07	240.57	269.78	241.34	219.58	210.88	201.18	188.08	179.58	173.60	174.38	163.06	160.25	155.49	147.29	144.60	136.89	-
OPEX																								
Mining	C$M	984.7	-	-	47.4	56.6	57.5	54.8	54.4	54.6	56.9	56.4	54.5	54.8	53.4	55.8	56.0	54.2	53.7	52.8	52.8	50.3	7.7	-
	C$/tonne mined	38.13	-	-	42.84	39.30	39.91	38.03	37.79	37.95	39.54	39.14	37.88	38.07	37.10	38.76	38.90	37.62	37.29	36.66	36.67	34.92	32.06	-
Processing	C$M	651.7	-	-	27.2	36.3	36.3	36.3	36.3	36.3	36.3	36.3	36.3	36.3	36.3	36.3	36.3	36.3	36.3	36.3	36.3	36.3	7.8	-
	C$/tonne milled	33.13	-	-	33.13	33.13	33.13	33.13	33.13	33.13	33.13	33.13	33.13	33.13	33.13	33.13	33.13	33.13	33.13	33.13	33.13	33.13	33.13	-
Tailings & DMS Rejects	C$M	56.5	-	-	2.4	3.1	3.1	3.1	3.1	3.1	3.1	3.1	3.1	3.1	3.1	3.1	3.1	3.1	3.1	3.1	3.1	3.1	0.7	-
	C$/tonne milled	2.87	-	-	2.87	2.87	2.87	2.87	2.87	2.87	2.87	2.87	2.87	2.87	2.87	2.87	2.87	2.87	2.87	2.87	2.87	2.87	2.87	-
G&A	C$M	321.3			17.6	18.0	18.1	18.2	18.1	18.1	17.7	17.8	17.6	17.1	17.1	17.0	17.0	17.0	17.0	16.8	16.8	16.3	8.0	-
	C$/tonne milled	16.33	-	-	21.47	16.43	16.49	16.64	16.49	16.49	16.15	16.22	16.07	15.66	15.66	15.48	15.55	15.52	15.52	15.32	15.39	14.92	34.03	-
Total OPEX	**C$M**	**2,014.1**	-	-	**94.6**	**114.0**	**114.9**	**112.4**	**111.9**	**112.1**	**114.0**	**113.5**	**111.6**	**111.4**	**110.0**	**112.2**	**112.5**	**110.6**	**110.1**	**109.0**	**109.1**	**106.0**	**24.3**	-
	C$/tonne milled	**102.38**	-	-	**115.14**	**104.11**	**104.97**	**102.64**	**102.18**	**102.40**	**104.15**	**103.68**	**101.87**	**101.73**	**100.45**	**102.45**	**102.70**	**100.99**	**100.55**	**99.52**	**99.60**	**96.84**	**102.83**	-
Net Operating Income	**C$M**	1,945.6	-	-	136.5	176.1	149.0	151.0	183.5	152.1	126.4	117.4	108.7	94.6	86.6	77.9	78.5	68.0	65.4	61.3	52.2	52.3	8.0	-
	C$/tonne	98.90	-	-	166.16	160.80	136.10	137.93	167.60	138.95	115.43	107.19	99.31	86.36	79.13	71.15	71.68	62.07	59.71	55.96	47.69	47.76	34.06	-
CAPEX																								
Mining	C$M	318.2	-	58.2	31.2	23.2	21.1	28.3	21.7	22.4	14.5	16.8	15.5	10.3	9.2	9.3	7.1	7.2	9.4	6.6	3.7	2.3	0.2	-
Site Development	C$M	8.2	2.3	5.3	0.1	-	0.1	-	0.1	-	0.1	-	0.1	-	0.1	-	0.1	-	0.1	-	0.1	-	-	-
Mineral Processing	C$M	90.6	13.5	65.4	-	0.8	0.4	1.1	0.8	0.4	1.1	0.8	0.4	1.1	0.8	0.4	1.1	0.8	0.4	1.1	-	-	-	-
Tailings Management	C$M	13.3	1.0	4.0	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	-	-
On-Site Infrastructure	C$M	61.4	15.0	40.1	4.1	0.1	0.3	0.1	0.0	0.1	0.3	0.1	0.0	0.1	0.3	0.1	-	0.1	0.3	-	0.1	-	-	-
Off-Site Infrastructure	C$M	1.2	0.5	0.5	-	0.0	-	0.0	-	0.0	-	0.0	-	0.0	-	0.0	-	0.0	-	0.0	-	-	-	-
Project Indirects	C$M	33.2	2.8	25.2	5.1	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Engineering & Project Management	C$M	18.8	5.6	11.8	1.5	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Owner Costs	C$M	5.6	0.3	5.3	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Closure	C$M	8.9	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	8.9
Subtotal	C$M	559.4	40.9	215.8	42.5	24.6	22.3	30.0	23.1	23.4	16.4	18.2	16.5	12.0	10.8	10.3	8.7	8.7	10.6	8.2	4.4	2.8	0.2	8.9
Contingency	C$M	58.5	4.6	41.1	3.5	1.4	1.1	1.1	0.6	0.5	0.4	0.4	0.3	0.4	0.3	0.4	0.4	0.4	0.3	0.4	0.2	0.1	0.0	0.8
Total CAPEX	**C$M**	**617.9**	**45.5**	**256.9**	**46.0**	**26.1**	**23.4**	**31.1**	**23.7**	**23.9**	**16.8**	**18.6**	**16.8**	**12.5**	**11.2**	**10.7**	**9.1**	**9.0**	**10.9**	**8.6**	**4.5**	**2.9**	**0.2**	**9.7**
	C$/tonne	**31.41**																						
Pre-Production	**C$M**	**302.3**	**45.5**	**256.9**																				
Sustaining	**C$M**	**315.6**			**46.0**	**26.1**	**23.4**	**31.1**	**23.7**	**23.9**	**16.8**	**18.6**	**16.8**	**12.5**	**11.2**	**10.7**	**9.1**	**9.0**	**10.9**	**8.6**	**4.5**	**2.9**	**0.2**	**9.7**
Working Capital	C$M	0.0		15.8																			(15.8)	

ZincX Resources Corp.
Akie Project, British Columbia, Canada
PEA Economic Model

LEGEND
link
input cell
calc

			Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9
Net Pre-Tax Cash Flow	US$M	1,022.3	-35.0	-209.9	69.7	115.5	96.7	92.3	123.0	98.7	84.4	76.1	70.8
	C$M	1,327.7	-45.5	-272.6	90.5	150.0	125.6	119.9	159.8	128.2	109.6	98.8	92.0
Cumulative Net Pre-Tax Cash Flow	C$M		-45.5	-318.1	-227.6	-77.6	48.0	167.9	327.7	455.9	565.5	664.3	756.3
Taxes	C$M	458.0	-	-	2.7	30.2	26.3	28.8	43.3	42.4	35.5	33.1	31.0
Net Post-Tax Cash Flow	US$M	669.6	-35.0	-209.9	67.6	92.2	76.5	70.1	89.7	66.0	57.1	50.6	46.9
	C$M	869.6	-45.5	-272.6	87.8	119.7	99.3	91.1	116.5	85.8	74.1	65.7	61.0
Cumulative Net Post-Tax Cash Flow	C$M		-45.5	-318.1	-230.3	-110.6	-11.3	79.8	196.3	282.0	356.2	421.9	482.9

		Y10	Y11	Y12	Y13	Y14	Y15	Y16	Y17	Y18	Y19	Y20
Net Pre-Tax Cash Flow	US$M	63.2	58.1	51.7	53.4	45.4	42.0	40.6	36.7	38.0	18.2	-7.5
	C$M	82.1	75.5	67.2	69.4	58.9	54.5	52.7	47.7	49.4	23.6	-9.7
Cumulative Net Pre-Tax Cash Flow	C$M	838.4	913.9	981.1	1,050.5	1,109.4	1,163.9	1,216.6	1,264.3	1,313.7	1,337.4	1,327.7
Taxes	C$M	27.4	25.3	23.0	23.9	20.4	19.5	18.5	9.0	18.3	2.5	(3.2)
Net Post-Tax Cash Flow	US$M	42.1	38.6	34.0	35.0	29.6	27.0	26.4	29.8	24.0	16.3	-5.0
	C$M	54.7	50.1	44.2	45.5	38.5	35.0	34.2	38.7	31.1	21.2	-6.5
Cumulative Net Post-Tax Cash Flow	C$M	537.6	587.7	631.9	677.4	715.9	750.9	785.2	823.8	855.0	876.1	869.6

ECONOMIC INDICATORS		
Pre-Tax Results		
Pre-Tax NPV	US$M	499.4
	C$M	648.5
Pre-Tax NPV	US$M	1,022.3
	C$M	1,327.7
IRR	%	35.0%
Payback	Years	2.6
Post-Tax Results		
Post-Tax NPV	US$M	308.5
	C$M	400.6
Post-Tax NPV	US$M	669.6
	C$M	869.6
IRR	%	27.0%
Payback	Years	3.2

Source: JDS (2018)

24 Adjacent Properties

There are two properties adjacent to the Akie property: Fluke and Elf. Both properties are considered advanced prospects hosting known stratiform sphalerite, galena, pyrite and barite mineralization and interpreted to be situated at the identical stratigraphic horizon as the Cardiac Creek deposit. They are currently controlled by the Cirque Operating Corp. (100%), a joint venture between Teck Resources Ltd. (50%) and Korea Zinc Company Ltd. (50%). The following section is an unabridged excerpt from a previous technical report entitled "*Geology, Diamond Drilling and Preliminary Resource Estimation, Akie Zinc-Lead-Silver Property, Northeast British Columbia, Canada*" by Donald G. MacIntyre and Robert C. Sim (2008) which covers the historical work on the Elf and Fluke properties. Exploration activities conducted by Teck Resources in 2013 and 2014 have also been summarized from recent assessment reports. Sample data collected on the Fluke and Elf properties has not been used in the estimate of mineral resources for the Cardiac Creek Deposit.

24.1 Fluke Property

The Fluke property covers a northwest-trending synclinal keel of Gunsteel strata that is bounded by Silurian Siltstone to the southwest and middle Devonian limestone to the northeast (Roberts, 1978). The Silurian rocks have been thrust northeastward over the Gunsteel syncline. In 1978, the property was staked by Cyprus Anvil Mining Corporation (Cyprus Anvil) to cover a small showing of laminar-banded pyrite with galena-sphalerite-rich bands that are exposed in a small northeast flowing tributary of the Akie River. Several nodular barite beds also crop out on the property. At surface, the mineralized interval is about 1 m thick and dips to the west. The host rocks are intensely deformed, carbonaceous cherty argillite and siliceous shale of the late Devonian Gunsteel Formation. Assays as high as 15% Zn+Pb and 35 g/t Ag have been reported. Cyprus Anvil drilled the property in 1980, 1981 and 1982. Only one drill hole intersected sulphide mineralization at approximately 200 m down-dip from the surface showing (Paradis et al., 1998). Recently, Teck Resources conducted a couple of limited exploration programs on the Fluke property. In 2013, a small soil sampling program was conducted over the known Fluke and Pook showings to determine the preferred soil horizon for future soil geochemistry surveys. A total of 96 samples were collected (Rasmussen and Thiessen, 2013). In 2014, Teck Resources contracted Geotech Ltd. to conduct an airborne VTEM geophysical survey over the Fluke property. A total of 83.3 line km were flown along 200 m spaced flight lines oriented at an azimuth of 50°. The results of this survey produced a number of linear northwest-southeast EM conductors that generally agreed with the known geology (Loughrey, 2015a and 2015b). No new drilling took place as part of this recent exploration work.

24.2 Elf Property

In 1978, the Elf property was staked by Cyprus Anvil to cover an area of moderately anomalous stream sediment geochemistry and the occurrence of a boulder of white barite containing high-grade galena and sphalerite in Elf Creek (Roberts, 1979). Subsequent soil sampling resulted in the discovery of an outcrop of bedded barite with high-grade bands of galena and sphalerite on the heavily timbered south-facing slope north of Elf Creek. The mineralized zone has been exposed on surface by trenching and is up to 4 m thick. A sulphide-rich sample from this zone assayed 14.1% Zn, 25% Pb and 106 g/t Ag (MacIntyre, 1998). Host rocks are carbonaceous cherty argillite and siliceous shale of the Gunsteel Formation. In 1979 and 1980, the property was drill tested. Drill holes intersected laminar-banded pyrite at depth; barite-sulphide mineralization similar to the surface showing was not intersected. The best drill intersection contained 13.8% Zn+Pb with 27 g/t Ag over 11 m (Paradis et al., 1998). Drilling and surface mapping suggest the Elf mineralization is contained within a steeply dipping, overturned fold limb that is over thrust to the west by Silurian dolomitic siltstone. Intense folding and structural imbrication of the Gunsteel host rocks has made defining the geometry of the mineralized interval difficult. In 1995, exploration on the Elf property resulted in the discovery of two additional mineralized showings referred to as the Joel Creek and Ian Creek showings consisting of laminated to disseminated pyrite with nodular to disseminated barite (Henry et al., 2014). In 2013 and 2014, Teck Resources conducted two limited exploration programs on the Elf property similar in nature to those on the Fluke property. In 2013, a small soil sampling program was conducted over the known Elf showing and surrounding area to determine the preferred soil horizon for future soil geochemistry surveys. A total of 649 samples were collected. New lead anomalies were outlined southeast of the Elf showing (Henry et al., 2014). Henry et al. (2014) also references earlier sampling taken on the Elf showing that returned 0.22% Zn, 10.46% Pb, and 22.58 g/t Ag over 4 m. In 2014, Teck Resources contracted Geotech Ltd. to conduct an airborne VTEM geophysical survey over the Elf property. A total of 228 line km were flown along 200 m spaced flight lines oriented at an azimuth of 50°. The results of this survey produced a number of linear northwest-southeast EM conductors that generally agreed with the known geology (Loughrey, 2015c and 2015d).

25 Other Relevant Data and Information

There are no additional relevant data, information or explanation necessary to make this report understandable and not misleading.

26 Interpretations and Conclusions

It is the conclusion of the QPs that the PEA summarized in this technical report contains adequate detail and information to support the positive economic outcome shown for the project. Standard industry practices, equipment and design methods were used in the PEA.

The Akie Project contains a substantial zinc, lead and silver resource that can be mined by underground methods and recovered with DMS and conventional flotation processing.

Based on the assumptions used for this preliminary evaluation, the project is considered to be economic and should proceed to the pre-feasibility (PFS) stage.

There is a likelihood of improving the project economics by identifying additional mineral resources within the development area that may justify increased mine production or extend the mine life.

To date, the QPs are not aware of any fatal flaws for the Project.

26.1 Risks

As with most mining Projects, there are many risks that could affect the economic viability of the Project. Many of these risks are based on lack of detailed knowledge and can be managed as more sampling, testing, design, and detailed engineering are conducted. Table 26-1 identifies what are currently deemed to be the most significant internal Project risks, potential impacts, and possible mitigation approaches.

The most significant potential risks associated with the project are uncontrolled dilution, uncontrolled groundwater inflow in the mines, lower metal recoveries than those projected, operating and capital cost escalation, permitting and environmental compliance, unforeseen schedule delays, changes in regulatory requirements, ability to raise financing and metal price. These risks are common to most mining projects, many of which can be mitigated with adequate engineering, planning and pro-active management.

External risks are, to a certain extent, beyond the control of the project proponents and are much more difficult to anticipate and mitigate, although, in many instances, some risk reduction can be achieved. External risks are things such as the political situation in the project region, metal prices, exchange rates and government legislation. These external risks are generally applicable to all mining projects. Negative variance to these items from the assumptions made in the economic model would reduce the profitability of the mine and the mineral resource and reserve estimates.

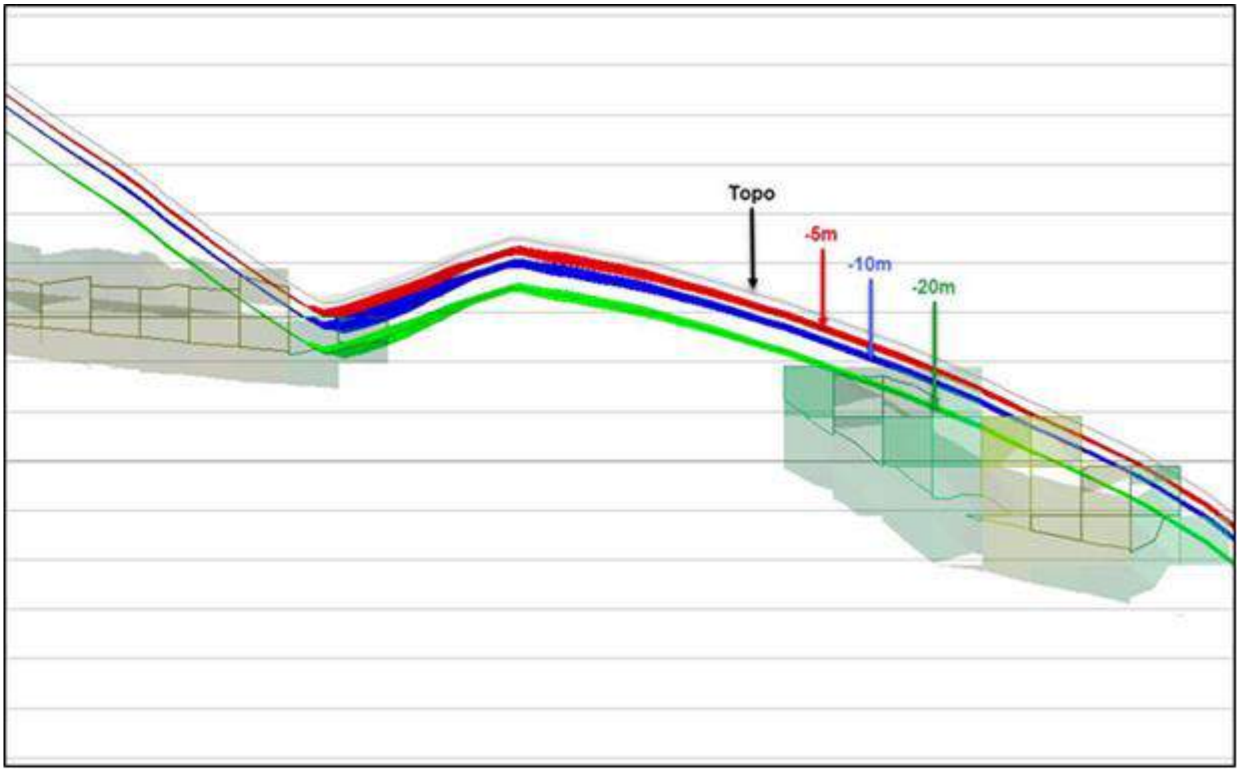
26.1.1 Crown Pillar

The mine design is optimistic with regard to extraction at the top of the orebody. As some stopes are located within 6 m of surface, a crown pillar may be necessary to prevent surface subsidence.

Figure 26-1 shows a cross section of the near-surface stopes. They vary from 5 m to 15 m in thickness, averaging 9.1 m. A crown pillar of 20 m to 30 m should be adequate for such stope widths, subject to the ground quality of the stope backs.

There is approximately 150 kt of inferred resource at an average grade of 6.37 % ZnEqv within 20 m of surface, and 230 kt of inferred resources at an average grade of 6.45% ZnEqv within 30m of surface. As this grade is marginally above the cut-off grade, very little positive cash flow is generated in the economic model from these tonnes. Their impact on the NPV and IRR is further reduced by their inclusion in the final two years of the production forecast.

Figure 26-1: Cross Section of Near-Surface Stopes



Source: JDS (2018)

Table 26-1: Main Project Risks

Risk	Explanation / Potential Impact	Possible Risk Mitigation
Dilution	Higher than expected dilution can have a severe impact on project economics. The mine must ensure accurate drilling and blasting practices are implemented to minimize dilution from wall rock, backfill and other low-grade mineralized zones.	A well planned and executed grade control plan is necessary immediately upon commencement of mining.
Water Inflow	The management of water on-site is a critical component of the project design. Basic assumptions were made for surface and underground water flows based on preliminary drilling and hydro-geologic information.	Continued collection and analysis of data relating to underground, and surface water needs to be continued on-site over the near-term to enhance the local hydrological knowledge.
Metallurgical Recoveries	While it is believed that the various programs of sampling and metallurgical test work conducted to date are appropriate to support a PEA, factors other than process conditions, such as dilution, plant ramp-up that could lead to reduced metal recovery and / or increased processing OPEX costs. If LOM, metal recoveries is lower, or costs higher, than estimated, the Project economics would be negatively impacted.	Additional sampling and test work should be conducted in the next project phase. Early process team recruitment and training, implementation of good quality instrumentation and process control.
CAPEX and OPEX	The ability to achieve the estimated CAPEX and OPEX costs are important elements of Project success. If OPEX increases then the mining cut-off grade would increase and, all else being equal, the size of the optimized pit would reduce yielding fewer mineable tonnes.	Active investigation of potential cost-reduction measures would assist in the support of reasonable cost estimates.
Timely Approval of Project Authorizations	The ability to secure all of the permits to build and operate the project is of paramount importance. Failure to secure the necessary permits could stop or delay the project.	The development of close relationships with the local communities and government along with a thorough Environmental and Social Impact Assessment and a project design that gives appropriate consideration to the environment and local people is required. Maintain direct control with a clear solution.
Development Schedule	The Project development could be delayed for a number of reasons and could impact Project economics. A change in schedule would alter the Project economics.	Select EPCM firm and develop detailed construction schedule
Acid Rock Drainage	Acid Rock Drainage at the Project site could pose problems during permitting due to its adverse environmental effects.	Continue with rigorous monitoring program and highlight the fact that there are naturally acidic waters in un-mined areas in the valley during the permitting process.
Materials Balance	The TMF embankment and many pads, roads, and foundations are constructed with mined material (overburden and mine rock), that could be potentially acid generating (PAG) and the production of mine rock according to the mine plan may not be sufficient to provide the capacity needed for all uses.	Early production/excavation of mine rock (non-mineralized) from the pit to assure an adequate supply of construction material
Smelter Location	The assumed smelter location of Trail may not have capacity to accept concentrates from the project. Overseas smelting may increase concentrate shipping costs.	Early negotiations to secure Trail smelter capacity for the project’s concentrates.
Availability of Experienced and Skilled Operating and Maintenance Personnel	Providing employment opportunities to the local and Indigenous communities is an objective of the Company. However, during the key early operating years there may be a need to acquire skilled and seasoned employees outside of the regional area.	Use of sophisticated screening techniques to ensure those recruited have the necessary attitude and aptitude to succeed and provide a comprehensive training program for those new to the industry.

Source: JDS (2018)

26.2 Opportunities

There are significant opportunities that could improve the economics, timing, and/or permitting potential of the project. The major opportunities that have been identified at this time are summarized in Table 26-2, excluding those typical to all mining projects, such as changes in metal prices, exchange rates, etc. Further information and assessments are needed before these opportunities should be included in the project economics.

Table 26-2: Main Project Opportunities

Opportunity	Explanation	Potential Benefit
Expansion of Mineral Resources	The mineral resource has not been fully delineated and there is an opportunity to expand the mineable resource as well as discover new mineralized zones.	Increased mine life.
Project Strategy and Optimization	With additional detailed planning and a series of strategic option reviews the Project may be able to add value.	Planning and executing the Project with the optimum mine design/schedule and processing systems would result in the maximum possible value to shareholders and other economic stakeholders.
Potential to Purchase Good Used Equipment	There is considerable used equipment on the market that could be utilized.	Capital cost reduction

Source: JDS (2018)

27 Recommendations

27.1 Recommended Work Programs

27.1.1 Metallurgy and Processing

Trade off study to determine if DMS is economical.

Recommendations for additional metallurgical test work are listed below:

- Phase I – three global composites representing the proposed mine plan for Year 0 to 1, 1 to 3 and 3 to end of mine, be prepared and used for flowsheet optimization. The composites will be subjected to mineralogical analysis, comminution test work including fine grinding specific energy requirements, DMS, flotation tests including locked cycle tests and settling and filtering assessments.
- Phase II – 25 composites representing discrete continuous intervals of mineralization to be used to assess variability in the deposit. The samples will undergo mineralogical analysis as well as comminution test work. The optimized flowsheet and parameters established in the Phase I program will be used as the basis for the flotation test work in Phase II to establish metallurgical performance.

27.1.2 Mining and Geotechnical Studies

Recommendations for the next phase of geotechnical work for the Project are summarized below:

- Complete geotechnical characterization program for underground mine and infrastructure including geotechnical core drilling and oriented core and/or televiewer;
- Complete laboratory strength testing program on core samples obtained from mine geotechnical core drilling;
- Carryout geotechnical mapping on relevant surface outcrops in the mine area;
- Prepare a trade-off study for conveyor transport to Portal One;
- Paste and cemented backfill strength and characteristic testing to support PFS level backfill plant and distribution system design; and
- Trade-off studies for diesel vs electric/battery equipment, longhole drill hole size and haulage options.

27.1.3 Infrastructure

Recommendations for the next phase of infrastructure work on the project are as follows:

- Water balance, geochemistry, and basic water treatment plant design, with supply and installation costs;
- Geotechnical investigation of crusher, plant, and site infrastructure locations;

- Investigation of all inputs for power, level of refinement for PFS, and PFS level of engineering for power plant supply and installation, with study on LNG supply, storage, transportation and powerplant (design / supply / installation costs), with supplier proposals;
- Concentrate shipping and handling, with study and site visits for rail handling facilities, port handling facilities, ocean shipping, and ore handling trucks, including study to determine backhaul to mine with operational suppliers in ore trucks;
- Road study between Mackenzie and site, including determining snow clearing costs, and any portions of road maintenance and snow clearing for forest service roads;
- Investigate camp and site buildings costs using PFS level proposals from suppliers for supply and install; and
- Personnel transport investigation, including trade-off study of developing airstrip to handle larger passenger craft, 40 person, i.e. Dash 8 – 100 vs 19 seat Beechcraft.

27.1.4 Geochemistry and Environment

Recommendations for future geochemistry and environment studies include:

Meteorology and Climate

- Update the meteorology monitoring program for project area; and
- Conduct a winter snow survey for a complete season.

Air Quality

- Develop and implement Air Quality monitoring programs for the project area.

Wildlife

- Conduct field surveys to validate the existing records for species and populations within and adjacent to the project area.

Hydrology

- Expansion of hydrometric network to include project affected hydrologic subzones for hydrology and water quality.

Aquatic Sediment and Aquatic Life

- Expanded fisheries, benthic invertebrates and periphyton studies.

Hydrogeology

- Expansion of the groundwater monitoring network to update the hydrogeology model.

Geochemistry

- Expansion of field bin program; and
- Expand kinetic testing program based on the expected waste materials and storage conditions.

27.1.5 Waste and Water Management

Recommendations for the next phase of engineering for the Project are summarized below:

- Complete a detailed Best Available Technology (BAT) assessment for waste and water management in future studies. The assessment will confirm the preferred location, tailings management technology and water management strategy;
- Complete site investigation programs at the TMF and Process Plant Site to support future designs and to comply with regulatory requirements;
- Complete testing on embankment construction materials to confirm material parameters;
- Complete testing on DMS reject and tailings materials to confirm suitability for proposed management strategy, and estimate material parameters for stability modelling and confirm design assumptions (dry density, specific gravity, etc.);
- Complete seepage and stability analyses for TMF and WMP to confirm designs comply with regulatory requirements for static and seismic stability;
- Develop a full closure plan for the waste and water management facilities based on the final design configuration;
- Optimize the water balance to incorporate updated runoff and process flow estimates;
- Conduct sensitivity analysis on the water balance to consider the effect of wet and dry cycles on the annual water balance surplus; and
- Revise the Mine Waste Disposal Alternatives Assessment to comply with provincial and federal guidance as more information becomes available.

27.2 Costs

It is estimated that a pre-feasibility study and supporting field work would cost approximately $30.4 million. A breakdown of the key components of the next study phase is as follows in Table 27-1.

Table 27-1: Estimated Costs to Advance project to Pre-feasibility Stage

Component	Estimated Cost ($C M)	Comment
Resource Drilling	5.0	Conversion of indicated to measured resources. Drilling will include holes combined for resource, geotech and hydrogeology purposes.
Metallurgical Testing	0.6	Comminution, DMS, flotation optimization, variability testing, tailings dewatering, concentrate filtration, mineralogy, minor element analysis.
Underground Development	20.0	Access for underground drilling and possible bulk sample.
Geochemistry	0.5	Acid Base Accounting (ABA) tests and humidity cell testing to determine acid generating potential of rock and tailings.
Waste & Water Site Investigation	0.8	Site investigation drilling, sampling and lab testing.
Geotechnical, Hydrology & Hydrogeology	1.0	Drilling, sampling, logging, test pitting, lab tests, etc.
Engineering	1.5	PFS-level mine, infrastructure and process design, cost estimation, scheduling & economic analysis.
Environment	1.0	Baseline investigations including, water quality, fisheries, wildlife, weather, traditional land use & archaeology.
Total	**30.4**	Excludes corporate overheads and future permitting activities.

Source: JDS (2018)

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29 Units of Measure, Abbreviations and Acronyms

Symbol / Abbreviation	Description
'	minute (plane angle)
"	second (plane angle) or inches
°	degree
°C	degrees Celsius
3D	three-dimensions
A	ampere
a	annum (year)
ac	acre
Acfm	actual cubic feet per minute
ALT	active layer thickness
ALT	active layer thickness
amsl	above mean sea level
AN	ammonium nitrate
ARD	acid rock drainage
Au	gold
AWR	all-weather road
B	billion
BD	bulk density
Bt	billion tonnes
BTU	British thermal unit
BV/h	bed volumes per hour
bya	billion years ago
C$	dollar (Canadian)
Ca	calcium
cfm	cubic feet per minute
CHP	combined heat and power plant
CIM	Canadian institute of mining and metallurgy
cm	centimetre
cm^2	square centimetre
cm^3	cubic centimetre
cP	centipoise
Cr	chromium
Cu	copper
d	day
d/a	days per year (annum)
d/wk	days per week

Symbol / Abbreviation	Description
dB	decibel
dBa	decibel adjusted
DGPS	differential global positioning system
DMS	dense media separation
dmt	dry metric ton
DWT	dead weight tonnes
EA	environmental assessment
EIS	environmental impact statement
ELC	ecological land classification
ERD	explosives regulatory division
FEL	front-end loader
FOC	fisheries and oceans Canada
ft	foot
ft^2	square foot
ft^3	cubic foot
ft^3/s	cubic feet per second
g	gram
G&A	general and administrative
g/cm^3	grams per cubic metre
g/L	grams per litre
g/t	grams per tonne
Ga	billion years
gal	gallon (us)
GJ	gigajoule
GPa	gigapascal
gpm	gallons per minute (us)
GSC	geological survey of Canada
GTZ	glacial terrain zone
GW	gigawatt
h	hour
h/a	hours per year
h/d	hours per day
h/wk	hours per week
ha	hectare (10,000 m^2)
ha	hectare
HG	high grade
HLEM	horizontal loop electro-magnetic
hp	horsepower
HPGR	high-pressure grinding rolls

Symbol / Abbreviation	Description
HQ	drill core diameter of 63.5 mm
Hz	hertz
ICP-MS	inductively coupled plasma mass spectrometry
in	inch
in^2	square inch
in^3	cubic inch
INAC	Indigenous and Northern Affairs Canada
IRR	internal rate of return
JDS	JDS Energy & Mining Inc.
K	hydraulic conductivity
k	kilo (thousand)
kg	kilogram
kg	kilogram
kg/h	kilograms per hour
kg/m^2	kilograms per square metre
kg/m^3	kilograms per cubic metre
KIM	kimberlitic indicator mineral
km	kilometre
km/h	kilometres per hour
km^2	square kilometre
kPa	kilopascal
kt	kilotonne
kV	kilovolt
kVA	kilovolt-ampere
kW	kilowatt
kWh	kilowatt hour
kWh/a	kilowatt hours per year
kWh/t	kilowatt hours per tonne
L	litre
L	mine level
L/min	litres per minute
L/s	litres per second
LDD	large-diameter drill
LG	low grade
LGM	last glacial maximum
LOM	life of mine
m	metre
M	million
m/min	metres per minute

Symbol / Abbreviation	Description
m/s	metres per second
m^2	square metre
m^3	cubic metre
m^3/h	cubic metres per hour
m^3/s	cubic metres per second
Ma	million years
MAAT	mean annual air temperature
MAE	mean annual evaporation
MAGT	mean annual ground temperature
mamsl	metres above mean sea level
MAP	mean annual precipitation
masl	metres above mean sea level
Mb/s	megabytes per second
mbgs	metres below ground surface
Mbm^3	million bank cubic metres
Mbm^3/a	million bank cubic metres per annum
mbs	metres below surface
mbsl	metres below sea level
mg	milligram
mg/L	milligrams per litre
min	minute (time)
mL	millilitre
mm	millimetre
Mm^3	million cubic metres
MMER	metal mining effluent regulations
MMSIM	metamorphosed massive sulphide indicator minerals
mo	month
MPa	megapascal
Mt	million metric tonnes
MVA	megavolt-ampere
MW	megawatt
NAD	North American datum
NG	normal grade
Ni	nickel
NI 43-101	national instrument 43-101
Nm^3/h	normal cubic metres per hour
NPAG	Non-potentially acid-generating
NQ	drill core diameter of 47.6 mm
NRC	natural resources Canada

Symbol / Abbreviation	Description
OP	open pit
OSA	overall slope angles
oz	troy ounce
P. Eng.	Professional engineer
P.Geo.	professional geoscientist
Pa	Pascal
PAG	potentially acid generating
Pb	lead
PEA	preliminary economic assessment
PFS	preliminary feasibility study
PGE	platinum group elements
PMF	probable maximum flood
ppb	parts per billion
ppm	parts per million
psi	pounds per square inch
QA/QC	quality assurance/quality control
QP	qualified person
RC	reverse circulation
RMR	rock mass rating
ROM	run of mine
rpm	revolutions per minute
RQD	rock quality designation
s	second (time)
S.G.	specific gravity
Scfm	standard cubic feet per minute
SEDEX	sedimentary exhalative
SFD	size frequency distribution
SFD	size frequency distribution
SG	specific gravity
t	tonne (1,000 kg) (metric ton)
t	metric tonne
t/a	tonnes per year
t/d	tonnes per day
t/h	tonnes per hour
TCR	total core recovery
TFFE	target for further exploration
TMF	tailings management facility
tph	tonnes per hour
ts/hm^3	tonnes seconds per hour metre cubed

Symbol / Abbreviation	Description
US	united states
US$	dollar (American)
UTM	universal transverse mercator
V	volt
VEC	valued ecosystem components
VMS	volcanic massive sulphide
VSEC	valued socio-economic components
w/w	weight/weight
wk	week
wmt	wet metric ton
WMP	water management pond
WRSF	waste rock storage facility
µm	microns
µm	micrometre
Zn	zinc

Scientific Notation	Number Equivalent
1.0E+00	1
1.0E+01	10
1.0E+02	100
1.0E+03	1,000
1.0E+04	10,000
1.0E+05	100,000
1.0E+06	1,000,000
1.0E+07	10,000,000
1.0E+09	1,000,000,000
1.0E+10	10,000,000,000