



PRELIMINARY ECONOMIC ASSESSMENT

PC SILVER MINE NORTHWEST TERRITORIES

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NOTICE

JDS Energy & Mining, Inc. prepared this National Instrument 43-101 Technical Report, in accordance with Form 43-101F1, for Honey Badger Silver Inc. 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.

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1 EXECUTIVE SUMMARY

1.1 Introduction

JDS Energy & Mining Inc. (JDS) was commissioned by Honey Badger Silver Inc. (Honey Badger Silver or the Company) to complete a Preliminary Economic Assessment (PEA) of the Prairie Creek Silver Mine (PC Silver Mine, or the Project), a silver-zinc-lead-copper resource development project owned by Honey Badger Silver located in the Mackenzie Mountains of the Northwest Territories. The PEA was prepared following the guidelines of the Canadian Securities Administrators' National Instrument 43-101 and Form 43-101F1.

1.2 Project Description

The Property is located in the Northwest Territories, Canada, near the Yukon border, in the Mackenzie Mountain Range that has an average physical relief of approximately 300 m, consisting of low mountains with moderate to steep sides and intervening narrow valleys. The Project site is located at an elevation of 870 m above mean sea level. Valleys are well-incised, and the area is located within the alpine forest-tundra section of the boreal forest.

Four styles of base metal mineralization have been identified on the Project: quartz vein, stratabound, stockwork and Mississippi Valley-type. Only the first three styles have been found in potentially economic quantities to date. Base metal mineral showings occur along the entire 16 km north to south length of the anticline covered by the main group of mining leases.

The PEA contemplates a restart of the underground PC Silver Mine at an average rate of approximately 2,400 tonnes per day (t/d), followed by dense media separation upgrading, and milling at approximately 1,500 t/d post-dense media separation over a 22-year mine life.

The site includes an existing mill, airstrip, maintenance shop, office complex and approximately 5 km of underground workings.

The proposed project infrastructure includes an on-site power plant and bulk fuel storage, water management and treatment facilities, and an accommodation camp.

1.3 Location, Access and Ownership

The PC Silver Mine (historically known as Prairie Creek) is located in the Mackenzie Mountains of the Northwest Territories at approximately 61°33'N, 124°48'W, roughly 90 km from Nahanni Butte. The Project site occupies an enclave that is surrounded by, but excluded from, Nahanni National Park Reserve. Tenure comprises mining and surface leases totaling approximately 7,485 hectares.

The nearest communities and approximate distances include Nahanni Butte, 90 km southeast; Fort Liard, 170 km south; and Fort Simpson, 185 km to the east. Yellowknife, the capital and administrative centre of the Northwest Territories, is approximately 500 km to the east. The town

of Fort St. John, British Columbia (BC), 650 km to the south of the Project, is the primary air charter point for incoming cargo and out-of-territory workers.

Access to the site is provided by charter aircraft, generally from Fort St. John, BC. or Fort Simpson, Northwest Territories, both of which are serviced by scheduled commercial airlines. A 1,000 m gravel airstrip is located on the flood plain of Prairie Creek, approximately 1 km north of the Project site.

The Liard highway, which connects Fort Nelson, BC to Fort Simpson, Northwest Territories, is the closest major transportation route to the Property.

The Company has Impact and Benefit Agreements with Nahᓂᓐ Dehé Dene Band and Łı́ıdlı́ Kúé First Nation, and a Benefit Agreement with Acho Dene Koe First Nation.

There is a 1.2% net smelter return (NSR) royalty payable to Versamet on the property and a 1% NSR royalty to Resource Capital Funds (RCF) affiliate RCF VI CAD LLC.

1.4 History, Exploration and Drilling

1.4.1 History

The original discovery of mineralization on the Project was made by a local trapper in 1928. The first claims were staked in 1958, and intermittent exploration was carried out until the Property was acquired by Cadillac Explorations Limited (Cadillac Explorations) in 1966. Between 1966 and 1969, Cadillac Explorations carried out trenching on several mineralized zones and underground exploration was commenced as a follow-up to trench results.

In 1970, the property was optioned to Penarroya Canada Limited (Penarroya), and the underground development was extended. Approximately 5,800 m of surface drilling and preliminary metallurgical testing were also carried out.

In 1980, Kilborn Engineering Limited completed a Feasibility Study for Cadillac Explorations, the results of which prompted the decision to put the underground mine (then called Cadillac Mine) into production. Between 1980 and 1982, extensive mine development took place, a mill and concentrator were erected, and a new camp was established. The winter road connecting the Project to the newly established Liard highway was also constructed. Two more underground levels and extensive underground workings were subsequently developed. Construction activities continued until May 1983 when Cadillac Explorations went into bankruptcy after approximately C\$64 million (1982 value) had been expended on the Project. Thereafter, site maintenance was taken over by Procan, which acquired Cadillac Explorations' interest in the Project through bankruptcy proceedings in 1984.

In 1991, Nanisivik Mines Limited (Nanisivik) acquired the Project from Procan and Canadian Zinc Corporation (Canadian Zinc) acquired a 60% interest in the Project from Nanisivik. In 2004, Canadian Zinc acquired a 100% interest in the Project.

In December 2022, RCF acquired NorZinc, the parent company of Canadian Zinc, by way of a Plan of Arrangement. In April 2026, Honey Badger Silver acquired 100% of Canadian Zinc Corporation from RCF, thus securing 100% ownership of the property.

Numerous historical resource estimates have been reported for the Main Zone deposits. The Main Zone in this report refers to Zones 1, 2, and 3. Initially these estimates were for the Main Quartz Vein (MQV) only but later incorporated the Stratabound Massive Sulphide (SMS) and Stockwork-style (STK) mineralization, as they were discovered. These historical resource estimates have been superseded by the estimate described in Section 14 of this Technical Report.

There has been no commercial production from the property, despite trial mining having been carried out in 1982.

1.4.2 Exploration

Honey Badger Silver has not done any drilling on the Project. The following is a summary of exploration completed by previous owners.

In the 1960s, the Main Zone became the focus of exploration and is where the majority of exploration has occurred since.

Historically, exploration was focused on what is now termed the MQV, with secondary targets being STK mineralization and SMS mineralization. The known Mississippi Valley Type (MVT) showings occurring in a more remote part of the Project report lower grades than either vein or strata bound mineralization, and as such they have not been the focus of significant exploration to date.

Surface exploration has included pitting and trenching; underground exploration has included channel sampling along drifts and crosscuts as well as sampling of development muck.

Limited geophysical exploration has been conducted on the Project. Canadian Zinc applied several geophysical techniques including UTEM, TDEM and gravity to assist with identifying new exploration targets.

1.4.3 Drilling

Honey Badger Silver has completed no exploration drilling on the property. The following is a summary of work completed by Canadian Zinc. The metres drilled during drill programs completed since 1992 are summarized in Table 1-1.

Table 1-1: Exploration Drill Summary Table from 1992 to Present

Year	No. of Holes	Metres	Highlights
1992	22	6,322	Discovery of previously unknown SMS mineralization by diamond drilling. Discovery hole (PC-92-008) ran 10.60% Zn, 5.29% Pb, 44.37 g/t Ag, over 28.40 m.
1993	31	8,432	Tested for further SMS Mineralization. Extended MQV by intersecting 18 m of vein 170 m below workings.
1994	31	11,113	Extension of Main Zone, more SMS lenses in Zone 5.
1995	36	10,082	-
2001	5	1,711	Diamond drilling program designed both to increase confidence in 1998 resource estimate and to identify new high-grade areas. Possibility of high-grade shoots recognized.
2004	27	5,944	MQV drilling which intersected significant mineralization. Step out on the vein hit narrow but high-grade intersections. SMS exploration outside Main Area.
2006	19	2,393	Phase 1 driving of decline tunnel and U/G drilling commences on MQV. Drilling of Zone 8 mineralization investigated.
2007	53	11,141	Phase 1 U/G program confirms vein grades. Decline extended, Phase 2 drilling. Gate claims drilling and Zone 8, 9 and 11 show poor results.
2010	4	2,694	Deep drilling in Casket Creek (for MQV) and proximal to resource drilling.
2011	30	5,926	Deep drilling in Casket Creek (wedging) and proximal to resource drilling.
2012	11	5,628	Deep drilling in Casket Creek and proximal to resource drilling.
2013	5	1,472	Deep drilling and proximal to resource drilling.
2015	21	5,548	Underground drilling - MQV and STK infill and extension, channel samples taken.
2020	2	1,130	Exploration for, MQV and STK intercepts.
2021	1	736	Exploration for, MQV and STK intercepts.
2022	24	5,928	Metallurgical and geotechnical drilling into existing mine area (not exploration).
Total	323	86,383	-

Source: GMRS (2026)

1.5 Geology and Mineralization

The Project is located within a westward-thickening wedge of sedimentary carbonate rocks of mid-Proterozoic to mid-Jurassic age that was deposited along the paleo-continental margin of western North America (Mackenzie Platform). The Prairie Creek Embayment paleo-basin is interpreted to have developed as a half-graben controlled by a north-trending fault with down-drop to the west.

In the immediate area of the Project, north trending faulting and folding are apparent. The most significant fold structure is the fault-bounded, north-striking, doubly plunging Prairie Creek anticlinal structure, which is the host to the PC Silver Mine mineralization.

Four styles of base metal mineralization have been identified on the Project: quartz vein, stratabound, stockwork and Mississippi Valley-type. Only the first three styles have been found in potentially economic quantities to date. Base metal mineral showings occur along the entire 16 km north to south length of the anticline covered by the main group of mining leases.

The most significant style of mineralization is the quartz vein-type, on which the previous underground workings had been developed. The MQV has been exposed by underground development and diamond drilling over a horizontal strike length of 2.3 km (Main Zone). The MQV trends at an azimuth of approximately 20° and dips between vertical and 40° east, with an average dip of 65°. The MQV consists of massive to disseminated galena and sphalerite with lesser pyrite and tennantite-tetrahedrite in a quartz-carbonate-dolomite sheared matrix. The galena and tennantite-tetrahedrite also carry economically significant silver values. This vein style of mineralization has been identified throughout the entire 16 km length of the mining leases with surface trenches and diamond drilling.

STK mineralization occurs as a series of narrow, massive, sphalerite-galena-tennantite veins striking at about 40° azimuth that occupy tensional or dilatant-type fractures within a structural offset of the MQV. This mineralization has developed in sub-vertical tensional openings formed obliquely to the initial primary fault movement along the main vein structure. STK has been exposed in both diamond drilling and underground development.

SMS mineralization occurs intermittently at the base of the trend of the PC Silver Mine vein system over a strike length of more than 800 m. The SMS mineralization occurs as semi-massive sphalerite-galena-pyrite replacement located close to both the vein system and the axis of the PC Silver Mine antiform but has not yet been intersected by underground development. The MQV structure carries fragments of the SMS indicating the vein mineralization to be younger in age.

MVT lead-zinc mineralization is exposed on the Project in surface showings marginal to the basin and consists of cavity-filling type breccias in dolostone with host fragments rimmed with colloform sphalerite-marcasite-galena healed with carbonate.

1.6 Metallurgical Testing and Mineral Processing

Metallurgical testwork was performed on PC Silver Mine beginning in the 1960's, although none of the earliest studies, including information relating to the existing circuit design, are currently available. Following this and continuing sporadically into the 1980's, and as recently as 2016, more testing using a modified flotation scheme for high oxide feed was conducted. There are three major principal types of mineralization consisting of MQV, STK, and SMS, with MQV identified as the most significant style of mineralization for the Project. The pre-2017 metallurgical testwork was performed on MQV and SMS mineralization by several different laboratories. For MQV the historic testing had been limited to samples readily available from the bulk sampling areas in mine adits closer to surface. This material contained more highly oxidized mineralization. MQV was then described as primarily massive to semi-massive galena and sphalerite, with varying degrees of cerussite (lead oxide) and smithsonite (zinc oxide) in a quartz-carbonate-dolomite matrix. The SMS samples, which were obtained from exploration drilling, were described as more finely grained and with lower sulphide oxidation content.

In 2017 a comprehensive testing program was conducted on the STK and high and low grade samples from the MQV mineralization zones. The program included mineralogy, dense media separation, comminution, and dewatering testwork. The recoveries determined from that testwork and subsequently used for this study can be found in Table 1-2.

Table 1-2: Estimated Metallurgical Recoveries and Concentrate Grades

Parameter	Unit	Dense Media Separation	Concentrates	
			Pb Concentrate	Zn Concentrate
Ag Recovery	%	97	$1.0086 \times 99.31 \times (\text{Pb} - \text{PbO}) / \text{Pb}$	$1.0771 \times \text{Pb recovery to lead concentrate} - \text{Ag recover to lead concentrate}$
Pb Recovery	%	97.5	$99.31 \times (\text{Pb} - \text{PbO}) / \text{Pb}$	0
Zn	%	96.5	0	$91.31 \times (\text{Zn} - \text{ZnO}) / \text{Zn}$
Cu Recovery	%	97.2	95	0
Concentrate Grade				
Mass	%	63	Variable	Variable
Ag	%	Variable	Variable	Variable
Pb	%	Variable	60	Variable
Zn	%	Variable	Variable	60
Cu Recovery	%	Variable	Variable	Variable

1.7 Mineral Resource Estimate

Table 1-3 summarizes the mineral resource estimate for the three mineral zones MQV, STK and SMS, at a silver equivalent (AgEq) cut-off grade of 100 grams per tonne (g/t).

Table 1-3: PC Silver Mine Mineral Resource Estimate at 100 g/t AgEq Cut-off

Domain	Classification	Tonnes	AgEq g/t	Ag g/t	Cu %	Pb %	Sb ppm	Zn %
MQV	Measured	856,000	702	209	0.40	11.44	1,020	12.89
MQV	Indicated	5,407,000	566	164	0.33	10.79	1,494	9.38
MQV	M&I	6,263,000	585	170	0.34	10.88	1,429	9.86
MQV	Inferred	4,958,000	679	185	0.49	8.18	2,429	14.98
STK	Measured	128,000	401	97	0.20	4.15	1,028	10.32
STK	Indicated	4,727,000	221	49	0.10	2.60	522	5.69
STK	M&I	4,855,000	226	50	0.11	2.64	536	5.81
STK	Inferred	3,256,000	221	53	0.10	3.15	502	5.11
SMS	Indicated	510,000	345	54	0.02	5.15	83	9.72
SMS	M&I	510,000	345	54	0.02	5.15	83	9.72
SMS	Inferred	255,000	325	48	0.01	4.35	73	9.67
ALL ZONES	Measured	984,000	663	194	0.37	10.49	1,021	12.56
ALL ZONES	Indicated	10,644,000	402	108	0.21	6.88	995	7.76
ALL ZONES	M&I	11,628,000	424	115	0.23	7.19	997	8.16
ALL ZONES	Inferred	8,469,000	492	130	0.33	6.13	1,617	11.03

Notes:

1. Mineral Resources are stated as of June 21, 2026;
 2. Stated at a cut-off grade of 100 g/t AgEq based on prices of C\$ 6.34/lb for copper, C\$ 1.28/lb for lead, C\$ 1.76/lb for zinc and C\$ 51.39/oz for silver;
 3. Average processing recovery factors of 85% for copper, 85.7% for zinc, 86.5% for lead, and 94.6% for silver;
 4. Average payables of 85% for copper, 85% for zinc, 87.9% for lead, and 85% for silver;
 5. $\text{AgEq} = (\text{grade of Ag in g/t} + [(\text{grade of copper in ppm} * \text{price of copper in C\$/g} * \text{recovery of copper in \%} * \text{payable copper in \%}) + (\text{grade of lead in \%} * \text{price of lead in C\$/lb} * 22.046 * \text{recovery of lead in \%} * \text{payable lead in \%}) + ((\text{grade of zinc in \%} * \text{price of zinc in C\$/lb} * 22.046 * \text{recovery of lead in \%} * \text{payable lead in \%})) \text{ price of silver in C\$/g} * \text{recovery of silver in \%} * \text{payable silver in \%})$; and
 6. Numbers may not compute exactly due to rounding.
- Source: GMRS (2026)

1.8 Mineral Reserve Estimate

There are no Mineral Reserve Estimates for the Project as it has not advanced to a pre-feasibility study level needed to estimate Reserves.

1.9 Mining

The PC Silver Mine is a proposed underground operation comprising three mineralized zones, (MQV, STK, and SMS). The MQV is the largest mining zone, extending approximately 2.3 km along strike, and is characterized by thin, steeply dipping geometry. The STK zone is also steeply dipping but generally wider and the SMS zone is wide and flat lying.

The MQV and STK zones will be mined using longitudinal retreat stoping, primarily with downhole drilling, while the SMS will use a combination of longitudinal transverse longhole stoping and mostly upwards drilling. Underground production will ramp up from an initial level of 1,600 t/d to a steady state rate of 2,400 t/d, or approximately 876,000 t/y. The mine life is estimated to be approximately 22 years.

Approximately 5 km of underground workings have been developed across the Project, including the 883 L, 930 L, and 970 L levels and associated portals. These workings will be incorporated into the mine plan where practical to reduce new development requirements. The proposed mine has been divided into four mining zones for increased scheduling and mining efficiencies, each accessed by a dedicated ramp. Dense media separation will be used to upgrade mine feed prior to milling, while tailings will primarily be returned underground as paste backfill.

The optimization parameters used in developing the underground mining inventory are summarized in Table 1-4.

Table 1-4: Stope Optimization Parameters

Parameter	Unit	Values
Revenue, Smelting, and Refining		
Zinc Price	US\$/lb	1.27
Lead Price	US\$/lb	0.92
Silver Price	US\$/oz	37.00
Copper Price	US\$/lb	5.57
Exchange Rate	USD:CAD	0.72
Zinc Payable	%	85.0
Lead Payable	%	87.9
Silver Payable	%	85.0
Copper Payable	%	96.5

Parameter	Unit	Values
Silver Eq (AgEq)	g/t	488
Indicative Rock Value	CAD\$/tonne	634
Net Smelter Royalty	%	2.2
Operating Cost Estimate		
Mining Cost - Underground Only	CAD\$/t mined	89.73
Processing Cost	CAD\$/t milled	44.29
G&A	CAD\$/t milled	20.15
Site Services	CAD\$/t milled	29.18
Transportation	CAD\$/t milled	34.50
Sustaining Capital Cost Estimate	CAD\$/t milled	33.25
Total Operating Cost Estimate	CAD\$/t milled	217.84
Recovery and Dilution		
Estimated Mining Dilution for Stope Optimization Input	%	12.0
Equivalent Linear Overbreak/Slough	m	0.5 + 0.5
External Mining Dilution for Mine Planning	%	5.0
Mining Recovery	%	95
Mining Dimensions		
Minimum Dip (MQV & STK)	°	55
Minimum Dip (SMS)	°	0
Minimum Mining Width	m	3
Stope Height (MQV & STK)	m	20
Stope Height (SMS)	m	Variable
Stope Length	m	5 to 20
Mill Recovery		
Zinc Recovery	%	85.7
Lead Recovery	%	86.5
Silver Recovery	%	94.6
Copper Recovery	%	85.0
Cut-off value		
Diluted/Recovered Cut-Off Value (includes royalty)	CAD\$/t	262.60
Diluted/Recovered mill breakeven cut-off value	CAD\$/t	151.04
Other Parameters		
Resource Classification	-	M+I+I
Mill Throughput	t/yr	876,000

Primary access will be through the existing 870 L portal, with additional access provided by the 930 L and 970 L portals. A main decline and four internal ramps will provide access to four mining zones (A, B, C, and D). Each zone is further subdivided into production areas, with mining generally advancing from the bottom upward and completed stopes paste filled to complete the mining cycle. The life of mine (LOM) mineralized material, average grades and contained metal by primary commodity are summarized in Table 1-5.

Table 1-5: LOM Inventory Summary

Parameter	Unit	LOM Total/Grade	Unit	Contained Metal
Mined Mineralized Material	kt	18,588		
Zinc	%	8.52	kt	1,583
Lead	%	6.14	kt	1,141
Silver	g/t	112.5	koz	67,212
Copper	%	0.25	kt	47

1.10 Recovery Methods

The process design for PC Silver Mine prioritizes utilization of the existing mill facility and equipment to minimize capital costs.

The flowsheet incorporates crushing, dense media separation, grinding, flotation, thickening, and filtration to produce two separate concentrates: a lead concentrate and a zinc concentrate.

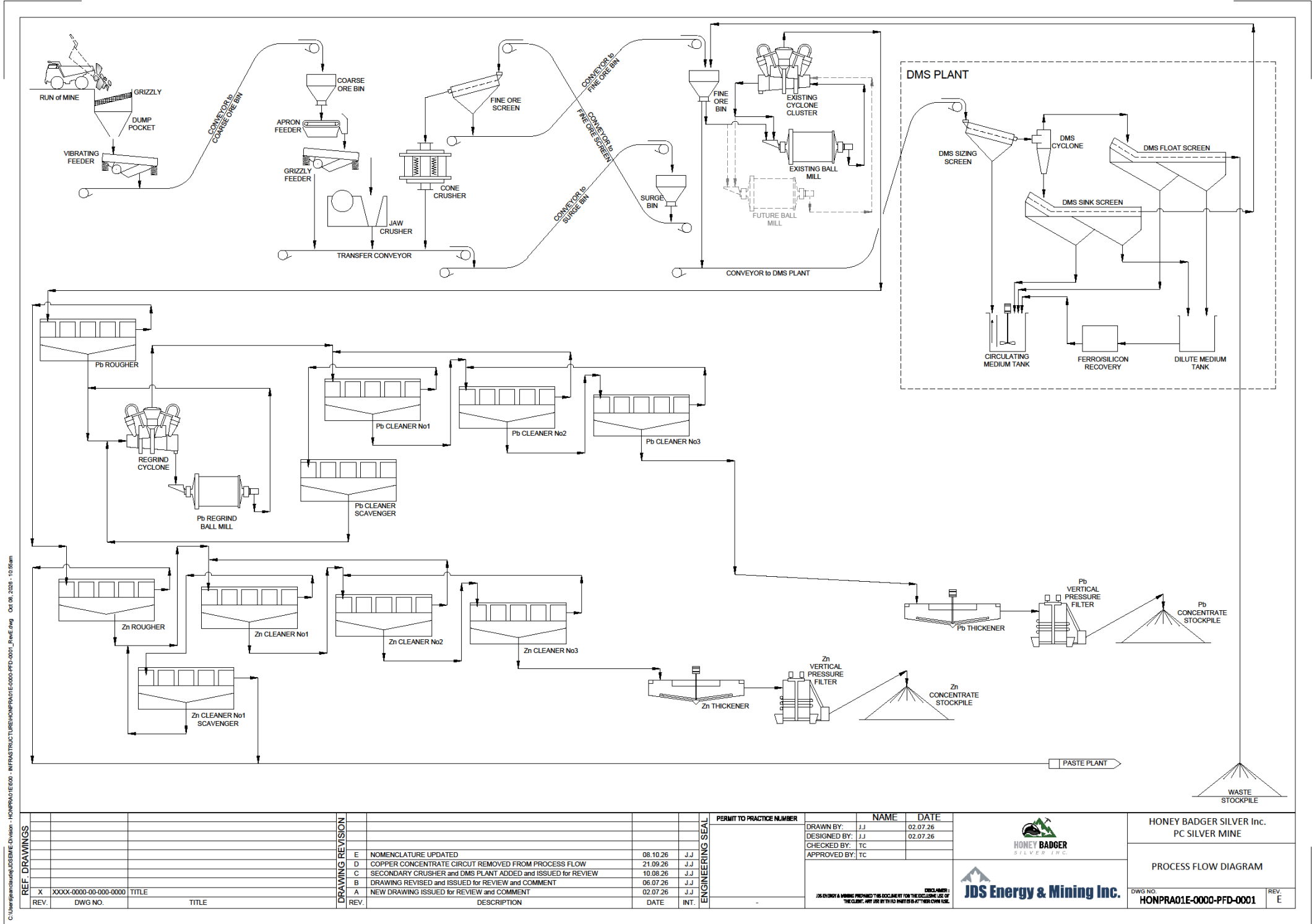
The flowsheet is based on processing design criteria detailed in Table 1-6 and is shown in Figure 1-1.

Table 1-6: Process Design Criteria

Criteria	Description	Units	Design
Plant Throughput		kt/d	2,400
		Mt/a	876,000
Crusher Availability		%	70
Crusher Throughput		t/h	143
Crusher Selection		Size	24 x 36
		Number	1

Criteria	Description	Units	Design
Dense Media Separation Plant Rejection	Mass of Feed Rejected	%	36.5
	Silver Recovery	%	97.40
	Lead Recovery	%	97.50
	Zinc Recovery	%	95.60
	Copper Recovery	%	97.20
	Iron Recovery	%	100.00
Mill/Flotation Availability		%	92
Mill Throughput		t/h	69
Physical Characteristics	BWI	kWh/t	13
Primary Grind Size	P ₈₀	µm	150
Lead Concentrate Grind Size	P ₈₀	µm	40
Head Grade (Ave)		%Pb	6.14
		%Zn	8.52
		% Cu	0.25
		g/t Ag	112.5
Flotation Recovery		%Pb	87.0
		%Zn	86.9
		% Cu	95
		% Ag	93.8
Concentrate Grade	Lead	%	60
	Zinc	%	58

Figure 1-1: Process Flowsheet



1.11 Infrastructure

The PC Silver Mine is an established remote site that will be upgraded to support continuous underground mining and processing at 2,400 t/d.

A 170 km All-Season Road (ASR) will be built over three years to connect the site to the Northwest Territories Highway 7 corridor. It will include a seasonal barge and ice bridge crossing of the Liard River, while a winter road will provide access until construction is finished. The existing 1,000 m gravel airstrip will be used for personnel transport, priority freight, and medical evacuation.

Power will be generated onsite by eight 1,500 kW diesel generators. They will be rented for the first two years and then replaced with dual fuel units purchased. Diesel and liquified natural gas (LNG) will be stored onsite in the existing onsite tanks with a combined capacity of 6.8 million litres.

The site will have freshwater supply, potable and wastewater treatment, and firewater systems. Surface runoff and contact water will be collected, diverted, and treated before discharge.

The vast majority of the tailings will be returned to the underground mine as paste backfill, with a nominal portion stored, temporarily on surface until placed underground. Waste rock will be hauled from the mine to a surface waste rock pile.

A new 250-room camp will replace the deteriorated existing camp. It will be supported by new modular administration, mine dry, and laboratory buildings, a new warehouse and laydown area, and the existing maintenance shop will be rehabilitated. See Figure 1-2 and Figure 1-3 for general site layout details.

PLAN

NOTES:
1. ALL DIMENSIONS IN MILLIMETRES, ELEVATIONS IN METRES

PRELIMINARY
NOT TO BE USED FOR CONSTRUCTION

1:5,000

NO.	DWG NO.	TITLE	DATE	BY
1	26062-0000-C-GAD-001	Site Layout Plan	2025-07-27	R. KULAR
2				
3				
4				
5				
6				
7				
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NO.	DESCRIPTION	DATE	BY
1	ISSUED FOR INFORMATION	2025-07-27	R. KULAR
2	ISSUED FOR INFORMATION	2025-08-19	R. KULAR
3	ISSUED FOR APPROVAL	2025-08-19	R. KULAR
4	ISSUED FOR REVIEW	2025-07-29	R. KULAR

PROJECT NO.	26062
DISCIPLINE	CIVIL
SCALE	1:5000
DRAWN BY	R. KULAR
DATE	2025-07-27
DESIGNED BY	R. KULAR
CHECKED BY	C. PATEL
DATE	2025-08-17
APPROVED BY	C. PATEL
DATE	2025-08-17

HONEY BADGER SILVER INC.

giffen CONSULTING LTD.

PRAIRIE CREEK MINE PEA STUDY

SITE LAYOUT MINE PAD & ACCESS ROAD PLAN VIEW

DWG NO. 26062-0000-C-GAD-001

PAGE 1-14

NOTES:
1. ALL DIMENSIONS IN MILLIMETRES, ELEVATIONS IN METRES

PLAN

PRELIMINARY
NOT TO BE USED FOR CONSTRUCTION

1:1,000

10 0 10 20 30 40 50m

NO.	DATE	DESCRIPTION	BY
1	2025-05-17	ISSUED FOR INFORMATION	RR
2	2025-05-19	ISSUED FOR INFORMATION	RR
3	2025-05-14	ISSUED FOR APPROVAL	RR
4	2025-07-28	ISSUED FOR REVIEW	RR

NO.	DATE	DESCRIPTION	BY
1	2025-05-17	ISSUED FOR INFORMATION	RR
2	2025-05-19	ISSUED FOR INFORMATION	RR
3	2025-05-14	ISSUED FOR APPROVAL	RR
4	2025-07-28	ISSUED FOR REVIEW	RR

PROJECT NO: 26062
DISCIPLINE: CIVIL
SCALE: 1:1000
DATE: 2025-07-27
DRAWN BY: R. KILAR
DESIGNED BY: R. KILAR
CHECKED BY: C. PATEL
APPROVED BY: C. PATEL
Giffen Consulting Ltd. PREPARED THIS DOCUMENT FOR THE EXCLUSIVE USE OF THE CLIENT. ANY USE BY THIRD PARTIES IS AT THEIR OWN RISK.

HONEY BADGER
SILVER INC.

giffen
CONSULTING LTD.

PRairie Creek Mine
PEA Study
SITE LAYOUT
MINE & CAMP PAD
PLAN VIEW

DWG NO: 26062-0000-C-GAD-002

PAGE 1-15

1.12 Environment and Permitting

The Project is in the southern Mackenzie Mountains (870 m above mean sea level) in southwest Northwest Territories. The Project site facilities are situated on the eastern side of and adjacent to PC Silver Mine, about 43 km upstream from its confluence with the South Nahanni River, and approximately 7 km upstream of the point where PC Silver Mine crosses the boundary of the expanded Nahanni National Park Reserve. The South Nahanni River flows into the Liard River near Nahanni Butte, 100 km downstream from PC Silver Mine. The Liard River merges with the Mackenzie River at Fort Simpson, a further 175 km downstream.

The Project is within the claimed traditional territories of the Nahʔą Dehé Dene Band and the Łı́ıdlı́ Kúę First Nation, both historically part of the Dehcho First Nations. The Dehcho First Nations have been in land claim negotiations since late 1990s. The nearest community is Nahanni Butte, home of the Nahʔą Dehé Dene Band, located approximately 90 km to the southeast of the Project site.

The PC Silver Mine and All-Season Access Road are both fully permitted for operation.

Canadian Zinc Corp, a 100%-owned subsidiary of Honey Badger Silver Inc., currently holds the land use permits and water licences necessary to advance the Project. The all-season access road crosses through the jurisdictions of both the Mackenzie Valley Land and Water Board and Parks Canada, and thus both agencies have issued land use permits and water licences.

Several agreements are currently in place with government agencies:

- Nahanni National Park Reserve / Parks Canada Memorandum of Understanding with regard to the expansion of the Nahanni National Park Reserve and the development of the PC Silver Mine;
- A Collaboration Agreement with the Government of the Northwest Territories to ensure effective co-operation regarding public transportation infrastructure that will support the PC Silver Mine and public needs; and
- A MOU with the Northwest Territories Power Corporation to examine the supply of electrical power for the development and operation of the PC Silver Mine.

1.12.1 Closure and Reclamation Planning

Closure activities at the PC Silver Mine site are envisaged to occur in three phases, as follows:

- Phase I – On-site and off-site reclamation of all facilities not required for long-term monitoring and water treatment;
- Phase II – Post-closure monitoring (13 years) and water treatment (six years, if required); and
- Phase III – Final reclamation of post-closure facilities.

Following Phase III at the Project site, reclamation of the access road will occur.

An interim Closure and Reclamation Plan has been approved under the mine's water licence. The licence specifies incremental financial security amounts for mine closure based on development milestones.

1.13 Capital and Operating Cost Estimates

Costs are expressed in Canadian dollars (C\$ or \$) and do not include allowances for escalation or exchange rate fluctuations. Pre-production capital costs amount to \$667 million. Total LOM capital costs are estimated to be \$1,150 million. Individual contingency rates were applied to each of the pre-production capital cost categories to reflect the level of engineering effort undertaken and the estimate/engineering accuracy. This resulted in a blended contingency rate of 16%, or \$91.8 M for the initial capital cost estimate. Contingency was applied to sustaining costs at a blended overall rate of 10%, resulting in a total of \$42.6 million which includes a 30% contingency applied to closure costs. Sustaining (capital costs are estimated to be \$483 million including that contingency cost, with closure costs comprising \$49 million of that amount. Pre-production and LOM Capital Costs are summarized below in Table 1-7.

Table 1-7: Summary of Capital Cost Estimate

Capital Costs	Pre-Production (\$M)	Sustaining / Closure (\$M)	Total (\$M)
Site Development	5.7	-	5.7
Mining	64.6	330.3	394.9
Mineral Processing	66.0	-	66.0
Tailings & Waste Rock Management	15.4	2.9	18.4
On-site Infrastructure	44.7	46.9	91.6
Off-site Infrastructure	205.2	-	205.2
Project Indirects	114.9	5.0	119.9
Engineering & EPCM	41.2	6.5	47.7
Owner's Costs	17.5	-	17.5
Closure	-	49.0	49.0
Subtotal	575.3	440.6	1,015.9
Contingency	91.8	42.5	134.3
Total Capital Costs	667.1	483.1	1,150.2

Operating costs include all costs associated with owner-operated mining, processing, and general & administration costs up to the production of concentrate onsite. Mine operating costs incurred during the pre-production are capitalized and form part of the capital cost estimate.

Total operating costs over the LOM are \$3,101 million, with average annual operating costs of \$141 million. The LOM operating costs are summarized in Table 1-8 below. The total operating unit cost is \$166.83/t processed. Operating costs are presented in Canadian dollars and do not include allowances for escalation or exchange rate fluctuations.

Table 1-8: Summary of Operating Cost Estimate

Operating Costs	\$/t Milled	LOM (\$M)
Mining	88.99	1,654
Processing	39.33	731
G&A	38.52	716
Total	166.83	3,101

The main assumptions used in the preparation of the basis of estimate for the operating costs are shown in Table 1-9.

Table 1-9: Main Operating Cost Estimate Component Assumptions

Item	Unit	Value
Electrical power cost Y-1-Y2	\$/kWh	0.46
Electrical power cost Y3+	\$/kWh	0.35
Average power consumption	MW	4.29
Overall power consumption (all facilities)	kWh/t processed	43
Diesel cost (delivered)	\$/litre	1.61

1.14 Economic Analysis

An economic model was developed to estimate annual cashflows and evaluate sensitivities for the Project. All costs, metal prices, and economic results are reported in Canadian currency (C\$ or \$) unless stated otherwise.

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 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.

This Technical Report contains forward-looking information regarding projected mine production rates, mine construction schedule, and forecasted resulting cash flows. The mill head grades are based on 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, to obtain major equipment or skilled labour on a timely basis, or 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 reader is cautioned that the metals prices and exchange rates used in this study are only estimates based on consensus prices per the Bank of Canada, and there is no guarantee that they will be realized if the Project is taken into production. The prices of silver, lead, zinc, and copper are based on many complex factors and there are no reliable methods of predicting their long-term prices.

At the base case metal prices (US\$ 50.00 per oz Ag, US\$0.90 per lb Pb, US\$1.50 per lb Zn, and US\$5.00 per lb Cu) the Project generates an after tax NPV_{8%} of \$1,172 and an after tax IRR of 29.3%. Payback on initial capital cost estimate is 3.1 years.

The economic results for the Project, based on the assumptions as outlined above, are presented below in Table 1-10.

Table 1-10: Summary of Economic Results

Parameter	Unit	Pre-Tax Results	After-Tax Results
NPV _{0%}	\$M	4,934	3,172
NPV _{5%}	\$M	2,683	1,683
NPV _{8%}	\$M	1,915	1,172
IRR	%	37.0	29.3
Payback period	Production years	2.7	3.1

This economic assessment is preliminary in nature and includes the use of inferred mineral resources that are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the preliminary economic assessment will be realized.

1.14.1 Sensitivities

Sensitivity analyses were performed using the foreign exchange rates, metal prices, mill head grade, capital cost estimate, and operating cost estimate as variables. The value of each variable was changed plus and minus 25% independently while all other variables were held constant. The results of the sensitivity analyses are shown in Table 1-11. The Project is most sensitive to exchange rate followed by Zinc price.

Table 1-11: Sensitivity Analysis Results

Variable	After-Tax NPV _{8%} (\$M)		
	-25% Variance	0% Variance	+25% Variance
Silver Price	900	1,172	1,446
Lead Price	995	1,172	1,350
Zinc Price	816	1,172	1,529
Foreign Exchange Rate	2,029	1,172	659
Operating Cost Estimate	1,377	1,172	968
Capital Cost Estimate	1,378	1,172	967

In addition to the base case as outlined above, three alternate price scenarios were analyzed based on metal spot prices, 12-month high prices and 36-month low prices respectively. The results of these analyses are shown below in Table 1-12.

Table 1-12: Additional Price Scenario Results

		Base Case	Spot Prices*	12-month High**	36-month Low***
Pre-Tax Results	NPV _{0%}	4,934	7,300	12,214	778
	NPV _{5%}	2,683	4,054	6,942	272
	NPV _{8%}	1,915	2,951	5,147	93
	IRR	37.0%	48.2%	69.1%	10.1
	Payback	2.7	2.1	1.3	6.0
Post-Tax Results	NPV _{0%}	3,172	4,678	7,813	528
	NPV _{5%}	1,683	2,559	4,403	136
	NPV _{8%}	1,172	1,836	3,240	-5
	IRR	29.3%	38.2%	54.8%	7.9%

		Base Case	Spot Prices*	12-month High**	36-month Low***
	Payback	3.1	2.4	1.6	6.2

Notes:

*Spot prices refers to closing prices on August 31, 2026 (Ag = US\$ 66.41/oz, Pb = US\$ 0.90/lb, Zn = US\$ 1.50/lb, Cu = US\$ 5.00/lb, FX = 0.72).

**12-month high prices refer to the highest closing prices reported between September 2025-September 2026 (Ag = US\$ 121.67/oz, Pb = US\$ 0.95/lb, Zn = US\$ 1.84/lb, Cu = US\$ 6.92/lb, FX = 0.7019)

***36-month low prices refer to the lowest closing prices reported between September 2023-September 2026 (Ag = US\$ 21.59/oz, Pb = US\$ 0.85/lb, Zn = US\$ 1.04/lb, Cu = US\$ 3.36/lb, FX = 0.7450)

1.15 Conclusions

It is the conclusion of the QPs that the PEA summarized in this Technical Report contains adequate preliminary detail and information to support a PEA-level report. Standard industry practices, equipment and design methods were used in the PEA.

Based on the assumptions used for this preliminary evaluation, the Project shows positive economics and should proceed to a Feasibility Study stage.

The most significant potential risks associated with the Project are uncontrolled dilution with waste rock and rock from different mineralized zones, operating and capital cost escalation, permitting and environmental compliance, unforeseen schedule delays, changes in regulatory requirements and 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 proactive management.

To date, the Qualified Persons (QP) are not aware of any fatal flaws for the PC Silver Mine.

1.16 Recommendations

It is recommended that the PC Silver Mine proceed to the Feasibility Study stage in line with Honey Badger Silver's desire to advance the Project.

It is estimated that a Feasibility Study and supporting field work would cost approximately \$5.0 million. A breakdown of the key components of the next study phase is as follows in Table 1-13.

Table 1-13: Estimated Cost of Proposed Programs to Support a Feasibility Study

Program Description	Estimated Total Cost (\$M)
Drilling	0.3
Mine Geotechnical (excludes drilling)	0.4
Metallurgical Testing	0.5
Waste & Water Management	0.5
Process Plant Assessment	0.5
Environment & Permitting	0.4
Feasibility Study Engineering and Technical Report	2.4
Total	5.0

2 INTRODUCTION

The PC Silver Mine is a silver-zinc-lead-copper resource development project located in the Mackenzie Mountains of the Northwest Territories. Honey Badger Silver Inc. (Honey Badger Silver) is the 100% owner of the mining and surface leases totaling approximately 7,485 hectares. Honey Badger Silver Inc. is a Canadian based mining company listed on the TSX Venture Exchange (TSX-V: TUF) with its corporate office located at 215 Spadina Avenue, Toronto, ON, M5T 2C7.

JDS Energy & Mining Inc. (JDS) was commissioned by Honey Badger Silver to lead a PEA for the Prairie Creek Silver Mine (PC Silver Mine, or the Project). The PEA was prepared following the guidelines of the Canadian Securities Administrators' National Instrument 43-101 and Form 43-101F1.

The reporting of the PEA was done using guidance from the NI 43-101 Standards of Disclosure for Mineral Projects. The classification of the Mineral Resource Estimate (MRE) is consistent with the 2014 Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards (2014 CIM Definitions). In completing the MRE, the Author uses general procedures and methodologies that are consistent with industry standard practices, including those documented in the 2019 CIM Estimation of Mineral Resources & Mineral Reserves Best Practice Guidelines (2019 CIM Guidelines).

The Effective Date of this Technical Report is June 21, 2026.

2.1 Qualified Persons

2.1.1 Scope of Work

This Technical Report summarizes the work of several consultants with the scope of work for each company listed below, which combined, comprises the total Project scope.

JDS Energy & Mining Inc. (JDS):

- Establishing the economic framework for the PEA;
- Establishing metal recovery values;
- Development of the conceptual flowsheet;
- Mine engineering, design and scheduling;
- Development of detailed flowsheets, specifications and selection of process equipment;
- Design oversight related to site infrastructure, access roads, airstrip, power generation, water treatment, accommodation camp and other ancillary facilities;

- Estimating mining, process plant, G&A and site services Operating and Capital Cost Estimates;
- Estimating closure requirements and costs;
- Preparing a financial model and conducting an economic evaluation including sensitivity and Project risk analyses;
- Interpreting the results and making conclusions that lead to recommendations to improve Project value and reduce risks; and
- Developing and compiling the Technical Report and integrating sub-consultant report sections.

Global Mineral Resources Services (GMRS):

- Deposit geology and mineralization;
- QA/QC, data verification; and
- Mineral Resource Estimation.

Knight Piésold Ltd. (KP):

- Tailings management facility and waste rock management facility design;
- Overall Project water balance;
- Water Management, including design of diversions and ponds; and
- Environment, Permitting, Social and Community Impacts.

T Engineering (T Eng):

- Paste backfill system design & engineering; and
- Estimation of backfill plant and underground distribution system Operating and Capital Cost Estimates.

2.1.2 Qualifications and Responsibilities

The results of this PEA 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 Honey Badger Silver and the Qualified Persons (QPs). The QPs are being paid a fee for their work in accordance with normal professional consulting practice.

The following individuals, by virtue of their education, experience and professional association, are considered QPs as defined in the NI 43-101, and are members in good standing of

appropriate professional institutions / associations. All Qualified Persons used for this study are independent of Honey Badger Silver. The QPs are responsible for the specific report sections as listed in Table 2-1.

Table 2-1: QP Responsibilities

QP	Company	QP Responsibility / Role	Report Section(s)
Michael Makarenko, P.Eng.	JDS	Project Management, Mining, Infrastructure and overall report compilation and all sections not covered by other QPs	1.1-1.3, 1.8, 1.9, 1.11, 1.15, 1.16, 2, 3, 12.2, 12.3, 15, 16 (except 16.3 and 16.8), 18 (except 18.6, 18.7), 23-28
Tad Crowie, P.Eng.	JDS	Metallurgy and Recovery methods	1.6, 1.10, 12.4, 13, 17
Michael Levy, P.Eng.	JDS	Mine geotechnical characterization and recommendations	12.5, 16.3
Tysen Hantelmann, P.Eng.	JDS	Market Studies, Capital & Operating Costs, Economic Analysis	1.13, 1.14, 19, 21, 22
Greg Mosher, P.Geo.	GMRS	History, drilling, geology, mineral resource estimate	1.4-1.5, 1.7, 4-11, 12.1, 14
Les Galbraith, P.Eng.	KP	Tailings Management Facility & Waste Rock Pile design, Water Management	12.6, 18.6, 18.7
Richard Cook, P.Geo.	KP	Environment & permitting	1.12, 20
Yen Jui (Bernie) Ting, P.Eng.	T Eng	Backfill engineering	16.8

2.2 Terms of Reference

The report supports disclosures by Honey Badger Silver in a news release dated September 17, 2026 entitled, “Honey Badger Silver Announces Preliminary Economic Assessment; The PC Silver Mine Has Potential to be a Significant Silver and Critical Minerals Producer”.

2.3 Sources of Information

This Technical Report is based on the following sources of information:

- Discussions with Honey Badger Silver personnel;
- Observations during June 2026 site visit to the PC Silver Mine;
- Inspection of the existing process plant, site infrastructure and drill core;
- Past reports and information provided by Honey Badger Silver;

- Review of exploration data collected by Honey Badger Silver; and
- Additional information from public domain sources.

Information regarding the Property accessibility, climate, local resources, infrastructure, and physiography, exploration history, regional property geology, deposit type, recent exploration and drilling, metallurgical testwork, and sample preparation, analyses, and security for previous drill programs etc. (Sections 5 through 13) have been sourced from previous technical reports and internal reports and updated where required. The Authors believe the information used to prepare the current Technical Report is valid and appropriate considering the status of the Project and the purpose of the Technical Report.

2.4 Site Visit

A summary of the site visit in accordance with National Instrument 43-101 guidelines is detailed in Table 2-2.

Table 2-2: QP Site Visits

Qualified Person	Company	Date	Accompanied by	Description of Inspection
Michael Makarenko, P.Eng.	JDS	June 16, 2026	<ul style="list-style-type: none"> • Ron Halas, COO • Sean Tucker, Site Manager • Zack Rockson, Geologist 	<ul style="list-style-type: none"> • Inspection of site infrastructure & buildings, existing mine portals, process plant, tailings management facility and proposed waste rock pile location • Inspection of selected drill core and logs
Tad Crowie, P.Eng.	JDS	June 16, 2026	<ul style="list-style-type: none"> • Ron Halas, COO • Sean Tucker, Site Manager • Matt Kovacs, HSE 	<ul style="list-style-type: none"> • Inspection of site infrastructure & buildings • Inspection of existing process plant and equipment
Michael Levy, P.Eng.	JDS	n/a	n/a	<ul style="list-style-type: none"> • Michael did not conduct a site visit
Tysen Hantelmann, P.Eng.	JDS	n/a	n/a	<ul style="list-style-type: none"> • Tysen did not conduct a site visit
Greg Mosher, P.Geo.	GMRS	June 16, 2026	<ul style="list-style-type: none"> • Ron Halas, COO • Sean Tucker, Site Manager • Zack Rockson, Geologist 	<ul style="list-style-type: none"> • Inspection of selected drill core and logs

Qualified Person	Company	Date	Accompanied by	Description of Inspection
Les Galbraith, P.Eng.	KP	June 16, 2026	<ul style="list-style-type: none"> Ron Halas, COO Sean Tucker, Site Manager 	<ul style="list-style-type: none"> Inspection of site infrastructure & buildings, existing mine portals, process plant, tailings management facility and proposed waste rock pile location
Richard Cook, P.Geo.	KP	n/a	n/a	<ul style="list-style-type: none"> Richard did not conduct a site visit
Yen Jui (Bernie) Ting, P.Eng.	T Eng	n/a	n/a	<ul style="list-style-type: none"> Bernie did not conduct a site visit

2.5 List Of Previous Relevant Technical Reports

Previously filed technical reports on the Project are as follows:

- Prairie Creek NI 43-101 Technical Report for Honey Badger Silver Inc., prepared by Ausenco Engineering Canada ULC., with an effective date of March 11, 2026;
- Prairie Creek Property Preliminary Economic Assessment NI 43-101 Technical Report for NorZinc Ltd., prepared by Ausenco, with an effective date of October 15, 2021;
- Prairie Creek Property Feasibility Study NI 43-101 Technical Report for Canadian Zinc Corporation, prepared by AMC Mining Consultants, with an effective date of September 28, 2017;
- Prairie Creek Property Prefeasibility Update NI 43-101 Technical Report (Amended and Restated) for Canadian Zinc Corporation, prepared by AMC Mining Consultants, with an effective date of March 31, 2016; and
- Prairie Creek Property Northwest Territories, Canada Technical Report for Canadian Zinc Corporation, prepared by AMC Mining Consultants, with an effective date of June 15, 2012.

2.6 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 Canadian dollars (C\$ or \$) unless otherwise noted.

Frequently used abbreviations and acronyms can be found in Section 28. This report includes technical information that required subsequent calculations to derive subtotals, totals and weighted averages. Such calculations inherently involve a degree of rounding and consequently introduce a margin of error. Where these occur, the QPs do not consider them to be material.

3 RELIANCE ON OTHER EXPERTS

The QP's opinions contained herein are based on information provided by Honey Badger Silver and others throughout the course of the study. The QPs have taken reasonable measures to confirm information provided by others and take responsibility for the information. The QPs used their experience to determine if the information from previous reports was suitable for inclusion in this Technical Report and adjusted information that required amending.

Non-QP specialists relied upon for specific advice are listed below, along with the extent of their involvement and sections of the report to which their input applies:

Don Smith – Resource Logistics Consulting

- Freight and transportation cost estimates contained in an e-mail and Excel spreadsheet “Logistics Costs – Prairie Creek – 2026.xlsx” dated August 7, 2026. The information contributed to Sections 21 and 22 of this Technical Report.

Andrew Fleming, CPA, CA – Accountant Taxation Services with MNP

- Provided input to the cash flow model related to Northwest Territories and Canadian taxation requirements summarized in Section 22.4 and reviewed and edited the tax model calculations in the cash flow model used in Section 22.5. Mr. Fleming's review and advice were in a series of e-mails and Excel spreadsheet reviews and edits August to September 2026.

Howard Okamura – Concentrate Marketing Consulting

- Howard Okamura, an experienced concentrate marketing consultant, provided a market assessment and developed a marketing strategy for the zinc and lead concentrates anticipated to be produced at the PC Silver Mine. His advice for potential concentrate smelter terms were received in documents via e-mails which were then applied as inputs into the economic model.

The QP only reviewed the land tenure in a preliminary fashion and has not independently verified the legal status or ownership of the Property or any underlying agreements or obligations attached to ownership of the Property. However, the QP has no reason to doubt that the title situation is other than what is presented in Section 4 of this Technical Report. The QP is not qualified to express any legal opinion with respect to Property titles or current ownership.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 Introduction

The Project is located in the Northwest Territories, Canada, near the Yukon border, at latitude 61° 33' North and longitude 124° 48' West. The nearest communities and approximate distances include Nahanni Butte, 90 km southeast; Fort Liard, 170 km south; and Fort Simpson, 185 km to the east. Yellowknife, the capital and administrative centre of the Northwest Territories, is approximately 500 km to the east. The town of Fort St. John, British Columbia, 700 km to the south of the mine, is the primary air charter point for incoming cargo and out-of-territory workers. Figure 4-1 shows the location of the Project within the Northwest Territories and relative to various population centers and mining operations. The total area of all land holdings, including mining leases and surface leases at PC Silver Mine, is 7,485.3 hectares (ha).

Figure 4-1: Location of PC Silver Mine



Source: NorZinc (2023)

The Project site, (Figure 4-1), is located within the watershed of the South Nahanni River, approximately 48 km upstream of the point where PC Silver Mine joins the South Nahanni River. The current boundary of the expanded Nahanni National Park Reserve is approximately 7 km downstream and 18 km upstream of the historical mine site. Since the expansion of the Nahanni National Park Reserve, the Project is located within an approximate 300 square kilometers (km²) area of territorial land that is now surrounded by, but not included in, the expanded Nahanni National Park Reserve.

4.2 Property Description and Ownership

The Project is located on Territorial Land and is administered by the GNWT. The planned ASR from the Nahanni Butte access road to PC Silver Mine will traverse territorial land and the Nahanni National Park Reserve which is administered by Parks Canada. Authorities for any proposed mine and the road include the federal departments of Environment and Climate Change Canada, Transport Canada, and the department of Fisheries and Oceans. The authority over the use of water and management of waste on territorial land is through the Mackenzie Valley Land and Water Board.

The property tenure consists of mining leases and surface leases which are currently 100% held by Honey Badger Silver Inc. through Canadian Zinc Corporation (CZC) and were issued by the GNWT. All leases, as listed in Table 4-1 are currently in good standing.

Honey Badger Silver completed an acquisition on April 15, 2026 to acquire 100% of the issued and outstanding shares of CZC from Resource Capital Fund VI L.P.

4.3 Mineral Tenure

The Mining Leases are renewable on a 21-year basis and currently have expiry dates ranging from September 2030 to August 2041. Mineral Leases are listed in Table 4-1. The mining leases are located as shown in Figure 4-2.

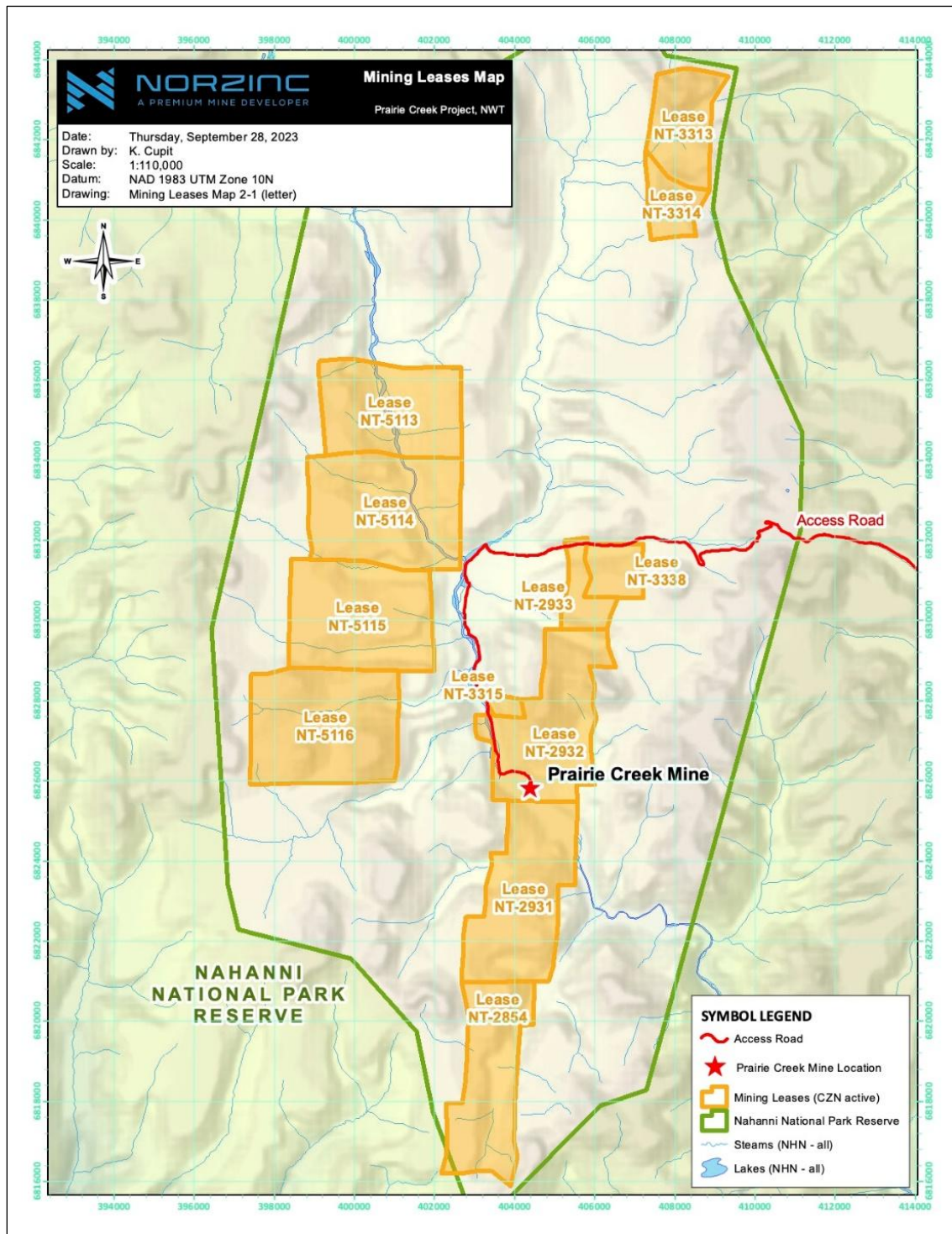
The Gate 1 to 4 Mineral Claims were staked in 1999. In August 2010 a perimeter land survey of these claims was completed resulting in an adjusted total surface area of 2,776 ha. New mining leases for the Gate Claims were received on February 16, 2011, dated September 9, 2009, and have a term of 21 years, until September 9, 2030.

Table 4-1: Mineral Tenure Plan

Lease Name	Lease Number	Holder	Type of Claim	Expiration Date	Area (ha)
Mine site	95F/10-5-5	CZN	Surface Lease	March 31, 2027 (renewed annually)	113.6
Airstrip	95F/10-7-4	CZN	Surface Lease	March 31, 2027 (renewed annually)	18.2
Total Surface Leases					131.8
Zone 8-12	NT 2854	CZN	Mineral Lease	August 21, 2040	743.0
Zone 4-7	NT 2931	CZN	Mineral Lease	August 4, 2041	909.0
Zone 3/Main Zone	NT 2932	CZN	Mineral Lease	August 4, 2041	871.0
Rico West	NT 2933	CZN	Mineral Lease	August 4, 2041	172.0
Samantha	NT 3313	CZN	Mineral Lease	July 12, 2031	420.0
West Joe	NT 3314	CZN	Mineral Lease	July 12, 2031	196.0
Miterk	NT 3315	CZN	Mineral Lease	July 12, 2031	43.5
Rico	NT 3338	CZN	Mineral Lease	July 16, 2031	186.0
Gate 1	NT 5113	CZN	Mineral Lease	September 8, 2030	794.0
Gate 2	NT 5114	CZN	Mineral Lease	September 8, 2030	1,039.0
Gate 3	NT 5115	CZN	Mineral Lease	September 8, 2030	944.0
Gate 4	NT 5116	CZN	Mineral Lease	September 8, 2030	1,036.0
Total Mining Leases					7,353.5
Grand Total					7,485.3

Source: Ausenco (2026)

Figure 4-2: Mineral Tenure Plan

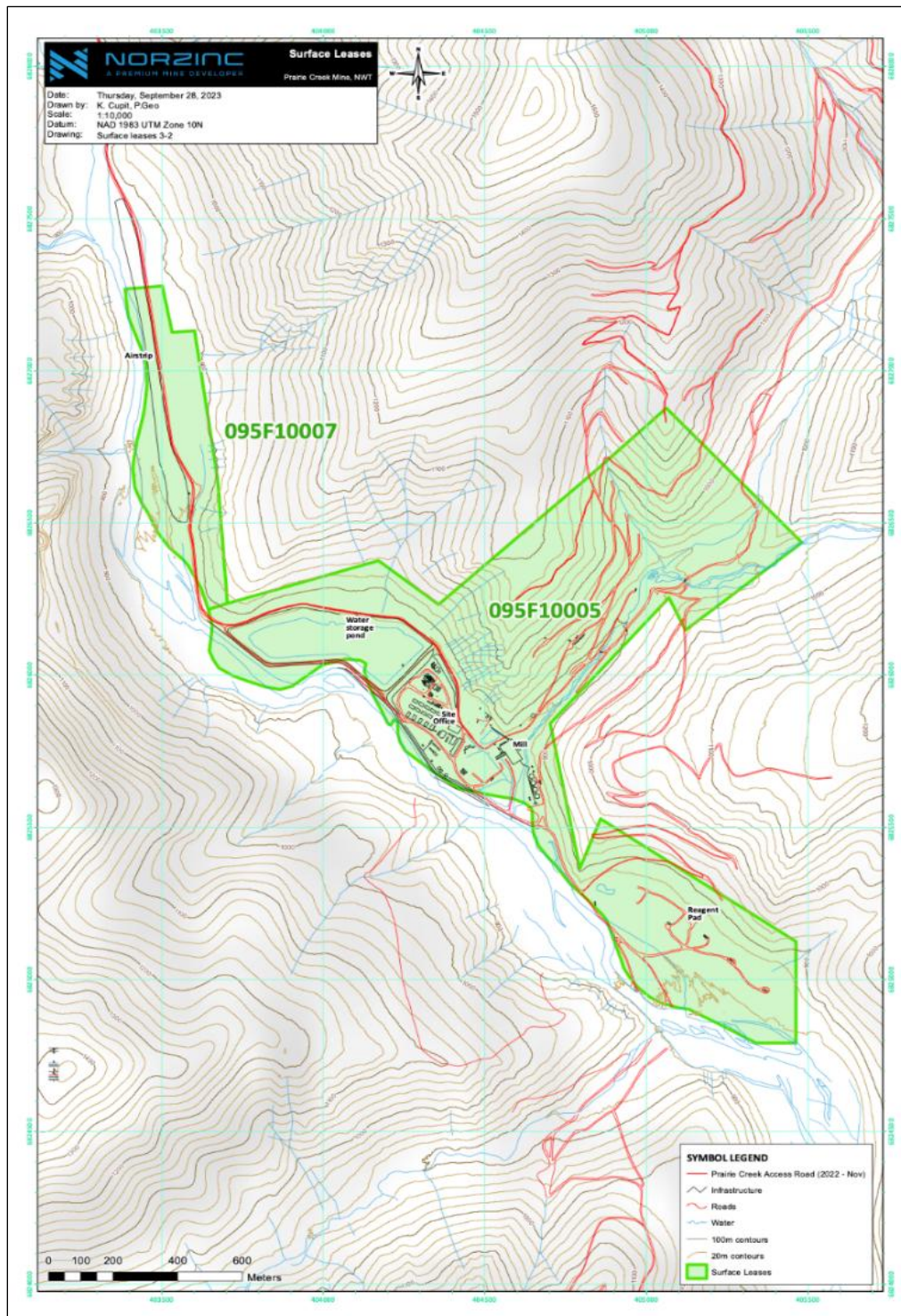


Source: NorZinc (2023)

4.4 Surface Rights

The current surface leases, containing the historical mine infrastructure, were originally granted by Aboriginal Affairs and Northern Development Canada (AANDC now CIRNAC) on a renewable, ten-year basis and, since devolution of some Federal powers to the Northwest Territories on April 1, 2014, are now administered by the GNWT. Presently the surface leases are held in a recurring annual overholding tenancy which is renewed on March 31st of each year and are still in good standing as of the effective date of the Report. These leases will remain until Honey Badger Silver negotiates new leases for operations. The location of surface rights is shown in Figure 4-3.

Figure 4-3: Location of Surface Rights



Source: NorZinc (2023)

4.5 Water Rights

The previous owner NorZinc was issued a Water Licence (MV2021L2-0004) for the PC Silver Mine from the Mackenzie Valley Land and Water Board, and Water Licences from Parks Canada for the ASR. The Mackenzie Valley Land and Water Board (MVLWB) also issued a Water Licence for the Indian Affairs Branch land, located 14 km near the Liard River. This land is administered by CIRNAC.

Table 4-2: Water Licence

Authorization Name	Licence	Authorizing Regulator	Expiration Date
Water Licence	PC2014L80006	Parks Canada	November 21, 2029
Water Licence (non-Federal)	MV2014L8-0006	MVLWB	November 12, 2039
Water Licence (Federal IAB)	MV2019L8-0002	MVLWB	November 12, 2039

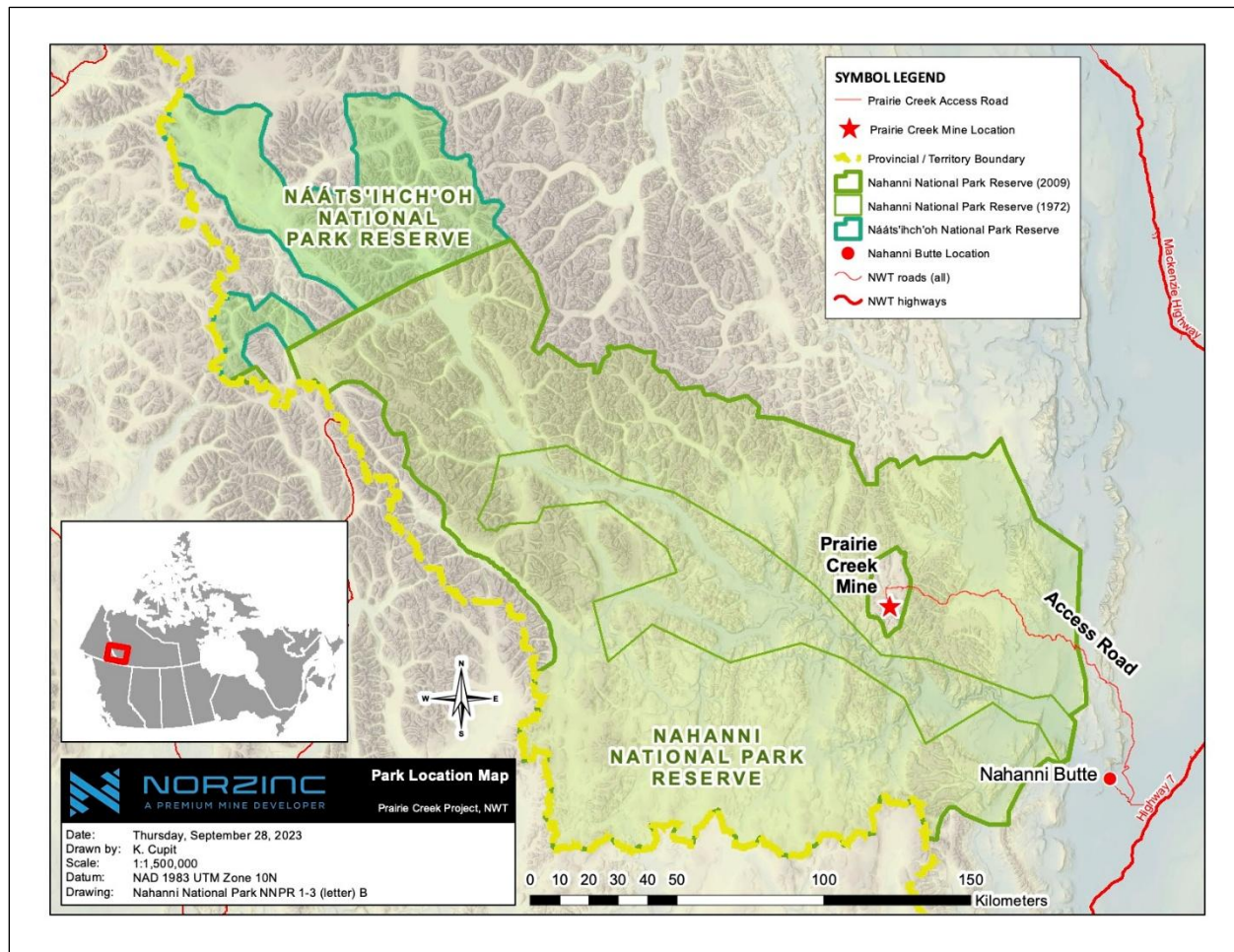
Source: Ausenco (2026)

4.6 Nahanni National Park Reserve

The Nahanni National Park Reserve was created in 1972 specifically for the purpose of setting aside the South Nahanni River for wilderness recreational purposes. Exploration activity at the Project had been ongoing for many years prior to 1972 and underground development was well advanced at that time.

In June 2009, new legislation was enacted by the Canadian Parliament entitled “An Act to amend the *Canada National Parks Act* to enlarge Nahanni National Park Reserve of Canada” to provide for the expansion of the Nahanni National Park Reserve. The Nahanni National Park Reserve was expanded to 30,000 km², making it the third largest National Park in Canada. The enlarged park covers most of the South Nahanni River watershed and encircles the territorial land hosting the Project mineral leases. The Project and surrounding area of approximately 300 km² were excluded from the expanded Nahanni National Park Reserve, as depicted in Figure 4-4, and the *Parks Act* was amended in Parliament to allow the right of road access through the Nahanni National Park Reserve into the PC Silver Mine. While not affecting PC Silver Mine, the Nááts'ihch'oh National Park Reserve was proclaimed in 2014 and adjoins Nahanni National Park Reserve to the northwest to further protect the South Nahanni watershed.

Figure 4-4: Property in Relation to the Expanded Nahanni National Park Reserve



Source: NorZinc (2023)

The exclusion of the Project from the Nahanni National Park Reserve expansion area brought clarity to the land use policy objectives for the region. In July 2008, Parks Canada Agency (Parks Canada) and Canadian Zinc entered a memorandum of understanding (MOU), valid for three years, covering mutual cooperation with regard to the expansion of the Nahanni National Park Reserve and the development of the PC Silver Mine. In March 2012, the MOU was renewed for a further period of three years wherein Parks Canada and Canadian Zinc agreed to work collaboratively to achieve their respective goals of managing the Nahanni National Park Reserve and an operating PC Silver Mine. Subsequently, the MOU was renewed in November 2015 for a period of five years, and again in September 2021 for another five years. At the time of writing this report the extension had been sent to Park Canada for signature.

4.7 Royalties and Encumbrances

The Property is subject to the following royalties:

- A 1.2% NSR royalty payable to Sandstorm Gold on the Project. Sandstorm Gold and Equinox Gold created a company called Sandbox Royalties in 2022 (Sandbox Royalties, 2022). In 2024, Sandbox Royalties entered into an agreement with B2Gold and changed their name to Versamet Royalties (Versamet Royalties, 2024); and
- A 1% NSR royalty payable to RCF VI CAD LLC.

4.8 Existing Environmental Liabilities

Existing environmental liabilities at the Project are presently covered through various security deposits posted to the relevant government agencies. The two existing surface leases, which contain all existing infrastructure, are covered by a reclamation and closure plan together with a security deposit. This plan describes limited reclamation liability to allow further exploration of the site with a view to future commercial operations and the assumption of all site liabilities. If Honey Badger Silver does not proceed with operations, the majority of the existing site liabilities will be the responsibility of the Crown.

On May 22, 2015, the MVLWB approved amendments to security payments relating to the issued Land Use Permit and Water Licence for operations at Prairie Creek. The amended payments reflected the existing liability at the site based on the closure plan associated with the surface leases, and staged future payments based on specific developments as the Project is advanced into production. On August 19, 2015, the GNWT confirmed that CZC had posted an additional security of \$1,550,000 (additional to the previously posted \$250,000 security) consistent with the MVLWB ruling. With the renewal of the mine land use permit and water licence by the MVLWB in 2021, the total current security was raised to \$2,391,986, which is on deposit with the GNWT. It is anticipated that Honey Badger Silver will take on these obligations.

4.9 Permitting Considerations

As noted in Sections 4.3, 4.4 and 4.5, the Project holds all necessary permits for commercial operations, and the current land tenure instruments are in the process of being updated for same. Each of the permits for the historical mine and road have conditions and requirements to fulfill, including structure designs and management plans, which are currently underway. For the historical mine site, this includes design of the water storage pond and waste rock pile. A full list of historical mine site submission requirements is listed at the back of Water Licence MV2021L2-0004. For the access road, designs and management plans were completed and approved for the Phase 1 winter road, which was implemented in early 2023 and allowed the completion of the geotechnical investigations needed to support Phase 2 ASR construction.

4.10 Social Licence Considerations

The Project is located on land claimed as their traditional territory by the Nahʔą Dehé Dene Band. The Łíídlıı Kúé First Nation also claims this land as part of their traditional territory. Dehcho First Nations is engaged in ongoing land settlement negotiations with the Government of Canada and the Government of the Northwest Territories (GNWT) in what is referred to as the Dehcho Process. However, the Nahʔą Dehé Dene Band is not represented by the Dehcho First Nations in this process but opted to represent themselves and conduct their own negotiations regarding land claims.

CZC has Impact Benefit Agreements (IBAs) signed with the Nahʔą Dehé Dene Band, Łíídlıı Kúé First Nation and Acho Dene Koe First Nation to ensure benefits are provided to the Indigenous Governments and communities with which they work. CZC also has a Socio-Economic Agreement with the GNWT for benefits to the north through training, employment, and businesses. These agreements remain in effect and will be reassigned to Honey Badger Silver.

4.11 Project Risks and Uncertainties

There are no foreseeable risks or uncertainties regarding property ownership, surface/legal access, environmental issues, or social licence. Property ownership and land tenure is assured, permit requirements are in the process of being addressed in relation to the environment, and CZC has benefit agreements with the necessary Indigenous groups who are fully supportive of the development.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Accessibility

Year-round access to the mine site is provided by charter aircraft, generally from Fort St. John, BC, or Fort Simpson, Northwest Territories, both of which are serviced by scheduled commercial airlines. A 914 m gravel airstrip is located on the flood plain of Prairie Creek, approximately 1 km north of the mine site.

The Liard highway, which connects Fort Nelson, BC to Fort Simpson, Northwest Territories, is the closest major transportation route to the property. A 170 km long winter road from the Blackstone crossing on the Liard highway was constructed in 1980. During the winters of 1981 and 1982 the road was used to transport the bulk of the building materials, supplies, and equipment into the mine site, which enabled the construction of the extensive infrastructure that is currently in place. About 700 loads per season of material, plant, machinery, equipment, and supplies were transported during this period. A proposed new ASR will use much of the old winter road route with some key re-alignments, such as changing the eastern terminus to join with the Nahanni Butte access road which connects to the Liard Highway.

5.2 Climate

The climate in the general project area is sub-Arctic and is characterized by long, cold winters with moderate snowfall, and short but pleasant summers. A climate station was established immediately to the south of the mine site and measures precipitation, temperature, wind speed and wind direction. A mean annual temperature of minus 5.4 °C (high of 27.1 °C and low of minus 44.2 °C) was recorded in 2022 with annual rainfall of 326 mm.

Historically the site opens from May to November when water treatment of underground water is required. When construction commences and the ASR is completed, access to the Project site is planned to be year-round.

5.3 Local Resources and Infrastructure

5.3.1 Local Resources

The hamlet of Nahanni Butte is the closest settlement to the Project (90 km by air). It has an airstrip but is remote and can offer only a limited labour force. Fort Simpson and Fort Liard are the next closest Northwest Territories communities and have businesses that could be a part of the procurement process to provide services such as labour, some heavy equipment, and supplies. Fort Nelson, BC (340 km south of Project site) is located adjacent to both a railhead and the Alaska Highway and can provide additional support.

5.3.2 Infrastructure

5.3.2.1 Utilities

Electrical power on-site is currently provided by diesel-powered generators. There are several generators available to bring online, depending on demand, and include a CAT 3412 750 kilowatt (kW), Isuzu 150 kW and a John Deere 75 kW. A diesel storage tank farm is located on site and capable of storing up to 6.8 million litres of diesel fuel. Potable water is extracted from an on-site well. A sewage treatment plant was constructed but has not been commissioned.

5.3.2.2 Formerly Intended Tailings Impoundment Area

The (unused) tailings impoundment was designed by Golder Associates and was constructed in 1982 in conjunction with the surface construction and historical mine development activities.

The tailings impoundment was originally designed to store the fine fraction of the tailings following hydro-cyclone size separation, whereas the coarse fraction was to be pumped back to the underground mine as backfill.

Current plans are that the existing large pond, originally intended for tailings disposal, will be reconfigured, relined, and recertified to form the two-celled water storage pond described in Section 18 (see Figure 18-3). In its reconfigured role, Cell A will serve as the contingency tailings storage facility, as it will be designed to receive up to 50,000 tonnes of the whole tailings stream, while Cell B will serve as the water storage facility for the site.

5.3.2.3 Communications

All outside communications from the site are via satellite. On site, radios are used to link surface work crews; a Femco telephone system has been installed underground.

5.3.2.4 Existing Mine Buildings

Most of the mine site surface facilities were constructed in 1982, including a prefabricated administration building that contains office, mine dry, first aid and warehouse facilities, as well as a maintenance building and storage building. These buildings are in good condition. Trailer accommodations and kitchen facilities were built to support a 200-person construction crew, but these are in disrepair and are planned to be replaced.

5.3.2.5 Processing Plant

A processing plant, also constructed in 1982, consists of a crusher rated at that time to handle 1,360 t/d of material, and a grinding and flotation circuit to produce separate lead and zinc concentrates that is rated at 1,000 t/d. The flotation circuit is partially constructed with some pieces of equipment not yet installed. Two Larox filters were installed for concentrate filtration

and two conventional thickener tanks were constructed for dewatering the tailings slurry in preparation for the tailings backfill circuit that was never completed. Upon mine closure in 1983, the processing plant was incomplete.

A powerhouse, which contains four Cooper Bessemer 1.1 megawatt (MW) generators and switching facilities, was constructed but never operated.

5.4 Physiography

The Project is located in the Mackenzie Mountain Range with topography that varies in elevation from approximately 870 to 1,170 masl, consisting of low mountains with moderate to steep sides and intervening narrow valleys. The mine site area is located at an elevation of 870 masl. Valleys are well-incised, and the area is located within the Alpine forest-tundra section of the boreal forest, characterized by stunted fir and limited undergrowth. The trees that grow at the lower elevations give way to mossy open Alpine-type country at higher elevations.

6 HISTORY

6.1 Activities and Ownership – 1928 to 1970

The original discovery of mineralization on the Project was made by a local trapper in 1928, at what is now known as the Zone 5 showing, a mineralized vein exposed in the south bank of Prairie Creek. Mr. Poole Field staked the first Mineral claims, and in 1958 a limited mapping program was undertaken by Fort Reliance Minerals Limited. The claims lapsed in 1965 and were re-staked and subsequently conveyed to Cadillac Explorations in 1966. Cadillac Explorations also acquired a 73,892 ha regional Prospecting Permit.

Between 1966 and 1969, trenching was carried out on several mineralized zones and underground exploration was commenced in the Main Zone and Zones 7 and 8 as follow-up to trench results. Underground workings in Zone 7 consisted of a 280 m drive northward, collared approximately 325 vertical metres below the surface trenches. Similarly, in Zone 8, a 240 m long underground drive was collared in 1969 and driven south, opposite the Zone 7 portal, to undercut the surface vein showings exposed in the trench 300 m vertically above the tunnels. Mineralization was encountered in both drives but was of lower grade than results obtained in the Main Zone. Both Zone 7 and 8 portals have been blocked by sloughed debris, and the drives are now inaccessible.

Cadillac Explorations' Prospecting Permit expired in 1969, and 2,695 ha (210 claims) were selected by Cadillac Explorations and brought to lease. The property was optioned to Penarroya in 1970 and the then-existing underground development in the Main Zone was extended. Approximately 5,800 m of surface drilling and preliminary metallurgical testing were also carried out. Penarroya discontinued its work late in 1970, at which time Cadillac Explorations resumed full operation of the Project.

6.2 Activities and Ownership – 1971 to 1991

In 1975, Noranda Exploration Company Limited optioned the southern portion of the Project, and drilled eight holes, before dropping its option in the same year. Cadillac Explorations, however, continued to develop the Main Zone underground workings and in 1979 re-sampled the crosscuts. A winter road from Camsell Bend to the site was used in the mid-1970s to transport equipment and supplies.

An independent feasibility study was completed in 1980 for Cadillac Explorations by Kilborn Engineering Limited, the results of which prompted the decision to put the mine (then called Cadillac Mine) into production. In December of 1980, Procan Exploration Company Limited (Procan), a company associated with Herbert and Bunker Hunt of Texas, agreed to provide financing for construction, mine development and working capital necessary to attain the planned production of 907 t/d.

Between 1980 and 1982, extensive mine development took place. Cadillac Explorations acquired a 907 t/d mill and concentrator from Churchill Copper, which was dismantled and transported to the site. The mill and concentrator were erected, and a new camp was established. The winter

road connecting the Project to the newly established Liard highway was also constructed and over 700 loads of supplies were transported to site. Two more underground levels and extensive underground workings were subsequently developed. In 1982, the Project received a Class A Water Licence and, with a Land Use Permit also, was fully permitted for production. In early 1982, the price of silver collapsed. Construction activities continued until May 1982 when they were suspended due to lack of financing, which forced Cadillac Explorations into bankruptcy in May 1983, after a total of approximately C\$64 million (1982 value) had been expended on the Project. Thereafter, site maintenance was taken over by Procan, which acquired Cadillac Explorations' interest in the Project through bankruptcy proceedings in 1984.

6.3 Ownership Post – 1991

In 1991, Nanisivik acquired the Project from Procan. Pursuant to an option agreement dated August 23, 1991, Canadian Zinc (then known as San Andreas Resources Corporation), acquired a 60% interest in the Project from Nanisivik.

Subsequently, pursuant to a March 29, 1993 Asset Purchase Agreement that superseded the 1991 Option Agreement, Canadian Zinc acquired a 100% interest in the mineral properties and a 60% interest in the plant and equipment, subject to a 2% NSR royalty in favor of Procan. In January 2004, Canadian Zinc acquired all of Procan's (which had become Titan Pacific Resources Limited) interest in the plant and equipment, including the 2% NSR royalty, thereby securing a 100% interest in the Project.

In December 2022, RCF acquired NorZinc, the then parent company of Canadian Zinc by way of a Plan of Arrangement.

In April 2026, Honey Badger Silver acquired 100% of Canadian Zinc Corporation from RCF, thus securing 100% ownership of the property.

6.4 Historical Resource Estimates

Numerous historical resource estimates have been reported for the Main Zone deposits. The Main Zone in this report refers to Zones 1, 2, and 3. Initially these estimates were for the MQV only, but later incorporated the SMS and STK mineralization, as they were discovered. The chronology of historical resource estimates is shown in Table 6-1.

Table 6-1: Exploration History

Year	Company	Zone Estimated		
		Vein	Stratabound	Stockwork
1970	Behre Dolbear & Company for Pennarroya Canada	Yes	-	-
1972	James & Buffam	Yes	-	-
1980	Kilborn Engineering Limited	Yes	-	-

Year	Company	Zone Estimated		
		Vein	Stratabound	Stockwork
1983	Procan Exploration	Yes	-	-
1993	Cominco Engineering	Yes	Yes	
1995	Simons Mining Group	Yes	Yes	Yes
1998	MRDI Canada	Yes	Yes	Yes
2007	MineFill Service Inc.	Yes	Yes	Yes
2012	AMC Mining Consultants (Canada) Ltd.	Yes	Yes	Yes
2015 March	AMC Mining Consultants (Canada) Ltd.	Yes	Yes	Yes
2015 September	AMC Mining Consultants (Canada) Ltd.	Yes	Yes	Yes
2021	Global Mineral Resource Services	Yes	Yes	Yes

None of the estimates completed prior to 2007 was prepared in accordance with NI 43-101. The 2007 estimate prepared by MineFill was used in a PEA prepared by SNC Lavalin in 2011, and originally reported in the Stone DMR and Godden SJ 2007, Technical Report on the Prairie Creek Silver Mine, Northwest Territories October 12, 2007, prepared by MineFill Service Inc.

The most recent historical estimate was disclosed in the 2021 Preliminary Economic Assessment prepared for Canadian Zinc and filed on SEDAR. The historical estimate remains relevant and reliable, although commodity prices have changed considerably for some of the metals.

For the 2021 estimate, a single block model was created to encompass the three mineral domains: MQV, STK, and SMS. Grades were interpolated into the blocks using ordinary kriging (OK). For a block to have been classified as Measured, it was necessary that a minimum of 24 composites be located within the volume of the search ellipse with a maximum of two composites per hole, equivalent to 12 drillholes or channel samples. The MQV and STK domains contained Measured resources; in both, the Measured blocks immediately surrounded the underground development in which channel sampling was carried out. For a block to have been classified as Indicated, it was necessary that a minimum of 10 composites be located within the volume of the search ellipse, with a maximum of 2 composites/ hole or the equivalent of five drillholes. For a block to have been classified as Inferred, it was only necessary that a minimum of four composites be located within the volume of the search ellipse with a maximum of two composites/hole or the equivalent of two drillholes.

The 2021 resource estimate has been superseded by the estimate described in Section 14 of this Technical Report.

6.5 Production

There has been no production from the property, despite trial mining having been carried out in 1982. During the trial mining period, a mineralized material stockpile was created in the main yard near the mill and is estimated to include approximately 10,000 tonnes of material. While historical reports indicate this stockpile was mostly from shrinkage stope development, it has not been evaluated and, since it has been weathering for 40 years, has been given no value as a mineral resource.

7 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional Geology

The Project is located within a westward-thickening wedge of sedimentary rocks of mid-Proterozoic to mid-Jurassic age that were deposited along the paleo-continental margin of western North America.

The Project area is underlain by lower Paleozoic-age clastic sedimentary strata that were deposited in the Prairie Creek Embayment, an eastward incursion of the Selwyn Basin into the western edge of the Mackenzie Platform. Figure 7-1 shows the Prairie Creek regional geology.

During the Ordovician through to Devonian periods, the western edge of the platform represented the western margin of the North American continent, and during this time, shallow-water carbonates were deposited on the Mackenzie Platform while deep-water clastic sediments were contemporaneously deposited in the basin to the west. The Prairie Creek Embayment is interpreted (Morrow and Cook, 1987) to have developed as a half-graben controlled by a north-trending fault with down-drop to the west.

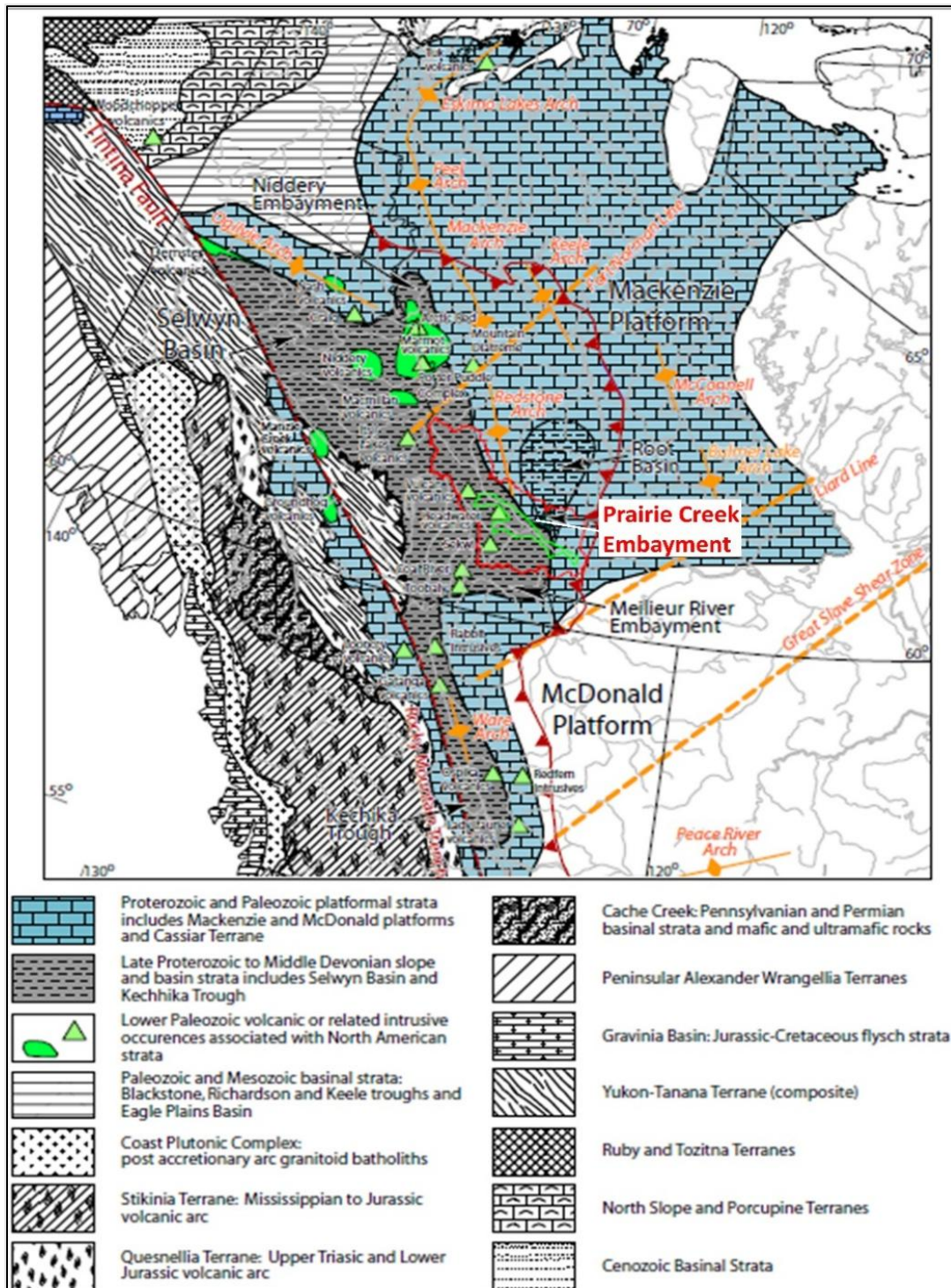
Sedimentation into the Prairie Creek Embayment ended in mid-Jurassic time when the eastward collision of an island-arc terrane led to imbrication and folding of the sedimentary succession and to the intrusion of widespread post-tectonic plutons of Cretaceous-age.

The structural style of deformation varies with lithology: thick, predominantly carbonate units form large structures whereas thin-bedded clastic units form repeated small folds and fault panels. These variations are evident in the Prairie Creek Embayment where three phases of deformation have occurred.

The earliest phase corresponds to regional north-south folding. These folds are cut by steep-dipping wrench faults that were subsequently reactivated as high-angle reverse faults. The reverse faults are post-dated by shallow, north-trending thrust faults that predominantly occur within the carbonate platform.

The present margins of the Prairie Creek Embayment are defined by the Tundra Thrust to the east and the Manetoe Thrust 20 km to the west. These thrusts juxtapose shallow-water shelf carbonates against deeper-water basin sedimentary rocks of the Embayment.

Figure 7-1: PC Silver Mine Regional Geology Plan



Source: Paradis (2007)

7.2 Project Geology

7.2.1 Stratigraphy

The lower Paleozoic strata exposed in the Project area are divisible into four major subdivisions that reflect abrupt changes in patterns of sedimentation related to the inception, growth, and filling of the Prairie Creek Embayment. In ascending stratigraphic order these subdivisions are: (1) Sunblood Platform, (2) Mount Kindle-Root River assemblage, (3) Prairie Creek Assemblage, and (4) Funeral-Headless assemblage.

The Sunblood Platform consists of shallow-water, argillaceous limestone and dolomite of the Sunblood Formation of middle Ordovician age. In the PC Silver Mine area, the Sunblood Formation is unconformably overlain by dolostones of the Whittaker Formation and, west of PC Silver Mine, the Sunblood Formation is conformably overlain by basal shales of the Road River Formation.

The Mount Kindle-Root River assemblage is comprised of the Whittaker, Road River, and Root River Formations of Late Ordovician to Devonian age. The Mount Kindle Formation, the shallow-water equivalent of the Whittaker, is not present in the PC Silver Mine area. The Whittaker Formation is divided into three members: (1) dark-grey, silty to sandy limestone of middle to upper Ordovician age (muOw1, Lower Whittaker), (2) fine-grained quartzite of middle to upper Ordovician age (muOw2, Middle Whittaker), and (3) laminated, dark-grey, fine-crystalline dolostone of upper Ordovician to Silurian age (OSW3, Upper Whittaker), that hosts sulphide mineralization at PC Silver Mine. The Silurian-Devonian age Road River Formation conformably overlies the Whittaker Formation and is comprised of graptolite-bearing shale and argillaceous dolostone and hosts sulphide mineralization as part of the PC Silver Mine deposit. The Silurian-age Root River Formation is comprised of light-grey, vuggy, micritic dolostone, and is interpreted to be the shallow-water equivalent of the Cadillac Explorations Formation.

The Upper Whittaker has been divided into seven lithological sub-units based on detailed information obtained from diamond drilling (Table 7-1). From stratigraphic bottom to top these sub-units are the Massive Dolomite (OSW3-1), Mottled Dolomite (OSW3-2), Lower Chert Nodular Dolomite (OSW3-3), Lower Spar (OSW3-4), Upper Chert Nodular Dolomite (OSW3-5), Upper Spar (OSW3-6), and Interbedded Chert-Dolomite (OSW3-7). The thickness of individual units varies broadly because contacts are generally gradational.

The Prairie Creek Assemblage of Silurian to Devonian age is variable in both lithology and thickness, which reflects the inception and growth of the Prairie Creek Embayment.

The assemblage is comprised of lower and upper Cadillac Explorations Formation phases. The lower phase marks the onset of the Embayment during Early Devonian time and is comprised of orange-weathering siltstone and carbonate debris flows. The upper Cadillac Explorations phase encompasses strata deposited in the Embayment throughout early Devonian time and comprises the Sombre and Arnica Formations as well as the pink shale member of the Cadillac Explorations Formation.

The Funeral-Headless assemblage of middle Devonian age records the disappearance of the Embayment and is comprised of shale, dolostone and limestone.

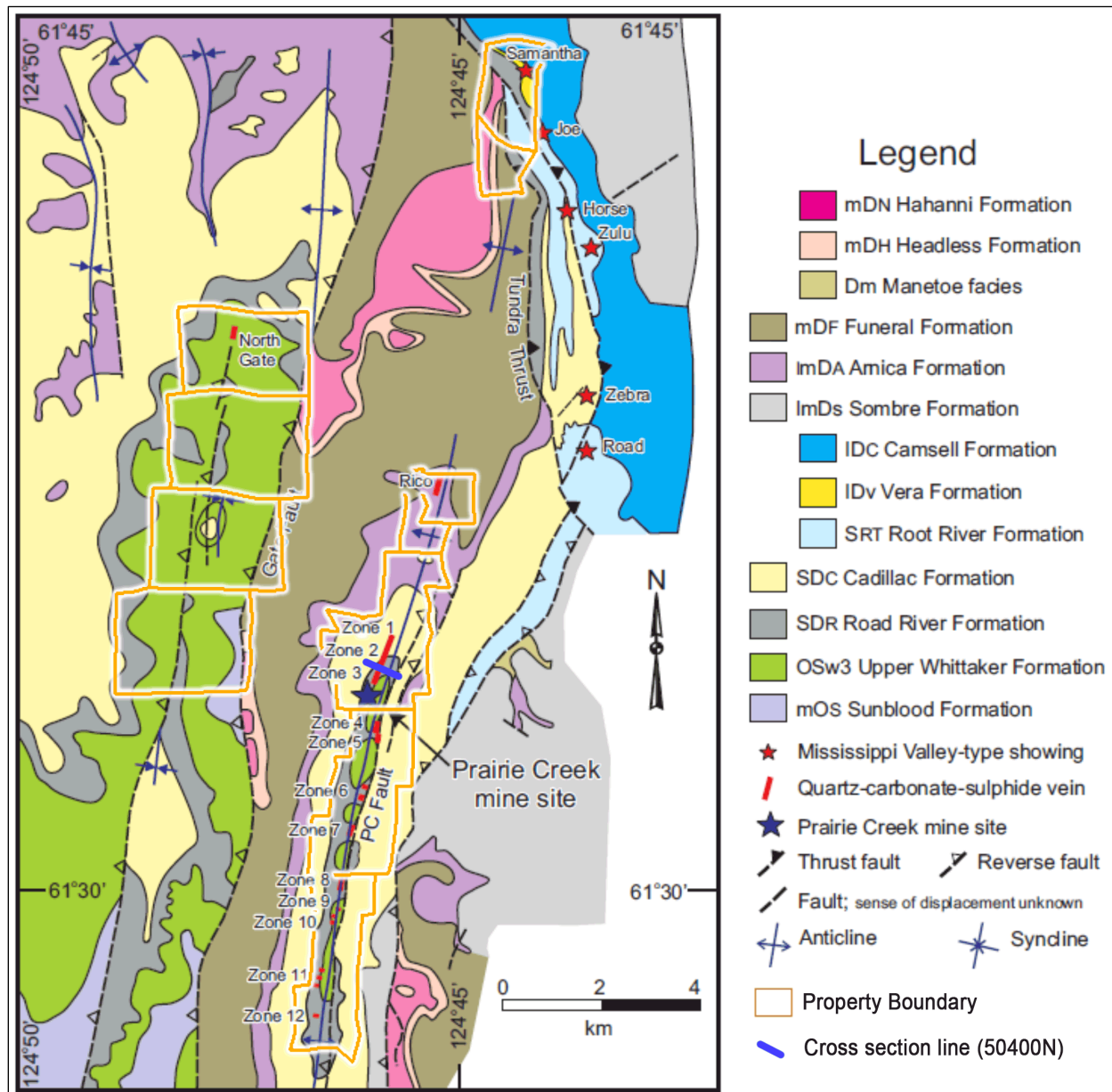
Table 7-1 summarizes PC Silver Mine's stratigraphy. Figure 7-2 is a simplified geological map of the Project area and Figure 7-3 is a representative cross-section through the main historical mine site area showing both stratigraphy and mineralization.

Table 7-1: Lithology Table

Formation	Code	Thickness (m)	Description
Arnica	ImDAb	200 to 250	Finely crystalline black nodular and banded cherty dolostone and limestone with white quartz-carbonate crackle veining.
Cadillac Explorations	SDC	300 to 350	Grey, thinly banded siltstone/shale with minor debris flow.
Road River	SDr	230 to 280	Mid-dark grey graphitic argillaceous bioclastic dolostone (graptolites common, occasional crinoids and brachiopods). Marker horizon near base – possible debris flow.
Upper Whittaker	OSW3-7	50 to 55	Interbedded chert-dolostone unit. Well-bedded, black to mid-grey cherts interbedded with dolostone. Chert content decreases with depth. Algal mat-type structures and possible dolomitized anhydrite towards base.
	OSW3-6	11 to 25	Upper Spar unit. Massive bioclastic, mid-grey, fine grained dolostone with white spar-filled cavities. Bioclastic material is fine grained and comminuted (crinoids, brachiopods).
	OSW3-5	55 to 100	Upper chert nodule-dolostone unit. Massive to poorly bedded weakly bioclastic, fine- to medium- grained dolostone. Mid-grey to black chert nodules.
	OSW3-4	9 to 24	Lower Spar unit (like the Upper Spar unit).
	OSW3-3	40 to 60	Lower Chert Nodule-dolostone unit (like Upper chert-nodule dolostone unit).
	OSW3-2	20 to 30	Mottled dolostone unit. Fine grained dolostone with spheroidal mottled texture and chert. Unit is host to SMS deposits. Disseminated fine-grained pyrite common.
	OSW3-1	20 to 30	Grey massive dolostone with minor chert nodules.
Middle Whittaker	muOW2	40 to 50	Grey gritty dolostone with some sand size grit units with greenish, shaley partings.
Lower Whittaker	muOW1	+50	Chert Nodule dolomite.

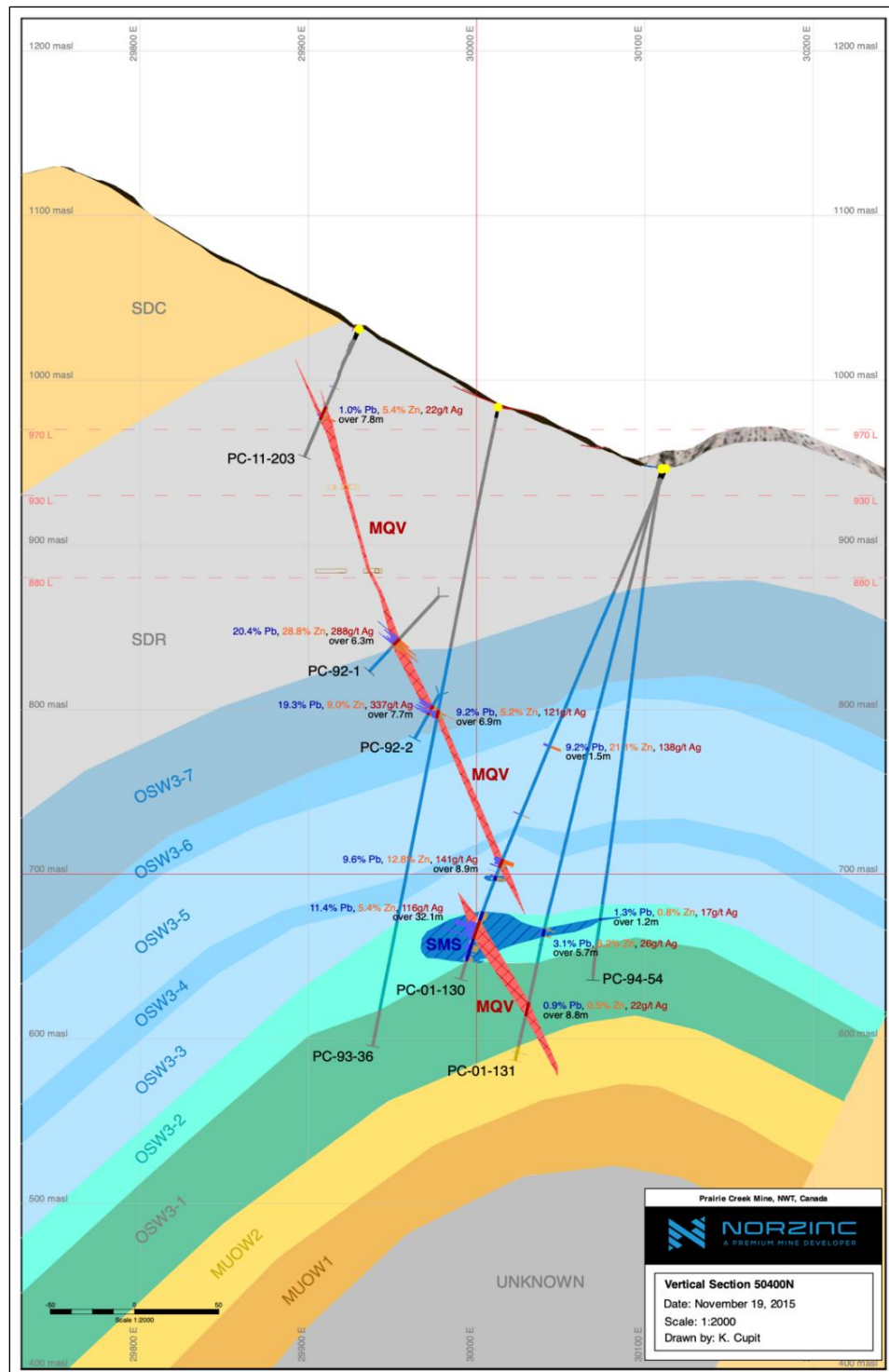
Source: Ausenco (2026)

Figure 7-2: Project Geology Plan



Source: Paradis (2007); modified by NorZinc (2023)

Figure 7-3: Simplified Vertical Cross Section of PC Silver Mine Main Zone



Source: NorZinc (2023)

7.2.2 Structure

In the immediate area of the Project, fault and fold axes trend north-south. The most significant fold is the gently, doubly plunging Prairie Creek Anticline (PCA), which is the locus of all of the immediate PC Silver Mine mineralization. Windows of Upper Whittaker and Road River Formation strata are exposed through the overlying Cadillac Explorations Formation along the axis of the PCA.

The PCA is bounded to the east by the Prairie Creek Fault and to the west by the Gate Fault. Both are west-dipping thrusts. The Prairie Creek Fault is up to 40 m thick and has a dextral displacement of approximately 1,500 m.

Leases in the northern part of the Project straddle the Tundra Thrust, which separates strata of the PCA to the west from platformal strata to the east (Figure 7-2). The platformal sediments are relatively undeformed and comprise a stratigraphic sequence starting with the Road River Formation that is overlain, from oldest to youngest, by the Root River, Camsell and Sombre Formations. Mississippi Valley type (MVT) mineralization is hosted in biohermal reefs of the Root River Formation, or facies equivalent.

In the historical Mine area, the continuation of the Tundra Thrust separates the PCA from the marginal platform, approximately 2 km east of the historical mine site. The platformal sequence in this area is dominated by a thick assemblage of Sombre Formation dolomites.

To the west of the historical Mine area, four contiguous Gate mining leases overlie lithological assemblages similar to those found in the Prairie Creek Assemblage (Figure 7-2). Grassroots exploration was carried out in this area for the presence of mineralization similar to that found in the historical mine area. The Whittaker and Road River Formations occur within the Gate Leases as relatively flat lying to gently dipping units and compared to the PCA, the prospective Whittaker Formation is more extensively exposed.

7.3 Mineralization

Exploration has located numerous base metal occurrences on the Project that can be grouped into four styles of mineralization:

- MQV mineralization is typically a 1 to 4 m wide polymetallic base metal sulphide vein, occurs in a north-trending, 16 km-long corridor. Vein showings were referred to historically as Zones 1 through 12, as shown in Figure 7-2. The Main Zone, which includes other styles of mineralization in addition to the MQV, is found in historical Zones 1, 2, and 3. Vein Zones 4 to 12 extend discontinuously for about 10 km to the south of the Main Zone. The Rico showing is located approximately 4 km north of the Main Zone;
- STK mineralization is associated with the MQV and does not represent a true stockwork but rather a series of tensional splays from the MQV. STK mineralization is exposed underground in the 883 mL, has been intersected in drillholes, and observed in surface showings;
- SMS mineralization is associated with several of the vein zones and occurs near the currently known lower limits of vein mineralization. Vein mineralization contains fragments of SMS

indicating that the deposition of SMS pre-dated vein formation. SMS mineralization is not exposed on surface or underground and is known only from drillholes; and

- MVT showings in the northern section of the Project are developed over a distance of approximately 10 km and from north to south are referred to as the Samantha, Joe, Horse, Zulu, Zebra, and Road showings (Figure 7-2).

7.3.1 MQV Mineralization

Vein-type mineralization occurs within cherty dolomites of the Ordovician-Silurian age Upper Whittaker Formation and shaly dolomites of the lower Road River Formation, along the axial plane of the PCA.

MQV type mineralization is comprised of massive to disseminated galena and sphalerite with lesser pyrite and tetrahedrite-tennantite in a quartz-carbonate-dolomite matrix. Oxidation is variably developed, yielding mainly cerussite (lead oxide) and smithsonite (zinc oxide); tetrahedrite-tennantite has undergone only minor oxidation. Silver is present in solid solution with tetrahedrite-tennantite and to a lesser extent with galena. Veins dip steeply to the east; widths generally vary from less than 0.1 m up to 5 m, with an average horizontal thickness of approximately 2.7 m.

The MQV is the most extensively developed of the known mineral zones. Underground development and diamond drilling have demonstrated the continuity of the MQV over a horizontal strike length of 2.3 km. The MQV trends approximately north south and dips between vertical and 40° east (average dip is 65° east). It remains open to the north and may continue for a further 4 km to the Rico showing. Diamond drilling has indicated continuity to a depth of at least 450 masl.

Mineralization is best developed in the more competent, brittle units of the Lower Road River and Whittaker Formations; graphitic shale in the mid and upper parts of the Road River Formation is less competent and contained veins are poorly developed. For example, at the end of 930 mL the MQV can be seen to dissipate into the middle Road River shales. Also, the vein does not appear to be well developed in the shales of the Cadillac Explorations Formation lying stratigraphically above the Road River Formation. Figure 7-4 shows the MQV exposed underground.

Preliminary structural evidence suggests that the various mineralized vein showings may be structurally linked as a series of en échelon segments comprising a single, but structurally complex, mineralized vein structure. An en échelon vein structure could offer a simple explanation for apparent offsets between the various vein showings.

Figure 7-4: MQV Exposed Underground



Source: NorZinc (2023)

7.3.2 STK Mineralization

Towards the end of the 930 mL at Crosscut 30, a series of narrow (average 0.3 m wide), massive sphalerite-galena-tennantite veins are developed at about 40° to the average trend of the MQV. These sub-vertical veins range in thickness from 0.1 m to 0.5 m, have no apparent alteration halo, and are separated from each other by unmineralized dolomite. The veins are locally offset and cut-off by fault planes and are difficult to correlate at the present level of information. This style of mineralization is referred to as STK, although it does not represent a true stockwork but rather a series of parallel splays off the MQV.

To date, STK-style mineralization has been located in the immediate area surrounding the Crosscut 30 exposure in the 930 mL workings, and through diamond drilling. Similar, but lesser-developed STK may be present along the initial entranceway of the 930 mL portal, considering the similar mineralization styles and strike direction of sulphide veinlets present. Furthermore, drillhole PC-95-125 intersected multiple occurrences of sulphides at the northernmost end of the defined MQV that are now interpreted to be potential STK candidates. These more speculative

occurrences could be related to the same structural controls and influences responsible for the STK occurrence near Crosscut 30. Figure 7-5 shows the STK mineralization.

Figure 7-5: STK Mineralization Showing Separate Distinct High-Grade Sub-Vertical Veins in 930 mL



Source: NorZinc (2021)

7.3.3 SMS Mineralization

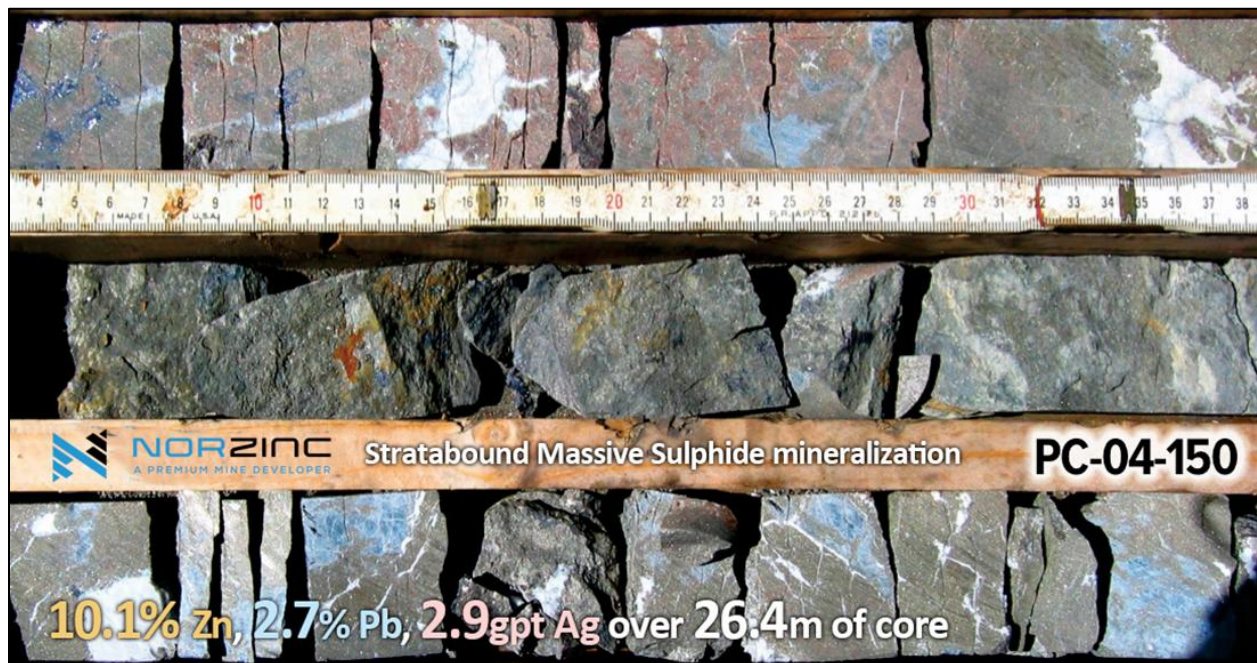
SMS mineralization was discovered by Canadian Zinc in 1992 while testing the depth extent of the MQV. To date, intermittent occurrences of SMS mineralization have been intersected in drillholes over a strike length of more than 800 m in the Main Zone, as well as in Zones 4, 5, and 6 (Figure 7-6).

Mineralization is generally fine-grained, banded to semi-massive, and comprises massive fine-grained sphalerite (typically pink, versus the brown sphalerite observed in other mineralization styles present at PC Silver Mine), coarse-grained galena and disseminated to massive pyrite. Silver is contained in solid solution within both galena and sphalerite, and the SMS mineralization

contains no tennantite-tetrahedrite, very little copper, half as much galena, but substantially more iron sulphide/pyrite than typical vein mineralization. Fragments of SMS mineralization occur in vein mineralization indicating that the SMS predates the veins.

The majority of SMS mineralization occurs within the Mottled Dolomite unit of the Whittaker Formation (OSW3-2, see Table 7-1), which the mineralization replaces without any significant alteration. SMS sulphides are developed close to both the vein system and the axis of the PCA and are presumably older than the vein mineralization (Figure 7-6). An apparent thickness of up to 28 m of SMS mineralization has been intersected in MQV drillholes, approximately 200 m below the 883 mL.

Figure 7-6: SMS Mineralization Showing Massive Sphalerite and Pyrite in Drill Core



Source: NorZinc (2021)

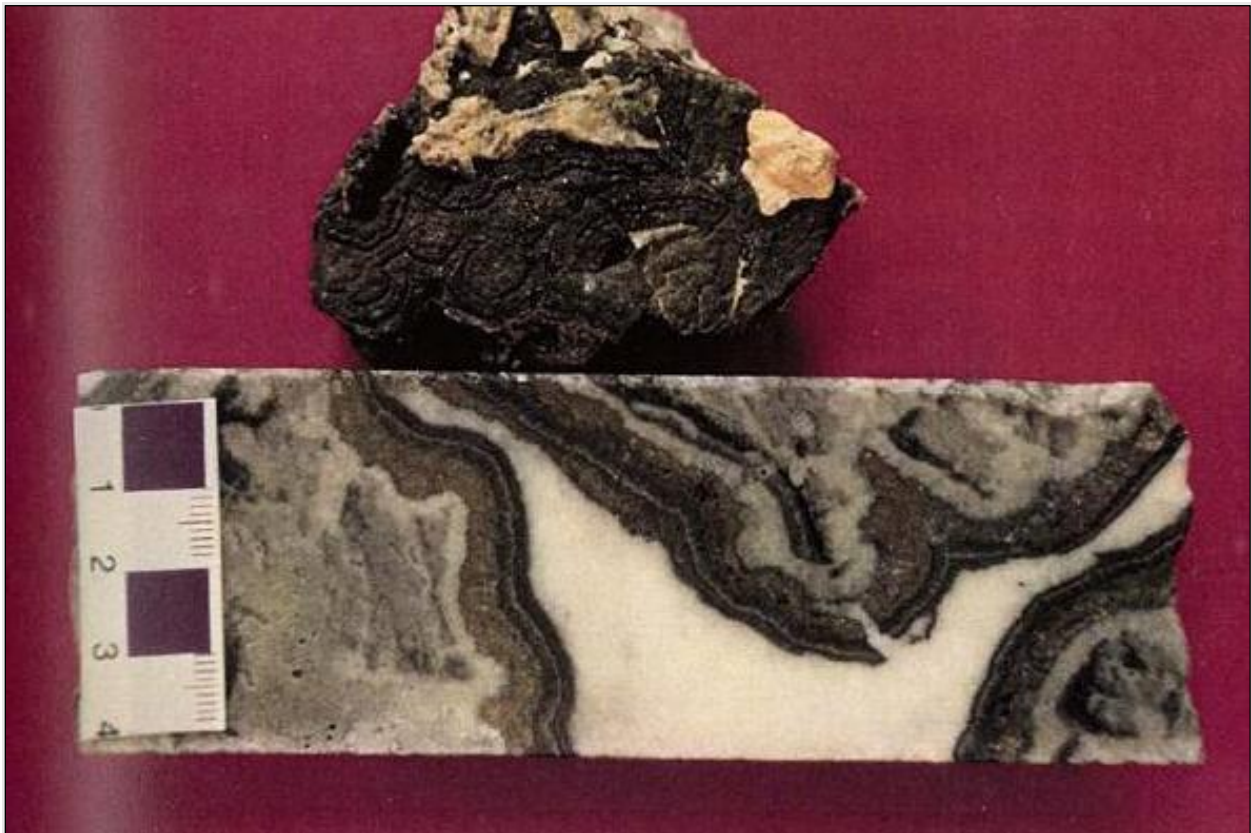
7.3.4 MVT Mineralization

MVT mineralization found on the northernmost mining leases is comprised of colloform rims of sphalerite, brassy pyrite-marcasite, and minor galena, with or without later dolomite infilling. The mineralization appears to occur discontinuously within coarse biohermal reefs of the Root River Formation, and always at approximately the same stratigraphic horizon. It appears to be classic MVT mineralization insofar as it occurs in open cavity-type settings.

Although the MVT showings appear to plot roughly on strike with the main zone MQV mineralization 10 km to the south, understanding of the MVT zone remains limited and is overshadowed by the economic potential of the other mineralization styles and occurrences in the main zone. As a result, the MVT mineralization showings remain underexplored.

An example of MVT style of mineralization showing the colloform nature of sulphide rimming fragments of local dolomite in chip and drill core from the Zebra showing is shown in Figure 7-7.

Figure 7-7: MVT Style Mineralization Showing Colloform Sphalerite Rimming Dolomite Fragment from Zebra Showing



Source: NorZinc (2021)

7.4 Comments on Geological Setting and Mineralization

Understanding of the nature of mineralization is supported by observations in underground exposures and demonstrated by diamond drilling across the Project.

8 DEPOSIT TYPES

8.1 Introduction

Four main styles of base metal mineralization have been identified on the Project:

- Hydrothermal Quartz Veins (MQV);
- STK;
- SMS; and
- MVT.

In the following generic descriptions, the STK style is discussed with the MQV because, as represented on the Project, it is directly related to, and a subset of, the MQV.

8.1.1 Hydrothermal Quartz Veins (MQV and STK)

The hydrothermal quartz veins of the MQV (and STK), are characteristic of polymetallic base metal veins, the salient characteristics of which follow (modified after Lefebure and Church, 1996):

- Tectonic settings: these veins occur in virtually all tectonic settings except oceanic, including continental margins, island arcs, continental volcanic and cratonic sequences;
- Depositional environment/geological setting: In sedimentary host rocks veins are emplaced along faults and fractures in sedimentary basins dominated by clastic rocks that have been deformed, metamorphosed, and intruded by igneous rocks. Veins postdate deformation and metamorphism;
- Age of mineralization: Proterozoic or younger;
- Host/associated rock types: these veins can occur in virtually any host. Most commonly the veins are hosted by thick sequences of clastic sedimentary, or intermediate to felsic volcanic rocks;
- Deposit form: typically, steeply dipping, narrow, tabular, or splayed veins. Commonly occurs as sets of parallel and offset veins. Individual veins vary from centimeters up to more than 3 m wide and can be followed from a few hundred to more than 1,000 m in length and depth;
- Texture/structure: compound veins with a complex paragenetic sequence are common. A wide variety of textures exist, including cockade texture, colloform banding and crustiform textures, and are locally drusy. Veins may grade into broad zones of stockwork or breccia.

Coarse-grained sulphides as patches and pods, and fine-grained disseminations are confined to veins;

- Mineralization: (principal and subordinate): galena, sphalerite, tetrahedrite-tennantite, other sulphosalts including pyrargyrite, stephanite, bournonite and acanthite, native silver, chalcopryite, pyrite, arsenopyrite, and stibnite. Silver minerals often occur as inclusions in galena. Native gold and electrum are in some deposits. Rhythmic compositional banding is sometimes present in sphalerite. Some veins contain more chalcopryite and gold at depth and gold grades are normally low for the amount of sulphides present;
- Gangue mineralogy: (principal and subordinate): in sedimentary host rocks: carbonates (most commonly siderite with minor dolomite, ankerite and calcite), quartz, barite, fluorite, magnetite, and bitumen;
- Alteration mineralogy: macroscopic wall rock alteration is typically limited in extent (measured in metres or less). The metasediments typically display sericitization, silicification, and pyritization. Thin veining of siderite or ankerite may be locally developed adjacent to veins;
- Weathering: galena and sphalerite weather to secondary lead and zinc carbonates and lead sulphate. In some deposits supergene enrichment has produced native and horn silver;
- Mineralization controls: regional faults, fault sets, and fractures are an important mineralization control; however, veins are typically associated with second order structures. In igneous rocks the faults may relate to volcanic centres. Significant deposits restricted to competent lithologies. Dikes are often emplaced along the same faults and in some camps are believed to be roughly contemporaneous with mineralization. Some polymetallic veins are found surrounding intrusions with porphyry deposits or prospects; and
- Genetic models: historically these veins have been considered to result from differentiation of magma with the development of a volatile fluid phase that escaped along faults to form the veins. More recently, researchers have preferred to invoke mixing of cooler, upper crustal hydrothermal, or meteoric waters with rising fluids that could be metamorphic, groundwater heated by an intrusion, or expelled directly from a differentiating magma.

8.1.2 Stratabound SMS Mineralization

The Irish carbonate-hosted, lead-zinc deposits (e.g., Lisheen, Galmoy, and Silvermines) may be the most appropriate analogy for the SMS. A brief description of this class of deposit follows (modified after Hoy 1996).

- Tectonic setting: platformal sequences on continental margins which commonly overlie deformed and metamorphosed continental crustal rocks;
- Depositional environment/geological setting: adjacent to normal growth faults in transgressive, shallow marine platformal carbonates. Also, commonly localized near basin margins;
- Age of mineralization: known deposits are believed to be Paleozoic in age and younger than their host rocks. Irish deposits are hosted by Lower Carboniferous rocks;

- Host/associated rock types: hosted by thick, non-argillaceous carbonate rocks. These are commonly the lowest pure carbonates in the stratigraphic succession. They comprise micritic and oolitic beds, and fine-grained calcarenites in calcareous shale, sandstone, and calcarenite succession. Underlying rocks include sandstones or argillaceous calcarenites and shales. Iron formations, comprising interlayered hematite, chert, and limestone, may occur as distal facies to some deposits;
- Deposit form: deposits are typically wedge shaped, ranging from over 30 m thick adjacent to, or along growth faults, to 1 to 2 cm bands of massive sulphides at the periphery of lenses. Economic mineralization rarely extends more than 200 m from the faults. Large deposits comprise individual or stacked sulphide lenses that are roughly concordant with bedding. In detail, however, most lenses cut host stratigraphy at low angles. Contacts are sharp to gradational. Deformed deposits are typically elongate within and parallel to the hinges of tight folds;
- Texture/structure: sulphide lenses are massive to occasionally well-layered. Typically, massive sulphides adjacent to faults grade outward into veinlet-controlled or disseminated sulphides. Colloform sphalerite and pyrite textures occur locally. Breccias are common with sulphides forming the matrix to carbonate (or as clasts). Sphalerite-galena veins, locally brecciated, commonly cut massive sulphides. Rarely, thin laminated, graded, and cross-bedded sulphides, with framboidal pyrite, occur above more massive sulphide lenses;
- Mineralization mineralogy: (principal and subordinate): sphalerite, galena; barite, chalcopryrite, pyrrhotite, tennantite, sulphosalts, tetrahedrite, and chalcopryrite;
- Gangue mineralogy: (principal and subordinate): dolomite, calcite, quartz, pyrite, marcasite; siderite, barite, hematite, magnetite; at higher metamorphic grades, olivine, diopside, tremolite, wollastonite, and garnet;
- Alteration mineralogy: extensive early dolomitization forms an envelope around most deposits which extends tens of metres beyond the sulphides. Dolomitization associated with mineralization is generally fine grained, commonly iron-rich, and locally brecciated and less well banded than limestone. Manganese halos occur around some deposits; silicification is local and uncommon. Iron occurs in more distal formations;
- Weathering: gossan minerals include limonite, cerussite, anglesite, smithsonite, hemimorphite, and pyromorphite;
- Mineralization controls: deposits are restricted to relatively pure, shallow-marine carbonates. Regional basement structures and, locally, growth faults are important. Orebodies may be more common at fault intersections. Proximity to carbonate bank margins may be a regional control in some districts;
- Genetic model: Two models are commonly proposed;
- Syngenetic seafloor deposition: evidence includes stratiform geometry of some deposits, occurrence together of bedded and clastic sulphides, sedimentary textures in sulphides, and, where determined, similar ages for mineralization and host rocks; and

- Diagenetic to epigenetic replacement: replacement and open-space filling textures, lack of laminated sulphides in most deposits, alteration and mineralization above sulphide lenses, and lack of seafloor oxidation.

8.1.3 MVT Mineralization

Salient characteristics of MVT mineral deposits are presented below (modified after Aldrick and Sangster, 2005):

- Tectonic settings: most commonly stable interior cratonic platform or continental shelf. Some deposits are incorporated in foreland thrust belts;
- Depositional environment/geological setting: host rocks form in shallow water, particularly tidal and subtidal marine environments. Reef complexes may be developed on or near paleo-topographic basement highs. The majority of deposits are found around the margins of deep-water shale basins. Some are located within or near rifts (Nanisivik, Alpine district);
- Age of mineralization: Proterozoic to Tertiary, with two peaks in Devonian to Permian and Cretaceous to Eocene time. Dating mineralization has confirmed the epigenetic character of these deposits. The difference between host rock age and mineralization age varies from district to district;
- Host/associated rock types: host rocks are most commonly dolostone, limestone, or dolomitized limestone. Locally hosted in sandstone, conglomerate or calcareous shale;
- Deposit form: highly irregular. may be concordant as planar, braided or linear replacement bodies. May be discordant in roughly cylindrical collapse breccias. Individual mineralized bodies range from a few tens to a few hundreds of metres in the two dimensions parallel with bedding. Perpendicular to bedding, dimensions are usually a few tens of metres. deposits tend to be interconnected thereby blurring deposit boundaries;
- Texture/structure: most commonly as sulphide cement to chaotic collapse breccia. Sulphide minerals may be disseminated between breccia fragments, deposited as layers atop fragments (snow-on-roof), or completely filling the intra-fragment space. Sphalerite commonly displays banding, either as colloform cement or as detrital layers (internal sediments) between host-rock fragments. Sulphide stalactites are abundant in some deposits. Both extremely fine-grained and extremely coarse-grained textured sulphide minerals may be found in the same deposit. Precipitation is usually in the order pyrite (marcasite) → sphalerite → galena;
- Mineralization mineralogy: (principal and subordinate): galena, sphalerite, barite, and fluorite. Some mineralization contains up to 30 ppm Ag. Although some MVT districts display metal zoning, this is not a common feature. The Southeast Missouri district and small portions of the Upper Mississippi Valley district are unusual in containing significant amounts of Ni-, Co- and Cu-sulphides;
- Gangue mineralogy: (principal and subordinate): dolomite (can be pinkish), pyrite, marcasite, quartz, calcite, gypsum;

- Alteration mineralogy: extensive fine crystalline dolostone may occur regionally, whereas coarse crystalline dolomite is more common close to mineralization. extensive carbonate dissolution results in deposition of insoluble residual components as internal sediments. Silicification (jasperoid) is closely associated with mineralized bodies in the Tri-State and northern Arkansas districts. Authigenic clays composed of illite, chlorite, muscovite, dickite, and or kaolinite accumulate in vugs; minor authigenic feldspar (adularia);
- Weathering: extensive development of smithsonite, hydrozincite, willemite, and hemimorphite, especially in non-glaciated regions (including upstanding hills or monadnocks). Large accumulations of secondary zinc minerals can be mined. Galena is usually much more resistant to weathering than sphalerite. Iron-rich gossans are not normally well-developed, even over pyrite-rich deposits;
- Mineralization controls: any porous unit may host mineralization. Porosity may be primary (rare) or secondary. Dissolution collapse breccias are the most common host although fault breccias, permeable reefs, and slump breccias may also be mineralized. Dissolution collapse breccias may form through action of meteoric waters or hydrothermal fluids. Underlying aquifers may be porous sandstone or limestone aquifers; the limestones may show thinning due to solution by ore-bearing fluids; and
- Genetic models: deposits are obviously epigenetic, having been emplaced after host rock lithification. mineralization-hosting breccias are considered to have resulted from dissolution of more soluble sedimentary units, followed by collapse of overlying beds. The major mineralizing processes appear to have been open-space filling between breccia fragments, and replacement of fragments or wall rock. The relative importance of these two processes varies widely among, and within, deposits. Fluid inclusion data show that these deposits formed from warm (75° to 200°C), saline, aqueous solutions are similar in composition to oil-field brines. Brine movement out of sedimentary basins, through aquifers or faults, to the hosting structures is the most widely accepted mode of formation.

Two main processes have been proposed to move mineralizing solutions out of basin clastics and into carbonates:

- 1) Compaction-driven fluid flow is generated by over-pressuring of subsurface aquifers by rapid sedimentation, followed by rapid release of basinal fluids; and
- 2) Gravity-driven fluid flow flushes subsurface brines by artesian groundwater flow from recharge areas in elevated regions of a foreland basin, to discharge areas in regions of lower elevation.

In addition to fluid transport, three geochemical mechanisms have been proposed to account for chemical transport and deposition of mineralization constituents:

- Mixing: base metals are transported by fluids of low sulphur content. Precipitation is affected by mixing fluids containing hydrogen sulphide; replacement of diagenetic iron sulphides; and/or reaction with sulphur released by thermal degradation of organic compounds;
- Sulphate reduction: base metals are transported together with sulphate in the same solution. Precipitation is the result of reduction of sulphate by reaction with organic matter or methane; and

- Reduced sulphur: base metals are transported together with reduced sulphur. Precipitation is brought about by change in pH, dilution, and/or cooling.

8.2 Comments on Deposit Type

MQV, STK, and SMS styles of mineralization all occur in the main zone, but the MQV style mineralization is of primary economic significance owing to appreciable grades of silver, zinc and lead compared to the other mineralization styles, as well as to overall estimated tonnages. Because of this, and the fact that the deposit remains open to the north even after decades of exploration in the region, all recent exploration efforts have centered around the main mineralized MQV zone.

9 EXPLORATION

9.1 Introduction

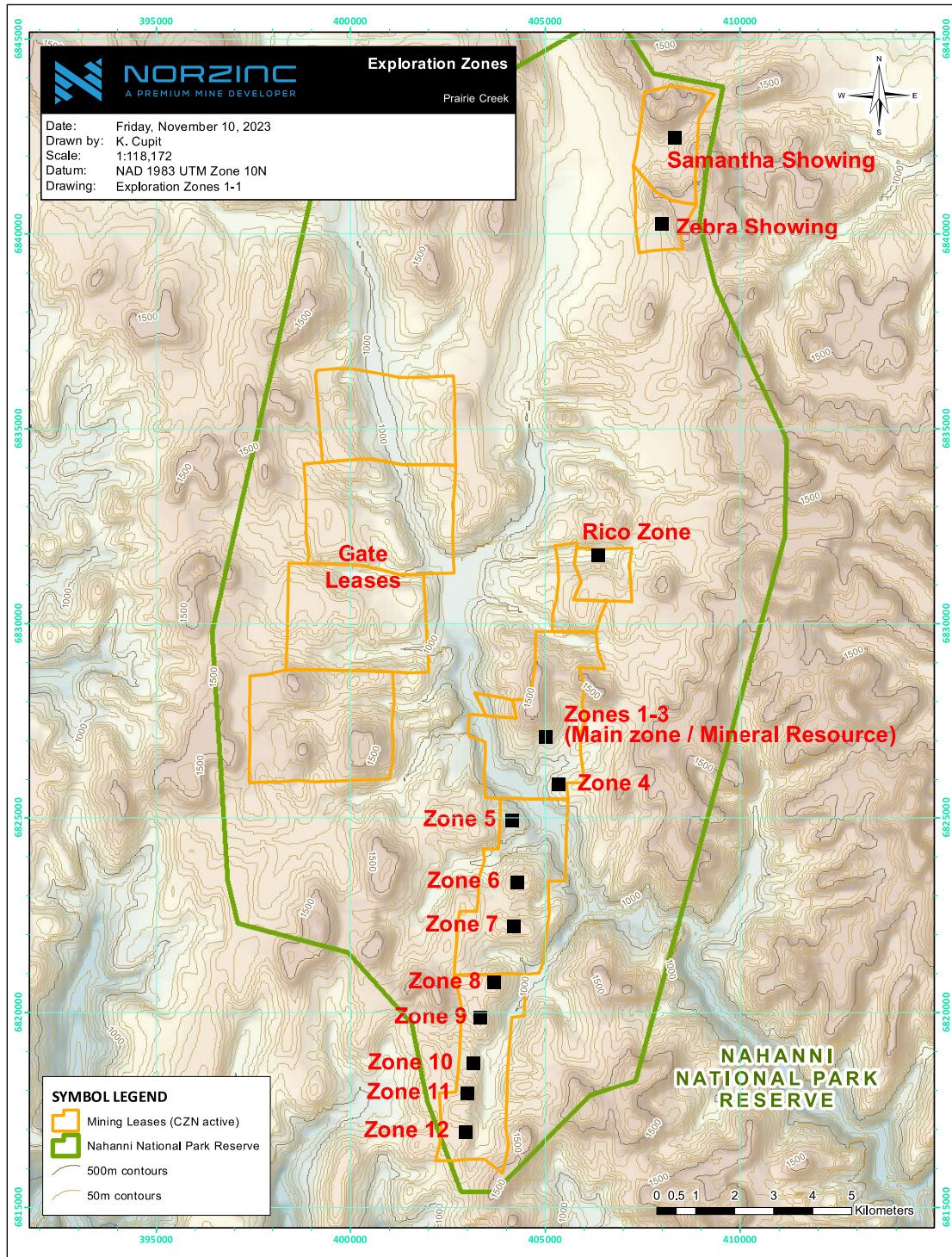
Honey Badger Silver has not done any exploration on the Project. The following is a summary of exploration completed by previous owners.

After the initial discovery in 1928 of a high-grade base metal vein exposed on the south side of PC Silver Mine, the focus of early exploration took place to the south of the historical mine site area, on large, well-exposed, surface vein base metal showings. It was not until the 1960s that significant mineralization was located in the Main Zone which shifted the focus of exploration to north of the mine site area, where the majority of exploration has occurred since.

Historically, exploration was focused on what is now termed the MQV, with secondary targets being STK mineralization and SMS mineralization. The known MVT showings occurring in a more remote part of the Project report lower grades than either vein or strata bound mineralization, and as such they have not been the focus of significant exploration to date.

Figure 9-1 shows the various zones referenced in this section and elsewhere in this report. Table 9-1 summarizes exploration work completed by Canadian Zinc since 1992. Drilling is further discussed in Section 10.

Figure 9-1: Summary of Exploration Work, 1992 to 2022



Source: NorZinc (2023)

Table 9-1: Summary of Exploration Work, 1992 to 2022

Year	No of holes	Metres	Highlights
1992	22	6,322	Discovery of previously unknown SMS mineralization by diamond drilling. SMS Discovery hole (PC-92-008) ran 10.60% Zn, 5.29% Pb, 44.37 g/t Ag, over 28.40 m.
1993	31	8,432	Tested for further SMS Mineralization. UTEM survey. Extended MQV by intersecting 18 m of vein 170 m below workings. Trench samples from Rico showing, in north showed grades of 18% Zn, 35% Pb, 242 g/t Ag in a vertical mineralized. Geological mapping in north claims (Sam).
1994	31	11,113	Extension of Main Zone, more SMS lenses in Zone 5, regional mapping. Rico Zone and Zebra showing (MVT) trenching, IP Ground Geophysics.
1995	36	10,082	Minor trenching and surface sampling.
1997	-	-	Channel sampling of previously un-sampled underground drift development.
1999	-	-	Gate Claims 1 to 4 were staked and geological mapping, soil and rock sampling, was carried out for geochemical analysis based on a large surface grid. Discovery of a mineralized vein in outcrop on Gate 1.
2001	5	1,711	Diamond drilling program designed both to increase confidence in 1998 resource estimate and to identify new high-grade areas. Possibility of high-grade shoots recognized.
2004	27	5,944	MQV drilling which intersected significant mineralization. Step out on the vein hit narrow but high-grade intersections. SMS exploration outside Main Area.
2005	-	-	Rehabilitation of underground workings, chip sampling of MQV underground.
2006	19	2,393	Phase 1 driving of decline tunnel and U/G drilling commences on MQV. Channel and round sampling. Drilling of Zone 8 mineralization investigated.
2007	53	11,141	Phase 1 U/G program confirms vein grades. Decline extended, Phase 2 drilling. Gate claims drilling and Zone 8, 9 and 11 show poor results.
2010	4	2,694	Deep drilling in Casket Creek (for MQV) and proximal to resource drilling.
2011	30	5,926	Deep drilling in Casket Creek (wedging) and proximal to resource drilling.
2012	11	5,628	Deep drilling in Casket Creek and proximal to resource drilling, Geophysical Gravity and EM surveys, LIDAR survey of property. Prospecting of full 16 km strike-length of MQV.
2013	5	1,472	Deep drilling and proximal to resource drilling, silt sampling.
2015	21	5,548	Underground drilling - MQV and STK infill and extension, channel samples taken.
2020	2	1,130	Exploration for Resources, MQV and STK intercepts.
2021	1	736	Exploration for Resources, MQV and STK intercepts.

Year	No of holes	Metres	Highlights
2022	24	5,928	Metallurgical and geotechnical drilling into existing mine area (not exploration).
Total	323	86,383	-

Source: Ausenco (2026)

9.2 Pits, Trenches, and Bulk Samples

In 1997, 231 channel samples were collected from 294 m of previously un-sampled MQV on the 883 mL and 930 mL. These samples gave a weighted average grade within vein limits of 17.2% Zn, 16.0% Pb, 330 g/t Ag, 0.8% Cu over a weighted average true width of 1.78 m.

This program brought the total of verifiable channel samples from Main Zone workings to 1,072, inclusive of channel samples collected by Cadillac Mines prior to 1982. The channel samples together form 393 composites, comprising 14 channel samples from 970 mL, 273 channel samples from 930 mL, and 106 channel samples from 883 mL.

In 2006, access to the new decline ramp was provided by the new Crosscut 883-07 that was driven as part of the 2006 underground exploration program. The MQV, with a true thickness of 6.5 m, was intersected about 12 m from the crosscut collar; and the walls of a 10 m intersection were channel sampled.

To obtain an overall grade comparison and dilution test, samples were also taken from each of the rounds excavated through the MQV intersection, including footwall and hanging wall material. After remixing the material twice, an estimated 20 kg of representative material was collected from each round, which was subsequently crushed on site to less than 1 cm in size, split into 2 kg samples and forwarded to the assay laboratory for analysis. Table 9-2 shows the weighted average grades of the rounds excavated in MQV.

Table 9-2: Weighted Average Grade of the Rounds in MQV

	Zinc (%)	Lead (%)	Silver (g/t)	Copper (%)
883-07 Rounds	19.0	16.4	250	0.5
Channels (excludes one outlier)	20.6	15.4	302	0.7

Source: Ausenco (2026)

In 2015, Canadian Zinc collected 22 channel samples comprised of 50 individual samples (63.6 aggregate metres) on the 930 mL to assess STK mineralization. The weighted average grade of all 50 samples is 8.3% Pb, 18.9% Zn, and 178 g/t Ag. Half of these samples were collected along

the strike of a mineralized STK vein exposed in the 930-Northwest Drift. The average grade of those samples is 9% Pb, 22.9% Zn and 223 g/t Ag.

No documentation was seen describing the channel sampling that has also been called chip sampling in some reports.

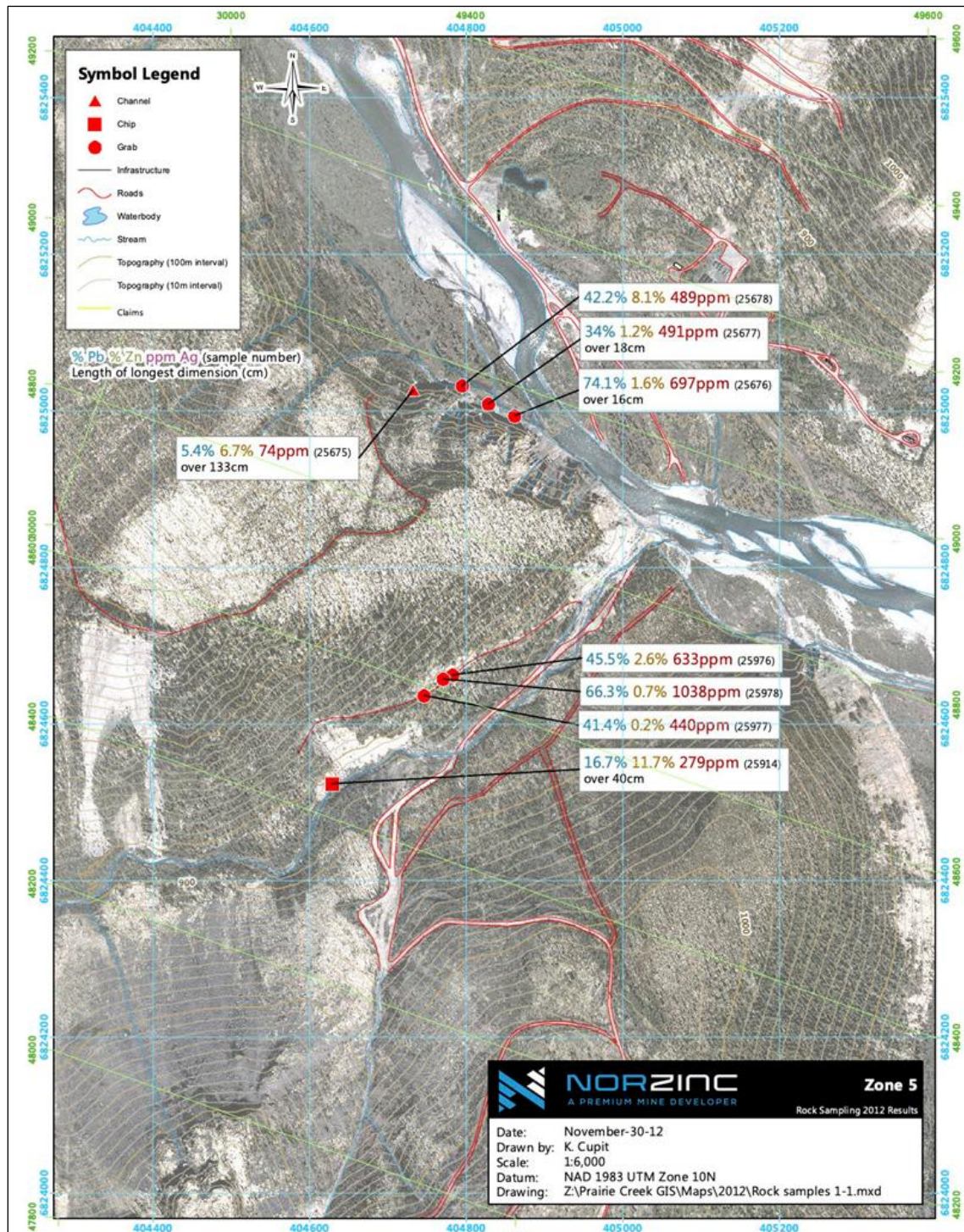
9.3 Grab Samples

Gate Mining Leases 1 to 4 were originally staked as claims in 1999 and converted to mining leases in 2008. During 2001, a small exploration program comprising geological mapping and soil and rock sampling was carried out over areas underlain by Whittaker Formation strata. This work resulted in the discovery of a vein in outcrop from which select grab samples contained grades similar to those previously established for the MQV: 820 g/t Ag, 3.5% Cu, 16% Pb, and 10% Zn. A large, 1,000 parts per million (ppm) zinc-in-soil anomaly was also located over favorable geology on the Gate 3 Mining Lease. Grab samples are selective and may not be representative of the true overall grade of the mineralization.

During 2007, Canadian Zinc carried out a helicopter-supported diamond drill program to test the soil anomalies within the Gate group and Zones 8, 9, and 11. This program returned very few significant mineral intersections.

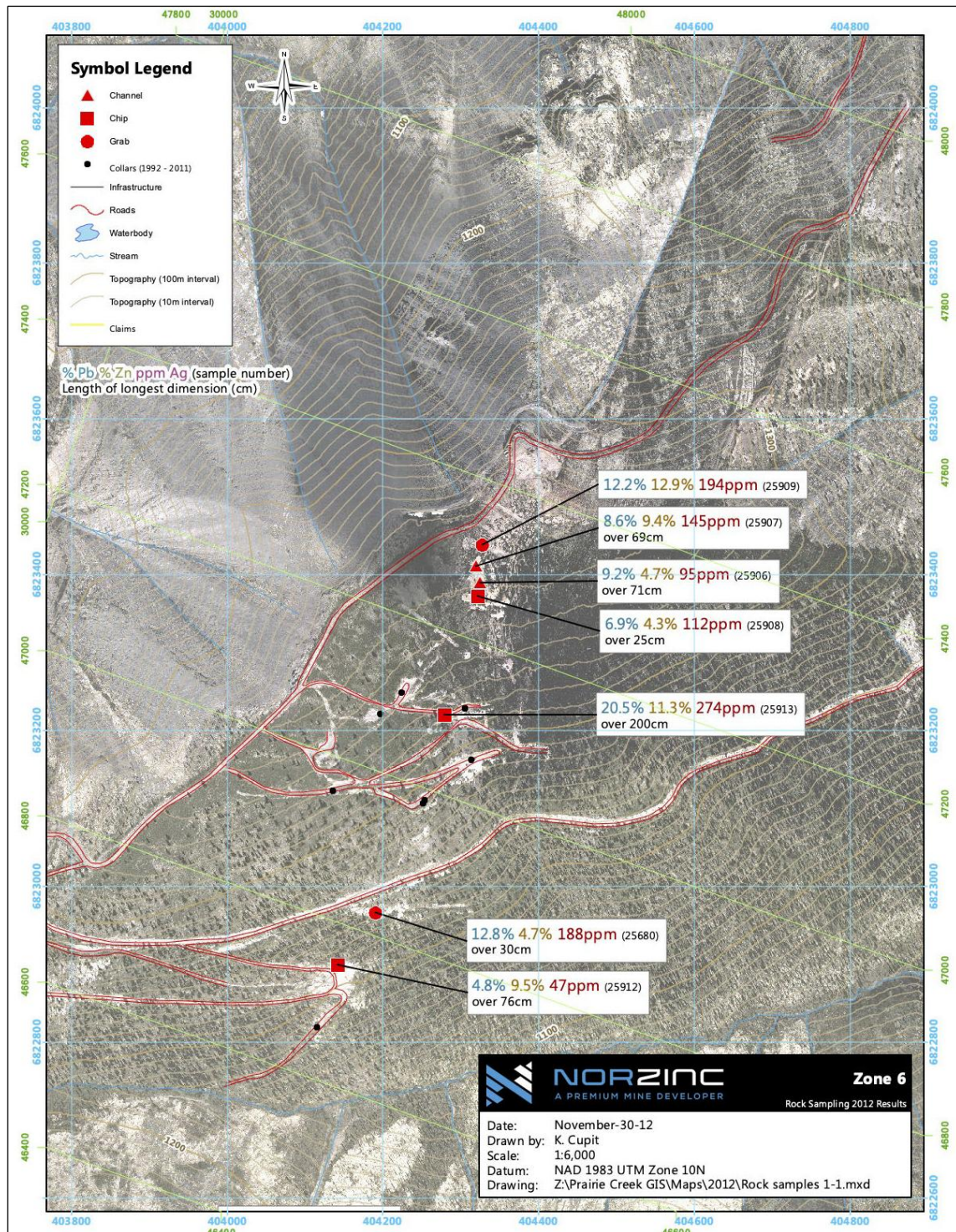
In 2012, Canadian Zinc conducted prospecting traverses throughout the contiguous MQV zones, with the intention to validate past sampling results and to identify potential new MQV targets with grab and field channel samples. The QP did not have access to the sampling methods used for the grab and field channel samples. Figure 9-2 through Figure 9-7 show channel and grab samples from the southern zones as part of the 2012 prospecting program.

Figure 9-2: Exploration Highlights from Zone 5



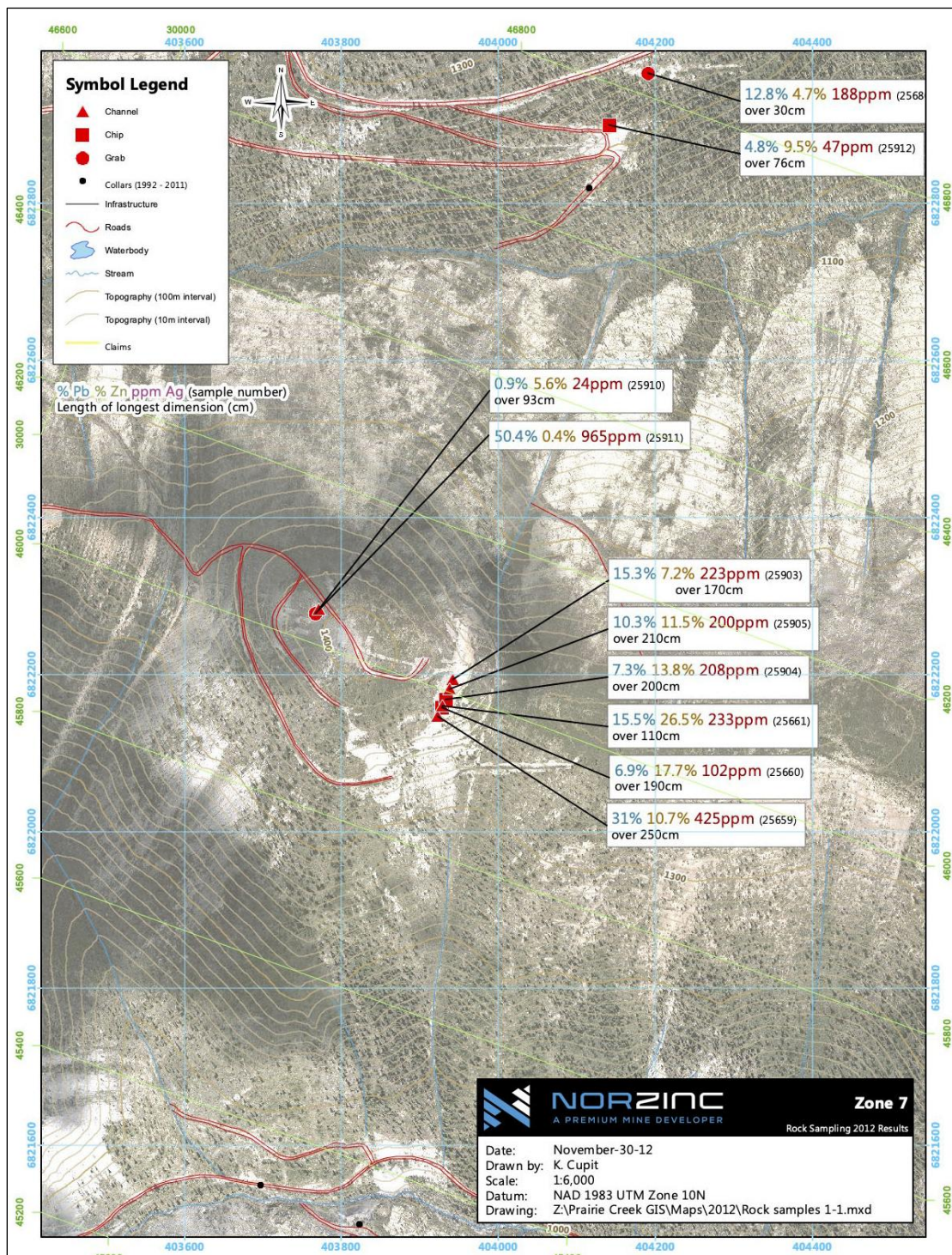
Source: NorZinc (2012)

Figure 9-3: Exploration Highlights from Zone 6



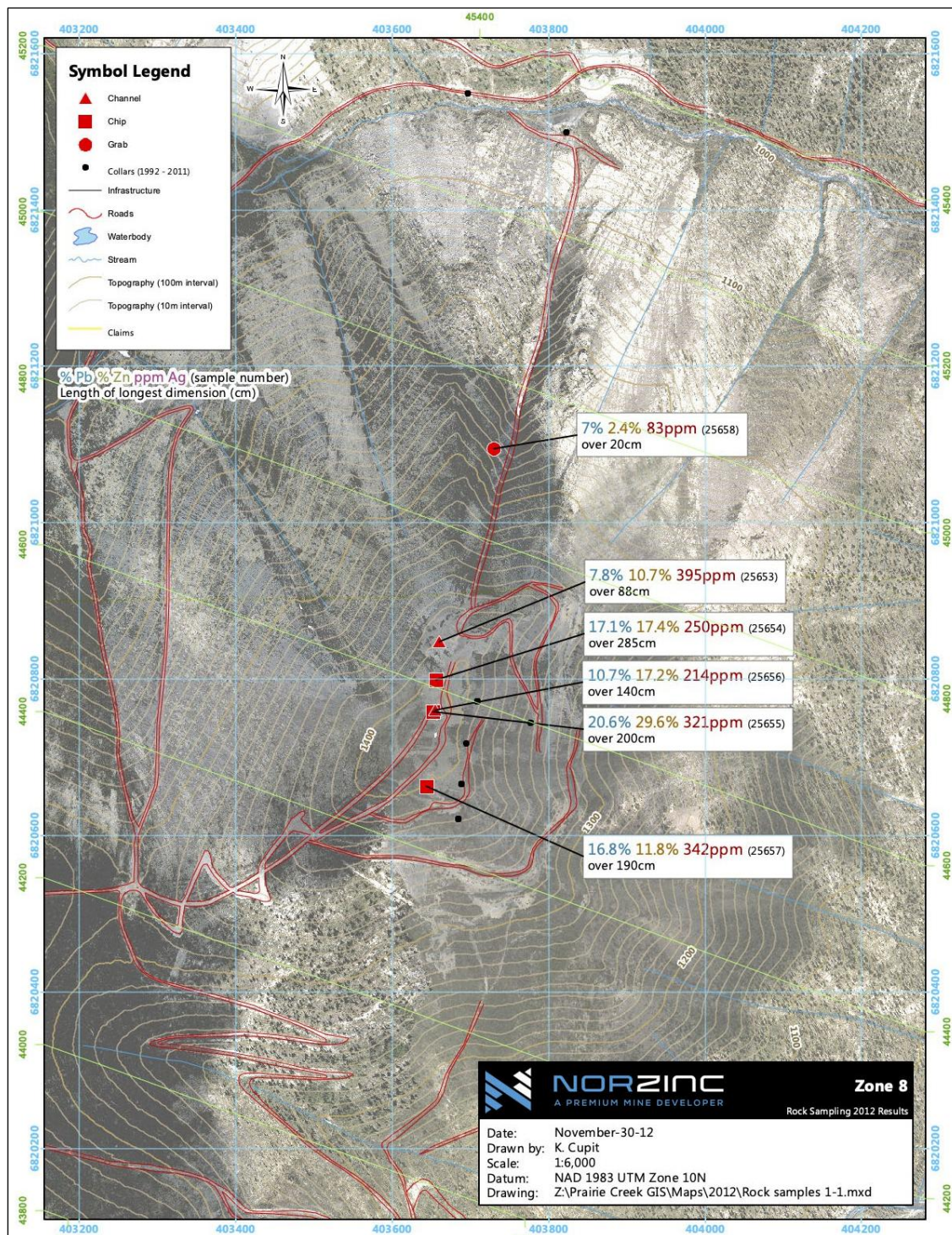
Source: NorZinc (2012)

Figure 9-4: Exploration Highlights from Zone 7



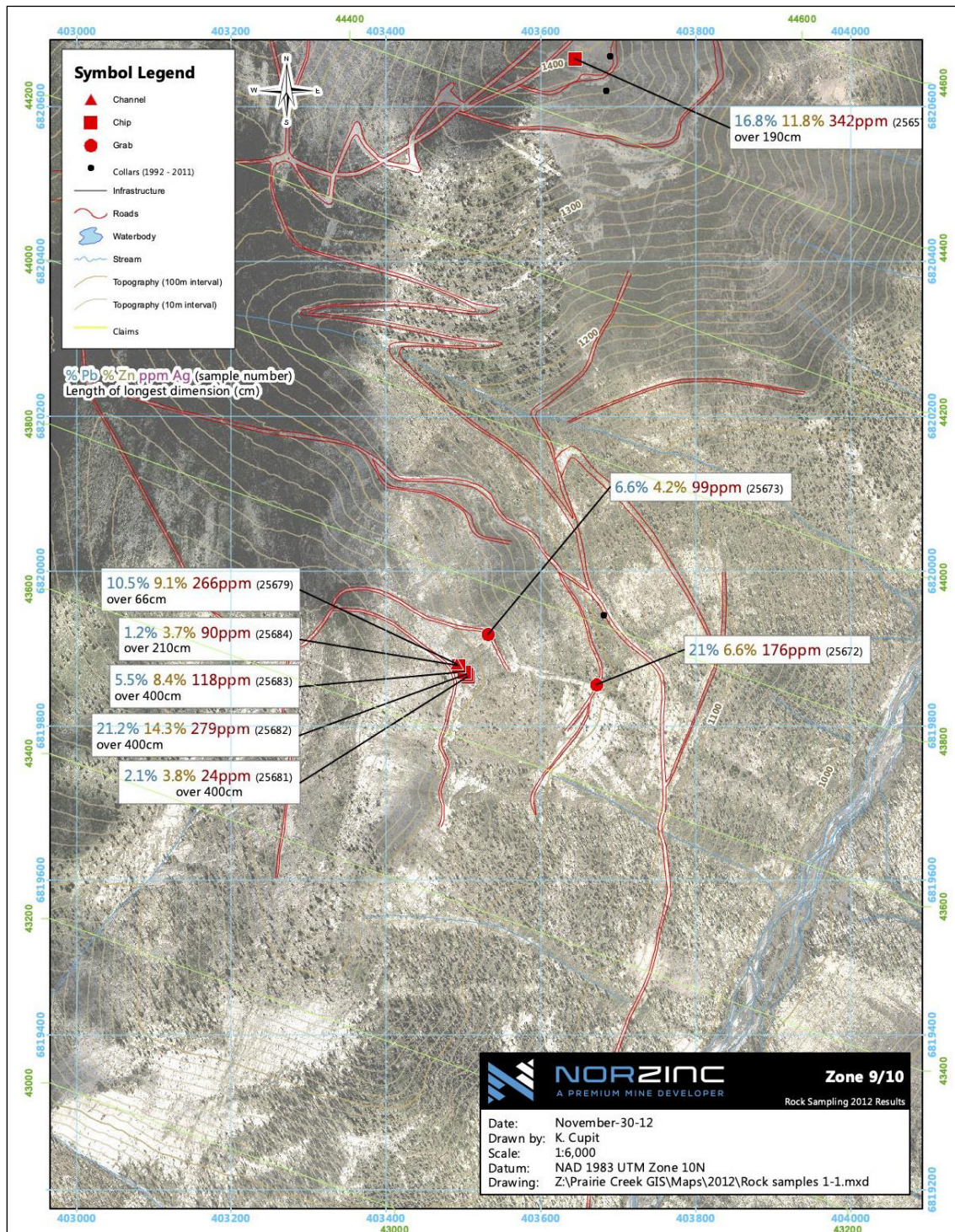
Source: NorZinc (2012)

Figure 9-5: Exploration Highlights from Zone 8



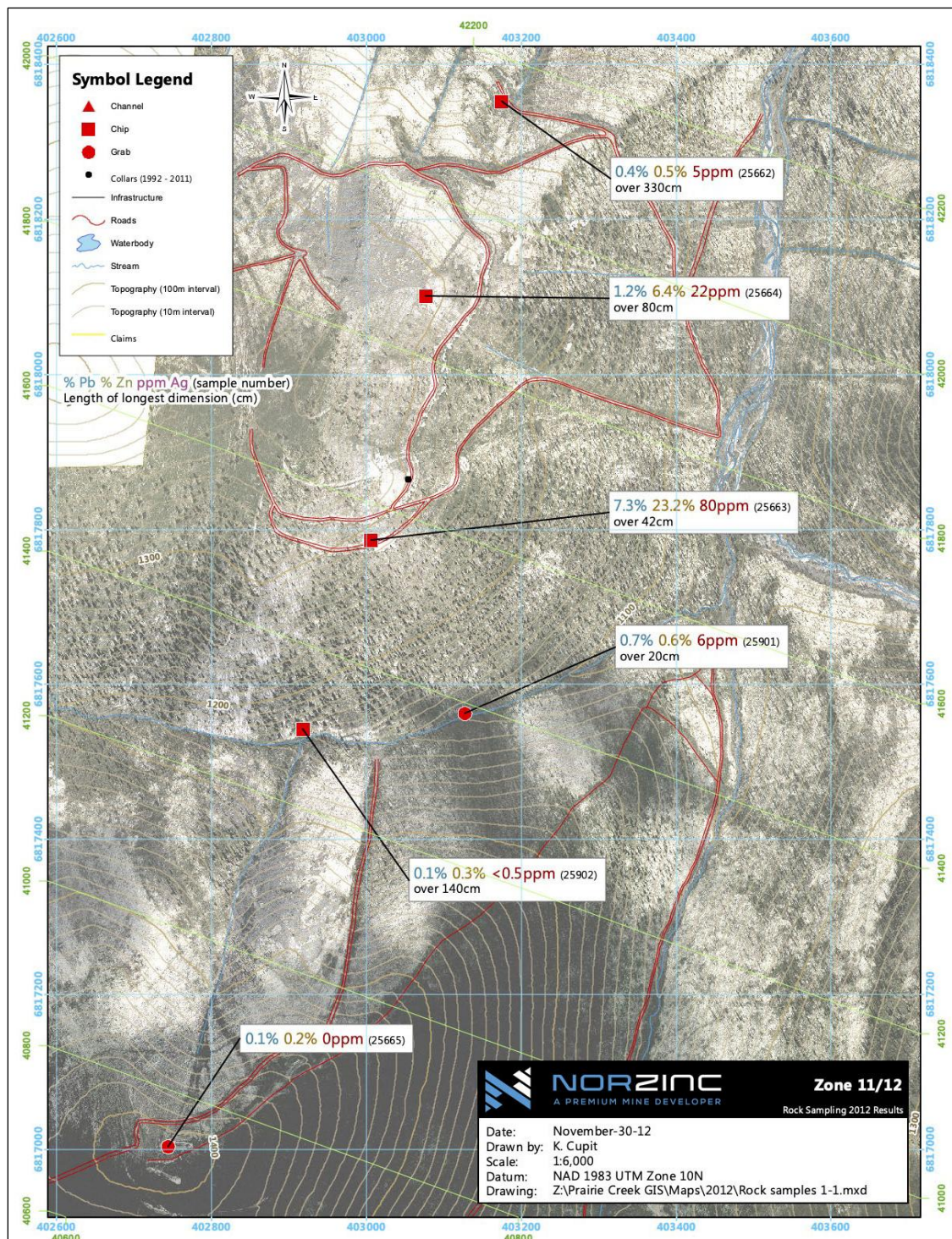
Source: NorZinc (2012)

Figure 9-6: Exploration Highlights from Zones 9 and 10



Source: NorZinc (2012)

Figure 9-7: Exploration Highlights from Zone 11 and 12



Source: NorZinc (2012)

9.4 Geophysical Surveys

Limited geophysical exploration has been conducted on the Project. Since Canadian Zinc acquired the Project, several geophysical techniques have been attempted to assist with identifying new exploration targets.

In 1993, SJ Geophysics Ltd. and Lamontagne Geophysics Ltd. conducted a small number of UTEM survey lines (coupled with a borehole EM survey) for Canadian Zinc in and around camp and the main zone, targeting flat-lying SMS style mineralization as a follow-up to SMS discoveries made in drillholes the previous year. Positive correlations to known SMS mineralization were observed particularly in borehole EM surveys, and several targets were identified for follow-up.

In 2012, a TDEM survey conducted by Ptarmigan Geophysics Ltd. collected 14.2 km of line survey data, and two borehole EM datasets, from a single loop in the main zone, to locate additional SMS mineralization targets. A large conductor response was detected in the western portion of the grid in later channels. This was tested with a drillhole in 2013 that did not intersect notable sulphide mineralization, and the conductor was therefore concluded to be the expression of formational graphite in the Road River (SDr) Formation. Additional minor targets were identified from earlier EM channels, near the boneyard and the road leading north from the boneyard, both of which remain untested to date.

Also in 2012, Quadra Surveys performed a gravity survey over parts of the main zone, and to the south of PC Silver Mine, to further assist with the identification of flat-lying SMS mineralization style targets. The data was largely inconclusive over areas of known SMS mineralization, perhaps due to the severity of the terrain and the depth of the mineralization. However, an anomaly was identified in the zone immediately south of PC Silver Mine, that represents a reasonable target for shallower- SMS mineralization and remains untested to date. Silt sampling in this same area also indicates the potential presence of MQV or SMS style mineralization, with anomalies in the nearby stream tributary spanning the target showing elevated concentrations of As, Ag, Cd, Pb, Sb, Hg, and Zn.

10 DRILLING

10.1 Introduction

Honey Badger Silver has completed no exploration drilling on the property. The following is a summary of work completed by Canadian Zinc.

The metres drilled during drill programs completed since 1992 are summarized in Table 10-1.

Table 10-1: Exploration Drill Summary Table from 1992 to Present

Year	No. of Holes	Metres	Highlights
1992	22	6,322	Discovery of previously unknown SMS mineralization by diamond drilling.
			Discovery hole (PC-92-008) ran 10.60% Zn, 5.29% Pb, 44.37 g/t Ag, over 28.40 m.
1993	31	8,432	Tested for further SMS Mineralization.
			Extended MQV by intersecting 18 m of vein 170 m below workings.
1994	31	11,113	Extension of Main Zone, more SMS lenses in Zone 5.
1995	36	10,082	-
2001	5	1,711	Diamond drilling program designed both to increase confidence in 1998 resource estimate and to identify new high-grade areas.
			Possibility of high-grade shoots recognized.
2004	27	5,944	MQV drilling which intersected significant mineralization.
			Step out on the vein hit narrow but high-grade intersections.
			SMS exploration outside Main Area.
2006	19	2,393	Phase 1 driving of decline tunnel and U/G drilling commences on MQV.
			Drilling of Zone 8 mineralization investigated.
2007	53	11,141	Phase 1 U/G program confirms vein grades. Decline extended, Phase 2 drilling.
			Gate claims drilling and Zone 8, 9 and 11 show poor results.
2010	4	2,694	Deep drilling in Casket Creek (for MQV) and proximal to resource drilling.
2011	30	5,926	Deep drilling in Casket Creek (wedging) and proximal to resource drilling.
2012	11	5,628	Deep drilling in Casket Creek and proximal to resource drilling.
2013	5	1,472	Deep drilling and proximal to resource drilling.
2015	21	5,548	Underground drilling - MQV and STK infill and extension, channel samples taken.
2020	2	1,130	Exploration for, MQV and STK intercepts.

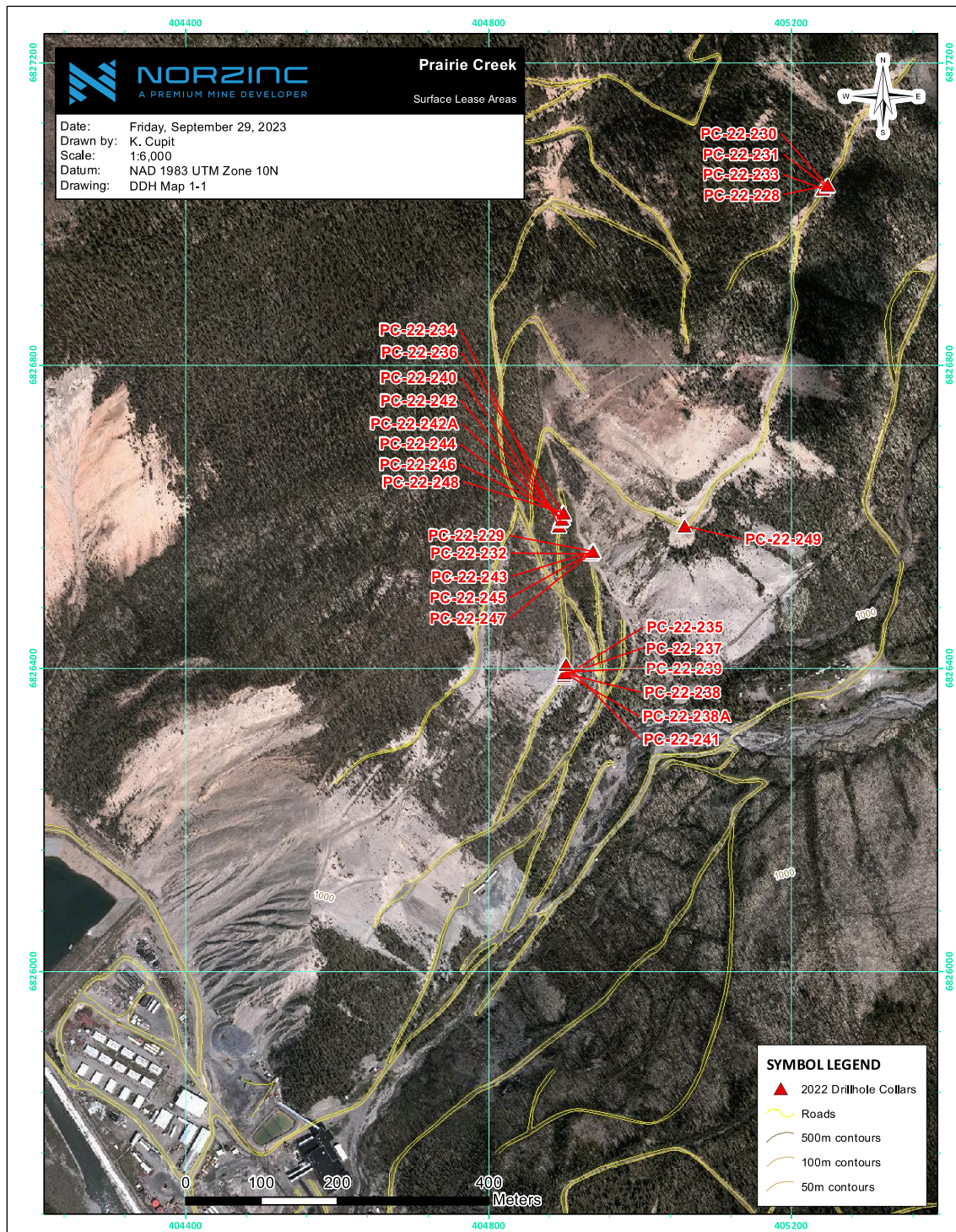
Year	No. of Holes	Metres	Highlights
2021	1	736	Exploration for, MQV and STK intercepts.
2022	24	5,928	Metallurgical and geotechnical drilling into existing mine area (not exploration).
Total	323	86,383	-

Approximately 19,244 m of drilling was carried out on the Project prior to 1992.

It should be noted that over 87% of the drilling tabulated in Table 10-1 was carried out in the Main Zone and Zone 4 area (Zones 1, 2, and 3 are now collectively referred to as the Main Zone, which is comprised of the MQV, SMS and STK zones that are referenced in this report).

Drill programs during 2010, 2011, and 2012 were primarily designed to test for extensions of mineralization to the north of the historical mine area. The 2013 drill program was principally designed to test for continuity of mineralization to the south of the Main Zone and to test an electromagnetic geophysical anomaly. The 2015 underground drill program was designed to assess the STK and adjacent MQV. Drill programs in 2020 and 2021 tested the continuity of the MQV and STK to the north of the Main Zone.

Figure 10-1: Drill Collar Location Plan



Source: NorZinc (2012)

10.2 Exploration Drilling Procedures

10.2.1 Drill Methods

Starting in 1992, surface diamond drilling was carried out using skid mounted Longyear Super 38 drills, owned by Canadian Zinc, to recover NQ diameter (47.6 mm) core. Core size was reduced to BQ size (36.5 mm) where difficult downhole conditions were encountered. In 2010 a new, higher-capacity hydraulic HTM-2500 diamond drill rig (Figure 10-2) was airlifted to the Project for use in the deep drilling program. Figure 10-1 shows this drill set up at Casket Creek during the deep drilling program.

Various drilling contractors have been engaged to run the Canadian Zinc drills. In 2007 Titan Drilling Limited of Yellowknife, Northwest Territories was contracted to carry out a surface drilling program using a Boyles helicopter-portable drill to recover NQ diameter core. Procon Mining and Tunnelling (Procon), who were contracted to continue the decline development work in 2005, sub-contracted Advanced Drilling Limited of Surrey BC, a subsidiary of Cabo Drilling Corporation of North Vancouver BC, to undertake the underground drilling programs. During smaller drill programs, Canadian Zinc has hired individuals to staff the drills as needed, as was the case in 2011 and 2013. In 2012, Cabo Drilling Corporation of North Vancouver was contracted to staff and supply the Canadian Zinc drills. During 2014-2015, Procon was contracted to manage the underground program and subcontracted DMAC Drilling Ltd. of Aldergrove, BC to carry out the diamond drilling. In 2020, Paycore Drilling of Valemount, BC was contracted to run the Canadian Zinc-owned HTM-2500 drill rig, with DMAC Drilling Ltd. returning in 2021 to continue drilling from the same drill pad as the previous year.

In 2022, NorZinc then the parent company of Canadian Zinc engaged Northtech Drilling Ltd. to provide two fly-in Hydracore diamond drill rigs and labour to operate them.

Underground drillholes have an intersection angle which is generally near normal to the planar MQV vein. As the SMS is sub-horizontal, the surface holes have an intersection angle which is near true width. All other surface holes have incidence angles with the MQV and STK veins ranging from approximately 10 to 60°.

Figure 10-2: HTM-2500 Skid-Mounted Diamond Drill Rig at Casket Creek



Source: NorZinc (2021)

10.2.2 Field Procedures

Surface drillhole collars were initially located by handheld GPS, and alignment was completed by Brunton compass sighting along pickets. Once aligned, the dip of the hole was set using an inclinometer placed on the rods. Underground drillhole collar locations were marked up using a total station instrument. The surveyor used spads placed in the development back for a reference line and marked the foresight and backsight on the walls of the drift with spray paint. The drill mast was aligned parallel to the foresight and backsight. A supervising geologist attended the drill site several times per day, as needed.

Drilled core was placed in wooden boxes with depth markers placed in the boxes at the end of each drilling run. The markers were labelled by the drillers in feet or metres to correspond with the specifications for the drill rods in use. Full drill core boxes were individually sealed with wooden lids securely nailed in place to prevent any spilling or shuffling of core during transit.

10.2.3 Surveying

The collars of completed surface drillholes were surveyed by qualified surveyors using a transit. Coordinates were recorded in Universal Transverse Mercator (UTM) projected to the North American Datum 1983 (NAD83) Zone 10N. Underground collars were surveyed using local mine grid coordinates that were then converted to UTM coordinates.

For the 2006 and 2007 drill programs, downhole surveys of both surface and underground holes were completed using a FLEXIT SmartTool instrument. Earlier surveys used an Icefield MI-3 tool and prior to 1995, a Pajari instrument was used. From 2010 onward, downhole survey measurements were completed using a Reflex EZ-Shot and were taken every 15 to 20 m instead of every 60 m as was previously the case. The completion of individual surveys was dependent on downhole conditions.

Raw survey data was processed by software that accompanied the survey tools. Output such as “depth in feet”, “depth in M”, “azimuth”, “dip”, “magnetic field strength” and “magnetic dip” were captured from the processed data and saved in the drillhole database.

10.2.4 Core Logging

All drill core logging was carried out at the Project site in a secure facility. Received core was laid out and a quick assessment was made to verify that all the boxes were intact, to confirm the drillhole identification data, and that the drillers’ depth markers were in good order (i.e., drill core mixing, or displacement did not occur during transport). If disruption was identified, the core was “fitted” together, and the depth markers were placed at the appropriate points by means of direct measurement and identification of the start/end points of successive drilling runs. The depth markers were then converted, if necessary, to metric measurements and aluminum tags were stapled to the outside end of each box noting drillhole number and the start and end depths represented by each box. Drill core recovery was calculated by comparing the drilled length with the actual core length recovered between depth markers. Rock quality designation (RQD) was calculated from the sum of the length of full-diameter drill core pieces over 10 cm, divided by the total length of the run. Between 2006 and 2015, rock mass ratings (RMRs) were calculated for 10 m envelopes around individual mineralized intersections, using industry standard methods.

All drill core was geologically logged using the standard lithologies identified in the stratigraphic sequence presented as Table 7-1. Geology logs, complete with written and coded descriptions of primary lithology, secondary lithology, alteration, oxide/sulphide mineralization and structure, were compiled and recorded in a drillhole database. Prior to core photography, which was done for two or three boxes at a time, sample intervals were marked on the core by the geologist responsible for that hole.

Prior to 2011, core logs were entered into Excel spreadsheet format for copying into a central database. Starting in 2011, Canadian Zinc switched to direct inputting into an MS Access database by way of a software package named GeoticLog. This core logging software allows for immediate error checking and reduces transcription errors. Data integrity checks (i.e., overlapping intervals, missing intervals, and duplicate samples) were performed via automated software checks nearer to the end of the season, and any problems were resolved as they were identified.

10.2.5 Core Recovery

Core recoveries were consistently recorded since 2006. Average recoveries were approximately 80% for the MQV and 97% for the SMS mineralization. Intervals of poor recovery in the MQV are associated with shearing and faulting and related fine-grained fault gouge that is easily washed out during the diamond drilling process. Rates of recovery are based on drill run lengths.

10.2.6 Bulk Density

No bulk density measurements have been collected from the drill core recently. Bulk density values used in later resource estimates were calculated on the basis of regression equations that were used for the 2007 and 2012 mineral resource estimates (see Section 6). These equations

were based on 231 measurements of drill core from the MQV made in 1998 and 54 measurements from sample pulps of SMS mineralization made in 2007. All samples collected in 2022 were submitted for pycnometer measurements.

10.3 Significant Drill Results and Interpretation

10.3.1 2010 Drill Program

During 2010, three holes, with an aggregate length of 2,694 m, were drilled in the Casket Creek area approximately 1.7 km north of the historical mine site. Hole PC-10-186 was drilled to a target depth of 1,557 m. This hole intersected the target Whittaker Formation, the principal host of mineralization, at a down-hole depth of 1,500 m. The stratigraphic information provided by this hole enabled the determination of a more precise location of the potential vein-hosting structure.

A second, wedged drillhole, PC-10-186W1, was directed from the upper part of PC-10-186 to the west toward the revised target location. This hole had to be abandoned after about 150 m, at an estimated depth of about 536 m, because of technical difficulties.

A third hole, PC-10-187, with a revised orientation, was collared at surface from the same drill pad and had reached a down hole depth of 652 m when weather conditions forced suspension of drilling for the year.

10.3.2 2011 Drill Program

The 2011 program had two objectives: continuation of the 2010 deep drilling program and testing for additional high-grade vein structures and for other, wider, SMS deposits adjacent to known mineralization within the historical mine area.

The Casket Creek program comprised four holes, including wedges, with an aggregate length of 2,513 m, and commenced with the completion of drillhole PC-10-187. This hole intersected significant vein-type lead-zinc mineralization that demonstrated the probable northward continuation of MQV-type mineralization from the historical mine area.

A wedge hole, PC-11-187W2, was drilled as an undercut to PC-10-187. This hole intersected mineralization 50 m below the PC-10-187 intercept. Intersected grades in PC-11-187 and PC-11-187W2 are shown in Table 10-2. Note that the thickness reported in the table is intersected rather than true thickness. As the dip of the drillholes and the attitude of the mineralization are both variable, true thickness cannot be estimated with confidence.

Table 10-2: Assay Results of 2011 Drill Program

Drillhole	From (m)	To (m)	Interval (m)	Pb (%)	Zn (%)	Ag (g/t)	Cu (%)
PC-10-187	1,348.36	1,348.88	0.52	4.92	5.90	34	0.034
PC-11-187W2	1,384.00	1,387.50	3.50	5.26	11.47	84	0.176

Source: Ausenco (2026)

The drill intercepts are approximately 100 m west of the PCA fold axis in a structural setting identical to that of the MQV in the Main Zone. It was concluded that a mineralized structure similar to the MQV in the main zone occurs under Casket Creek and may represent the northern continuation of the MQV.

A subsequent drillhole, PC-11-206, was designed to cut target stratigraphy 250 m below intersected depths at the Main Zone. The drilling of this hole was suspended at the end of October 2011 due to weather conditions and was completed in 2012.

10.3.3 2012 Drill Program

Eleven holes with an aggregate length of 5,628 m were drilled in 2012. Eight of these tested the Main Zone and one (PC-12-213) was drilled to the north of the Main Zone in Casket Creek.

Examples of significant intercepts from the 2012 program are listed in Table 10-3. Note that the thickness reported in the table is intersected rather than true thickness. As the dip of the drillholes and the attitude of the mineralization are both variable, true thickness cannot be estimated with confidence.

Table 10-3: Significant Assay Results of 2012 Drill Program

Drillhole	From (m)	To (m)	Interval (m)	Pb (%)	Zn (%)	Ag (g/t)	Cu (%)
PC-12-211	215.62	216.39	0.77	3.87	4.66	47.00	0.13
PC-12-212	211.44	212.75	1.31	4.60	5.91	27.70	0.01
PC-12-212	212.75	213.46	0.71	3.48	15.30	27.10	0.03
PC-12-212	213.46	214.26	0.80	8.20	21.10	50.00	0.04
PC-12-214	152.84	153.74	0.90	9.07	19.50	162.00	0.45
PC-12-214	305.00	306.00	1.00	31.90	3.25	393.00	0.37
PC-12-214	306.00	306.80	0.80	2.13	6.31	36.30	0.03
PC-12-215	575.59	576.34	0.75	14.00	0.15	103.00	0.02
PC-12-215	576.34	577.19	0.85	0.08	0.05	0.00	0.00
PC-12-215	578.51	579.35	0.84	36.90	6.30	268.00	0.04

Drillhole	From (m)	To (m)	Interval (m)	Pb (%)	Zn (%)	Ag (g/t)	Cu (%)
PC-12-215	579.35	580.27	0.92	0.08	0.02	0.70	0.00
PC-12-216	417.17	418.28	1.11	0.12	0.11	1.00	0.00
PC-12-216	418.28	419.59	1.31	4.02	10.30	2,059.00	9.37
PC-12-216	419.59	420.60	1.01	0.18	0.84	5.90	0.02
PC-12-217	463.6	464.60	1.00	18.10	3.96	157.00	0.01

Source: Ausenco (2026)

10.3.4 2013 Drill Program

Five holes, with an aggregate length of 1,472 m, were drilled in 2013. Three were drilled to test Zone 4 approximately 200 m south of the currently defined southern end of the MQV, and two were collared about 320 m apart to test a 900 m wide multi-channel electromagnetic anomaly identified in 2012. PC-13-220 was also designed to intersect projections of previously defined vein and STK mineralization. Note that the thickness reported in the table is intersected rather than true thickness. As the dip of the drillholes and the attitude of the mineralization are both variable, true thickness cannot be estimated with confidence.

Both holes are projected to have tested the main part of the geophysical anomaly at depth. Interpretations based on current data suggest that the EM anomaly is due to inherent natural variations in graphite content within the upper half of the Road River Formation. Table 10-4 lists intercepts with greater than 8% combined lead-zinc.

Table 10-4: Significant Assay Results from 2013 Drill Program

Drillhole	From (m)	To (m)	Interval (m)	Pb (%)	Zn (%)	Ag (g/t)	Cu (%)
PC-13-220	206.08	207.08	1.00	3.22	5.92	62.00	0.11
PC-13-220	207.93	209.00	1.07	5.00	8.76	84.00	0.09
PC-13-220	209.00	210.00	1.00	5.35	8.74	104.00	0.25
PC-13-220	210.00	211.00	1.00	14.40	21.50	191.00	0.32
PC-13-220	212.00	213.00	1.00	13.80	25.30	331.00	0.86
PC-13-222	373.60	374.60	1.00	16.60	1.59	125.00	0.05
PC-13-223	83.56	84.56	1.00	6.17	19.70	66.00	0.03
PC-13-224	28.85	29.60	0.75	23.00	20.70	268.00	0.02
PC-13-224	34.80	36.10	1.30	1.40	15.80	30.80	0.05
PC-13-224	47.24	48.24	1.00	5.61	8.84	97.00	0.03
PC-13-224	87.00	88.00	1.00	2.55	5.44	18.20	0.01
PC-13-224	105.22	106.22	1.00	1.92	6.16	15.80	0.00

Source: Ausenco (2026)

10.3.5 2015 Drill Program

During 2015, Canadian Zinc drilled 21 underground holes from the 883 mL decline as a series of vertically oriented fans to test the MQV STK Zones and to provide additional information regarding the spatial relationship between the two. The resulting assays indicate that the style and grades of the MQV Zone that have been encountered in the southern portion of the Zone continue to the north beyond and below the existing workings. As well, the drill program indicated that the STK Zone occupies an offset in the MQV and is largely bounded to the east and west by the MQV, suggesting that the STK formed as a result of deformation of the MQV. Representative assay results from the 2015 drill program are shown in Table 10-5. Note that the thickness reported in the table is intersected rather than true thickness. As the dip of the drillholes and the attitude of the mineralization are both variable, true thickness cannot be estimated with confidence.

Table 10-5: Representative Assay Results from 2015 Drill Program

Drillhole	From (m)	To (m)	Interval (m)	Pb (%)	Zn (%)	Ag (g/t)	Cu (%)
PCU-15-52	128.24	129.24	1.00	0.03	0.04	1	0.00
PCU-15-52	129.24	130.24	1.00	12.90	31.10	159	0.30
PCU-15-52	132.63	133.60	0.97	25.70	29.80	321	0.59
PCU-15-52	133.60	134.60	1.00	0.62	3.96	8	0.01
PCU-15-52	134.60	135.70	1.10	0.38	0.35	4	0.00
PCU-15-52	135.70	136.64	0.94	5.97	20.90	43	0.00
PCU-15-52	136.64	137.70	1.06	5.70	13.20	58	0.10
PCU-15-52	137.70	138.70	1.00	0.21	0.26	2	0.00
PCU-15-53	101.80	103.33	1.53	0.33	1.96	8	0.01
PCU-15-53	103.33	104.85	1.52	20.90	31.20	173	0.04
PCU-15-53	104.85	106.38	1.53	34.00	29.70	405	0.10
PCU-15-53	106.38	107.90	1.52	1.06	1.15	23	0.06
PCU-15-53	124.66	126.19	1.53	0.40	4.66	8	0.02
PCU-15-54	181.20	181.90	0.70	0.06	0.08	1	0.00
PCU-15-54	181.90	182.57	0.67	3.94	7.69	84	0.21
PCU-15-60	138.38	139.35	0.97	0.08	0.84	2	0.00
PCU-15-60	139.35	140.35	1.00	0.15	4.18	2	0.00
PCU-15-60	144.40	145.40	1.00	7.87	9.47	118	0.24
PCU-15-60	145.40	146.40	1.00	0.19	0.42	59	0.22
PCU-15-60	158.48	159.48	1.00	1.55	14.10	21	0.02
PCU-15-60	164.57	165.70	1.13	4.67	8.11	55	0.08
PCU-15-60	165.70	166.70	1.00	5.01	15.80	78	0.17
PCU-15-68	142.00	142.95	0.95	2.58	3.42	44	0.12

Drillhole	From (m)	To (m)	Interval (m)	Pb (%)	Zn (%)	Ag (g/t)	Cu (%)
PCU-15-68	142.95	144.48	1.53	0.07	0.05	2	0.00
PCU-15-68	144.48	146.40	1.92	0.08	0.06	1	0.00
PCU-15-68	146.40	147.35	0.95	1.39	12.10	44	0.13
PCU-15-72	282.72	283.73	1.01	11.50	0.38	121	0.12
PCU-15-72	285.16	286.20	1.04	3.02	10.20	51	0.13
PCU-15-72	286.20	287.53	1.33	0.28	5.60	11	0.03
PCU-15-72	292.98	293.83	0.85	0.56	8.37	25	0.08
PCU-15-72	296.15	296.88	0.73	1.36	1.01	18	0.03
PCU-15-72	299.92	300.98	1.06	1.84	0.97	14	0.00
PCU-15-72	302.28	303.26	0.98	0.08	0.02	1	0.00
PCU-15-72	309.52	310.59	1.07	1.16	1.13	10	0.01

Source: Ausenco (2026)

10.3.6 2020 and 2021 Drill Programs

In 2020, Canadian Zinc undertook an exploration drill program to test the continuity of the MQV and STK type mineralization adjacent to the Main Zone, and to be able to upgrade Inferred mineral resources to the Indicated category for future mineral resource estimates. Nearby existing holes suggested that the target area could host above-average silver grades and is proximal to the strong intercepts observed in hole PCU-15-72. All three 2020 and 2021 holes were from the same drill pad, with results summarized in Table 10-6. Note that the thickness reported in the table is intersected rather than true thickness. As the dip of the drillholes and the attitude of the mineralization are both variable, true thickness cannot be estimated with confidence.

Table 10-6: Representative Assay Results from 2020/2021 Drill Programs Drillhole

Drillhole	From (m)	To (m)	Interval (m)	Pb (%)	Zn (%)	Ag (g/t)	Cu (%)
PC-20-225	502.55	506.81	4.26	16.89	27.18	204	0.47
PC-20-225	506.81	508.79	1.98	8.44	5.58	88	0.16
PC-20-225	549.89	611.12	61.23	2.47	3.79	29	0.06
PC-20-226	593.8	594.8	1.00	10.1	21.3	391	1.6
PC-21-227	Hole is at 553 m and did not reach target depth						

Source: Ausenco (2026)

10.4 2022 Drill Program

In 2022, Canadian Zinc executed a complex drill, sampling, and data acquisition program to acquire sufficient geotechnical data to inform mining dilution estimates, to acquire enough material to perform confirmation metallurgical testwork, and to verify underground channel samples through surface drilling.

A total of 5,928 m was drilled over a four-month period in 24 drillholes using two Hydracore drill rigs provided and operated by Northtech Drilling Ltd. of Yellowknife. All the drilling was conducted in the Main Zone. No exploration drilling was conducted as part of this program.

The geotechnical investigation involved manual geotechnical core logging of current and past drillholes, obtaining 36 fresh drill core samples for rock strength testing, and detailed downhole surveying with acoustic and visual televiwers of 11 drillholes. Drillholes were selected from across the Main Zone to assist with building a comprehensive 3D geotechnical model through the main mineralized zone.

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1 Introduction

Honey Badger Silver has not done any sampling on the Project. The following summarizes the work done by Canadian Zinc.

11.2 Historical Core Sampling

Drill core was boxed at the drill rig by the drillers' helpers who securely nailed a wooden lid onto each filled core box. Underground core was transported by the drillers to the portal. Both underground and surface drill core boxes were picked up by a Canadian Zinc geologist and then transported in pick-up trucks or side-by-side vehicles, driven by a Canadian Zinc geologist, or authorized designate, to the secure, on-site drill core logging and sampling facility (referred to as the "core shack" or "core shed").

Rice sacks containing channel sample bags were transported to the surface by either the responsible geologist or an assistant under his or her supervision. The rice sacks were then transported in pick-up trucks driven by a Canadian Zinc geologist to the secure, on-site drill core logging and sampling facility.

The still-sealed core boxes were organized in order, from top to bottom of the hole, on large tables or racks outside the core shack from where they are brought inside for logging and sampling. A geologist marked appropriate sample intervals on the drill core of approximately 1 m length. After logging, core boxes were photographed three at a time and then cross-piled outside or set aside for sample processing within the core shack. A geotechnician cut the marked sample intervals of drill core longitudinally in half with a diamond saw, placing half of the material into a sample bag and the other half back into the core box.

Loosely consolidated material, if present and marked for sampling, was scooped out with a stainless-steel spoon, taking half of the representative material into the sample bag. A tag was placed into the sample bag listing an ID number, and another tag with the same ID number was stapled into the core box at the start of the sampled interval for later reference. All sample IDs were entered into the drillhole database with a sample-specific description for each, for later cross-referencing with assay results.

11.3 Sample Security and Storage

All drill core logging and sampling were supervised by a senior geologist. Only authorized personnel or those accompanied by an authorized person were allowed into the on-site drill core and sampling facility. The sampling facility was locked at all times when geologists or their assistants were not present.

Individual sample bags were sealed with plastic security zip ties and placed in rice sacks (50 pounds per bag). Requisition sheets were inserted into each rice sack, and each rice sack was

labelled with the assay lab shipping address. The sacks were sealed with larger plastic security zip ties and stored in the secure drill core and sampling facility prior to their transport off-site.

11.3.1 Transport

Samples were air-freighted in charter aircraft from the Project site to Fort Nelson, BC, or Fort Simpson, Northwest Territories. Prior to 2011, sealed rice sacks containing samples were transported by Greyhound bus to the Acme Labs assay laboratory in Vancouver, BC. From 2011 to 2020, sacks were delivered to AGAT Laboratories either at their sample drop-off location in Fort Nelson, or shipped to their lab in Mississauga, ON. For 2021 and onwards, sacks were either couriered or delivered by Canadian Zinc staff direct to ALS Geochemistry in Yellowknife, Northwest Territories, or sent by courier to ALS Geochemistry in North Vancouver, BC.

11.3.2 Drill Core Storage

Until 2025, boxes containing the main mineralized drill core intersections are stored in trailers adjacent to the core shack facility (Figure 11-1). However, those buildings were destroyed by adverse weather, and the core is now cross-stacked on pallets in the same area. Boxes containing unmineralized drill core are cross-piled in stacks (Figure 11-2) in the core storage area next to the boneyard near Harrison Creek and covered with stapled-down weatherproof geotextiles.

Figure 11-1: Stored Mineralized Drill Core Intersections at Main Site



Source: NorZinc (2021)

Figure 11-2: Stored Unmineralized Drill Core at Harrison Creek Site



Source: NorZinc (2021)

11.4 Analytical Laboratories

Acme Labs (ISO 9001-2000 accredited) has carried out the majority of the sample assaying since Canadian Zinc's first involvement with the Project in 1992 and was the primary assay lab until 2011.

From 2011 to 2020, sample assaying was conducted by AGAT Laboratories (ISO/IEC 17025:2005 accredited).

For 2021 and onwards, assays were completed by ALS Geochemistry (ISO/IEC 17025:2017 and ISO 9001:2015 accredited).

All labs used were independent of Canadian Zinc.

11.5 Sample Preparation and Analysis

In the analytical laboratory, samples were sorted and inspected for quality of use (quantity and condition); wet or damp samples are dried at 60°C. Samples were then crushed to 70% passing ten mesh (2 mm), homogenized, riffle split (250 g sub-sample) and pulverized to 95% passing 150 mesh (100 µm). The crusher and pulveriser were cleaned by brush and compressed air between routine samples. A granite wash was used to scour equipment after high-grade samples, between changes in rock color or at the end of each file. Granite was crushed and

pulverized as the first sample in each sequence and each granite sample was carried through to analysis to monitor background assay grades.

11.5.1 Assay Procedure

The grades of silver, copper, lead, zinc, antimony, mercury, cadmium, and arsenic as well as up to 29 additional elements, were determined for all samples by aqua regia digestion followed by an ICP-ES finish. Non-sulphide lead and non-sulphide zinc (which quantify lead oxides and zinc oxides, respectively) were assayed by ammonium acetate leach and AAS finish. Silver was analyzed by fire assay fusion. Overlimits were assayed as required.

11.6 Quality Assurance and Quality Control

The data was provided by Canadian Zinc as a series of Excel spreadsheets detailing the primary and QA/QC data and descriptions of the nature of QA/QC samples.

The provided data covered results from 1992 to 2022 and included core duplicates, coarse reject duplicates, pulp duplicates, Canadian Zinc submitted certified reference materials (CRMs) and blanks, together with laboratory submitted blanks and internal laboratory standards. During this period analysis was completed at three different laboratories:

- ACME Analytical Laboratories (now Bureau Veritas) – 1992-2011;
- AGAT Laboratories – 2012-2015; and
- ALS Global – 2020-2022.

Each laboratory has recognized certification to ISO/IEC 17025.

11.6.1 Standards

Canadian Zinc generated its own assay standard samples, in conjunction with Smee & Associates Consulting Limited of North Vancouver, BC (Smee). Standards were compiled from a shipment of mineralized samples sent by Canadian Zinc to CDN Resource Laboratories Limited in Delta, BC (CDN). CDN prepared three homogeneous pulps suitable for use as standard reference materials. The samples were dried, and the material was mechanically ground in a rod mill and then screened through a 200-mesh sieve, the plus 200 mesh fraction being discarded. The minus 200 fraction was mechanically mixed for 48 hours in a twin-shell V Blender rotating at approximately 20 revolutions per minute. The derived standards were bagged in lots of approximately 110 g in tin-top kraft bags that were then individually vacuum packed and heat-sealed in plastic bags.

Ten samples of each bagged and sealed standard were sent for round-robin analysis to Acme Labs (ISO 9001-2000 accredited), Chemex (ISO 9001-2000 accredited), Actlabs Limited in Ancaster, Ontario (ISO/IEC 17025 [Standards Council of Canada], which includes ISO 9001 and

ISO 9002 accreditations), Assayers Canada in Vancouver BC (ISO/IEC 17025 [Standards Council of Canada]) and SGS Lakefield (ISO 9001-2000 accredited).

In total there were 140 standards (certified reference materials or CRMs) submitted across all phases of drilling at PC Silver Mine. A breakdown of the number assayed by each of the three laboratories is presented in Table 11-1, and a summary of the certified values for Ag, Pb and Zn for each standard is presented in Table 11-2.

Table 11-1: Number of CRMs by Analytical Laboratory

Laboratory	Number of Standards		
	Standard 1	Standard 2	Standard 3
ACME	19	30	17
AGAT	19	20	27
ALS	8	-	-
Total	46	50	44

Source: Ausenco (2026)

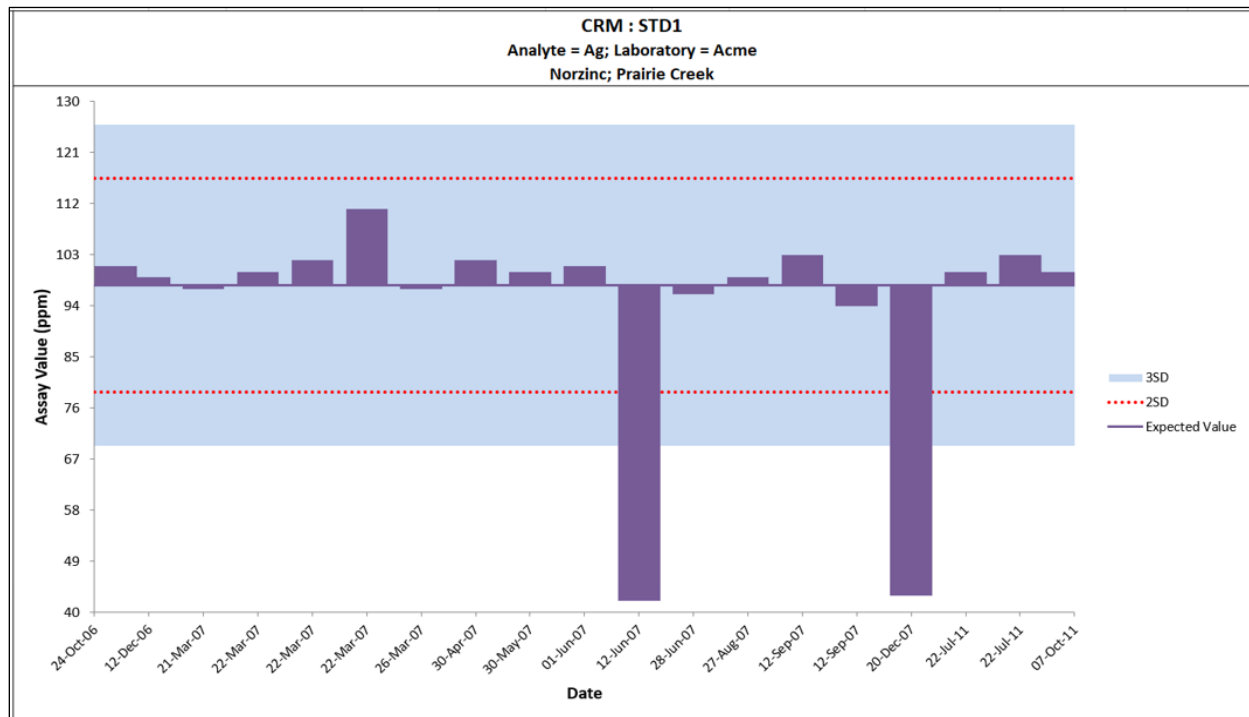
Table 11-2: Certified Values of CRMs

Standard ID	Ag		Pb		Zn	
	Certified Value (g/t)	Standard Deviation	Certified Value (%)	Standard Deviation	Certified Value (%)	Standard Deviation
Standard 1	97.6	9.4	6.88	0.20	4.17	0.22
Standard 2	221	19	12.52	0.62	11.6	0.98
Standard 3	367	15.4	24.8	1.90	20.32	1.74

Source: Ausenco (2026)

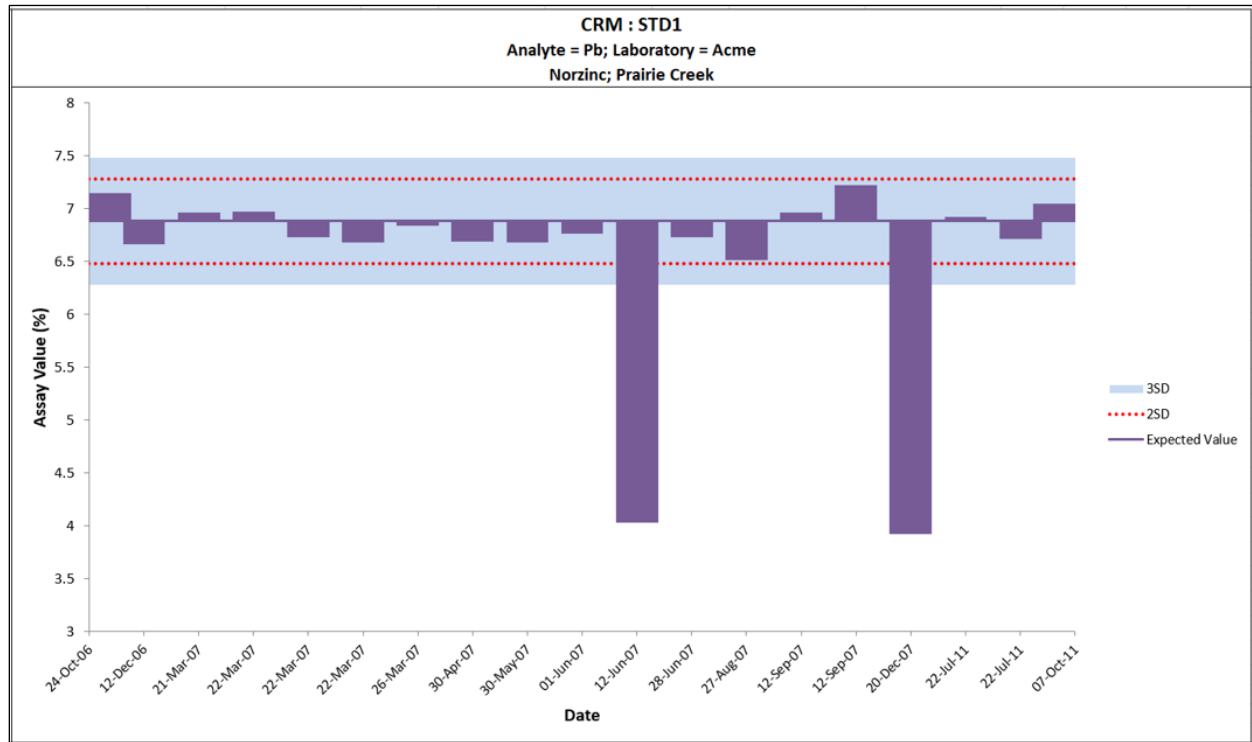
Assays of the standards show an acceptable performance, with only three results falling outside the control limit defined at three times the standard deviation. Control charts for each standard for Ag, Pb and Zn for the ACME laboratory are presented in Figure 11-3 through Figure 11-11.

Figure 11-3: Control Chart for Ag Reported for Standard 1 – ACME Laboratory



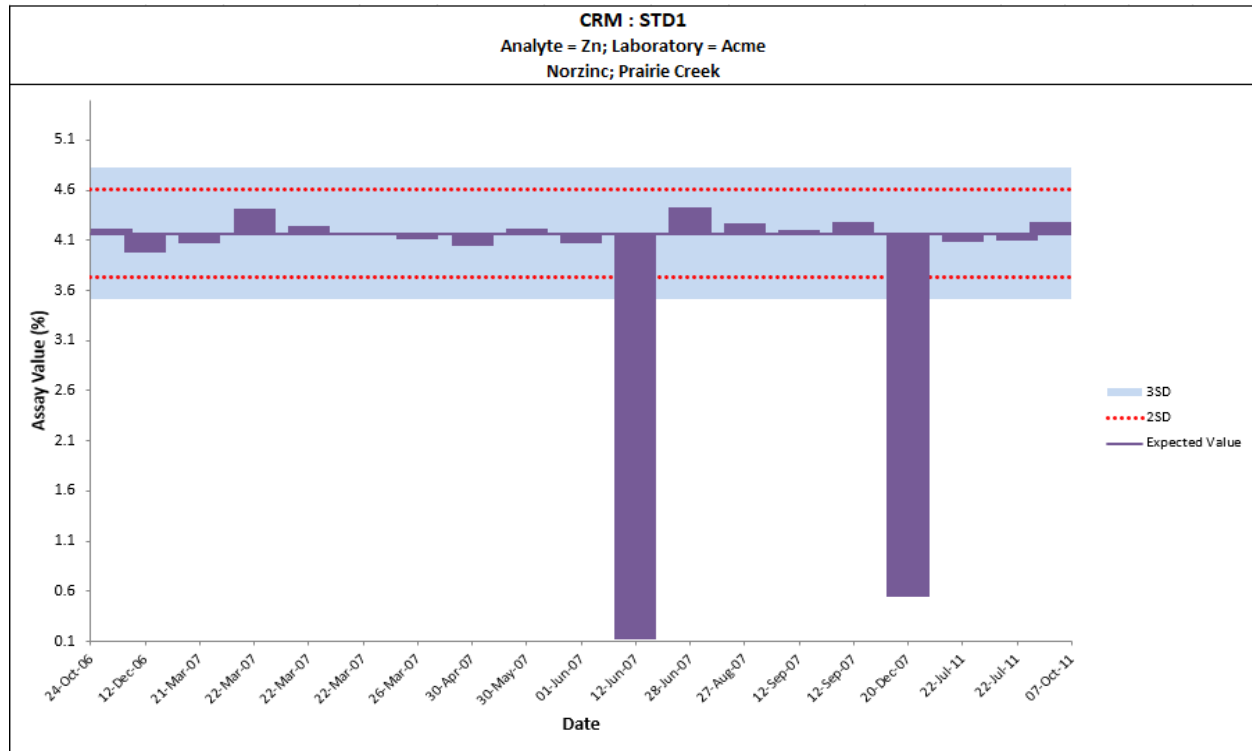
Source: Cube (2023)

Figure 11-4: Control Chart for Pb Reported for Standard 1 – ACME Laboratory



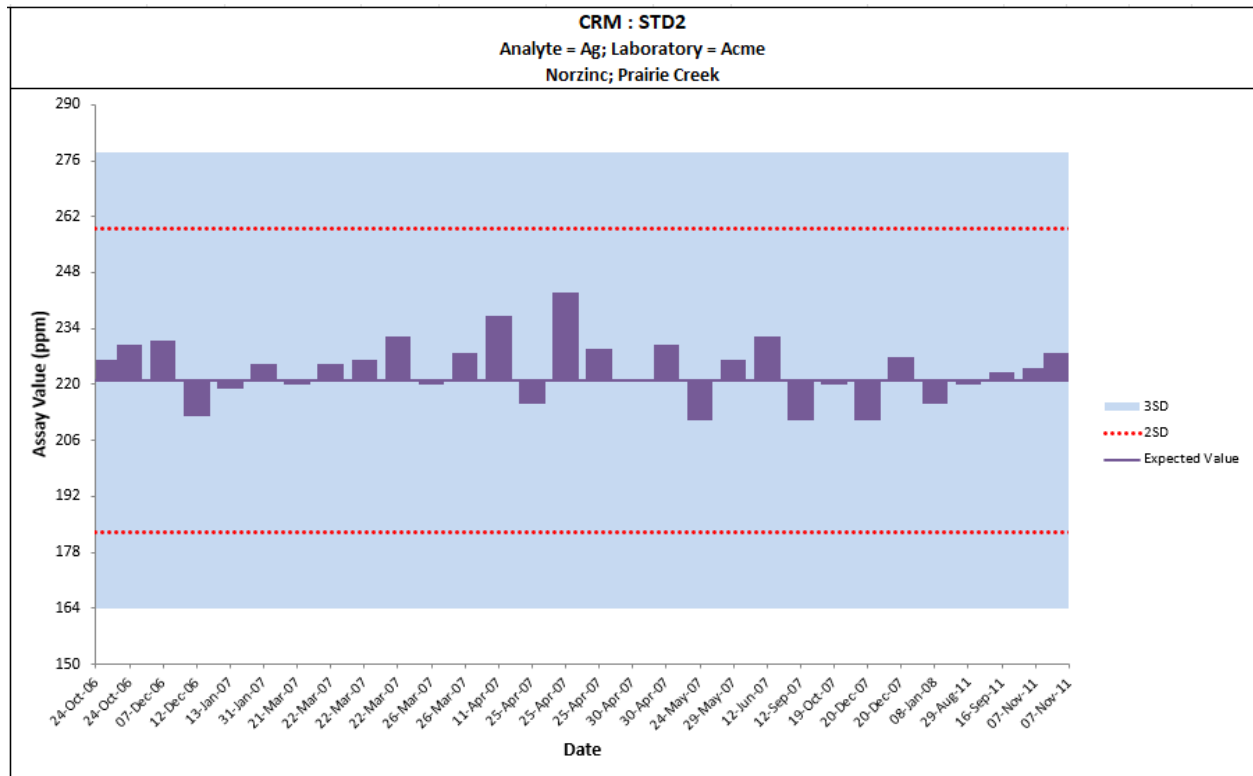
Source: Cube (2023)

Figure 11-5: Control Chart for Zn Reported for Standard 1 – ACME Laboratory



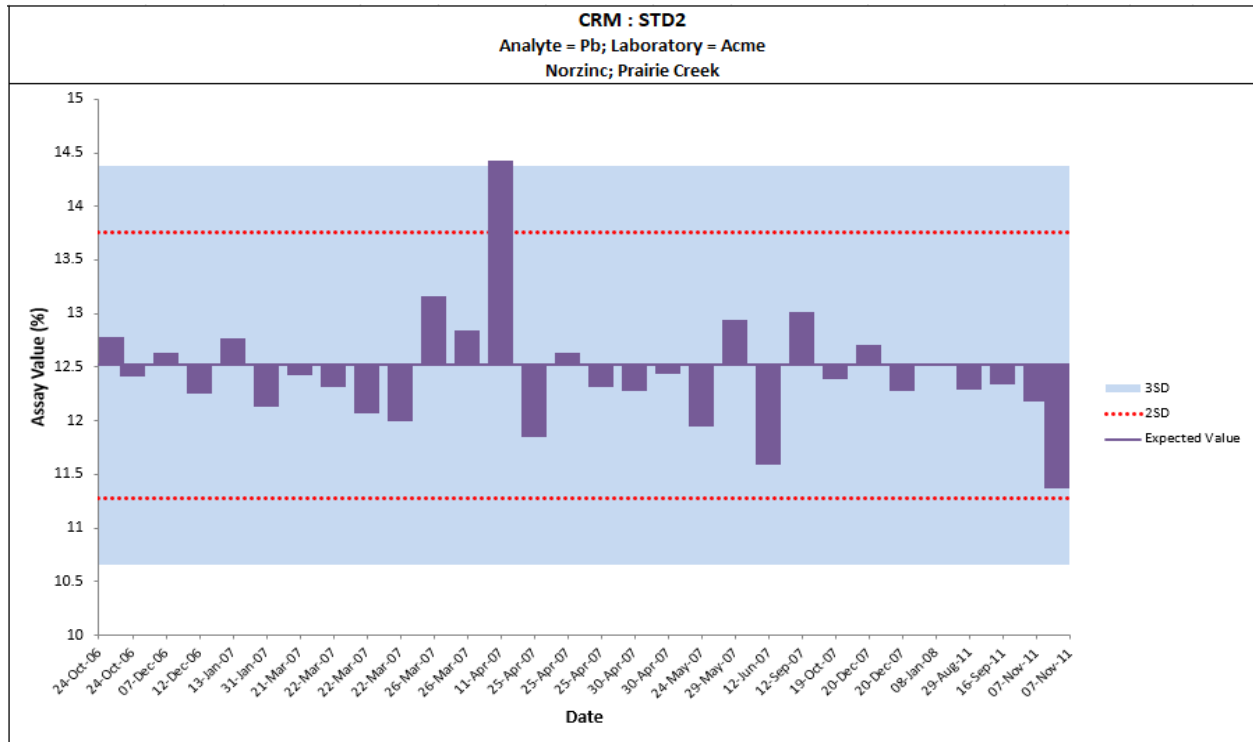
Source: Cube (2023)

Figure 11-6: Control Chart for Ag Reported for Standard 2 – ACME Laboratory



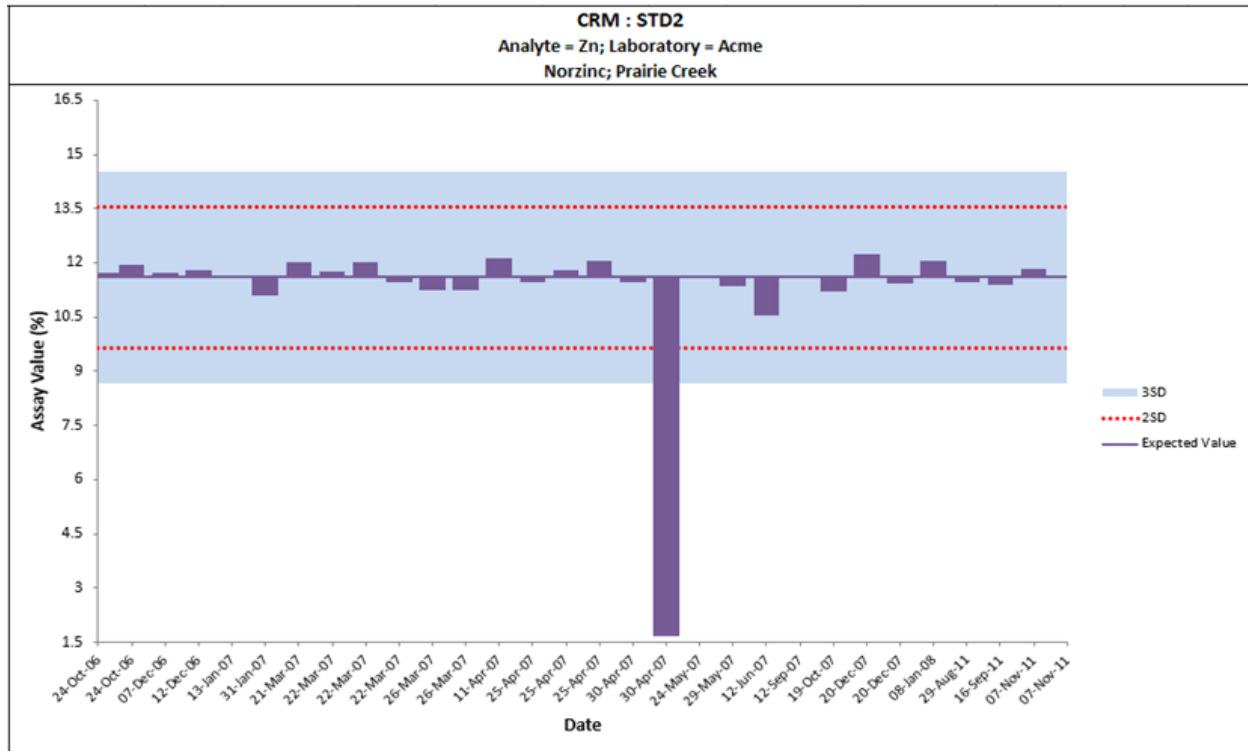
Source: Cube (2023)

Figure 11-7: Control Chart for Pb Reported for Standard 2 – ACME Laboratory



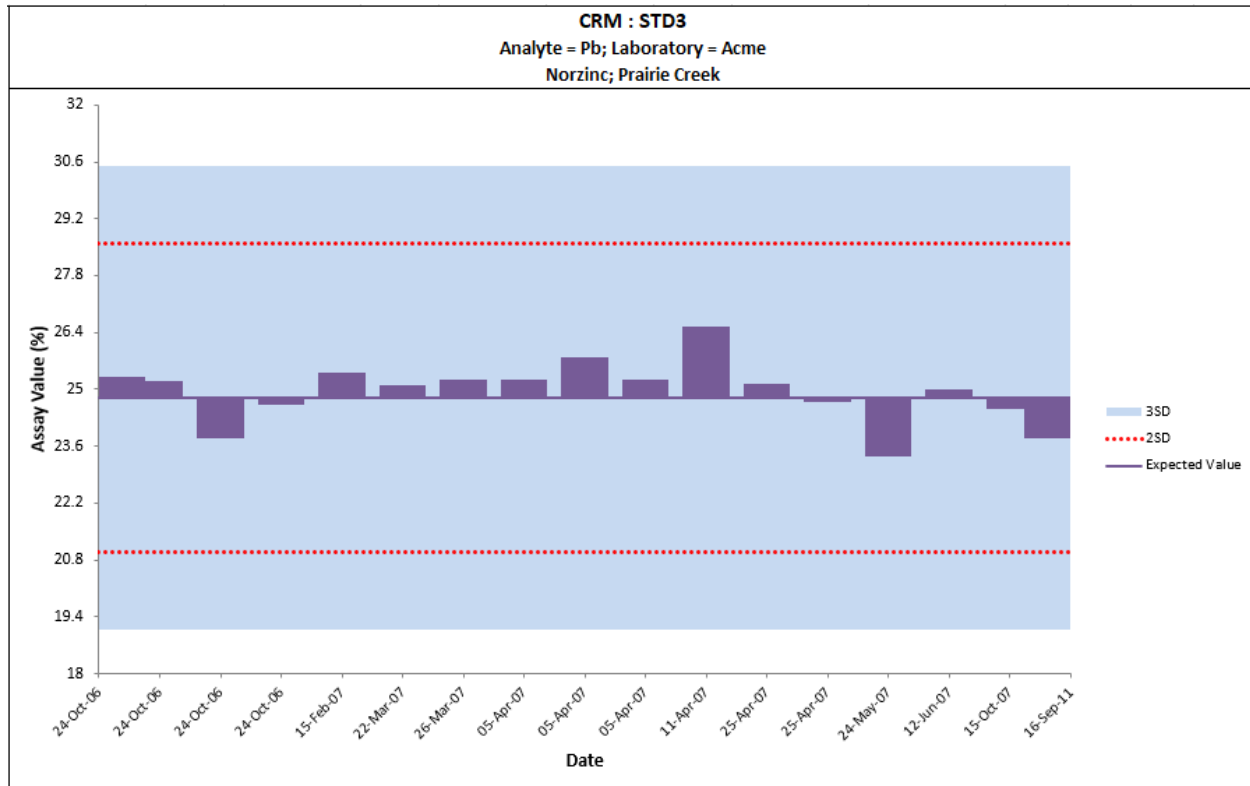
Source: Cube (2023)

Figure 11-8: Control Chart for Zn Reported for Standard 2 – ACME Laboratory



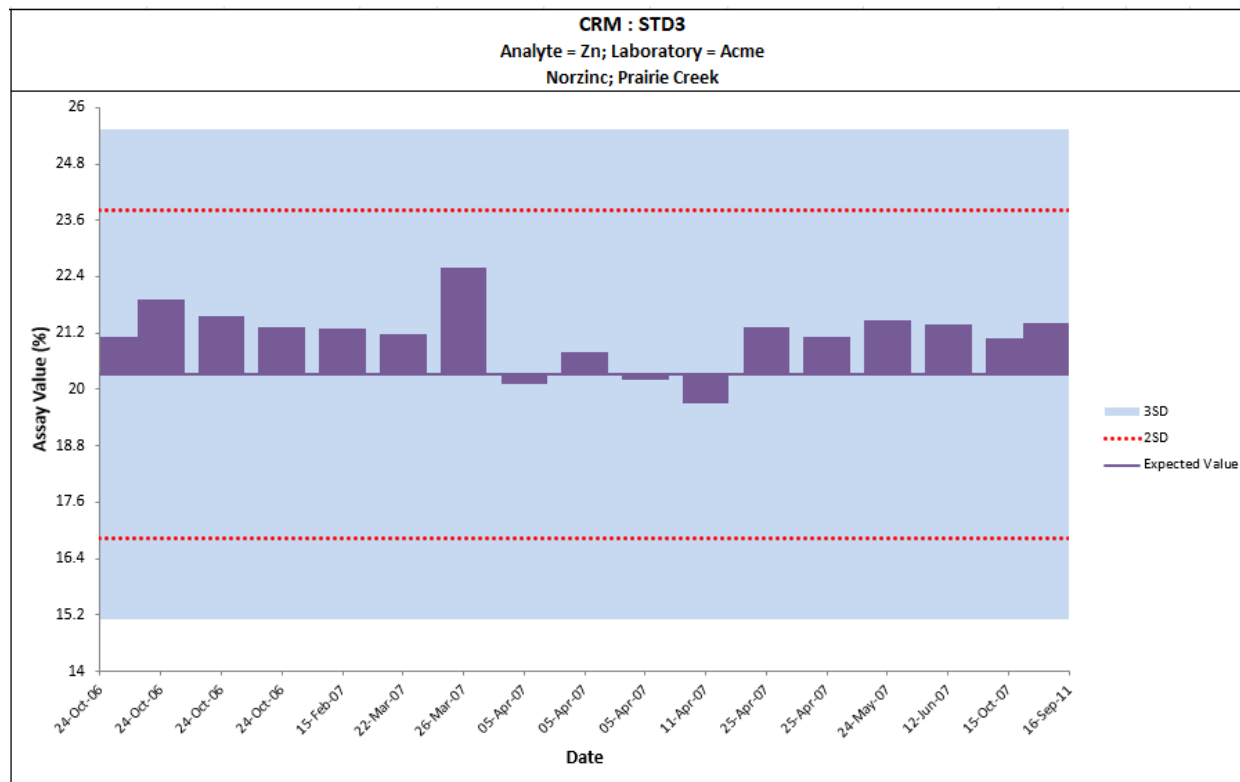
Source: Cube (2023)

Figure 11-10: Control Chart for Pb Reported for Standard 3 – ACME Laboratory



Source: Cube (2023)

Figure 11-11: Control Chart for Zn Reported for Standard 3 – ACME Laboratory



Source: Cube (2023)

11.6.2 Duplicates

A total of 282 duplicate sample assays has been reported for the Project. These duplicate analyses were reported from three different types of duplicates as outlined in Table 11-3.

Table 11-3: Number and Type of Duplicates by Year

Year	Duplicate Type		
	Field Duplicate	Coarse Reject Duplicate	Pulp Duplicate
1992	-	-	6
1993	-	-	37
1994	-	-	29
1995	-	5	10

Year	Duplicate Type		
	Field Duplicate	Coarse Reject Duplicate	Pulp Duplicate
1997	-	-	12
2001	-	5	5
2004	-	6	10
2006	-	8	14
2007	-	46	55
2011	9	-	-
2013	2	-	-
2015	17	-	-
2020	6	-	-
Total	34	70	178

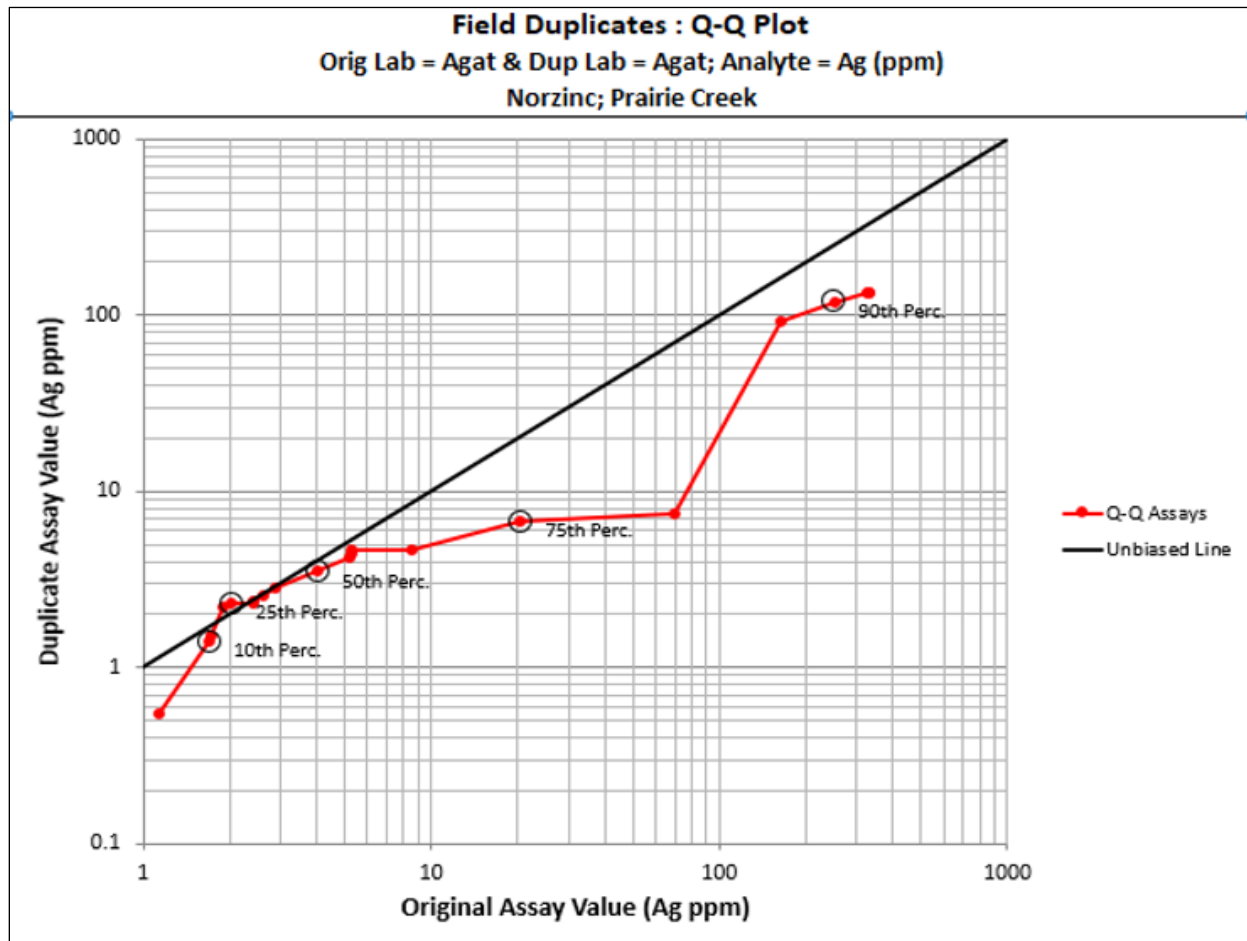
Source: Ausenco (2026)

11.6.2.1 Field Duplicates

Field duplicates comprise ¼ core duplicates. Because of the sample size difference between the primary and duplicate samples the results have limited value as QC samples. The presence of nuggety mineralization can also compromise the usefulness of this type of duplicate. Nevertheless, analysis of the duplicate performance was completed for these samples. Example Q-Q plots for Ag, Pb, and Zn from AGAT laboratory are presented in Figure 11-12 through Figure 11-14 respectively.

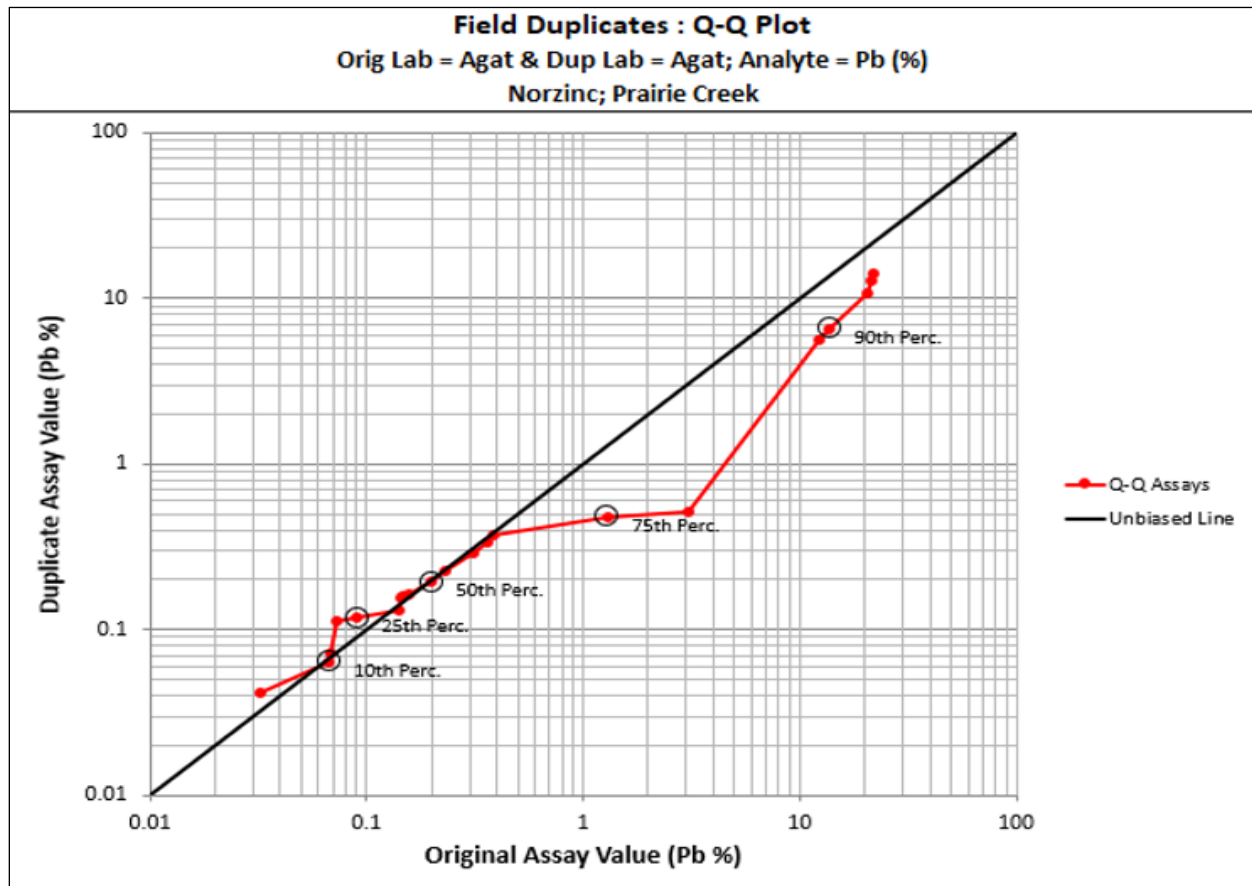
These charts show a significant bias between the original sample and the ¼ core duplicate result. As discussed, this is considered to be a consequence of the smaller duplicate sample size and the nuggety nature of mineralization, particularly for the Ag mineralization.

Figure 11-12: Q-Q Plot for Ag ¼ Core Field Duplicates – AGAT Laboratory



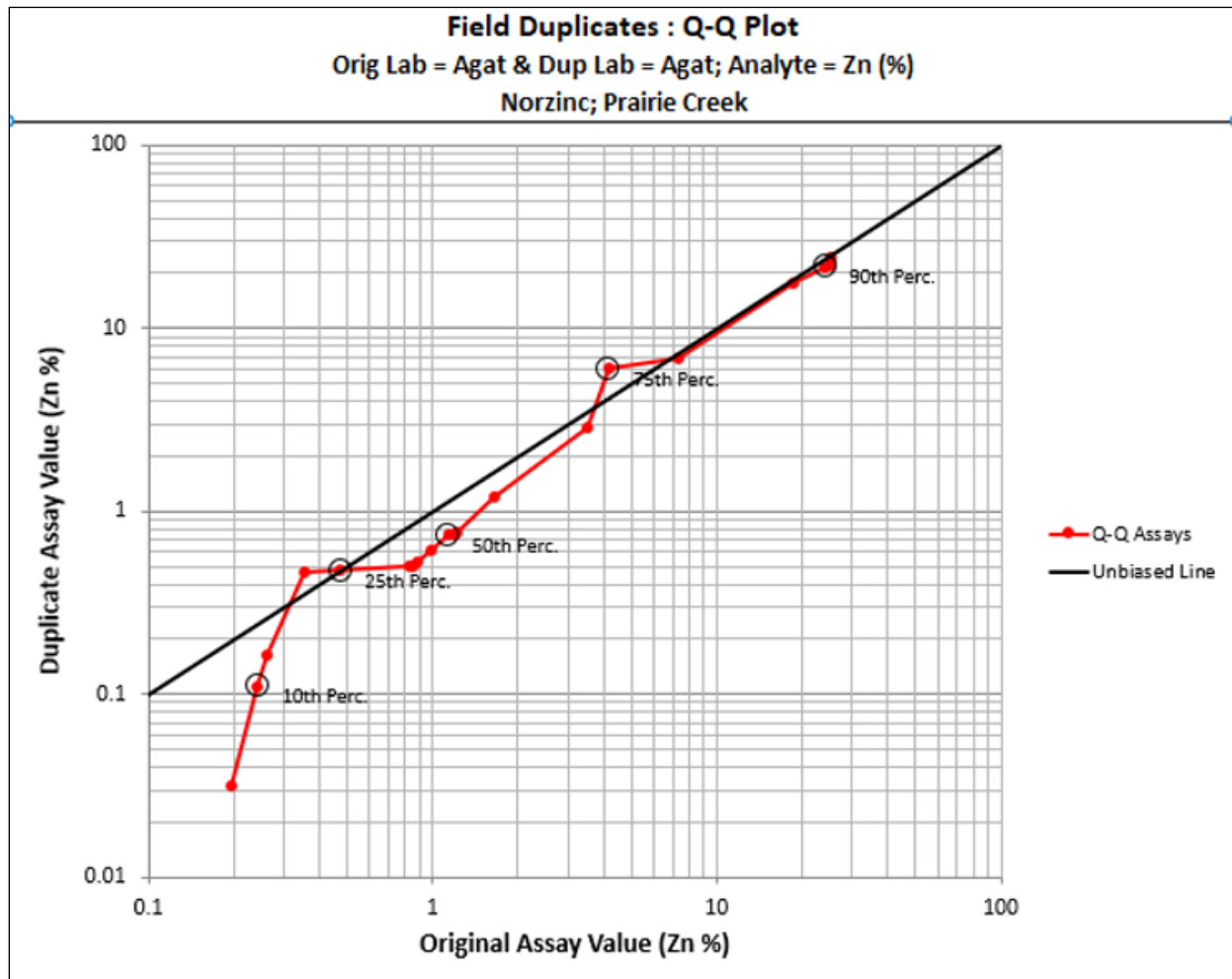
Source: Cube (2023)

Figure 11-13: Q-Q Plot for Ag ¼ Core Field Duplicates - AGAT Laboratory



Source: Cube (2023)

Figure 11-14: Q-Q Plot for Ag ¼ Core Field Duplicates – AGAT Laboratory



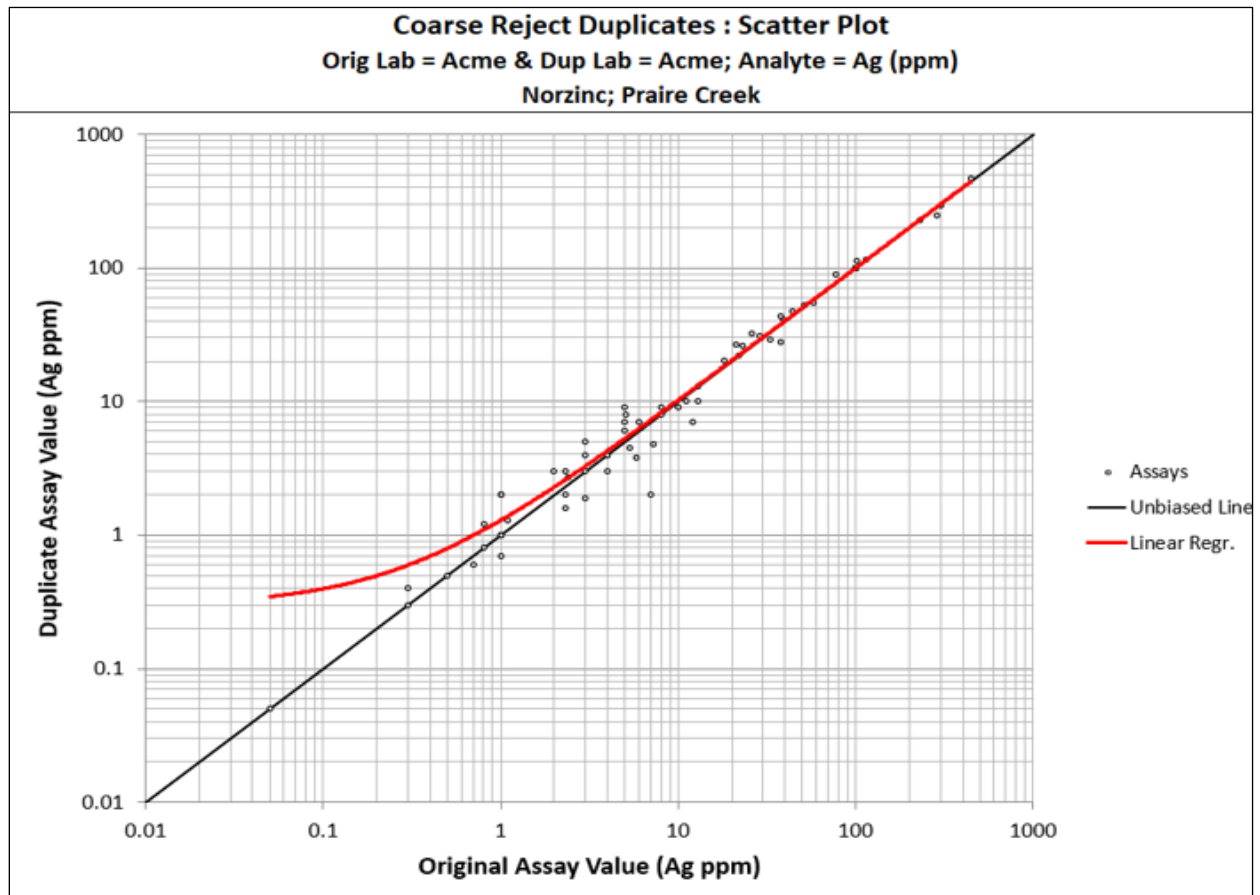
Source: Cube (2023)

11.6.2.2 Coarse Reject Duplicates

Coarse reject duplicates represent a secondary analysis of the rejected portion of the core sample following primary crushing and splitting. These rejects are commonly employed for diamond core as they avoid some of the issues related to nugget effect in ¼ core field duplicate analysis. As they are collected after the primary crushing stage, quantification of any potential laboratory contamination in the crushing stage is not possible.

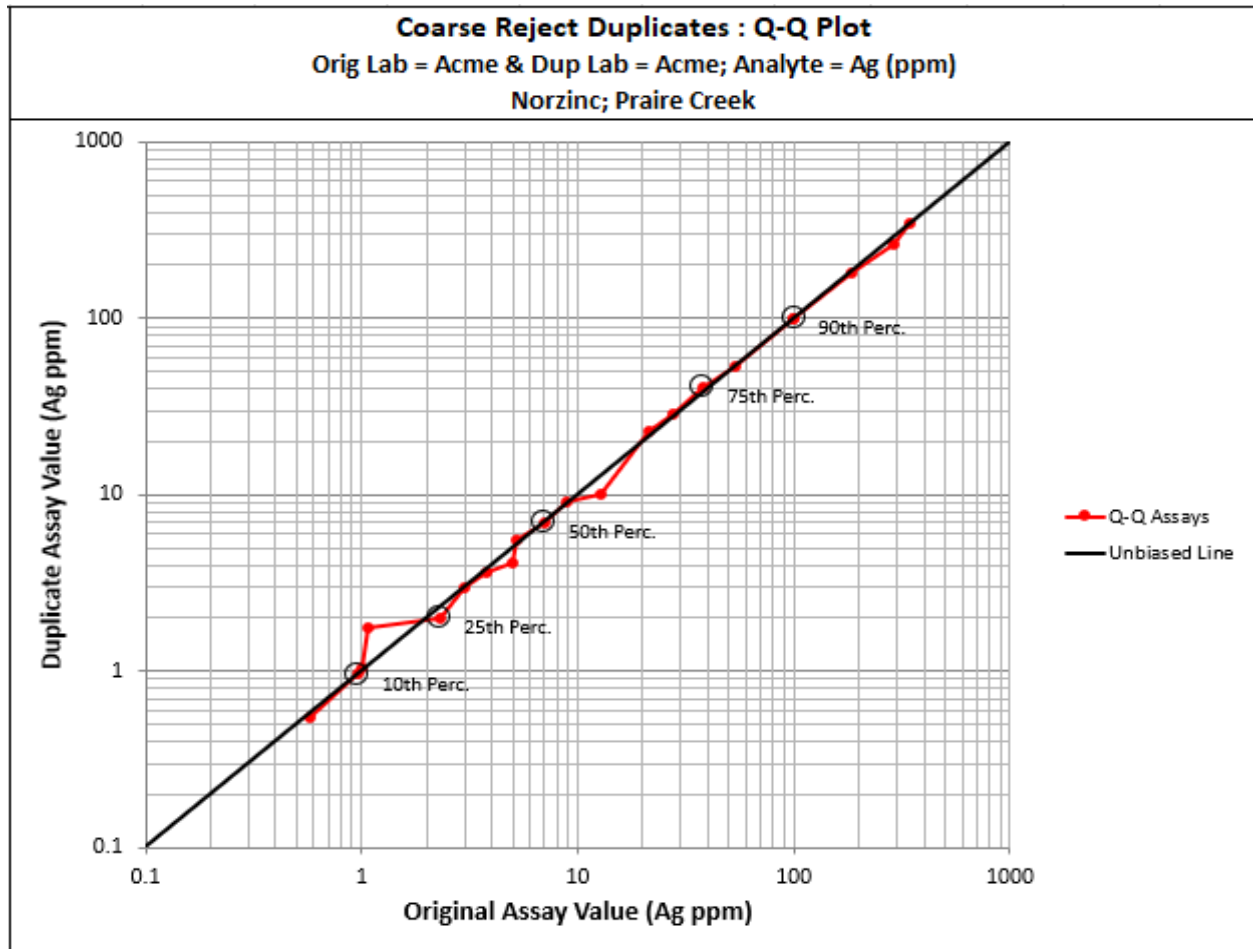
Review of results statistically and graphically suggests the coarse duplicates have performed well, with tight scatters and no perceptible bias in the Q-Q plots. Charts for Ag, Pb and Zn are presented in Figure 11-15 through Figure 11-20.

Figure 11-15: Scatter Plot for Ag Coarse Reject Duplicates – ACME Laboratory



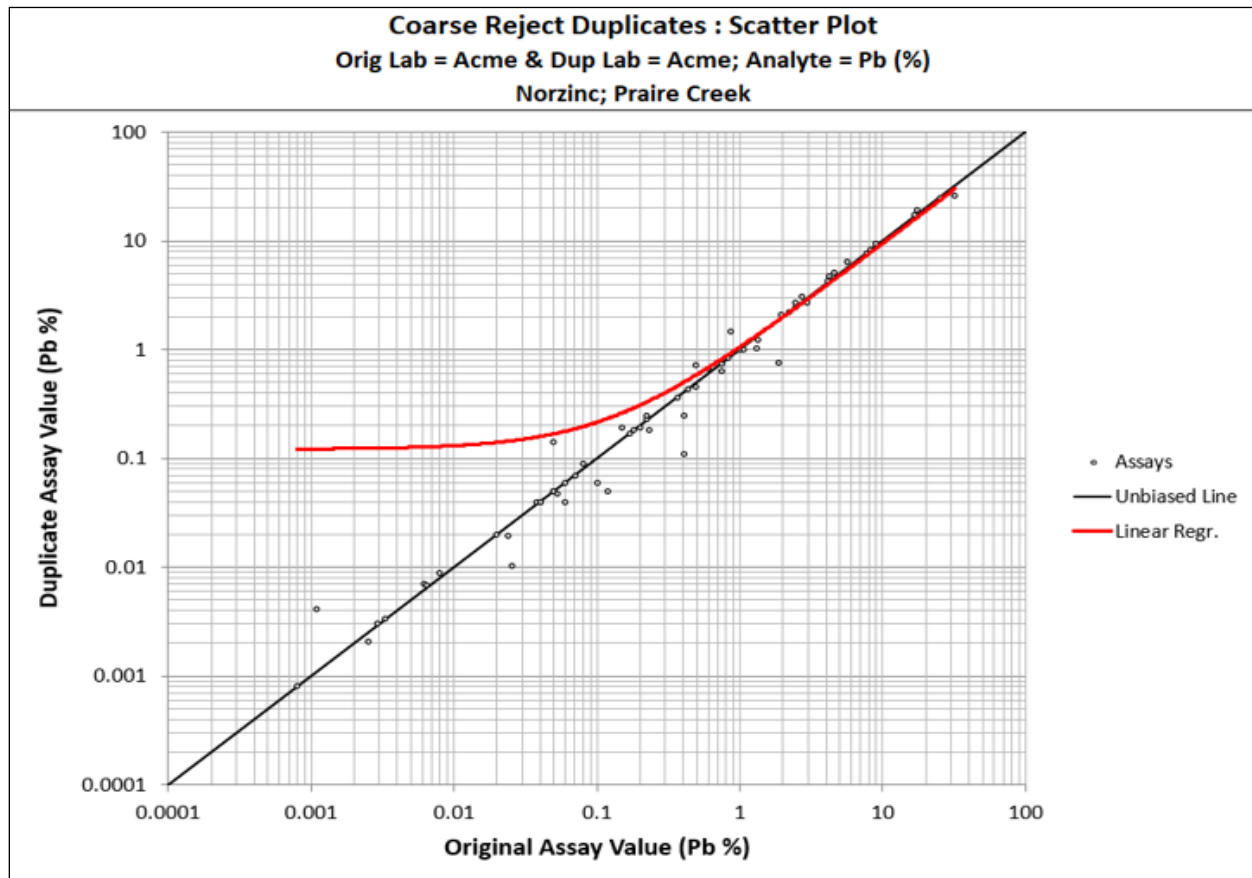
Source: Cube (2023)

Figure 11-16: Q-Q Plot for Ag Coarse Reject Duplicates – ACME Laboratory



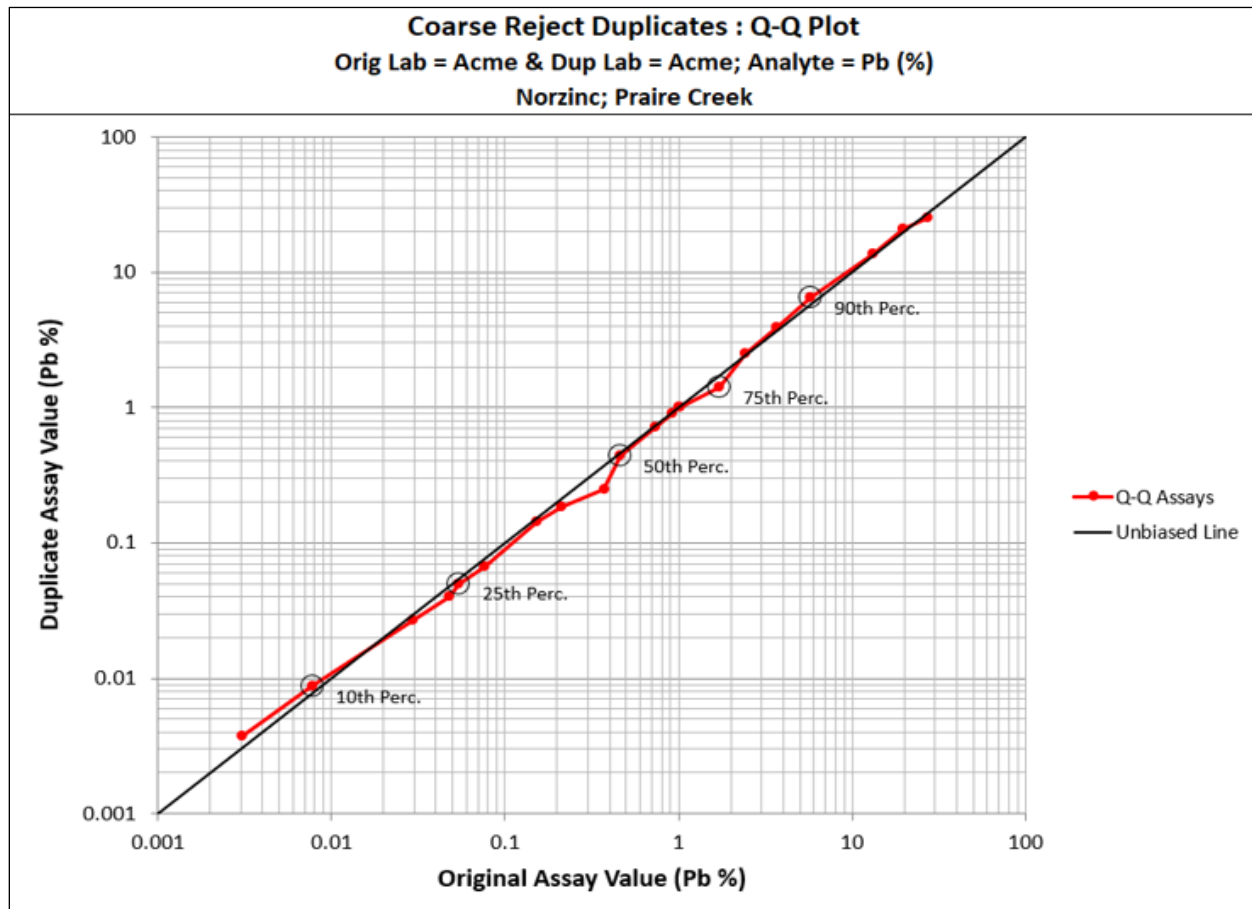
Source: Cube (2023)

Figure 11-17: Scatter Plot for Pb Coarse Reject Duplicates – ACME Laboratory



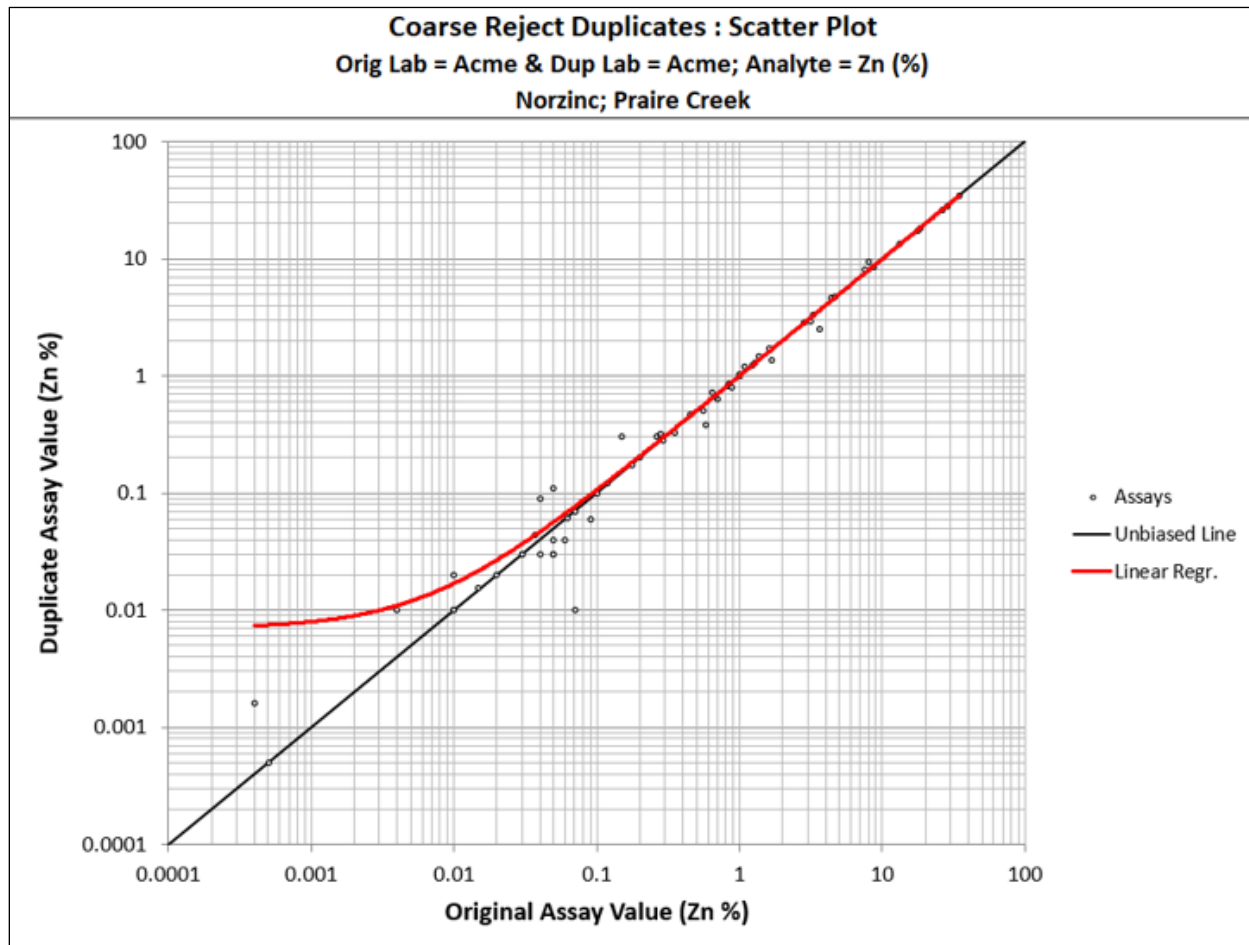
Source: Cube (2023)

Figure 11-18: Q-Q Plot for Pb Coarse Reject Duplicates – ACME Laboratory



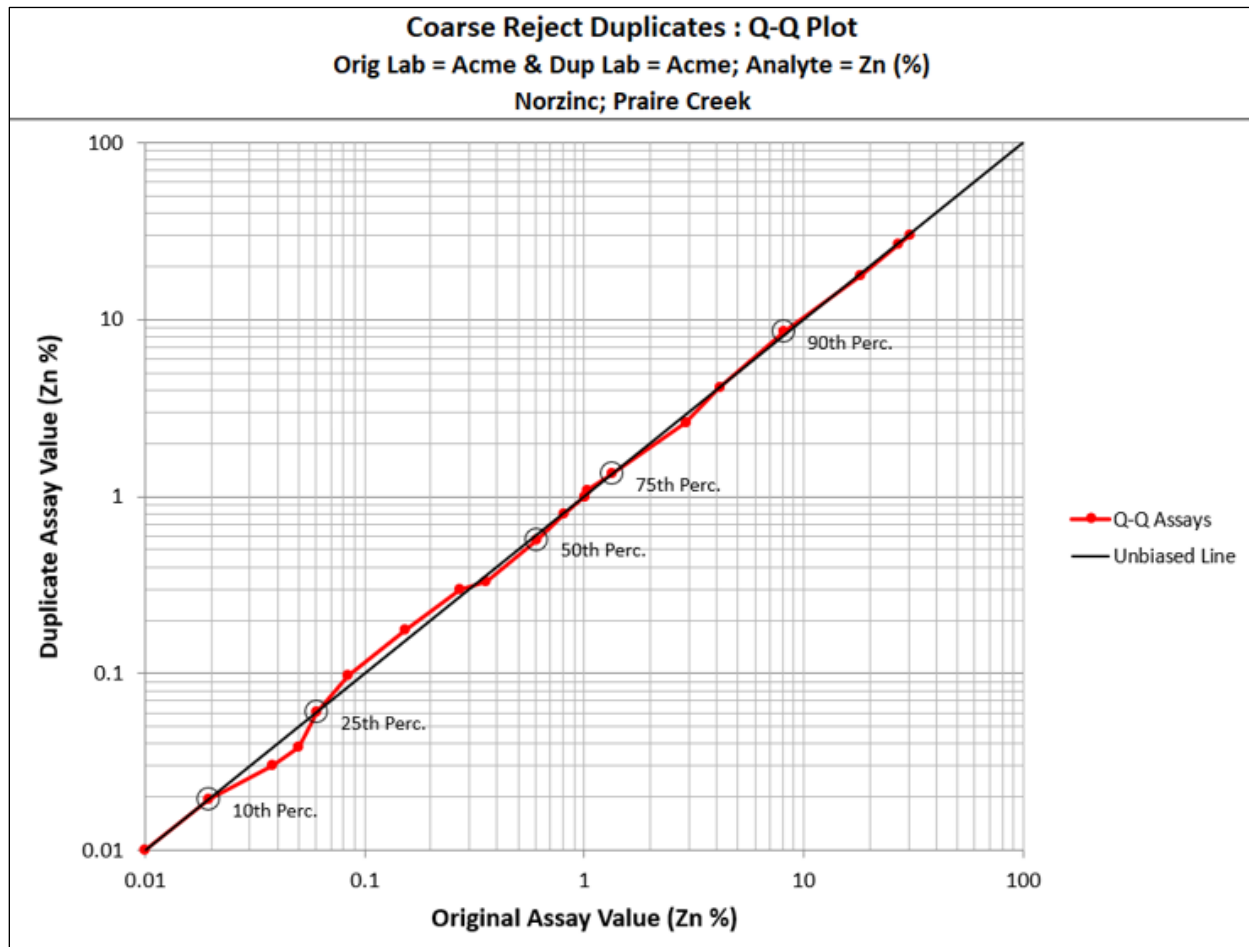
Source: Cube (2023)

Figure 11-19: Scatter Plot for Zn Coarse Reject Duplicates – ACME Laboratory



Source: Cube (2023)

Figure 11-20: Q-Q Plot for Zn Coarse Reject Duplicates – ACME Laboratory



Source: Cube (2023)

11.6.2.3 Pulp Duplicates

In total 178 pulp duplicates were submitted for analysis.

Although the Q-Q plots show no perceptible bias, the scatter plots have several outliers. When these were reviewed in more detail the large difference is inferred to be due to sample mix-ups. For example, there is an original Ag assay of approximately 1 g/t, but the repeat returned a value of approximately 375 g/t Ag. This is an unlikely result for a pulp sample. Overall, the performance of the pulp duplicates is acceptable.

11.6.3 Blanks

Information on blank performance for the Project includes both internal laboratory blanks, together with CRM blanks submitted as part of the assay dispatch. No review was completed of the internal blanks reported by the laboratory as these are not considered independent results.

The external blanks are described as “landscaping gravel” and as such are not a certified blank.

Review of the blank performance is complicated by the varying detection limits reported over time, but the results suggest the laboratories are maintaining a suitable standard of hygiene throughout the sample preparation process.

11.7 Comment on Sample Preparation, Analyses and Security

The following comments are made with respect to the quality of sampling and security processes applicable to the Project:

- Internal laboratory QC data was not analyzed as it is not considered to be independent;
- A review in terms of graphing the blank performance is difficult due to different detection limits over time and between laboratories. However, a visual inspection of the data indicates that there is no contamination;
- Coarse duplicates performed well indicating that the crushing and splitting process is satisfactory, however it is recommended that these be submitted blind to the laboratory;
- Only 34 ¼ core duplicates have been taken over the Project life, but this is not considered to be an issue as core duplicates have limited value in determining QA/QC performance. Coarse reject duplicates are more suitable, and these show acceptable performance;
- Pulps have performed satisfactorily, with no perceptible bias, however there are a few large outliers that indicate potential sample mix-ups;
- CRMs performed to an acceptable level; and
- The overall proportion of QA/QC samples against the total assays is approximately 10%.

Overall, it is the view of the qualified person that the sample preparation, security and analytical procedures are adequate, and the resulting assay data for the Project is fit for purpose and suitable for this Technical Report.

12 DATA VERIFICATION

12.1 Verification performed by QP – Greg Mosher

QP Greg Mosher, P.Geo., performed a random check of approximately 5% of the drillhole assays that were generated from the 2012 verification program until 2021, by comparing assay values in the database against the laboratory certificates. No discrepancies were found.

Canadian Zinc indicated to GMRS that the assay database in its entirety was rebuilt in early 2021 from original assay certificates as part of a comprehensive effort to re-import all certificate values to their certified levels of precision and to re-assay pulps for over limits in penalty elements, and in doing so corrected identified errors in mercury grades primarily from the 2006-2007 drill programs.

QP Greg Mosher, P.Geo., has conducted several site inspection visits. The latest was on June 16, 2026 for a period of one day. During that visit, the 870 mL portal area was inspected, and a review of mineralized core was conducted for each of the MQV, STK and SMS Zones. The previous visit was on October 8, 2021 during which the collar locations for the 2020/21 drillholes were inspected and photographed and GPS readings of the collar coordinates were collected. Mineralized intervals of drill core from hole PC-20-225 were examined and compared with written descriptions in the geology logs. Sample intervals recorded in the drill logs were also checked against the depth locations marked in the core boxes.

A total of ten pulp samples from various drill programs between 2011 and 2020 were collected and submitted to ALS in North Vancouver, BC. Samples were assayed for 41 elements using the analytical package ME-ICP41. Over limits for silver were re-run using GRA-21, over limits for mercury using HG-ICP42, and lead and zinc using ME-OG46h. All assay values for all elements compare closely; results for silver, lead, and zinc are shown in comparison with the original assay results in Table 12-1.

Table 12-1: Check Sample Results 2021

Original Samples					Check Samples (ALS 2021)		
Sample	Ag ppm	Pb %	Zn %	Lab	Ag ppm	Pb %	Zn %
25016	2	0.15	0.26	AGAT	3	0.17	0.36
25044	162	9.07	19.5	AGAT	173	10.5	24.3
25184	181	4.64	12.9	AGAT	175	4.84	11.2
25212	59	6.33	1.31	AGAT	53	5.87	1.29
25775	0	0.01	0.01	AGAT	1	0.01	0.01
25795	11	1.02	2.05	AGAT	10	1.02	2.12
398567	46	2.64	4.21	ALS	41	2.7	4.03

Original Samples					Check Samples (ALS 2021)		
Sample	Ag ppm	Pb %	Zn %	Lab	Ag ppm	Pb %	Zn %
1157285	124	2.21	0.65	ACME	117	2.37	0.69
1157496	17	0.62	0.03	ACME	16	0.69	0.04
D732517	433	10.05	21.3	ALS	441	10.3	21.6

Source: GMRS (2021)

12.1.1 Comment on Data Verification

The QP is of the opinion that the data is adequate for the purposes of this report.

12.2 Verification of Cost Information

Information and support data for the estimation of capital and operating costs was achieved through the following means:

- Site visit review into the condition of the surface facilities, processing plant, tailings storage facility conducted by QP Michael Makarenko, P.Eng., Tad Crowie, P.Eng., and Les Galbraith P.Eng.;
- Quotes from multiple suppliers and/or recent experience of major equipment costs;
- Multiple sources of labour costs were reviewed from other Canadian mines, an independent labour survey and experience corporate experience of JDS and Honey Badger Silver; and
- Electrical power cost estimates were summarized from power generation suppliers.

The QP Michael Makarenko, P.Eng. considers the cost information gathered for the report is suitable for a PEA.

12.3 Mining Data Verification

In support of the mine design, the block models were imported by the mining QP and validated against the Resource tables provided prior to any underground stope optimization and mine design. Geological wireframes were verified to match the spatial positioning of the block models. The site topography was validated against the geology surfaces provided with the block models.

The QP Michael Makarenko, P.Eng. is of the opinion that the data is adequate for the purposes of this PEA report.

12.4 Metallurgical Data Verification

The QP has reviewed the metallurgical testwork data and confirmed that standard metallurgical testwork protocols were used. The testwork was conducted at a lab that is well known for conducting quality metallurgical testwork. A review of testwork assays confirmed good assay reconciliation between tests and composite assays.

The QP Tad Crowie, P.Eng. is of the opinion that the data is adequate for the purposes of this PEA report.

12.5 Mine Geotechnical Data Verification

Geotechnical core logging data was collected from relevant intervals for several holes during historic exploration drilling campaigns. The level of logging oversight and QA/QC of the data are unknown. JDS has reviewed the data and spot-checked data against respective core photographs, where available, to confirm data quality and internal consistency. Although inconsistencies were noted, JDS considers the data overall to provide a reasonable representation of rock quality for the respective areas and generally reliable for a PEA level of study.

12.6 Waste & Water Management Data Verification

QP Les Galbraith P.Eng. visited the site on June 16, 2026. The site visit included observing the existing infrastructure, including the unused water storage pond constructed in 1982, and the planned locations of the waste rock pile and collection pond, and the filtered tailings stockpile. Geotechnical data collected in previous site investigations was reviewed to support the PEA evaluation of the existing water storage pond, the waste rock pile and collection pond, and the filtered tailings stockpile. The available data is considered adequate to support the current level of study. Additional geotechnical information is required to support the next level of study as discussed in Section 26.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

Metallurgical testwork was performed on PC Silver Mine beginning in the 1960's, although none of the earliest studies, including information relating to the existing circuit design, are currently available. Following this and continuing sporadically into the 1980's, and as recently as 2016, more testing using a modified flotation scheme for high oxide feed was conducted. There are three principal types of mineralization that represent the resource from the PC Silver Mine. These consist of the MQV, STK, and SMS, with the MQV as the most prominent style of mineralization. The pre-2017 metallurgical testwork was performed on MQV and SMS mineralization by several different laboratories. For MQV the historic testing had been limited to samples readily available from the bulk sampling areas in mine adits, closer to surface. This material contained more highly oxidized mineralization. MQV was then described as primarily massive to semi-massive galena and sphalerite, with varying degrees of cerussite (lead oxide) and smithsonite (zinc oxide) in a quartz-carbonate-dolomite matrix. The SMS samples, which were obtained from drilling, were described as more finely grained and with lower sulphide oxidation content. The samples used for testing prior to 2017 are not considered representative, as the oxide content was elevated well above that indicated in the most recent mine schedule. In 2017 less oxidized material for metallurgical testing was sourced from deeper zones in the resource that originated from a 2015 exploration drilling program.

A summary of the pre-2017 metallurgical test programs are discussed below in Section 13.1, while the 2017 test program is outlined in Section 13.2.

The information presented in this section has been substantially derived from the 2021 PEA Technical Report authored by Ausenco for this property. The content of this section has been reviewed and updated, where appropriate.

13.1 Historical Testwork

The metallurgical testwork conducted prior to 2017 test was performed principally on MQV and SMS mineralization. This work was limited to samples that could be obtained from the existing surface stockpiles and near surface underground mine workings. These contained more highly oxidized material often averaging from 4% to 8% lead oxide, and 3% to 7% zinc oxide. Based on the current mine plan, this average extent of oxidation would be more than double that of the highest oxide content of the corresponding base metals content expected for mill feed, and well above those expected for LOM. Consequently, most of the historic testwork related to flotation response is of limited value.

Historic testwork was summarized during previous technical reporting and is outlined in Table 13-1.

Table 13-1: Major Historical Metallurgical Test Programs

Year	Program ID	Laboratory	Flotation	Grindability	Mineralogy	Others
1960 - 1980	Unknown	Unknown	Unknown			
1980 - 1990	KM019, KM034, KM040, KM048, KM077, KM081, KM370, KM424, KM440, KM454, KM462, KM469, KM474, KM488, KM497	Kamloops/G&T	✓	✓	-	✓
1980	L.R.2252	Lakefield	✓	-	-	-
1982 -1983	N20481/ NP831003	CSMRI	✓	-	✓	✓
1993 -1994	X93-112/X94-006	Cominco	-	-	✓	-
1997	9197-01	Hazen	-	✓	-	-
1997	97-099	PRA	-	-	-	✓
1992	MT-9303	De Randt Corp	-	-	✓	-
2000	00-90	Harris Exploration Services	-	-	✓	-
2000	-	UBC	✓	-	-	-
2006	MS-06 Jun-001/ MS-06Aug-001	Terra	-	-	✓	-
2004 - 2009	10916/11098/12018	SGS-Lakefield	✓	✓	✓	✓
2011	SE-1389-TR	Outotec	-	-	-	✓
2013	50242-001	SGS-Lakefield	✓	-	-	✓
2014 - 2016	14002	GMR	✓	-	-	✓

Notes:

- a) Kamloops = Metallurgical Services Ltd/Kamloops Research & Assay Laboratory Ltd./G&T Metallurgical Services Ltd.
- b) De Randt Corp = De Randt Corp Mineral Technologies Group, Division of De Randt Corp Enterprises
- c) Terra = Terra Mineralogical Services
- d) Harris = Harris Exploration Services
- e) Hazen = Hazen Research Inc.
- f) Cominco = Cominco Exploration Research Laboratory
- g) Lakefield = Lakefield Research of Canada Limited
- h) SGS Lakefield = Lakefield Research Limited
- i) CSMRI = The Colorado School of Mines Research Institute
- j) UBC = University of British Columbia
- k) Outotec = Outotec (Canada) Ltd.

Early studies were not well documented or are not available, but beginning in 1980, a variety of test programs were performed to investigate metallurgical responses and have been reported. Among the principal programs were various studies including pilot plant testing of a sulphide

flotation circuit conducted by Colorado School of Mines Research Institute (CSMRI) in the early 1980's. Following this, the majority of the work was conducted at G&T and its predecessor laboratory.

Kamloops Metallurgy (KM), located in Kamloops, BC, which focused on flotation from 2004 through to 2009. During this time and more recently SGS (including a predecessor company Lakefield Research) performed considerable testing, primarily at their facilities in Lakefield, Ontario. The SGS testing included considerable bench testing as well as bulk sample evaluation using dense media separation, followed by flotation. Other laboratories performed additional programs as described in the following sub-sections.

13.2 2017 Test Program

The 2017 test program was performed through SGS Canada Inc., Mineral Services (SGS) at their Canadian facilities in both Burnaby, BC, and Lakefield, Ontario. The program was conducted in two phases for Canadian Zinc. Phase 1 evaluated low oxide material, from deeper within the resource on samples obtained from a drill program conducted in 2015. The Phase 1 final report was issued by SGS in July, 2017. SGS Project 16016-001 Final Report: - Dense Media Separation, Comminution, Flotation and Solid-Liquid Separation on Samples for the PC Silver Mine, Prepared for Canadian Zinc., July 28, 2017.

This was followed by a second phase of testing, which consisted of blended material to note the flotation response that would better represent oxide content in the initial years of production. The Phase 2 SGS report was issued on December 4, 2017 SGS Project 16016-001 Final Report II: - Flotation of Oxidized Samples for the Prairie Creek Zinc Lead and Silver Project., December 4, 2017.

The primary objectives of the 2017 test program were to determine the mineral processing response of a more representative sample than what had been conducted from the 1980's up until 2016. An additional objective at that time was to attempt to make use of the existing site infrastructure and equipment, along with the addition of dense media separation to the flowsheet.

Mineral processing testing was performed by SGS, primarily at their laboratory in Burnaby, BC. This included quantitative evaluation of minerals by scanning electron microscopy (QEMSCAN), grinding, flotation, and related studies. Procedures including dense media separation piloting, and liquid – solid separation, as well as some of the required analytical procedures were conducted at the SGS facility in Lakefield, Ontario.

The pre-concentration conditions were based on historical testwork and were incorporated into the 2017 program. This consisted of crushing to 12.7 mm (1/2") and screening fines at 1.4 mm (12 Tyler mesh). The -12.7 mm +1.4 mm material was subjected to dense media separation using a ferrosilicon media adjusted to SG 2.75. Dense media separation sinks and the screened -1.4 mm fines were combined and forwarded to evaluation using various differential flotation procedures. Based on the grades of the 2017 lab samples there were insufficient oxide lead/zinc minerals or copper present to justify evaluating separate circuits for recovering these minerals. Consequently, the differential flotation procedures focused on optimizing concentrating the galena and sphalerite. Follow-up settling and pressure filtration testing was performed on the final lead and zinc concentrates, as well as on the flotation tailing.

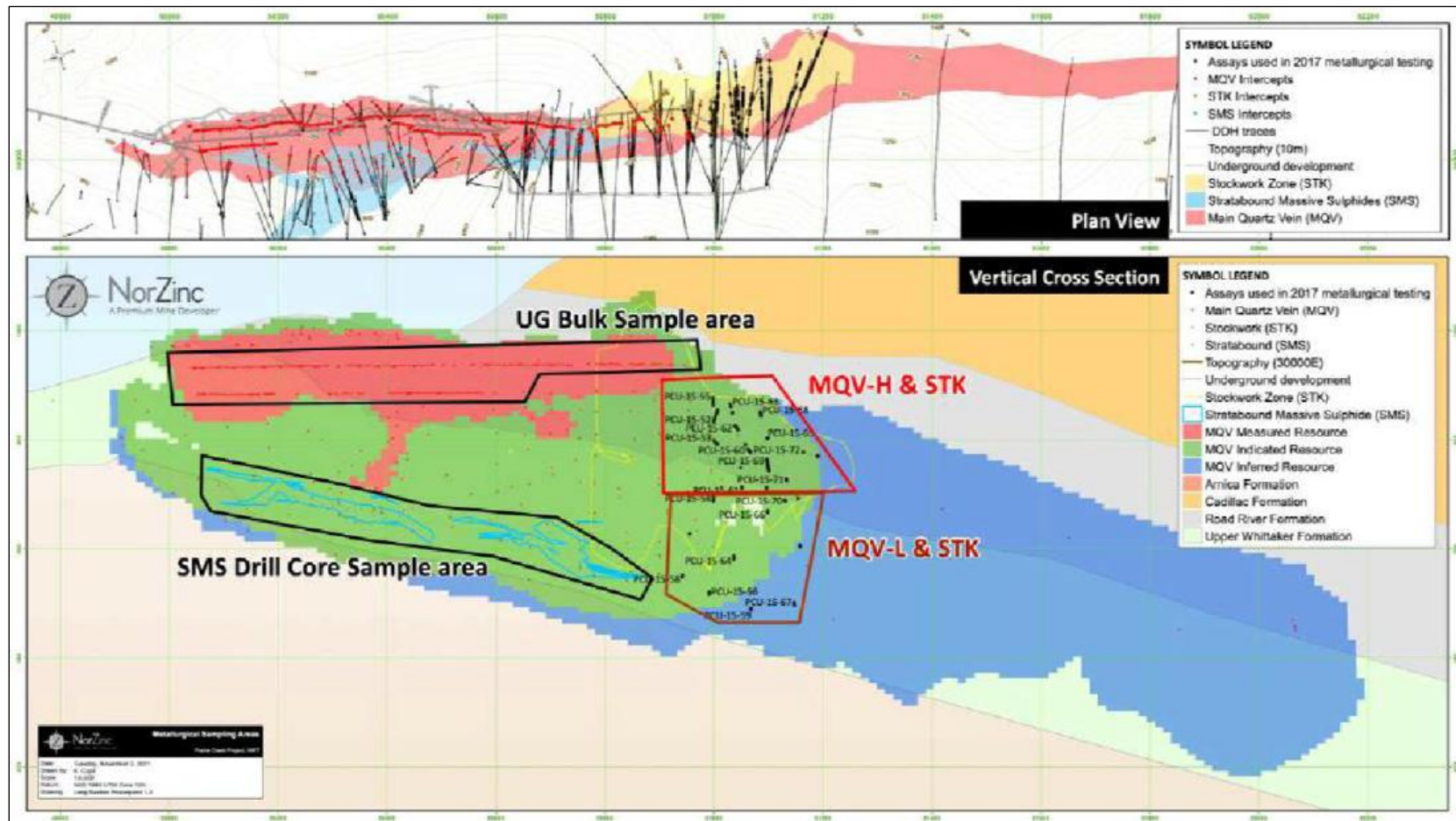
In January 2017, the Phase 1 program was initiated on samples representing previously untested zones of the resource. These consisted of deeper zones in the MQV and for STK material. The testwork focused on simplifying the historic lab flotation procedure, reagent scheme, and better optimizing the grind. At the conclusion of Phase 1, a modified flotation procedure has been developed.

Later in 2017 a second phase test program was conducted at SGS to evaluate a higher oxide feed content. This feed was considered more typical of what would be expected in the initial years of production. Representation for oxide content was based on the most recent available mine plan, which was correlated to lead oxide. Zinc oxide mineralization generally trended below that of lead. The mine plan that was referenced called for between 2% to 2.5% lead oxide during start-up and commissioning, which is expected to last 4-6 months. Following this, the first year of mill feed is anticipated to average on a monthly basis between 1.5% to 2% lead oxide, before dropping below 1.5% near the middle of Year 2. With some minor fluctuation, the oxide content of both lead and zinc after the initial period then generally decreases for the LOM. The Phase 2 testwork initially tested the earlier developed flowsheet, as well as additional modified oxide treatment procedures. As the Phase 2 program developed, it followed the similar flotation procedures developed during Phase 1.

13.2.1 Origin of Metallurgical Composite Sample

Three master composites originating from 2015 split drill core were primarily used for testing flotation procedures and flowsheet development in Phase 1. These composite identifications consisted of the STK, and two areas of greater depth in the MQV, broken out initially as upper MQV zone (MQV-H), and lower MQV (MQV-L) zone. The zones from which the samples were obtained are shown in Figure 13-1 below and also show the bulk sampling areas of MQV and SMS that were used in historical testwork.

Figure 13-1: Sample Locations



Note:

Figure prepared by Canadian Zinc, 2021.

Source: Ausenco (2021)

The program consisted of generating 3 area master composites, a preliminary assay reject composite, and 3 variability assay reject master composites:

- Master Composite STK
- Master Composite MQV-H
- Master Composite MQV-L
- Scoping Composite MQV-AR
- Variability Composite MQV-Fe
- Variability Composite MQV-Cu
- Variability Composite MQV-Pbox

The variability samples all originated from the MQV zone. These consisted of composites with higher iron content (MQV-Fe), higher copper content (MQV-Cu), and variable lead oxide content (MQV-Pbox), as compared to the MQV master composites. As with MQV-AR, the three variability composites originated from assay rejects. As a result of the finer particle size of minus 3.6 mm this did not allow the assay rejects to be subjected to dense media separation.

For the Phase 2 test program the objective was to evaluate samples with a higher oxide content than Phase 1. It was composited to better represent material scheduled for the initial 3-5 years of production. This was blended from the pre-existing composites from Phase 1 shown above, with additional elevated sulphide oxidation. The higher oxide material was primarily obtained from the existing underground mine workings at the 930 level of crosscut 11. This material was obtained in May 2017 and assayed 286 g/t Ag, 16% Pb and 33.6% Zn. The oxide content was 3.68% Pb oxide and 2.37% Zn oxide. Further re-blending of this material with other assay rejects that were archived was also performed. Ultimately there were three master composite samples used in the Phase 2 test program. These were labeled as Comp. PB1, PB2, and PB3, to target a composite oxide lead content of 1%, 2%, and 3%, respectively.

13.2.2 Composite Head Characterization

The six master composites from both the Phase 1 and Phase 2 study originated from specific spatial zones and corresponding variations in head grade, oxide content and mineralogy from material both obtained near surface and at greater depth in the resource than those tested prior to that time.

The head assays to the crushing circuit for metal values, and other elements of interest including those potentially deleterious to the process or smelter terms are provided in Table 13-2 below. These consist of the three master composites used in Phase 1, and three master composites in Phase 2 testing.

Table 13-2: Master Composite Head Analyses

Sample ID	Phase	Pb %	Zn %	Pb Oxide as Pb %	Zn Oxide as Zn %	Ag g/t	Cu %	Fe %	S %	S= %	Hg g/t	Sb g/t	As g/t
STK	1	3.86	7.46	0.43	0.029	58.6	0.18	0.956	5.19	4.85	117	680	558
MQV-H	1	14.7	16	0.39	0.051	222	0.4	0.316	10.1	9.82	436	2050	949
MQV-L	1	5.9	4.74	0.39	0.017	84	0.18	2.33	5.66	5.42	162	917	552
PB1	2	10.6	11.5	1.01	0.51	160	0.26	0.85	7.46	7.2	287	1290	105
PB2	2	11.9	12	2.04	1.21	170	0.27	0.77	7.05	7	378	1220	121
PB3	2	12.3	12.1	3.11	1.86	170	0.29	0.72	6.87	6.58	471	1180	125

Source: Ausenco (2021)

Composite grade variation included following the principal metals of value consisting of lead, zinc, and silver grades. Potential penalty elements reporting to the flotation concentrate, such as mercury, arsenic and antimony were analyzed. The variation in grade for potential detrimental minerals that can impact flotation response was also monitored. This includes iron (Fe) in relation to pyrite, and graphite as quantified by total organic carbon analyses. For the master composites these parameters were also re-analyzed following dense media separation treatment.

Similarly head assays for composites generated from assay rejects were analyzed and are provided in Table 13-3.

Table 13-3: Assay Reject and Variability Composites Head Analyses

Sample ID	Pb %	Zn %	Pb Oxide as Pb %	Zn Oxide as Zn %	Ag g/t	Cu %	Fe%	S %	TOC leco %	Hg g/t	Sb g/t	As g/t
MQV-AR	10.8	11.6	n/a	n/a	149	0.25	0.32	n/a	n/a	246	1400	<1000
MQV-Fe	6.11	2.1	0.38	0.008	137	0.37	5.28	7.95	0.15	106	1660	159
MQV-Cu2	11.3	14.6	0.32	0.022	236	0.65	1.08	9.8	0.21	273	3070	145
MQV-PbOx	11.9	11.2	0.4	0.017	133	0.17	0.47	7.3	0.4	158	847	< 70

Notes:

*n/a = not available

TOC = total organic carbon

Source: Ausenco (2021)

Mineralogical evaluation representing float feed for two of the master composites (MQV-H and STK) were performed by QEMSCAN. Note that this represents the material following rejection of majority of gangue minerals to the dense media separation float. The related information was issued by SGS on March 6, 2017. A synopsis of the results is provided below in Table 13-4 and Table 13-5 respectively for MQV-H and STK.

Table 13-4: Mineral Distribution Float Feed MQV-H

Fraction		Combined	+75 µm		-75 µm	
Mass Size Distribution (%)			33.7		66.3	
Calc. ESD Particle Size (µm)		21	85		16	
		Sample	Sample	Fraction	Sample	Fraction
	Sphalerite	30.6	12.9	38.2	17.7	26.7
	Galena	16.3	3.4	10.1	12.9	19.5
	Tetrahedrite	1.25	0.37	1.09	0.88	1.33
	Pyrite	0.39	0.05	0.16	0.34	0.51
	Other Sulphides	0.05	0.01	0.04	0.04	0.06
	Quartz	12.6	5.08	15.1	7.47	11.3
	Feldspars	0.17	0.04	0.12	0.13	0.19
Mineral Mass (%)	Micas	2.77	0.58	1.73	2.19	3.3
	Other Silicates	0.45	0.1	0.3	0.34	0.52
	Dolomite	34.7	11	32.7	23.7	35.8
	Calcite	0.39	0.12	0.37	0.26	0.4
	Oxides	0.16	0.01	0.04	0.14	0.22
	Other	0.18	0.02	0.05	0.16	0.24
	Total	100	33.7	100	66.3	100

Source: Ausenco (2021)

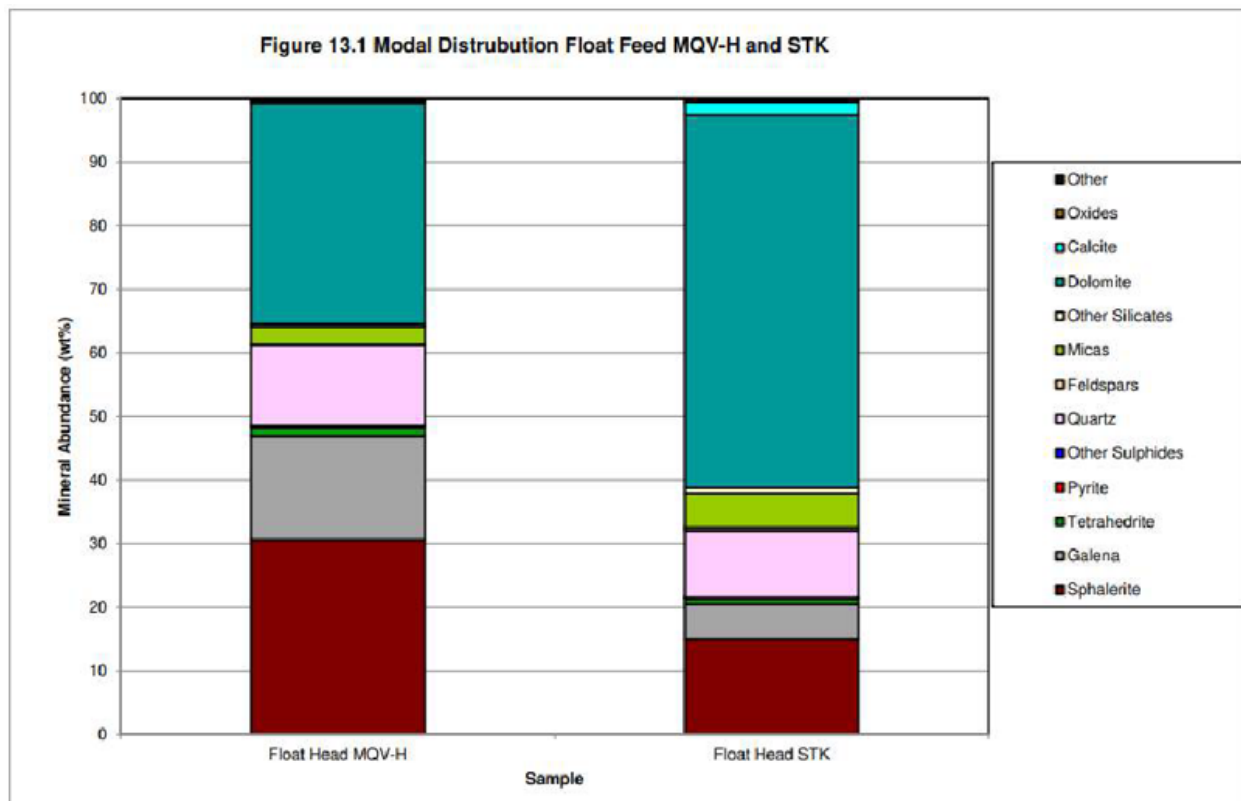
Table 13-5: Mineral Distribution Float Feed STK

Fraction Combined		+75 µm		-75 µm	
Mass Size Distribution (%)		30.8		69.2	
Calc. ESD Particle Size (µm)		86		15	
	Sample	Sample	Fraction	Sample	Fraction
Sphalerite	15	6	19.5	9.03	13
Galena	5.51	1.37	4.44	4.14	5.98
Tetrahedrite	0.81	0.25	0.82	0.56	0.81
Pyrite	0.17	0.03	0.11	0.13	0.19
Other Sulphides	0.01	0	0	0.01	0.02
Quartz	10.5	3.96	12.9	6.54	9.45
Feldspars	0.49	0.15	0.48	0.34	0.5
Micas	5.44	0.98	3.18	4.47	6.45
Mineral Mass (%)	Other Silicates	0.81	0.2	0.66	0.6
	Dolomite	58.7	17.1	55.5	41.6
	Calcite	1.99	0.68	2.21	1.32
	Oxides	0.27	0.03	0.09	0.24
	Other	0.3	0.04	0.13	0.26
	Total	100	30.8	100	69.2

Source: Ausenco (2021)

The results confirm MQV-H has significantly higher content of sphalerite and galena as compared to STK. The other main sulphide minerals present consist of tetrahedrite, followed by pyrite. The major gangue mineral remaining is dolomite. This is further represented in Figure 13-2 below.

Figure 13-2: Modal Distribution Float Feed MQV-H and STK



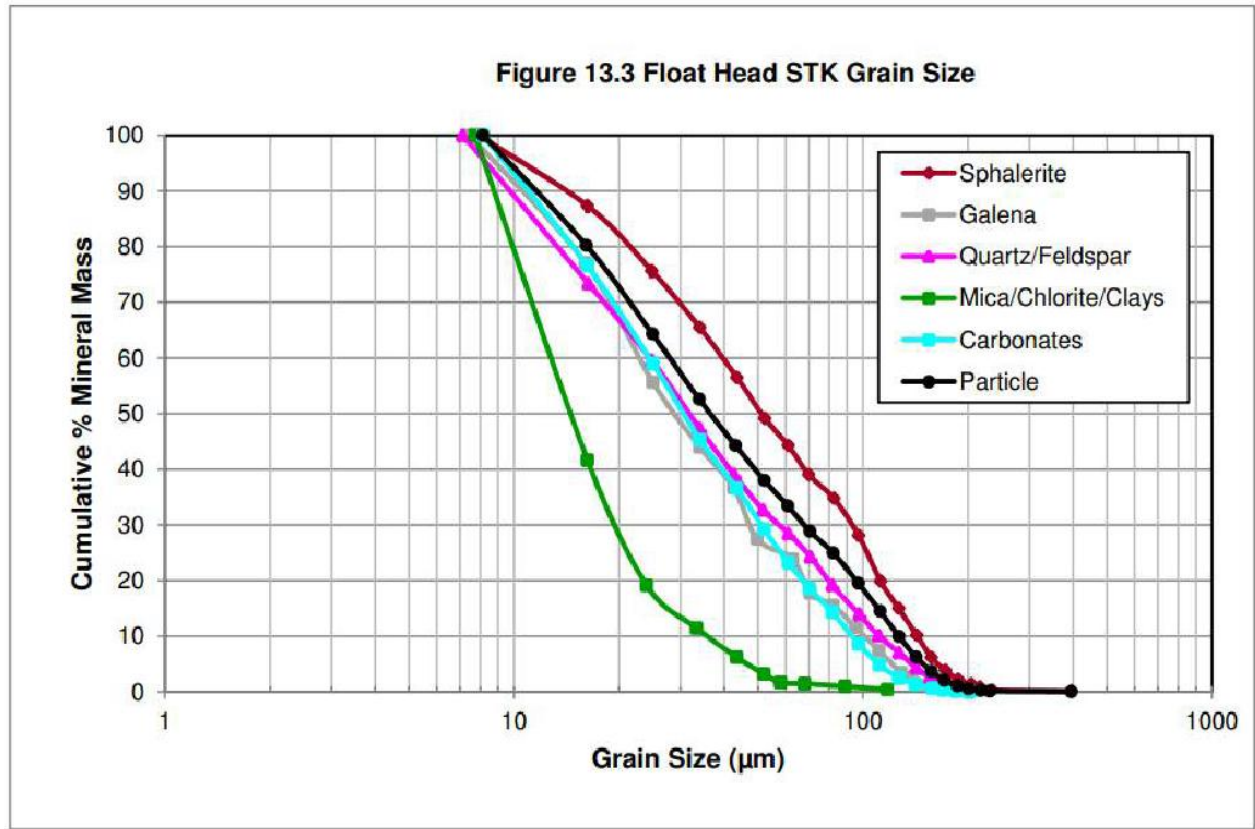
Note:

Figure prepared by SGS, 2017.

Source: Ausenco (2021)

Size distribution of the major minerals is provided in Figure 13-3 and Figure 13-4, respectively for STK and MQV-H.

Figure 13-3: Float Head STK Grain Size

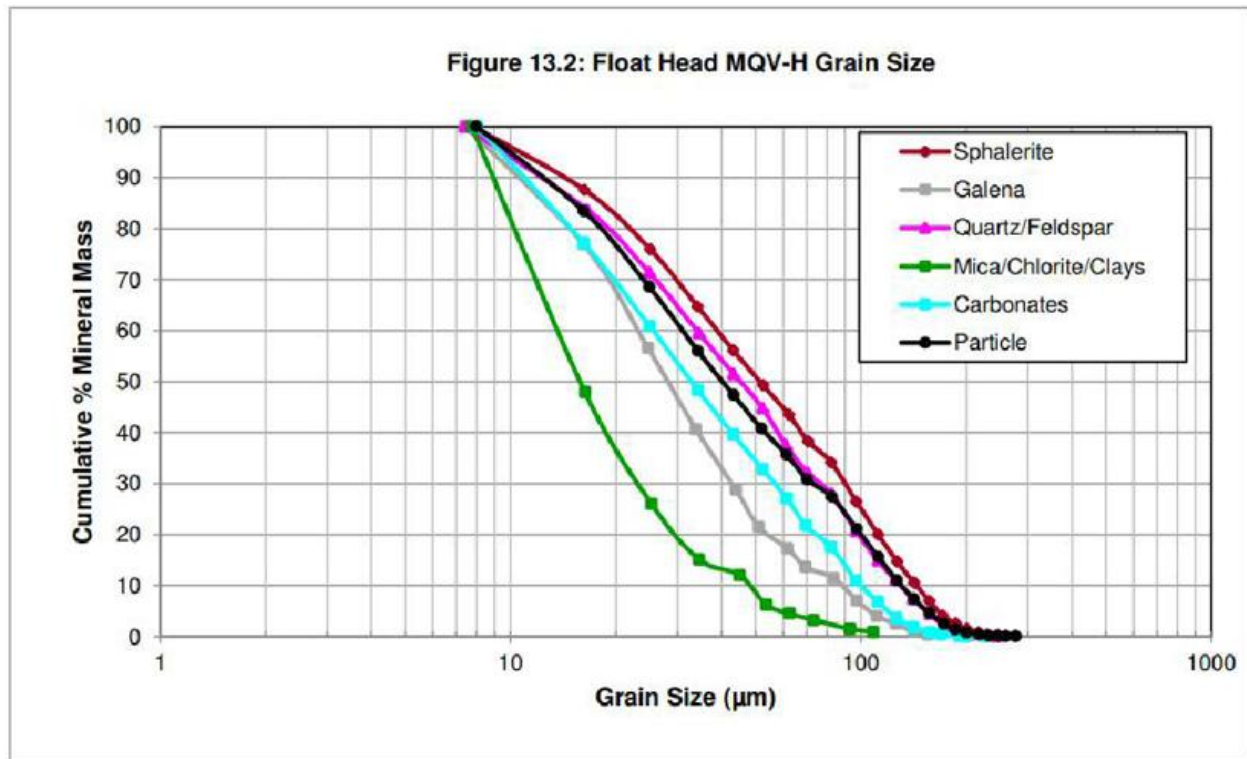


Note:

Figure prepared by SGS, 2017.

Source: Ausenco (2021)

Figure 13-4: Float Head MQV-H Grain Size



Note:

Figure prepared by SGS, 2017.

Source: Ausenco (2021)

Examination indicated that for both these master composites the galena and sphalerite particles showed good liberation, and the samples should respond well to standard flotation procedures at moderate grinds.

Further evaluation of physical characteristics of potential mill feed material was sent to Jenike and Johanson Laboratory in Mississauga, Ontario (Jenike). The material selected was described as a MQV blend of -6 mesh assay rejects, primarily consisting of sulphides in quartz veining and dolomite. The sample was observed to be free flowing (not appearing to be sticky) or have a high clay content or generate anomalous dust. Jenike advanced the material for basic physical testing including compressibility, flow function and wall friction.

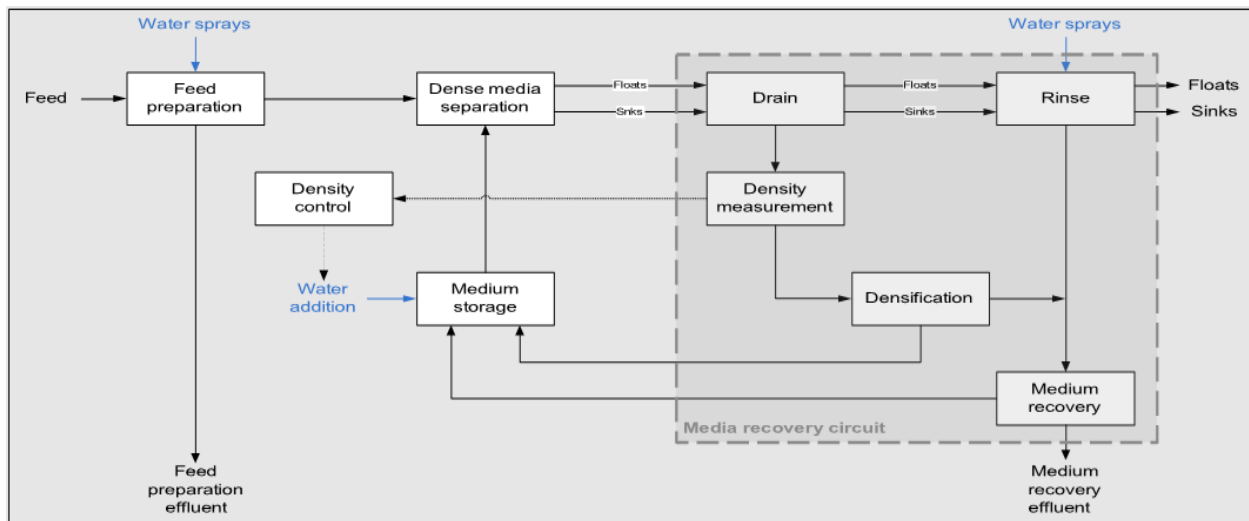
Among the findings of the Jenike study was the tested material provided a 50% particle size distribution of 0.83 mm, and a moisture saturation of 10.6%. Further studies related to developing flow properties, including minimum chute outlet sizes, and mass flow wall angles at various moisture content, bulk densities, and materials of construction. The findings are presented in Jenike and Johanson Ltd.: - Report 70809-1 Rev. 1, Canadian Zinc Prairie Creek, Northwest Territories, Flow Property Test Results for Zinc Lead Silver Ore, July 28, 2017.

13.2.3 Dense Media Separation

Dense media separation was performed on each of the Phase 1 master composites MQV-H, MQV-L and STK, using procedures that had been developed with previous metallurgical laboratory testwork. Dense media separation work was not performed on the Phase 2 samples which were limited by sample weight and focused on the flotation circuit response to a higher oxide feed representing the initial years of the projected mine schedule. The Phase 2 samples were blended to represent projected grade of flotation feed, following dense media separation.

Sample preparation and dense media separation pilot testing were performed at the SGS facilities in Lakefield, Ontario. After crushing the split drill core to a nominal 12.7 mm (1/2") crush, the fines were removed by screening at 1.4 mm. The minus 12.7 mm, plus 1.4 mm was pumped to the 200 mm Multotec cyclone at feed rates ranging from 185 kg/h to 223 kg/h. The dense media separation media incorporated ferrosilicon adjusted to SG 2.75. Ferrosilicon was subsequently recovered for reuse by magnetic separation. The dense media separation floats were removed as reject. The dense media separation sinks were combined with the screened fines and forwarded to further mineral testing procedures. A flowsheet of the dense media separation pilot plant circuit is provided, courtesy of SGS in Figure 13-5.

Figure 13-5: SGS Dense Media Pilot Plant



Note:

Figure prepared by SGS, 2017.

Source: Ausenco (2021)

The dense media separation sinks, and pre-screened fines were then blended and the resulting product ground in a laboratory mill in 2 kg batches to be used for flotation feed. The separate dense media separation products consisting of the sink, and rejects (float), as well as the fines removed prior to dense media separation are provided for each of the master composites in Table 13-6.

Table 13-6: Dense Media Separation Balance on Composite MQV-H

Product	Wt (kg)	Wt%	Assay					% Distribution				
			%Pb	%Zn	g/t, Ag	%S	%TOC	Pb	Zn	Ag	S	TOC
Dense Media Separation Sink	118.1	62.62	21.1	21.4	250	13.1	0.2	88.9	78.1	78.6	79.2	39.2
Dense Media Separation Float	43.2	22.91	0.36	0.49	10	0.47	0.43	0.55	0.65	1.15	1.04	30.8
Dense Media Separation Fines	27.3	14.48	10.8	25.2	279	14.1	0.66	10.5	21.3	20.3	19.7	29.9
Dense Media Separation Sink+Fines	145.4	77.09	19.2	22.1	255	13.3	0.29	99.4	99.3	98.9	99	69.2
Head (Calc.)	188.6	100	14.9	17.2	199	10.4	0.32	100	100	100	100	100

Notes:

TOC = total organic carbon

Source: Ausenco (2021)

Table 13-7: Dense Media Separation Balance on Composite MQV-L

Product	Wt (kg)	Wt%	Assay					% Distribution				
			%Pb	%Zn	g/t, Ag	%S	%TOC	Pb	Zn	Ag	S	TOC
Dense Media Separation Sink	67	53.3	8.49	6.2	139	7.58	0.11	56	59.5	59.1	64.3	34
Dense Media Separation Float	31.7	25.2	0.27	0.3	10	0.63	0.22	0.84	1.36	2.01	2.53	32.2
Dense Media Separation Fines	27.1	21.5	16.2	10.1	226	9.67	0.27	43.2	39.2	38.9	33.2	33.8
Dense Media Separation Sink+Fines	94.1	74.8	10.7	7.32	164	8.18	0.16	99.2	98.6	98	97.5	67.8

			Assay					% Distribution				
Product	Wt (kg)	Wt%	%Pb	%Zn	g/t, Ag	%S	%TOC	Pb	Zn	Ag	S	TOC
Head (Calc.)	125.8	100	8.08	5.55	125	6.28	0.17	100	100	100	100	100

Notes:

TOC = total organic carbon

Source: Ausenco (2021)

Table 13-8: Dense Media Separation Balance on Composite STK

			Assay					% Distribution				
Product	Wt (kg)	Wt%	%Pb	%Zn	g/t, Ag	%S	%TOC	Pb	Zn	Ag	S	TOC
Dense Media Separation Sink	177	56.5	4.35	9.3	95.4	5.37	0.33	64.4	66.3	63	66.4	49.8
Dense Media Separation Float	74.4	23.7	0.16	0.24	5.1	0.22	0.4	1	0.7	1.4	1.1	25.4
Dense Media Separation Fines	61.9	19.8	6.67	13.3	154	7.5	0.47	34.6	33	35.6	32.4	24.8
Dense Media Separation Sink+Fines	238.9	76.3	4.95	10.36	111	5.92	0.37	99	99.3	98.6	98.9	74.6
Head (Calc.)	313.3	100	3.81	7.96	85.5	4.57	0.37	100	100	100	100	100

Notes:

TOC = total organic carbon

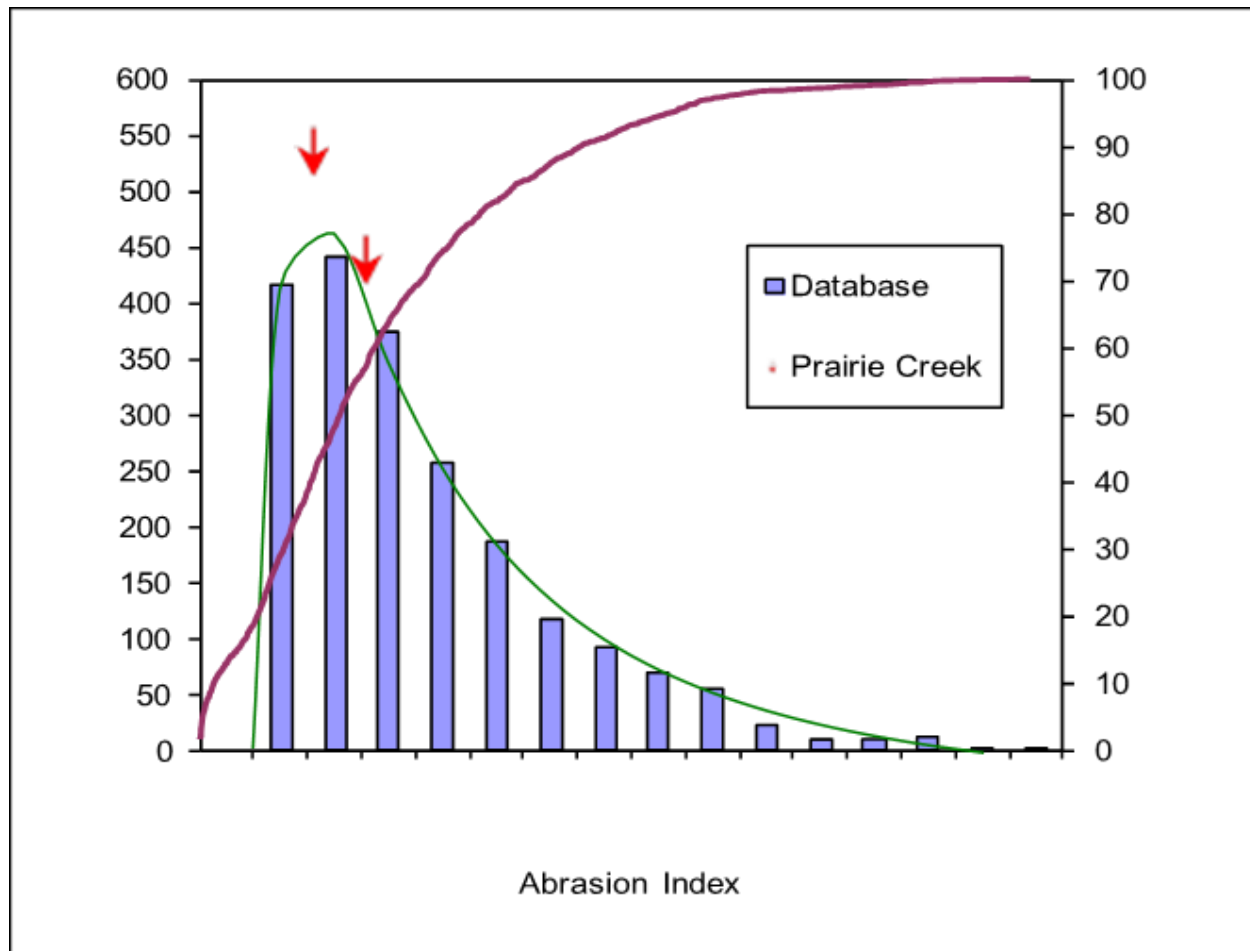
Source: Ausenco (2021)

The results show an excellent response of the three master composites to dense media separation. The combined fines with the dense media separation sinks averaged a recovery approaching 99% of the metal values, while rejecting close to a quarter of the feed mass prior to grinding.

13.2.4 Comminution

Abrasion (Ai) testing used for calculating grinding media consumption and liner wear rates were performed on the Phase 1 master composite sinks. The sinks were produced by heavy media separation with a media SG 2.75, on nominal 19 mm ($\frac{3}{4}$ ") material required for Ai testing. The two MQV composites were combined in a 50:50 weight ratio into a single Ai test head labelled as MQV-HL. STK was tested separately. The results gave an Ai of 0.205 g for the sinks produced from MQV- HL, and 0.108 g for STK. This would be considered relatively soft when compared to the SGS database as plotted in Figure 13-6 below.

Figure 13-6: Abrasion Index



Note:

Figure prepared by SGS, 2017.

Source: Ausenco (2021)

Comminution testing was limited by the particle size of the sample available. Each of the three Phase 1 master composite dense media separation sinks had a Bond Ball Mill Work Index (BBMWi) undertaken using a closing mesh size of 106 µm (150 Tyler mesh). Results are provided in Table 13-9.

Table 13-9: Bond Ball Mill Grindability Test Summary

Sample ID	Mesh of Grind	F ₈₀	P ₈₀	Gram per Revolution	Work Index (kWh/t)	Hardness Percentile	Category
		(µm)	(µm)				
MQV-H (Dense Media Separation sink)	150	2,485	81	1.41	13.9	45	Medium
MQV-L (Dense Media Separation sink)	150	2,294	82	1.45	13.8	44	Medium
STK (Dense Media Separation sink)	150	2,355	77	1.49	13	35	Moderately Soft

Source: Ausenco (2021)

The work index range of 13 to 14 kWh/t for the three master composites is noted to be significantly higher than the historical work index test results. This is hypothesized to be due to potential differences in mineralogy, including a higher quartzite content from deeper areas of the resource and to a lower extent of sulphide oxidation and weathering.

The dense media separation sinks, and -1.4 mm screened fines were combined to make up the flotation feed for each of the Phase 1 master composites. Each master composite float feed material, as well as the four assay reject composite blends had separate laboratory mill test grinds performed for estimating the various targeted primary grind sizes used in flotation, which is discussed in the following section.

13.2.5 Flotation

A series of flotation studies was performed to determine the response of composited mineral samples representing previously untested zones of the resource. In the case of the Phase 1 composites these had been pre-concentrated by the dense media separation procedures as discussed previously. The Phase 2 composites did not have dense media separation treatment as these were limited by the particle size distribution and weights of the sample. The Phase 2 composites were blended to more directly represent the targeted float feed blends (dense media separation sinks + fines).

The flotation studies consisted of:

- Open cycle Testing: With an initial objective to simplify the pre-2017 flotation flowsheet, including for reducing the reagent requirements, as well as using a coarser grind. Optimization was followed by final confirmation open cycle testing;

- Variability Testing: Composites were generated containing variable head characteristics which included increased iron content, increased copper content and a variable oxide content to note their response to the optimized float procedure;
- Ten (10) kg Batch Studies: Larger scale ten (10) kg batch tests were undertaken on a global mix of primarily the Phase 1 master composite dense media separation sinks and fines. The testing was performed to generate concentrate for smelter terms and tailing paste backfill testing undertaken by other parties. Minor variation in reagent addition was investigated when undertaking these procedures to better optimize a recommended dosage; and
- Locked Cycle Testing: Three tests were initially performed to better confirm the optimized open cycle response on the dense media separation product (sink + fines) of the Phase 1 master composites. To best represent the average grade of silver, lead and zinc in the resource the MQV-H and MQV-L were combined in a 50:50 weight ratio, labelled as MQV-HL. STK was tested as a separate resource zone. An additional fourth locked cycle test was performed on Comp. PB2 to represent the higher oxide feed expected during initial production.

Bench scale work including variability and locked cycle testing used 2 kg batch samples in a Denver D12 float machine at typically 33 wt.% slurry feed. Standard procedures consisted of depressing sphalerite and floating galena, followed by reactivating the sphalerite to produce a separate zinc sulphide concentrate. Flotation cleaning was accomplished in typically three stages. Following evaluation of regrinding, it was employed for the rougher lead concentrate prior to cleaning. No regrinding was used prior to cleaning zinc. A more detailed summary of the procedure and corresponding results of each set of tests are provided below.

13.2.5.1 Open Cycle Flotation Optimization

The Phase 1 portion was used to establish the optimized procedures. The initial scoping flotation procedures were performed on the MQV-AR composite to first repeat historic methods and to determine response. This moved to modifications including testing of new reagents and a more simplified cleaning circuit with fewer recycle streams. Test conditions initially used the historic 80% passing particle size (K_{80}) primary grind of $\sim 80 \mu\text{m}$. Based on the initial eight tests a procedure was then undertaken to test on the master composite dense media separation products that better represented flotation feed.

The initial tests performed on the master composites suggested using a reduced reagent scheme consisting of zinc sulphate (ZnSO_4) as a depressant, with the majority added in primary grinding. This was followed with a combination of two selective collectors produced by Solvay Corp. (A3418, A2410) that were used in the lead circuit. Sphalerite was reactivated with copper sulphate (CuSO_4) in the zinc circuit conditioning step, with sodium isopropyl xanthate (SIPX) used as the collector. Lime was typically used to modify pH generally evaluated at a range of 8 to 9.5 in the lead roughers, and at up to pH 10.5 in the lead cleaners. The pH in the zinc circuit was maintained at approximately 11, increasing up to 11.5 during cleaning. The pH was adjusted later in the study to correspond with iron control and depended on pyrite content of the feed. Use of soda ash as a modifier was also investigated and might be preferential for feeds with higher oxide content.

Initial optimization of the master composites continued with primary grinding evaluation. Primary grind product particle size included starting with a targeted K_{80} of 85 μm , based on historical work. This grind would have required major modifications and/or addition to the existing site grinding circuit, especially with the increasing hardness of the ball mill work index that was indicated as mining depth increased. Consequently, the testwork proceeded at evaluating coarser grind sizes that could more easily be accommodated with modifications to the existing circuit, continuing up to K_{80} of approximately 156 μm . The K_{80} 156 μm was selected based on modeling in 2017 by Ausenco which indicated the existing refurbished mill could accommodate this grind at 1,200 t/d throughput, assuming a BBMWi of 13.9 kWh/t. Since then, a higher throughput is envisioned, which would require the need for a redesigned grinding circuit.

The results of the primary grind testing are provided in Table 13-10.

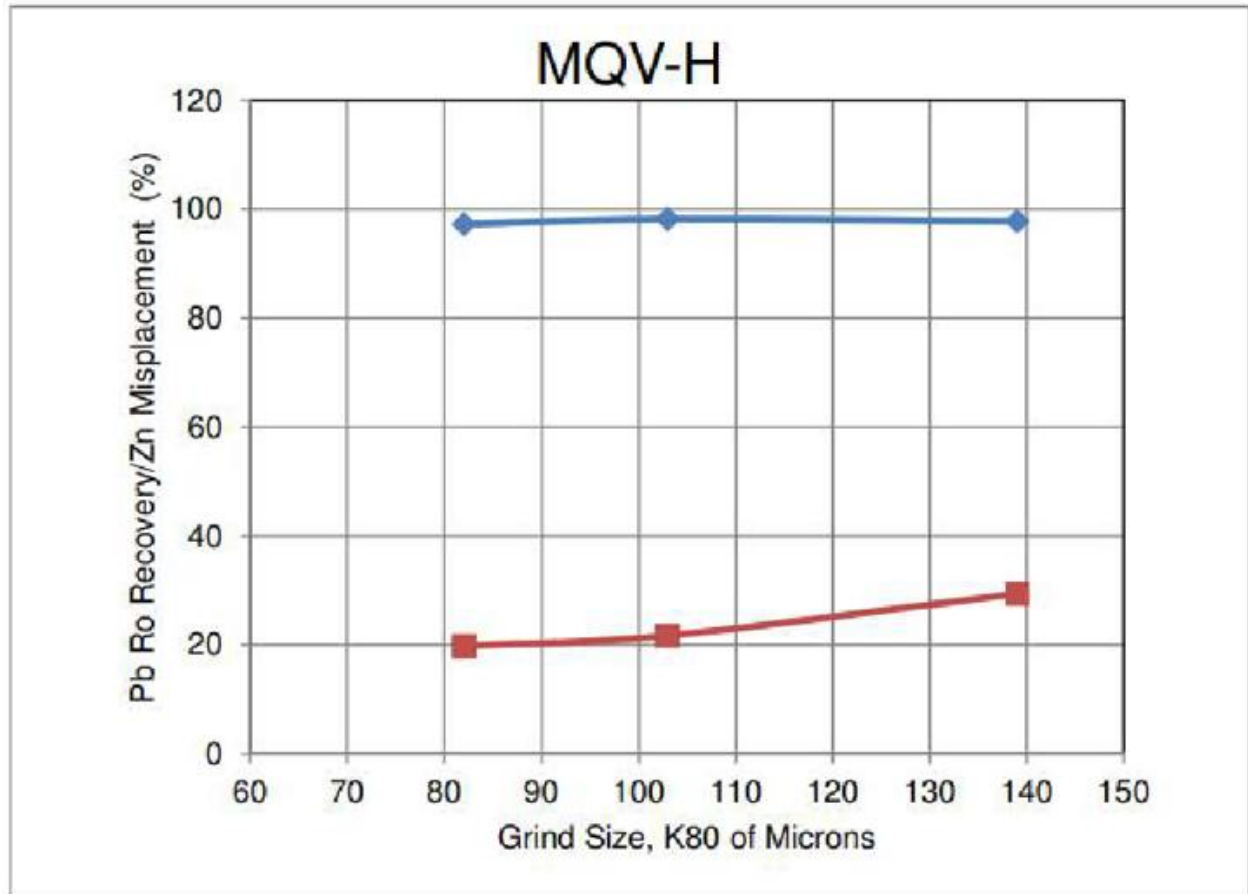
Table 13-10: Primary Grind vs Float Response

Comp.	Prim Grind	Pb Bulk	2nd Pb Conc.		2nd Zn Conc.		Final Tail Grade	
ID	K_{80} μm	Rec. (%)	% Rec	% Pb	% Rec	% Zn	%Pb	%Zn
MQV-H	83	97.2	91.3	61.0	67.9	64.7	0.78	2.0
MQV-H	103	95.8	92.1	57.5	69.7	63.7	0.37	1.23
MQV-H	139	97.8	91.0	54.3	67	62.3	0.57	1.24
MQV-L	77	98.3	90.9	55.3	68.9	47.4	0.14	0.12
MQV-L	100	97.6	91.3	52.6	71.7	42.0	0.17	0.14
MQV-L	130	97.2	91.3	45.4	67.3	43.5	0.37	0.19
STK	85	91.4	70.5	51.4	85.2	67.2	0.28	0.15
STK	107	94.2	59.8	45.6	86.1	66.4	0.18	0.11
STK	135	95.5	82.6	42.6	87.1	63.3	0.13	0.17

Source: Ausenco (2021)

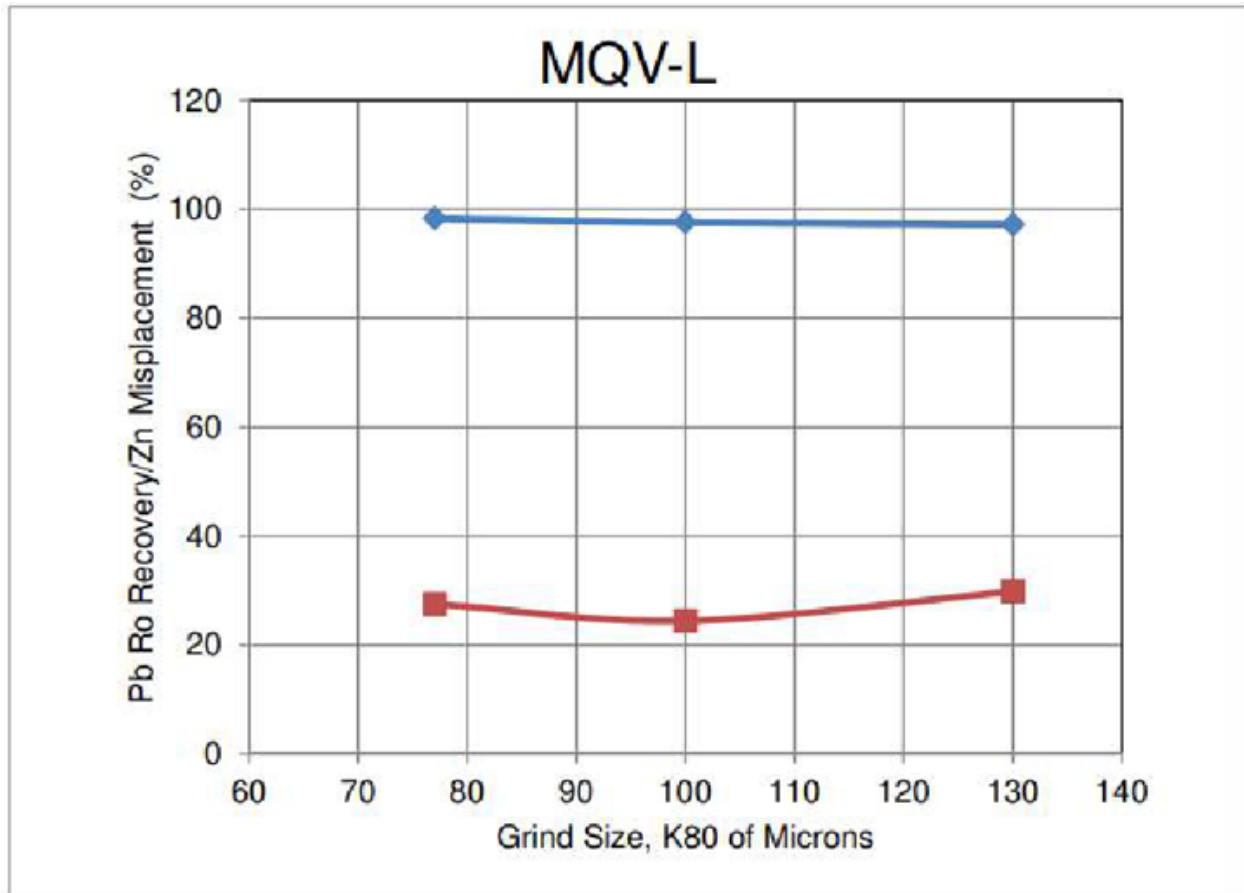
The grind data indicates that a significantly coarser primary grind from that used in most of the historical testwork could be applied. The bulk lead concentrate prior to cleaning showed consistent grades as did the final tailing. A lower zinc concentrate grade for MQV-L is attributed to the higher iron content (pyrite) in the feed. This was subsequently partially offset with increased lime addition. More detailed examination of the data showed the mass pull was not significantly affected within the range of the particle size distributions of the float feed that were studied. Silver recovery tended to follow that of lead. The trend while variable showed the final concentrate lead and zinc grades to be relatively static. There was some degradation of zinc recovery, and final lead concentrate grade at the coarsest primary grind, as indicated below in Figure 13-7 and Figure 13-8, respectively for composites MQV-H and MQV-L. This was attributed to minor liberation issues between galena and sphalerite prior to the first lead cleaner that should be assisted by regrinding the bulk lead concentrate if the coarser primary grind is employed.

Figure 13-7: Grind vs MQV-H Recovery



Note:
Figure prepared by SGS, 2017.

Figure 13-8: Grind vs MQV-L Recovery



Note:

Figure prepared by SGS, 2017.

The data suggested that a coarser grind than historically used can be accommodated with minimal effect on flotation response, if regrinding of the lead rougher concentrate is used to assist in improving the final lead concentrate grade. A follow-up examination on a coarser grind of $K_{80} \sim 156 \mu\text{m}$, along with a brief polish regrind was performed on combined MQV material labelled as MQV-HL. This material consisted of a 50% ratio of MQV-H, blended with an equal weight of MQV-L (50:50 ratio). A similar test was performed on STK, and the results provided in Table 13-11.

Table 13-11: Primary Grind (with regrind) vs Flotation Response

Comp.	Prim Grind	Pb Bulk	3rd Pb Conc.		2nd Zn Conc.		Zn Ro. Tail Grade	
ID	K ₈₀ µm	Rec. (%)	% Rec	% Pb	% Rec	% Zn	%Pb	%Zn
MQV-H+L	109	97.7	88.5	67.3	74.0	64.8	0.36	0.34
MQV-H+L	156	96.1	90.2	63.3	74.4	63.2	0.64	0.63
STK	107	93.8	64.8	58.6	84.9	64.0	0.22	0.12
STK	156	91.6	65.0	60.1	83.3	63.8	0.27	0.21

Source: Ausenco (2021)

If a brief regrind is incorporated with the coarser primary grind of K₈₀ ~156 µm, then the results as compared to the K₈₀ ~110 µm grind, still show some degradation to float response. Further study showed why this might be expected, as shown in Table 13-12.

Table 13-12: Comp MQV-HL Lead Rougher Float Kinetic Response

Stream	Retention	Size	Grade of Concentrate			Distribution (%)			
	Minutes	K ₈₀ µm	Pb	Zn	Fe	mass	Pb	Zn	Fe
Float Feed	-	156	12.1	12.9	1.73	100	100	100	100
Pb Ro #1 Con	6	74	54.8	5.2	1.91	19.9	89.7	8	21.9
Pb Ro #2+3 Con	6+6	54	9.18	12.1	3.63	7.7	5.8	7.2	16.2
Pb Ro. Tail	-	202	0.74	15.1	1.48	72.4	4.4	84.8	61.9

Source: Ausenco (2021)

The lead rougher concentrate was shown to be considerably finer than the corresponding ball mill product likely owing to a softer work index of the galena, as compared to the gangue minerals. Consequently, this resulted in the tailing having a coarser particle size distribution as shown from the K₈₀ when compared to the primary grind. The initial lead rougher (Pb. Ro1) concentrate also had a coarser K₈₀ as compared to the later portion (Pb Ro. 2+3), indicating the flotation kinetics could be hindered by overgrinding. Most of the lead was shown to report to the bulk float concentrate in the first 6 minutes. A coarser grind is also supported by the galena / sphalerite liberation particle size as outlined in the QEMSCAN data, previously presented.

Further effect on the primary grind on MQV is provided in the locked cycle testing discussed below.

13.2.5.2 Variability Flotation Testing

Variations to the optimized open cycle float procedure were undertaken to address specific characteristics of each of the major composites in both the Phase 1 and Phase 2 studies. Among the observations was that Comp. MQV-H having the higher grade required slightly extended retention times, although retention time was generally kept consistent for all open cycle tests. MQV-H also appeared to require additional collector suggesting further reagent optimization might be available to lower grade flotation feed material. Other factors showed that feed material with higher pyrite (MQV-L) as quantified by iron content, or higher graphite content (STK) as measured by total organic carbon analyses required a more specialized approach. As the extent of oxidation increased the response to the optimized Phase 1 flowsheet was more challenged. Attempts to improve overall lead recovery by addition of oxide lead flotation following sphalerite flotation produced low concentrate grade within the 1% to 3% lead oxide content of flotation feed.

The presence of graphite can cause excessive froth in the lead circuit. This can result in excessive reagent consumption and challenges in forwarding a heavy stiff concentrate with each cleaning stage. In addition, extra stages for reagent application can be necessary and ultimately a lower final lead concentrate grade results due to dilution from the graphite content. This was especially true if the graphite to lead ratio increased as evident in the STK master composite. A carbon pre-float was tested but resulted in unacceptable losses of metal values. The use of a graphite depressant A633, distributed by Solvay Corp. was then evaluated and worked well. The results on STK with A633 are compared to a similar test without using the depressant in Table 13-13.

Table 13-13: STK – Lead Flotation Response to Graphite Depressant (A633)

A633	3rd Cl. Pb Conc Grade					Pb 3rd Cl Conc Rec.		2nd Cl Zn Conc	
g/t	%Mass	%Pb	Ag, g/t	%Zn	%TOC	%Pb	%Ag	% Zn	%Zn Rec.
None	5.2	60.1	1452	2.1	2.1	65.0	83.0	63.8	83.3
250	5.2	70.5	1554	2.5	0.39	78.7	85.4	65.3	82.8

Notes:

TOC = total organic carbon

Source: Ausenco (2021)

Although the reagent addition was not optimized the results show both recovery and grade of the lead concentrate improved by the addition of A633. The zinc concentrate grade improved slightly with a stable recovery.

The open cycle results also showed that higher pH using lime as the modifier assisted with pyrite depression to improve the grades of both the lead and zinc concentrates. This is well documented in mineral processing literature and was shown to be particularly evident for MQV-L, which had the highest iron content of the three master composites.

Further variability testing included replacing lime with soda ash, as a pH modifier. This was evaluated as lime (calcium ions) may inhibit flotation of lead oxide. Based on historical testwork oxide lead flotation is anticipated to be incorporated for feeds earlier in the mine life that have a higher oxide content. While high oxide feed was not available for the 2017 Phase 1 test program the use of soda ash was evaluated as an alternate pH modifier, with results shown in the following table. Also included in the following table is the response from three separate variability composites that were evaluated to compare variation in their feed characteristics to the master composites. These variability composites were blended from assay rejects that were made available and outlined in composite origins discussed previously. Since the material was not subjected to dense media separation the feed had a higher ratio of gangue minerals. The variability samples included MQV-Fe, and MQV-Cu, which respectively contained a higher iron and copper content than any of the master composites. In addition, a test was performed on composite MQV-Pbox, which had a slightly elevated oxide lead content as compared to the other two variability samples.

The results of open cycle testwork evaluating pH with respect to pyrite depression included varying the iron head grade, pH modification with lime, along with a single test using soda ash. This data is presented in Table 13-14 and Table 13-15, respectively for the response on the lead and zinc concentrates.

Table 13-14: Lead Flotation Variability Response on Higher Iron Composites

Comp.	Modifier	pH range		Calc Head Grade / Wt. Ratio			Final Pb Conc % Rec.			Final Pb Conc Grade %		
Id	Used	Ro. Float	Cleaners	%Pb	%Fe	Fe:Pb	mass	Pb	Fe	%Pb	%Zn	%Fe
MQV-HL	Lime	9	10-10.5	12.4	1.7	0.14	17.6	90.2	17.2	63.3	4.76	1.7
MQV-HL	Soda Ash	8	10-Sep	12.2	1.8	0.14	16.9	85.5	32.1	61.6	4.4	3.3
MQV-L*	None	7.3	7.3	10.1	3.1	0.31	19.4	92.3	40.2	48.3	4.68	6.4
MQV-L	Lime	10-10.2	10.1-11	10	3.1	0.31	15.1	89.7	18.6	59.2	5.65	3.9
MQV-Fe	Lime	9	10.1-10.5	5.9	5.4	0.91	13.6	89.7	35.8	39.2	2.71	14.2
MQV-Fe**	Lime	~10.3	10.4-10.8	5.6	5	0.9	11.9	88.9	27.9	41.6	2.85	11.4

Notes:

*Finer primary grind used; **Collector & CuSO₄ dosage decreased

Source: Ausenco (2021)

Table 13-15: Zinc Flotation Variability Response on Higher Iron Composites

Comp.	Modifier	pH range		Calc Head Grade / Wt. Ratio			Final Zn Conc % Rec.			Final Zn Conc Grade %		
ID	Used	Ro. Float	Cleaners	%Zn	%Fe	Fe:Zn	Mass	Zn	Fe	%Zn	%Pb	%Fe
MQV-HL	Lime	11	11.2- 11.4	12.7	1.7	0.14	15.0	74.4	7.0	63.2	0.22	0.80
MQV-HL	Soda Ash	10	10.5	12.8	1.8	0.14	14.9	74.5	5.4	64.1	0.33	0.64
MQV-L*	None	11.2	11.6	5.9	3.1	0.53	9.2	67.5	27.0	42.8	0.51	9.1
MQV-L	Lime	11	11.4	6.2	3.1	0.50	5.9	57.4	7.6	60.1	0.32	4.0
MQV-Fe	Lime	11	11 - 11.1	2.2	5.4	2.4	5.8	57.6	27.3	21.8	0.72	25.3
MQV-Fe**	Lime	11.1	11.6-11.9	2.0	5.0	2.5	4.6	68.0	14.0	30.3	1.82	15.3

Notes:

*Finer primary grind used; **Collector & CuSO₄ dosage decreased

Source: Ausenco (2021)

The results suggest that higher pyrite (Fe) concentration can negatively impact concentrate grade, particularly as the ratio of Fe:Pb, or Fe:Zn increases. This can be countered to some extent with the use of lime to increase pH. Soda ash can be substituted for lime as a pH modifier prior to flotation of oxide lead providing iron content is moderate. Lime as the modifier should be used as pyrite content increases, but the effectiveness has upper limits, and alternate procedures may need to be investigated. Further evaluation into checking into additional iron depressants and/or decreasing collector dosage and CuSO₄ addition can be applied when galena or sphalerite content decrease in the feed with respect to the iron ratio.

The response of the variability composites MQV-Cu and MQV-Pbox are shown in Table 13-16.

Table 13-16: Lead Flotation Response – Phase 1 Variability Samples

Comp. ID	Final (3rd CI) Conc Recovery			Final (3rd CI) Conc Grade			
Lead Conc.	Lead	Zinc	Mass	% Pb	% Zn	%Fe	Hg, ppm
MQV-Pbox	79.3	1.2	12.2	76.3	1.1	0.15	104
MQV-Cu	88.4	4.6	15.6	62.8	4.4	1.25	306
Zinc Conc.	Zinc	Lead	Mass	% Zn	% Pb	%Fe	Hg, ppm
MQV-Pbox	80.9	0.5	13.8	66.6	0.43	0.27	924
MQV-Cu	80.8	0.3	18.5	64.6	0.18	1.05	1030

Source: Ausenco (2021)

Both MQV-Cu and MQV-Pbox responded well, and with a good comparison to the baseline testing done on the master composite dense media separation products. The MQV-Pbox composite was shown to still have too low an oxide content to sufficiently justify incorporating a separate oxide flotation circuit. Further variability evaluation of oxidized sample was conducted in Phase 2 on Comps. PB1, PB2, and PB3.

The response to the optimized float procedure developed in Phase 1 was evaluated in Phase 2 using the higher oxide samples. The use of specialty reagents and sulphidization was evaluated during flotation of galena. Ultimately, other than a modest increase in collector dose depending on extent of oxidation and a slightly increased pH in the lead rougher float to pH 9.2 there were no significant changes to the sulphide portion of the flotation procedures. The response of the final open cycle testing incorporated for Comp. PB1, PB2 and PB3 is provided in Table 13-17.

Table 13-17: Oxide Composites Open Cycle Flotation Response

Comp	Grind P ₈₀ (µm)		Calc. Head		Rougher Rec. (%)		3rd CI Pb Conc.			3rd CI Zn Conc.			Final Tail Grade	
ID	Prim.	Regrind	%Pb	%Zn	Pb	Zn	% Rec	%Pb	% Zn	% Rec	%Zn	% Pb	%Pb	%Zn
PB1	127	40	11.0	11.7	92.8	83.7	84.8	62.8	4.4	80.1	57.3	0.69	0.97	0.76
PB2	136	50	10.9	11.8	86.8	74.9	81.8	55.3	8.1	71.9	59.3	4.80	n/a	n/a
PB3	126	47	11.4	11.7	79.1	71.9	73.3	61.4	7.3	68.1	57.3	1.39	n/a	n/a

Notes:

*n/a= not available as final tail (Zn Ro. Tail) used for lead oxide float testing

Source: Ausenco (2021)

The data shows that for the less oxidized composite of up to 2% lead oxide the recovery losses of lead and zinc are manageable, although the final zinc concentrate grade does suffer slightly compared to the Phase 1 materials. For Comp. PB3 with 3.1% oxide lead, the sulphide flotation response becomes more challenging. While the lead and zinc concentrate grades remain acceptable, the recovery losses increase by up to 10%.

As a means of improving total lead recovery, several oxide lead flotation schemes were evaluated including the previously optimized historical procedure, and modified methods that followed the sulphide zinc flotation. The modified procedures often use sodium isobutyl xanthate as collector in various combination with specialty collectors including from the supplier Solvay Ltd. These collectors included OX-100, as well as SQ4, typically with DF067 as frother. In most cases sodium sulphide (Na₂S) was used as a sulphidizing agent, and soda ash incorporated as a pH modifier.

The best results using the specialty reagents were, not surprisingly, on the highest grade oxide composite sample PB3, which had a head grade of 12.3% total lead, of which 3.1% reported as oxide lead. This oxide content is above the highest one month mill head grade indicated in the mine plan. The current mine plan indicates the highest oxide content in mill feed occurs during commissioning and is provided approximately 2.5% Pbox. Using the separate lead oxide float circuit, the highest open cycle recovery to the oxide concentrate resulted in an additional 14% lead produced from Comp. PB3. This was providing that the froth was pulled hard by the technician. However, the corresponding lead oxide concentrate grade was only 20% with a 1:2 mass pull as related to the sulphide concentrate. When the two lead concentrates are recombined into a final lead concentrate, the grade would be such as it would be expensive to ship and difficult to market. The highest grade lead oxide concentrate for Comp. PB3 was 33% Pb, resulting in an additional 8.7% in improved lead recovery. Lower grade oxide feed samples showed decreasing grade and recovery relationships.

13.2.5.3 10 kg Batch Flotation

A final set of open cycle tests was undertaken during the Phase 1 program to produce enough product to provide concentrate samples to better establish smelter terms and to provide slurried

tailing for paste backfill testing (not part of the metallurgical test program). The resulting products were also used for settling and pressure filtration studies performed at SGS Lakefield, Ontario (discussed below). As the products were needed to be kept wet final the mass balance (including middling analyses) was not performed, although assay splits were taken for each of the two final concentrates, and two tailing (Zn 1st Cl scavenger tailing; and Zn rougher tailing) produced for each test. The blend for the main feed consisted of master composite dense media separation sink, plus fines in a weighted percent of 49% MQV-H, 36% MQV-L and 15% STK which corresponds to a calculated float feed grade of 9.5% Pb, 10.6% Zn, 147 g/t Ag, 1.2% Fe, and 288 ppm Hg. Later additional float feed was generated to make up additional material for further concentrate and tailing characterization.

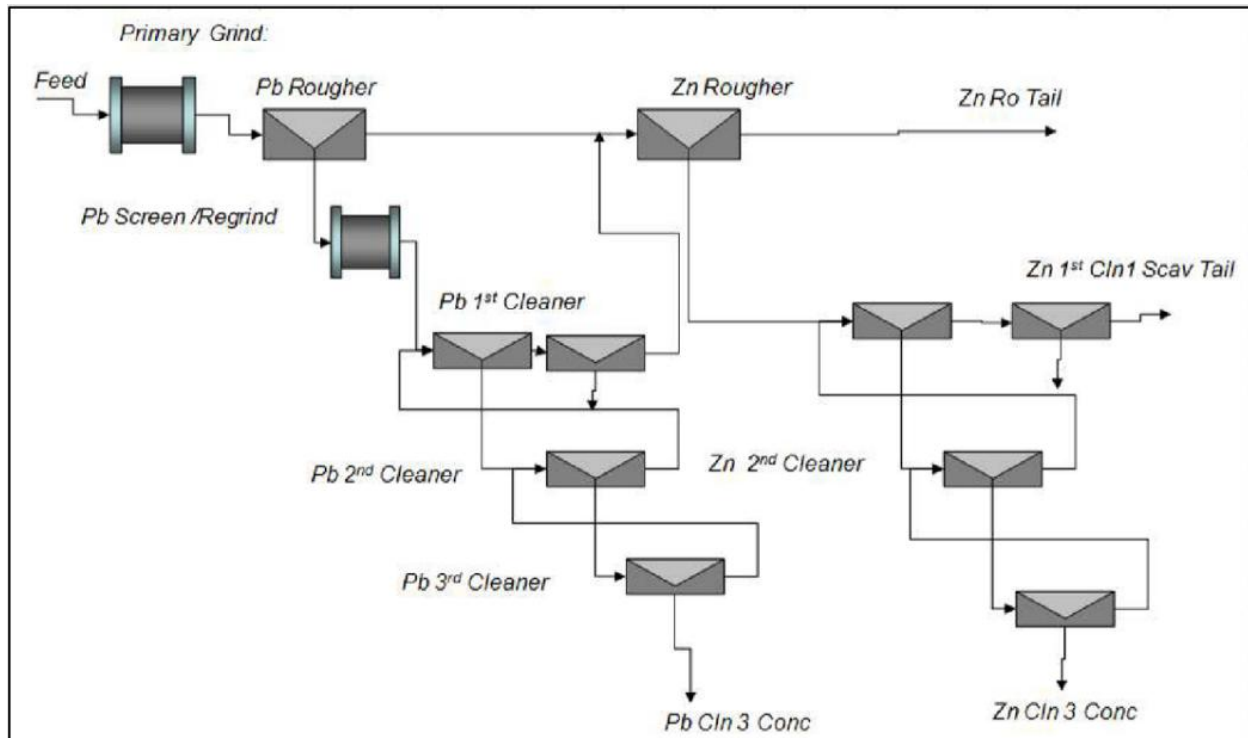
The 10 kg testing used a primary grind K_{80} of approximately 135 μm . Lime was used as the pH modifier. Reagent addition initially followed those used in bench scale work (including for locked cycle), but dosage reductions were undertaken as the testing proceeded. The results indicated that the various collector dosages when compared to the 2 kg open cycle tests could be reduced by about 30% for average expected feed grades. Addition of the zinc sulphate depressant, and copper sulphate activator were also modestly reduced without apparent negative consequences to the float response. The required float retention time, particularly during lead roughing were also observed to be less based on froth characteristics (i.e., color, mass pull). This is likely a result of changes in cell aeration with the larger 10 kg cells. Overall, the final concentrate grades appeared relatively consistent in the range of 55% to 65% Pb for the lead concentrate, and 60% to 65% Zn for the zinc concentrate.

13.2.5.4 Locked Cycle Flotation

The locked cycle testing was based on the flowsheet developed during the 2017 open cycle test program. Three locked cycle tests were performed in Phase 1 (LCT1, LCT2, LCT3) on the master composite dense media separation sinks plus screen fines, representing flotation feed. Two of the tests (LCT1, LCT3) were performed on MQV-HL, which was a 50:50 weighted blend of MQV-H and MQV-L. The blend was used to better represent average expected mill head grades for the MQV resource zone, representing the majority of mill feed for the LOM. The variation between the two tests was the primary grind. Another locked cycle test (LCT2) was performed on STK mineralization. A final locked cycle LCT 4, was performed in Phase 2 on Comp. PB2. This material represented the average feed expected during the initial three years of operation consisting of a higher oxide content, represented by 2% oxide lead, and the corresponding zinc and sulphur oxidation. Due to the lower oxidation content of the material, as compared to historic testwork neither a lead nor zinc oxide float circuit was incorporated.

The locked cycle procedure was also simplified as compared to the historic flotation flowsheet, with less scavenging and middling recycling incorporated, as well as reduced reagent requirements. The float circuit consisted of separately cleaning a bulk lead and bulk zinc rougher concentrate. There is no lead or zinc rougher scavenger requiring recycle. The lead rougher concentrate is sent to a brief regrind prior to cleaning in three stages. Both the first cleaner lead and zinc scavenger concentrate are respectively combined with their second cleaner tailing back to the first cleaner. Six cycles for each test were incorporated for each LCT. The flowsheet used for locked cycle testing is provided in Figure 13-9, courtesy of SGS.

Figure 13-9: Locked Cycle Flotation Flowsheet



Note:

Figure prepared by SGS, 2017.

Source: Ausenco (2021)

Both the STK and PB2 composites used an average K_{80} of approximately 135 μm . The remaining two locked cycle tests on MQV-HL were performed under similar conditions, with exception being the grind and cleaner float retention time used. Based on open cycle studies the feed particle size of the locked cycle flotation feed was varied between LCT1 versus LCT3 on sample Comp. MQV-HL to note the variation in response. The lower and upper grind range ($K_{80} \sim 110$ to 156 μm) were used to represent potential differences in particle size due to changes in the ball mill work index at depth, and to assist with design evaluation.

The reagents scheme was developed during the open cycle study. However, dosages can be further optimized to suite head grade and mineralogical characteristics. Zinc sulphate (ZnSO_4) was added as a sphalerite depressant, with the majority added in primary grinding. A graphite depressant A633 was added during lead rougher conditioning for STK, but not for MQV-HL. Any mill feed with an elevated Pb to total organic carbon ratio would likely benefit from A633 addition. A3418, and A241 were used as collectors in the lead circuit. This was followed by reactivating the sphalerite with CuSO_4 and using SIPX as collector in the zinc circuit. The pH modifier used was lime. For the locked cycle tests the targeted pH 9 was used in the lead roughers, increasing up to pH 10 during cleaning. The pH used in the zinc rougher circuit was approximately 11, increasing up to 11.5 in cleaning. STK had a lower iron content than MQV-HL and consequently

did not increase the pH in cleaning. The pH is primarily used to depress pyrite and can be adjusted depending on iron content of the float feed.

Table 13-18 includes the calculated heads generated from the four tests, which compared reasonably well to the assayed head for the master composites. Also shown in the following table are the tailing analyses.

Table 13-18: Locked Cycle - Calculated Head / Tailing Assay

Comp	Test	Calc. Head Grade				Zn 1st CI Tail Grade			Zn Ro. Tail Grade		
ID	No.	Pb, %	Zn, %	Fe, %	Ag, g/t	% Pb	% Zn	Ag, g/t	% Pb	% Zn	Ag, g/t
MQV-HL	LCT 1	12.2	13.3	2.4	176	1.38	1.21	17.2	0.81	0.80	12.0
STK	LCT 2	4.73	9.64	0.38	90	1.07	0.76	14.0	0.20	0.17	2.1
MQV-HL	LCT 3	12.4	12.9	1.7	185	1.08	0.98	15.3	0.58	0.46	5.7
PB2	LCT 4	11.0	11.4	0.51	152	3.49	2.64	33.0	2.16	1.59	20.1

Source: Ausenco (2021)

The grind and regrind 80% passing particle size as represented by the K_{80} , as well as the mass distribution from the final product streams are provided in Table 13-19.

Table 13-19: Locked Cycle – Mass Balance

Comp	Test	Grind	Regrind	Mass Distribution (%)			
ID	No.	~ K_{80} μ m	~ K_{80} μ m	Pb Con	Zn Con	Zn 1st Tail	Zn Ro. Tail
MQV-HL	LCT 1	156	47	22.0	18.7	9.8	49.4
STK	LCT 2	135	48	6.8	15.1	10.2	67.9
MQV-HL	LCT 3	109	48	19.1	19.0	13.9	48.0
PB2	LCT 4	135	41	16.1	15.9	7.8	60.3

Source: Ausenco (2021)

Based on the locked cycle results the material was shown to have responded well to produce stable concentrates at acceptable grades and recovery. The average of the final three cycles for the principal elements of interest reporting to the lead concentrate is summarized in the following Table 13-20.

Table 13-20: Locked Cycle – Lead Concentrate Assay and Recovery

Comp	Test	Pb Conc. Grade				% Recovery	
ID	No.	Pb, %	Ag, g/t	Zn, %	Fe, %	Pb	Ag
MQV-HL	LCT 1	52.5	762	6.3	2.42	94.5	91.7
STK	LCT 2	61.7	1281	3.0	0.42	88.9	99.1
MQV-HL	LCT 3	61.9	941	4.7	2.41	95.6	93.5
PB2	LCT 4	57.3	855	6.6	1.10	83.8	85.6

Source: Ausenco (2021)

The first locked cycle performed on MQV-HL was conducted at the coarser grind. The results provided for a lower lead concentrate grade of ~53% Pb. For LCT3, which was done at a finer grind the grade to the final lead concentrate increased to ~62% Pb. However, this was thought to be at least partly due from too long a residence time during final cleaning in the initial test. This is supported by a corresponding high lead recovery of 95% in LCT1. Consequently, for LCT 3 while done at a finer primary grind the final lead cleaning retention time was also reduced, from 3 minutes to 2 minutes per cycle. This resulted in a concentrate grade recovery relationship more in line with the expected optimized open cycle data. Silver grade and recovery responded similarly with modestly improved grade and recovery for the LCT3 procedures, providing a silver recovery in the lower ninety percent range.

For STK (see LCT2) despite a lower head grade the lead concentrate grade and recovery relationship was roughly maintained. The findings based on the composite tested would indicate that the expected grade for the lead concentrate would be in the lower sixty percent range, with recovery in the upper eighty percent range. The STK silver grade achieved was 1280 g/t, with excellent recovery of 99%.

Test LCT4 was performed on Composite PB2, which has a higher oxide content and better represents material during initial production. There is a corresponding drop in recovery in lead and silver recovery to the lead concentrate by 5 -10% as compared to LCF 1 and 3. There is also higher zinc reporting to the lead concentrate, although the concentrate grade remains just above 57% Pb. The results show that sulphide oxidation can have a notable effect on lead float response.

Zinc concentrate grade and recovery from the locked cycle procedures is summarized in Table 13-21.

Table 13-21: Locked Cycle – Zinc Concentrate Analyses and Recovery

Comp ID	Test No.	Zn %	Zn Conc. Grade		Fe %	% Recovery	
			Ag g/t	Pb %		Zn	Ag
MQV-HL	LCT 1	60.8	40	0.74	1.30	85.6	4.1
STK	LCT 2	61.1	30	1.85	0.38	95.8	0.57
MQV-HL	LCT 3	61.4	40	0.62	1.27	90.2	3.9
PB2	LCT4	58.0	53	1.31	0.65	80.6	5.3

Source: Ausenco (2021)

Zinc concentrate could be expected to grade approximately 60% Zn, with about 30-40 g/t Ag, in samples with a lower extent of oxidation. Recovery for MQV-HL is slightly more variable depending on the primary grind in the 85% to 90% range depending on the process conditions used. STK achieved zinc recoveries approximately 5% higher than MQV-HL. As with the lead oxidation of the MQV sample with Comp. PB2, showed a drop in zinc recovery of up to 10% Zn.

13.2.5.5 Characterization of Flotation Products

In addition to the metals of value (Pb, Zn, Ag), there are other elements that are present which can impact smelter terms resulting in potential penalty charges, or that could affect acceptability of the concentrate. Most notably the deportment and concentration of mercury, antimony, arsenic, and other detrimental elements are relevant.

The lead and zinc concentrates produced from the locked cycle testing were subjected to detailed chemical characterization. Also included are the blended concentrates produced from the 10 kg batch floats on feed originating from the mixed MQV/STK feed, as outlined previously.

Table 13-22: Lead Concentrate Analyses

Element	Comp. ID=	MQV-HL	STK	MQV-HL	PB2	10 kg Blends
	Test ID=	LCT 1	LCT 2	LCT 3	LCT4	F10-1 to 20
Ag	g/t	761	1288	889	927	939
As	%	0.33	0.67	0.38	0.33	0.1
Bi	ppm	<20	<30	<20	<10	<20
C (total)	%	2.31	2.38	1.35	2.07	1.01
Cd	ppm	380	271	346	417	299
Cl	%	<0.005	<0.005	0.005	0.007	0.008
Cu	%	1.5	3.7	1.9	1.7	2

Element	Comp. ID=	MQV-HL	STK	MQV-HL	PB2	10 kg Blends
	Test ID=	LCT 1	LCT 2	LCT 3	LCT4	F10-1 to 20
F	%	0.012	0.013	0.007	0.014	0.014
Fe	%	2.37	0.43	2.37	0.84	1
Hg	ppm	289	239	284	662	258
Pb	%	54.3	61.7	62.9	61.1	63.6
S (total)	%	13.6	11.4	14	13.4	12.8
Sb	%	0.76	1.27	0.7	0.61	1.01
TOC	%	0.51	0.65	0.54	1.02	0.69
Zn	%	5.98	3.05	5.25	5.64	4.09

Notes:

TOC = total organic carbon

Source: Ausenco (2021)

Table 13-23: Zinc Concentrate Analyses

Element	Comp. ID=	MQV-HL	STK	MQV-HL	PB2	10 kg Blends
	Test ID=	LCT 1	LCT 2	LCT 3	LCT4	F10-1 to 20
Ag	g/t	41	29	39	50	28
As	ppm	<200	<80	<100	119	<30
Bi	ppm	<20	<30	<20	<20	<20
C (total)	%	0.38	0.69	0.33	0.97	0.48
Cd	ppm	3390	3514	3557	221	3390
Cl	%	0.008	0.012	0.008	0.027	-
Cu	ppm	722	681	699	1270	545
F	%	0.003	0.006	0.003	0.012	<0.005
Fe	%	1.3	0.38	1.27	0.57	0.63
Hg	ppm	1520	988	1530	1390	1400
Pb	%	0.74	1.85	0.62	1.38	0.55
S (total)	%	31.7	30.6	32.7	29.4	31.3
Sb	ppm	170	<80	99	177	123
TOC	%	n/a	0.34	<0.05	0.22	0.12
Zn	%	60.8	61.4	61.4	58.6	63.4

Notes:

TOC = total organic carbon

Source: Ausenco (2021)

For the Phase 1 work, acid base accounting based on the Sobek method was performed on the zinc rougher tailing and provided a net modified neutralization potential of 634 tonnes equivalent CaCO_3 per thousand tonnes of material. The corresponding total sulphur content was 2.7% S, 2.2% as sulphide S, with a 9.0 paste pH. Chemical analyses on the combined final tailing from the final cycle of LCT3, provided for 2.3% total S, 1.6% Fe, 479 ppm Cu, 59 ppm As, 80 ppm Sb, and 34 ppm Hg.

Various physical characterizations including particle size analyses (PSA), angle of repose, as well as solid and bulk specific gravity were determined for final blended tailing, and for each of the two final concentrates. The combined (final) tailing was produced by blending the Zn 1st CI scavenger tailing; and Zn rougher tailing. The bulk SG moisture content was selected based on expected filter cake moisture content, with vacuum filter option assumed for the tailing and pressure filtration for the two concentrates. The products used for the characterization studies were produced from twenty 10 kg flotation tests discussed previously that were performed to generate the necessary material. The detailed data is presented in the May 2017 SGS report⁵, titled "Solid – Liquid Separation and Geotechnical Results". The physical characteristics of the various products are summarized in Table 13-24.

Table 13-24: Float Product Physical Characteristics

Material	Particle Size Analy.		SG	Bulk SG	Moisture wt. %	Bulk SG (Kg/L)	Angle Repose
	K ₈₀ µm	<20 µm (% vol)	Dry	Dry	(assumed)	@ % moisture	Avg @ % moisture
Combined Tailing	148	12.3	2.83	2.83	13.0	1.627	46 degrees
Zn Concentrate	135	18.9	4.03	4.03	5.2	2.083	46 degrees
Lead Concentrate	46	62.5	5.75	5.75	6.6	2.887	39 degrees

Source: Ausenco (2021)

The lead concentrate had the finest particle size, due to the relative softness of galena, as well as the fact it was reground prior to flotation cleaning. This likely contributed to this product having the lowest angle of repose at 39 degrees and a higher pressure filter cake moisture content as compared to the zinc concentrate.

13.2.5.6 Settling and Filtration Studies

The lead concentrate, the zinc concentrate, and final tailing produced from the 10 kg flotation tests, and characterized as described previously above, were used for solid / liquid separation studies. The related work was conducted at SGS Lakefield and outlined in a report titled Liquid Separation and Geotechnical Results, dated May 24, 2017. The combined (final) tailing was produced by blending the Zn 1st CI scavenger tailing; and Zn rougher tailing. The work included static and dynamic settling tests, as well as pressure filtration testing on all three materials. Vacuum filtration testing was also performed on the flotation tailing.

Initial scoping studies indicated the flocculant, Magnafloc 10, distributed by BASF worked well, improving settling characteristics. Depending on the material and conditions used improvements were observed at Magnafloc 10 dosage rates of 4 g/t to 15 g/t. The static settling results are provided in Table 13-25, with abbreviated data of the dynamic thickening provided in Table 13-26.

Table 13-25: Static Thickening Data

Sample ID	Dosage Flocc ¹ g/t	Feed ¹ %w/w	U/F ² %w/w	Unit Area m ² /(t/day)	ISR ³ m ³ /m ² /day	Supernatant ⁴ Visual	TSS ⁵ mg/L
Comb Zn Tailings	5	20	73	0.05	502	Clear	<10
Zn Conc	4	25	81	0.05	657	Clear	11
Pb Conc	6	30	80	0.05	337	Clear	18

Notes:

All values were calculated without a safety factor. Common conditions: Raked, ambient temperature. Magnafloc : BASF Magnafloc 10 flocculants.

¹Diluted Thickener Feed.

²Final Underflow Density.

³Initial Settling Rate.

⁴Supernatant Visual Clarity after 30 minutes of elapsed settling time.

⁵Supernatant Total Suspended Solids (TSS) after 30 minutes of elapsed settling time.

Source: Ausenco (2021)

Table 13-26: Dynamic Settling Data

Product Conditions	Unit Area m ² /(t/d)	Solids Loading t/m ² /h	Net Rise Rate m ³ /m ² /d	Underflow wt.% solids	O/F - TSS mg/L	Residence Time (h)
Combined Tailing	0.05	0.83	78.0	70.2	28	0.51
(10 g/t Magnafloc 10)	0.07	0.60	55.7	70.8	18	0.69
Feed @ 20 wt.% solids	0.09	0.46	43.3	71.0	13	0.89
Zinc Concentrate	0.04	1.04	72.5	79.7	74	0.58
(10 g/t Magnafloc 10)	0.05	0.83	58.0	79.3	47	0.73
Feed @ 25 wt.% solids	0.06	0.69	48.3	79.0	14	0.81
Lead Concentrate	0.05	0.83	57.0	80.8	17	1.04
(15 g/t Magnafloc 10)	0.07	0.60	40.7	81.4	18	1.45
Feed @ 25 wt.% solids	0.10	0.42	28.5	80.8	9	2.07

Source: Ausenco (2021)

The higher SG lead concentrate had the densest underflow at 80 wt.% solids, but with the longest residence time required. The data shows good settling characteristics for all three materials with



high bed compaction and thickener unit areas calculated at 0.04 to 0.1 m²/(t/d) depending on the conditions used.

Testori P 6583 TC polypropylene cloth was selected for use in the various filtration studies, after conducting scoping tests using various filter cloths. The thickened tailing was subjected to vacuum filtration testing with the summarized data presented in Table 13-27.

Table 13-27: Vacuum Filtration Data

		Operating Conditions					Filter Outputs				
Sample ID	Filter Cloth	Feed Solids %w/w	Vacuum Level, inch Hg	Form Time, s	Dry Time, s	Form/Dry Ratio	Cake Thickness, mm	2 Throughout, dry kg/m ² h	Cake Moisture, % w/w	Filtrate TSS	Cake Texture
Combined Zn Tails	Testori P6583 TC	70.0	20	14	1	11.50	53	21581	19.2	50	Wet
				8	2	4.67	37	22446	18.6	68	Wet
				8	5	1.75	37	17307	18.4	59	Wet
				8	13	0.64	37	10545	16.2	57	Wet
				8	40	0.21	38	4685	13.8	51	Wet
				5	120	0.04	22	1031	9.7	49	1DTT
				17	170	0.10	55	1739	12.4	30	DTT

Notes:

¹Dry to touch

²Examples of general filter throughout predictions versus test conditions using raw test data. Throughputs are calculated based on cycle time of form and dry only. Results are not for sizing of any specific type of filter. Refer to individual test results for additional sizing information.

Source: Ausenco (2021)

The results provided for a cake moisture content varying from 9.7 wt.% to 19.7 wt.%. Depending on the vacuum filtration equipment used, a one minute dry time would be expected to give a cake moisture content of ~12 wt.%. Cake surface cracking, or cake-wall separation were not observed, and the cakes were reported to have a clean release from the cloth.

Pressure filtration was performed on each of the three materials with a synopsis of the results provided in Table 13-28.

Table 13-28: Pressure Filtration Data

Material	Feed	Pressure	Form	Dry	Cake	Throughput	Filtrate	Cake
	wt.% solid	bar	Time (s)	Time (s)	Thick (mm)	kg/m ² h (dry)	TSS (mg/L)	wt.% moist.
Tailing	70	4.1	3	43	30	3358	197	6.1
	70	6.9	3	47	30	3684	85	6.1
Zinc	79	4.1	2	42	28	5488	164	4.3
	79	6.9	1	48	30.5	5221	180	3.8
Lead	81	4.1	15	44	20	4322	95	6.7
	81	6.9	15	81	25	3252	72	6.1

Source: Ausenco (2021)

Overall, the results show good filter response with the lead concentrate having the finest particle size distribution (see Table 13-28, above) providing for a lower throughput, and with the highest cake moisture content. Generally, the throughput and residual moisture were reported to be relatively insensitive to pressure levels. Cake surface cracking or cake-wall separation was not observed, but typically the filter cakes left a thin layer of solids on the cloth.

The water saturation level and porosity calculations were based on the moisture content levels that were targeted in the bulk density tests. The results are summarized in Table 13-29.

Table 13-29: Moisture Saturation and Porosity Calculation

Sample ID	Solids Content (%)	Water Content (%)	Wet Bulk Density (t/m ³)	Dry Bulk Density (t/m ³)	Density of Water at 20°C (t/m ³)	Solids Specific Gravity, Gs	Porosity N	Void Ratio e	Saturation Sr
Combined Zn Tailing	87.00	0.130	1.63	N/D	0.998	2.83	0.50	1.00	0.42
	93.00	0.070	1.40	N/D	0.998	2.83	0.54	1.16	0.18
	100.00	0.000	N/D	1.83	0.998	2.83	0.35	0.55	0.00
Zn Conc	94.80	0.052	2.08	N/D	0.998	4.03	0.51	1.04	0.21

Sample ID	Solids Content (%)	Water Content (%)	Wet Bulk Density (t/m ³)	Dry Bulk Density (t/m ³)	Density of Water at 20°C (t/m ³)	Solids Specific Gravity, Gs	Porosity N	Void Ratio e	Saturation Sr
	100.00	0.000	N/D	2.64	0.998	4.03	0.34	0.52	0.00
Pb Conc	93.39	0.066	2.89	N/D	0.998	5.75	0.53	1.13	0.36
	100.00	0.000	N/D	2.56	0.998	5.75	0.38	0.61	0.00

Notes:

Where, Porosity, n = (void space, liquid & gas)/(total volume including solid) Void, e = (void space, liquid & gas)/(Volume of solid)

Source: Ausenco (2021)

13.3 Summary

The 2017 metallurgical test program updated the preceding flotation studies to include additional mineral zones that had not previously been tested. This improved representation of the samples to better reflect the most recent mine plan. The 2017 study was successful in being able to establish a more conventional and simplified flotation flowsheet and reagent scheme than what had previously been proposed. The use of dense media separation was further supported by this most recent testwork. The revised flotation flowsheet eliminated some of the scavenging and recycle streams during differential flotation of galena and sphalerite and was able to justify increasing the particle size of flotation feed. An improved reagent scheme resulted in reducing the number of flotation reagents, as well as identifying readily available chemical products that are currently marketed by known suppliers. The metallurgical response of the 2017 work, as compared to earlier results, showed an improvement by having lower lead and silver values in the zinc concentrate, with most of the silver reporting to the lead concentrate. Mercury content remained elevated in the zinc concentrate and requires further verification to the mine plan. Alternate methods were also developed for variation in changing mill feed characteristics, particularly with respect to sulphide oxidation, as well as pyrite and graphite content.

In conjunction with the developing mine plan, a more representative range of oxide minerals was tested in 2017. The oxide content of the plant feed was shown to be significantly lower than previously anticipated. Due to this lower extent of sulphide oxidation in the 2017 metallurgical samples being tested, there was no anticipated need shown for separate oxide flotation circuits.

The overall process response of the PC Silver Mine material shows good separation and yields of payable metals using conventional mineral processing applications. The data allows for recovery projections for lead and silver reporting into a lead sulphide flotation concentrate and separately zinc reporting into a zinc sulphide flotation concentrate.

13.4 Metallurgical Assumptions

The current recovery equations include all of the aforementioned updates and are shown in Table 13-30.

Table 13-30: Recovery and Concentrate Grade Estimates

Parameter	Unit	Dense Media Separation	Concentrates	
			Pb Concentrate	Zn Concentrate
Ag Recovery	%	97	$1.0086 \times 99.31 \times (\text{Pb} - \text{PbO}) / \text{Pb}$	1.0771 * Pb recovery to lead concentrate - Ag recover to lead concentrate
Pb Recovery	%	97.5	$99.31 \times (\text{Pb} - \text{PbO}) / \text{Pb}$	0
Zn	%	96.5	0	$91.31 \times (\text{Zn} - \text{ZnO}) / \text{Zn}$
Cu Recovery	%	97.2	95	0
Concentrate Grade				
Mass	%	63	Variable	Variable
Ag	%	Variable	Variable	Variable
Pb	%	Variable	60	Variable
Zn	%	Variable	Variable	60
Cu Recovery	%	Variable	Variable	Variable

14 MINERAL RESOURCE ESTIMATE

14.1 Introduction

The current mineral resource estimate is an update of the estimate in the Preliminary Economic Study completed by Ausenco Engineering Canada Inc. for Canadian Zinc with an effective date of October 15, 2021. The current estimate includes updated metal prices, mining and processing costs, and metallurgical recoveries.

The data used for the current MRE includes the same wireframes of the three mineral domains, surface topography and underground development, as well as drillhole locations, downhole surveys, assays and geology that were used for the 2021 MRE. The QP, Greg Mosher, P.Geo. of Global Mineral Resource Services, completed the Mineral Resource estimate using Genesis software version 2.3.14 from SGS.

As received, minor overlaps existed between the STK and MQV and SMS and MQV wireframes. To avoid duplication of estimated resources in those overlapping volumes, the STK and SMS wireframes were clipped against the MQV wireframe so that the volume is attributed only to the MQV wireframe.

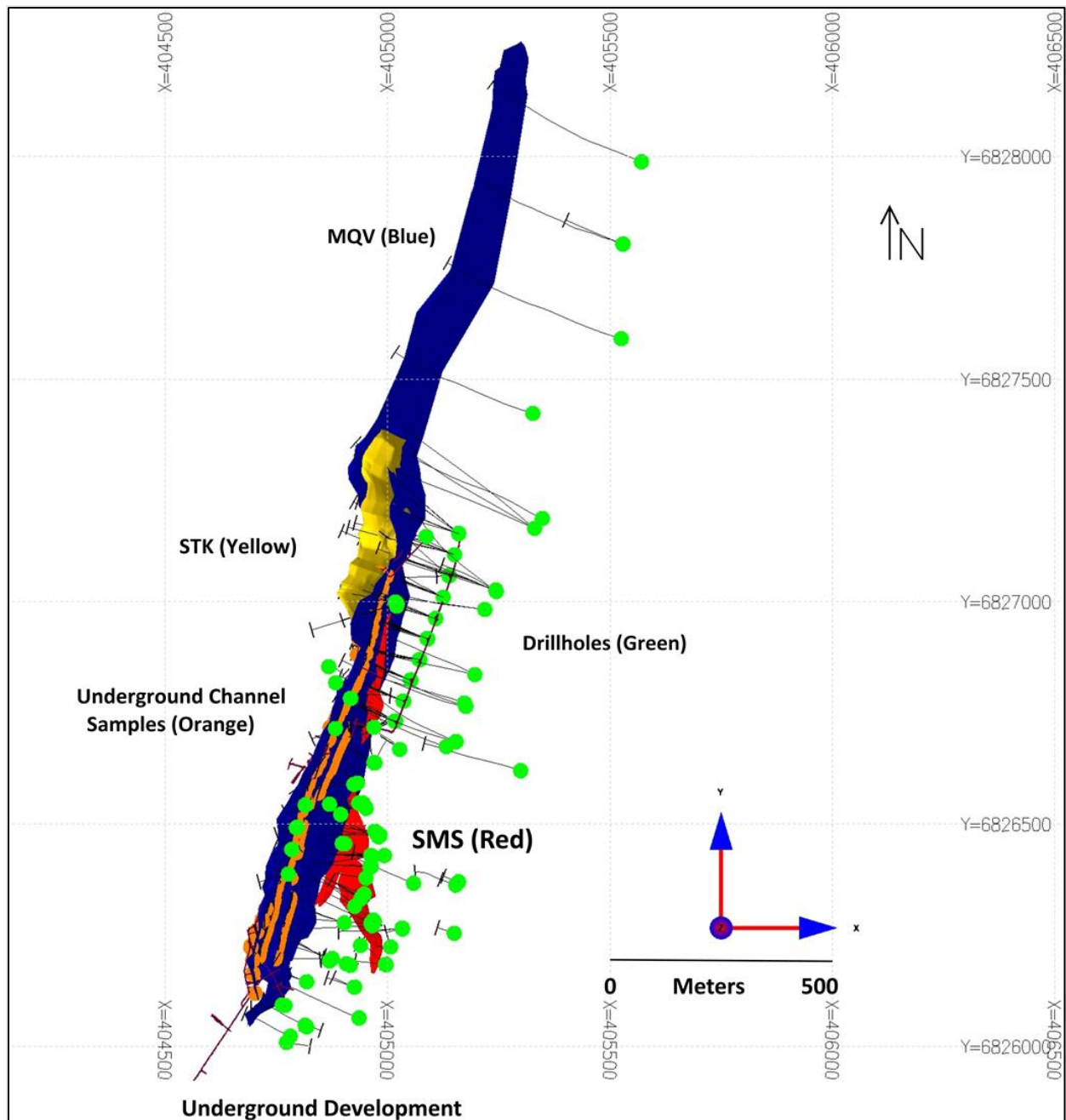
14.2 Exploratory Data Analysis

The mineral resource estimate is based on assays from all underground channel samples, surface and underground drill core collected by Canadian Zinc since 1992.

The dataset contains data for 302 surface and underground drillholes of which 220, with an aggregate length of 60,849 m, constrain the three mineral zone domains (MQV, SMS, and STK), and 370 channel samples (1,283 aggregate metres) from the MQV and STK Zones. The 220 drillholes contain 3,280 assays of which 875 are contained within the MQV, 897 within the STK and 391 within the SMS domain. The channel samples contain 918 assays, of which 678 are within the MQV domain and 66 within the STK. Channel samples are from the MQV and STK Zones and were taken from the three underground levels: 970 mL, 930 mL, and 883 mL.

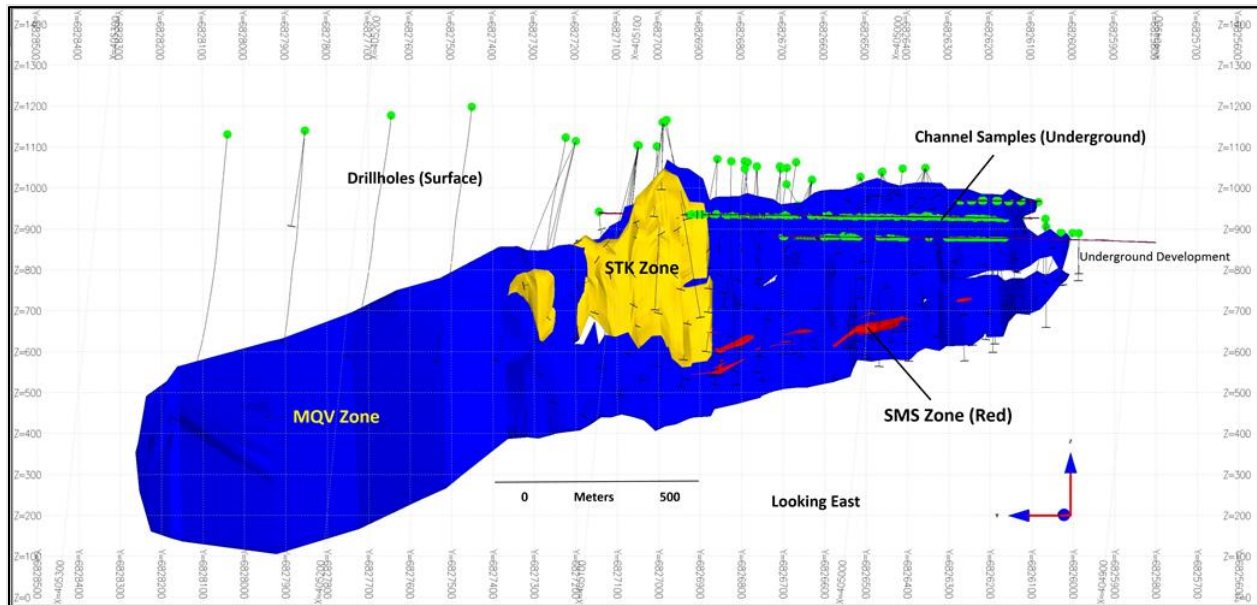
Plan and longitudinal views of the MQV, SMS and STK mineral domains are shown in Figure 14-1 and Figure 14-2. The underground channel samples are indicated on Figure 14-1.

Figure 14-1: PC Silver Mine Mineral Domains, Plan View



Source: Ausenco (2021)

Figure 14-2: PC Silver Mine Mineral Domains Longitudinal View



Source: Ausenco (2021)

Table 14-1 shows the number of assays or analyses available for each element of interest from channel and drillhole samples.

Table 14-1: PC Silver Mine Assay Count by Channel and Drillhole Samples

Element	Channel #	DDH #
Ag ppm	894	3,279
As ppm	304	3,280
Cd ppm	918	3,166
Cu ppm	753	3,187
Fe %	287	3,195
Hg ppm	918	3,161
PbO %	681	2,422
Pb %	916	3,214
Sb ppm	290	3,280
ZnO %	751	2,454
Zn %	918	3,238

Source: Ausenco (2021)

A significant number of channel samples were not analyzed for mercury and cadmium, both of which are of metallurgical significance. To ensure that there were sufficient values for those elements in the area of the channel samples, missing analyses were replaced with regression values using the regressions $Hg \text{ ppm} = Zn \text{ ppm} * 45.038$ and $Cd \text{ ppm} = Zn \text{ ppm} * 71.39$.

Descriptive statistics for silver, lead and zinc from drillhole and channel sample assays and corresponding composites are presented in Table 14-2. Compositing is described in Section 14.4.

Table 14-2: PC Silver Mine Assay and Composite Descriptive Statistics by Domain

MQV Assays Non-Zero	Ag_ppm	Pb_pct	Zn_pct	MQV Comps Non-Zero	Ag_ppm	Pb_pct	Zn_pct
Mean	190	11.23	10.97	Mean	184	10.94	10.66
Median	132	8.61	7.2	Median	149	8.98	8.33
Mode	3	0.02	0.02	Mode	69	0.08	10.35
Standard Deviation	208	11.37	11.57	Standard Deviation	158	8.82	9.23
Range	1848	69.87	64.11	Range	968	49.02	47.9
Minimum	0	0.01	0.01	Minimum	0	0.01	0.02
Maximum	1848	69.88	64.12	Maximum	968	49.03	47.92
Count	1540	1548	1554	Count	762	764	763

STK Assays Non-Zero	Ag_ppm	Pb_pct	Zn_pct	STK Comps Non-Zero	Ag_ppm	Pb_pct	Zn_pct
Mean	79	4.37	7.69	Mean	50	2.91	4.94
Median	21	1.13	2.53	Median	21	1.17	2.55
Mode	2	0.02	0.02	Mode	9	0.01	0.02
Standard Deviation	139	7.08	11.13	Standard Deviation	69	4.17	6.37
Range	1741	47.1	54.39	Range	658	31.95	36.94
Minimum	0	0.01	0.01	Minimum	0	0.01	0.01
Maximum	1741	47.11	54.4	Maximum	658	31.96	36.95
Count	962	945	951	Count	599	592	595

SMS Assays Non-Zero	Ag_ppm	Pb_pct	Zn_pct	SMS Comps Non-Zero	Ag_ppm	Pb_pct	Zn_pct
Mean	56	5.59	10.08	Mean	51	4.95	9.96
Median	32	3.23	8.18	Median	37	3.9	8.91
Mode	3	0.07	0.04	Mode	13	4.85	6.55
Standard Deviation	79	7.11	9.44	Standard Deviation	47	4.37	6.81
Range	707	63.2	50.97	Range	238	22.6	29.91
Minimum	0	0.01	0.01	Minimum	0	0.02	0.03
Maximum	707	63.21	50.98	Maximum	238	22.62	29.94

SMS Assays Non-Zero	Ag_ppm	Pb_pct	Zn_pct	SMS Comps Non-Zero	Ag_ppm	Pb_pct	Zn_pct
Count	391	389	391	Count	185	185	185

Source: Ausenco (2021)

Table 14-3 shows the correlation coefficients of silver, lead and zinc relative to a number of other elements for each of the three Mineral domains MQV, STK and SMS, as well as the underground channel samples. Although there are differences in magnitude, all domains have generally similar element affinities. Correlations greater than $r=0.75$ are highlighted.

Table 14-3: PC Silver Mine Mineral Correlation Coefficients

Element	Ag			Pb			Zn		
Ag	MQV	STK	SMS	MQV	STK	SMS	MQV	STK	SMS
Ag				0.77	0.68	0.90	0.42	0.64	0.48
As	0.78	0.88	0.19	0.39	0.30	0.20	0.40	0.51	0.28
Cd	0.51	0.66	0.44	0.39	0.57	0.46	0.95	0.98	0.93
Cu	0.83	0.89	0.69	0.43	0.32	0.68	0.40	0.52	0.36
Fe	-0.06	-0.02	0.10	-0.03	-0.02	0.12	-0.18	-0.09	0.37
Hg	0.56	0.59	0.29	0.42	0.48	0.33	0.80	0.81	0.75
PbO	0.45	0.45	0.60	0.39	0.57	0.69	0.12	0.42	0.57
Pb	0.77	0.68	0.90				0.33	0.55	0.49
Sb	0.87	0.89	0.63	0.47	0.33	0.61	0.40	0.52	0.08
ZnO	0.05	0.07	0.22	-0.03	0.03	0.24	0.16	0.13	0.63
Zn	0.42	0.64	0.48	0.33	0.55	0.49			

Source: Ausenco (2021)

The mineral resource estimate in the upper levels of the MQV and a small portion of the upper STK is supported by both channel and drill core samples; the estimation of grades in the balance of the MQV and STK zones is supported by drill core data only. The SMS mineral resource estimate is based on diamond drill core only.

14.3 Capping

Capping is the process of artificially reducing high values within a sample population that are regarded as statistically anomalous with respect to the population as a whole (outliers), to avoid the distorting influence these values would have on the statistical characteristics of the population if left at their full value. The risk in including atypically high values in a Mineral resource estimate

is that their contribution to the estimated grade will be disproportionate to their contribution to the tonnage, and therefore the grade of the Mineral resource as a whole will be overstated.

The appropriateness of capping of high assay values was investigated by the construction of cumulative frequency plots of silver, lead and zinc assay values. None of the distributions displayed any discernible breaks in the plots suggestive of separate populations of high values and, therefore, no capping of assay values was considered warranted.

14.4 Composites

Compositing of samples is done to overcome the influence of sample length on the contribution of sample grade (sample support). Both drill core and channel samples were composited to a length of 2.5 m. Approximately 97% of the drill core samples and 98% of the channel samples are 2.5 m in length or shorter. Descriptive statistics of composites are presented in Table 14-2. Composites were constrained by domain boundaries (MQV, STK and SMS) and the last composite within a domain was discarded if it was less than 20% of the nominal composite length.

14.5 Bulk Density

For the current estimate, bulk density values were estimated on the basis of regression equations that were used for the 2007 and 2012 estimates. These equations were based on 231 measurements of drill core from the MQV made in 1998, and 54 measurements from sample pulps of SMS mineralization made in 2007; no measurements were made on samples from the STK and the regression equation used for the STK domain is the same as for the MQV. The equation for the MQV and STK domains is: $(2.6466 + (\text{Pb}\% \times 0.0339) + (\text{Zn}\% \times 0.02) - (\text{Fe}\% \times 0.02))$. For the SMS domain, the equation is: $(2.777 + ((\text{Pb}\% + \text{Fe}\%) \times 0.0413))$.

14.5.1 Geological Interpretation

Canadian Zinc generated wireframe models for the three mineralized domains MQV, STK and SMS. These solids were reviewed for conformity to the lithological boundaries established by drilling and were observed to adhere to those boundaries. The wireframe models were used as provided except for the removal of minor overlaps. The mineral domains are illustrated in plan and longitudinal vertical views in Figure 14-1 and Figure 14-2.

14.6 Spatial Analysis

Variography of composited values was carried out using Sage 2001 software. A range of lag distances was tested, and 50 m was determined to be optimal with respect to maximizing the number of sample pairs used in the construction of the variogram. Consequently, all variograms and search ellipses were established on the basis of 50 m lag spacings. Separate variographic and search ellipse parameters were determined for each of the Mineral domains MQV, STK, and SMS, and for each of the elements/compounds: silver, arsenic, cadmium, copper, iron, mercury, lead, lead oxide, antimony, zinc and zinc oxide. All models were one-structure spherical. Because of the paucity of data, variograms and search ellipses for the SMS domain were constructed

using orientations and dimensions obtained from zinc variography. Table 14-4 contains the variography parameters and Table 14-5 the search ellipse parameters that were used in the estimate. The search ellipse dimensions were standardized to provide similar coverage for all elements. Note that the plunge is in the strike direction.

Table 14-4: PC Silver Mine 2021 Variogram Parameters

Domain	Element	Nugget	C ₁	Strike (m)	Cross-Strike (m)	Dip (m)	Azimuth (°)	Dip (°)	Plunge (°)
MQV	Ag	0.684	0.316	200	35	28	13	84	-3
MQV	As	0.61	0.39	200	74	57	18	61	7
MQV	Cd	0.685	0.315	200	44	6	15	61	2
MQV	Cu	0.769	0.231	147	98	77	4	82	-8
MQV	Fe	0.357	0.643	200	77	29	338	82	3
MQV	Hg	0.698	0.302	200	100	93	342	78	-10
MQV	PbO	0.193	0.807	95	41	22	181	-23	67
MQV	Pb	0.712	0.288	200	100	50	40	-53	34
MQV	Sb	0.436	0.564	200	90	7	21	76	14
MQV	ZnO	0.339	0.661	158	100	10	45	56	28
MQV	Zn	0.564	0.436	185	75	7	24	-56	28
STK	Ag	0.755	0.245	200	77	41	341	45	24
STK	As	0.751	0.249	200	100	5	356	90	0
STK	Cd	0.658	225	200	100	6	325	90	0
STK	Cu	0.775	0.231	200	53	14	304	78	10
STK	Fe	0.283	0.717	69	35	11	310	-66	22
STK	Hg	0.687	0.313	200	100	5	155	89	-1
STK	PbO	0.53	0.47	197	79	17	7	-64	2
STK	Pb	0.648	0.352	115	25	13	313	69	18
STK	Sb	0.722	0.278	200	100	4	330	90	0
STK	ZnO	0.427	0.573	200	13	12	2	90	0
STK	Zn	0.177	0.823	130	76	6	319	-62	27
SMS	All	0.231	0.769	15	14	12	10	-5	2

Source: Ausenco (2021)

14.7 Mineral Resource Block Model Parameters

Block model parameters are summarized in Table 14-5. A block size of width 2.5 m across strike length 15 m along strike, and height of 10 m reasonably captures the mineralization distribution in all three mineral domains.

Table 14-5: PC Silver Mine 2021 Block Model Parameters

Dimension	Number	Size (m)	Coordinates *	Minimum	Maximum
Columns	241	2.5	X	404 400	405 5000
Rows	169	15	Y	6 826 000	6 828 520
Levels	111	10	Z	0	1100
Rotation	15 Degrees Clockwise				

Notes:

Coordinates of Block Centroids * UTM NAD 83 Zone 10V

Source: Ausenco (2021)

14.8 Interpolation Plan

Grades were estimated for silver, lead, zinc, arsenic, cadmium, copper, iron, mercury lead oxide, and zinc oxide and were interpolated into the block model using OK.

For the MQV and STK domains, grades were interpolated into the block model in two passes. The first pass required a minimum of 24 composites within the volume of the search ellipse. The only portion of the two domains that could meet this requirement is within the underground development in which channel samples were collected at nominal five-metre intervals, therefore this pass almost exclusively captured only composites from the channels. This approach was taken because the channels preferentially sampled mineralization and therefore are of higher average grade than composites from drillholes and if they were not constrained, they would potentially inflate the interpolated grades of blocks outside the areas that were preferentially sampled. For the second pass, a minimum of four composites within the volume of the search ellipse was required for a grade to be interpolated into a block, with a maximum of two composites coming from a single drillhole. Therefore, a minimum of two drillholes was required to interpolate a grade which ensured that continuity of mineralization was demonstrated. The maximum number of composites was set at 24 (12 drillholes). For the SMS domain, grades were interpolated in a single pass with the same parameters as the second pass for the MQV and STK domains.

14.9 Silver Equivalency Formula

The Mineral resource is stated using a silver equivalent grade (AgEq.) as a cut-off that takes into account the economic contribution of silver, lead and zinc. The equivalency calculation, which expresses the combined value of silver, lead and zinc in terms of parts per million silver, was calculated as follows:

$$\text{AgEq\%} = (\text{Grade of Ag in ppm} + [(\text{Grade of lead in \%} * \text{Price of lead in \$/lb} * 22.046 * \text{Recovery of lead in \%} * \text{Payable lead in \%}) + (\text{Grade of zinc in \%} * \text{Price of zinc in \$/lb} * 22.046 * \text{Recovery of zinc in \%} * \text{Payable zinc in \%}) + (\text{Grade of Copper in ppm} * \text{Price of copper in \$/g} * \text{Recovery of copper in \%} * \text{Payable copper in \%})] / (\text{Price of silver in US\$/ppm} * \text{Recovery of silver in \%} * \text{Payable silver in \%}))$$

Metal prices were based on an assessment of three-year trailing averages and market forecasts and considering reasonable prospects for eventual economic extraction. Recoveries and payables were provided by JDS from internal studies. Parameters are summarized below in Table 14-6.

Table 14-6: Silver-Equivalency Equation Parameters

Item	Units	Value
Cu price	CAD\$/Lb	6.30
Zn price	CAD\$/Lb	1.76
Pb price	CAD\$/Lb	1.28
Ag price	US\$/Oz	37.00
Cu recovery	%	85.0
Zn recovery	%	85.7
Pb recovery	%	86.5
Ag recovery	%	94.6
Cu payable	%	95.0
Zn payable	%	85.0
Pb payable	%	87.9
Ag payable	%	85.0

Notes:

Exchange Rate: CAD\$1.00 = US\$0.72

22.046 = pounds/metric tonne/%

31.10348 = grams/Troy ounce

Source: GMRS (2026)

14.10 Mineral Resource Classification

The mineral resource was classified as Measured, Indicated and Inferred. For a block to be classified as Measured, it was necessary that a minimum of 24 composites be located within the volume of the search ellipse. The MQV and STK domains contain Measured resources; in both, the Measured blocks immediately surround the underground development in which channel sampling was carried out.

For a block to be classified as Indicated, it was necessary that a minimum of 10 composites be located within the volume of the search ellipse.

For a block to be classified as Inferred, it was only necessary that a minimum of four composites be located within the volume of the search ellipse.

Table 14-7 shows the search ellipse orientations and dimensions used for resource classification.

Table 14-7: PC Silver Mine Resource Classification Search Ellipse Parameters

Domain	Category	Strike	Cross-Strike	Dip	Azimuth (°)	Dip (°)	Plunge (°)	Minimum Comps	Maximum Comps
MQV	Measured	85	35	50	15	0	-15	24	24
MQV	Indicated	325	50	150	15	0	-15	10	24
MQV	Inferred	400	200	300	15	0	-15	4	24
STK	Measured	30	30	30	15	0	-15	24	24
STK	Indicated	70	70	70	15	0	-15	10	24
STK	Inferred	400	200	300	15	0	-15	4	24
SMS	Indicated	60	60	60	15	0	-15	10	24
SMS	Inferred	400	200	300	5	-10	-15	4	24

Notes:

Ellipse dimensions in metres.

Source: GMRS (2026)

Figure 14-3 shows a longitudinal vertical view of the silver-equivalent block grade distribution in the MQV domain; Figure 14-4, and Figure 14-5 show similar views for the SMS and STK Zones. Figure 14-6 shows the classification for the MQV Zone.

Figure 14-3: Silver-Equivalent Block Grade Distribution MQV Zone

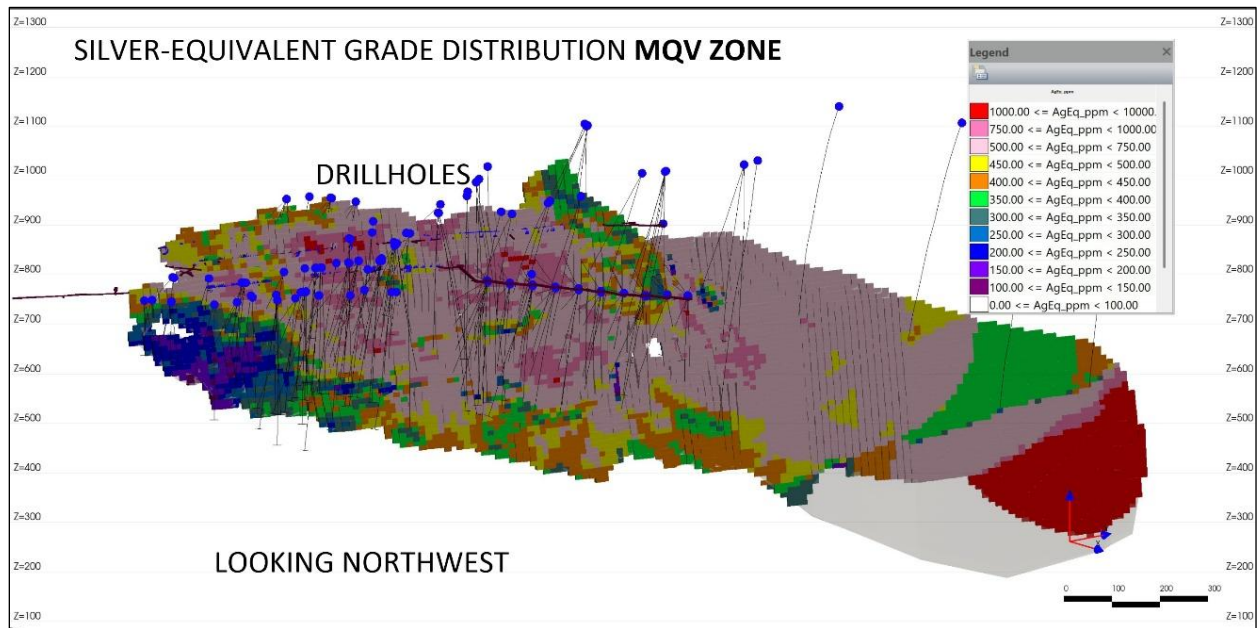


Figure 14-4: Silver-Equivalent Block Grade Distribution SMS Zone

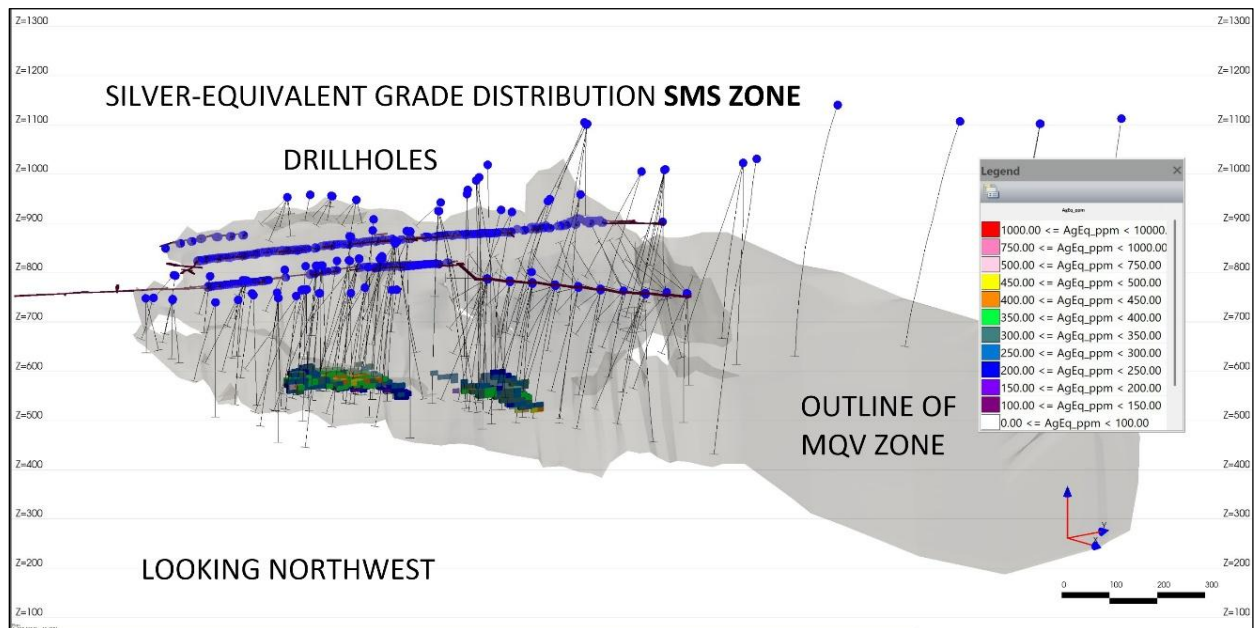


Figure 14-5: Silver-Equivalent Block Grade Distribution STK Zone

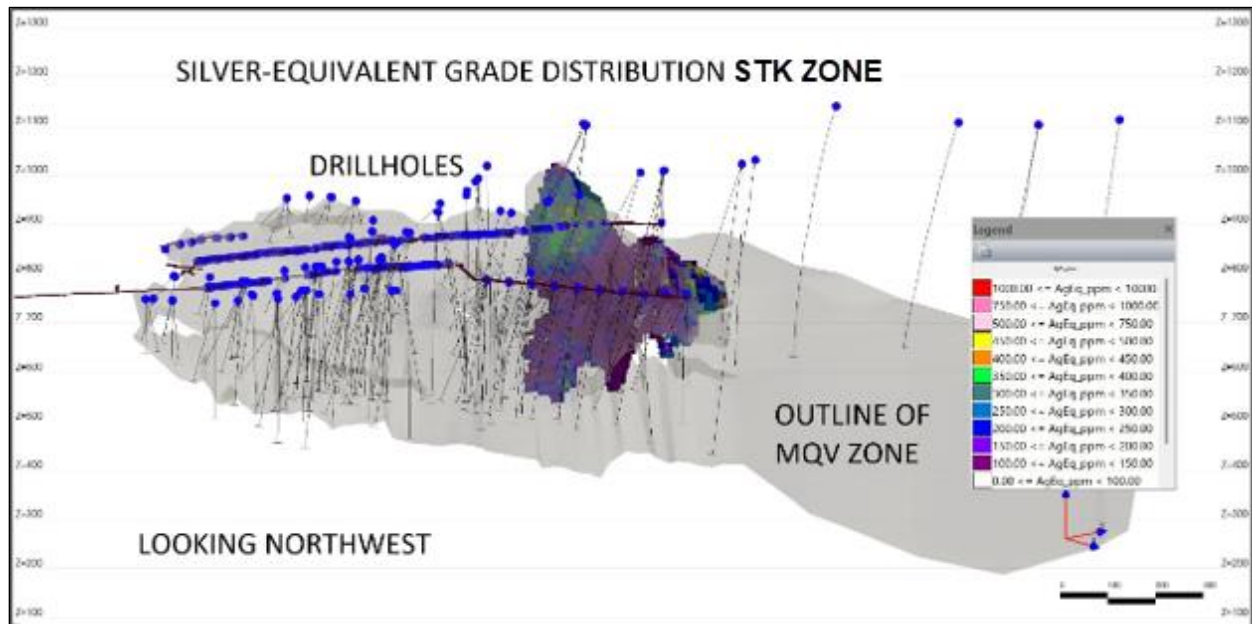
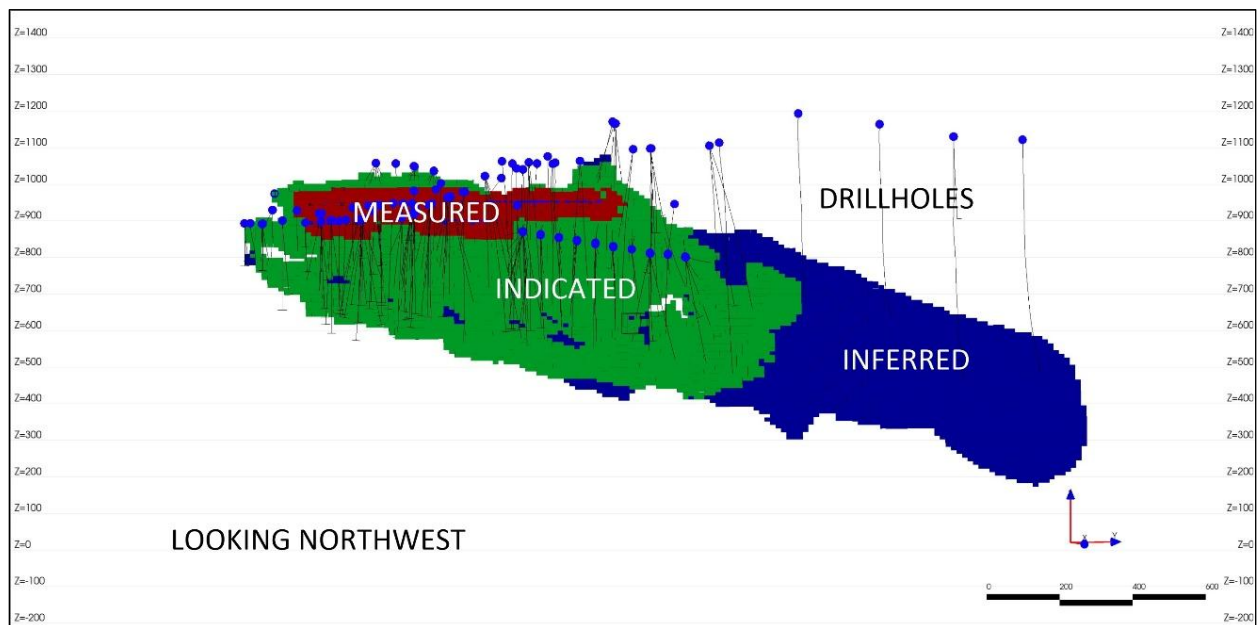


Figure 14-6: Mineral Resource Classification MQV Domain



14.11 Reasonable Prospects of Eventual Economic Extraction

It is anticipated that the PC Silver Mine deposit will be mined by underground methods, therefore it is necessary to constrain the estimated resource by mining shapes that meet the criteria for Reasonable Prospects of Eventual Economic Extraction (RPEEE). The parameters used to establish reasonable prospects are set out in Table 14-8. Metal prices are set out in Table 14-6.

Table 14-8: Reasonable Prospects of Eventual Economic Extraction Parameters

Parameter	Unit	Value
Revenue	CAD\$	$(Ag_ppm * Ag\ C\$/g * Recovery\ Ag\ \% * Payable\ Ag\ \% + (Cu_ppm * Recovery\ Cu\ \% * Cu\ C\$/g) + (Pb_ \% * Recovery\ Pb\ \% * Pb\ C\$/lb * 22.046) + (Zn_ \% * Recovery\ Zn\ \% * Zn\ C\$/lb * 22.046))$
Mining Cost	CAD\$/tonne	92.00
Processing Cost	CAD\$/tonne	45.00
Minimum stope width (x)	metres	5.00
Maximum Stope Width (x)	metres	5.00
Minimum Stope Length (y)	metres	15.00
Maximum Stope Length (y)	metres	30.00
Stope Height (z)	metres	20.00
Minimum Azimuth	degrees	0.00
Maximum Azimuth	degrees	360.00
Minimum Dip	degrees	0.00
Maximum Dip	degrees	90.00
Mining Recovery	%	95.00
Mining Dilution	%	12.00
Process Recovery Ag	%	94.60
Process Recovery Cu	%	85.00
Process Recovery Pb	%	86.50
Process Recovery Zn	%	85.70
Payable Ag	%	85.00
Payable Cu	%	95.00
Payable Pb	%	87.90
Payable Zn	%	85.00
Grams / Ounce 31.10348	Metal Prices and Recoveries as shown in Table 14-6	

14.12 Cut-off Grade

The mineral resource is stated in terms of silver-equivalent cut-off grades. The base case cut-off grade is taken as 100 g/t silver based on the formula: (Mining cost C\$/tonne + processing cost C\$/tonne)/(Silver price C\$/g * silver recovery % * silver payable %). Using the costs and prices set out in Table 14-6 and Table 14-8, this calculation equals 103 g/t. This has been rounded down to 100 g/t.

14.13 Mineral Resource Tabulation

Table 14-9 presents the mineral resource estimate for the three mineral zones MQV, STK and SMS, at a AgEq cut-off of 100 g/t. The upper portion of the table presents the mineral resources in each of the zones; the lower portion of the table shows the sum of those same mineral resources according to resource classification.

Table 14-9: PC Silver Mine Mineral Resource Summary at 100 AgEq g/t Grade Cut-off

Domain	Cut-off AgEq g/t	Classification	Tonnes	AgEq g/t	Ag g/t	Cu %	Pb %	Sb ppm	Zn %
MQV	100	Measured	856,000	702	209	0.40	11.44	1,020	12.89
MQV	100	Indicated	5,407,000	566	164	0.33	10.79	1,494	9.38
MQV	100	M&I	6,263,000	585	170	0.34	10.88	1,429	9.86
MQV	100	Inferred	4,958,000	679	185	0.49	8.18	2,429	14.98
STK	100	Measured	128,000	401	97	0.20	4.15	1,028	10.32
STK	100	Indicated	4,727,000	221	49	0.10	2.60	522	5.69
STK	100	M&I	4,855,000	226	50	0.11	2.64	536	5.81
STK	100	Inferred	3,256,000	221	53	0.10	3.15	502	5.11
SMS	100	Indicated	510,000	345	54	0.02	5.15	83	9.72
SMS	100	M&I	510,000	345	54	0.02	5.15	83	9.72
SMS	100	Inferred	255,000	325	48	0.01	4.35	73	9.67
ALL ZONES	100	Measured	984,000	663	194	0.37	10.49	1,021	12.56
ALL ZONES	100	Indicated	10,644,000	402	108	0.21	6.88	995	7.76
ALL ZONES	100	M&I	11,628,000	424	115	0.23	7.19	997	8.16

Domain	Cut-off AgEq g/t	Classification	Tonnes	AgEq g/t	Ag g/t	Cu %	Pb %	Sb ppm	Zn %
ALL ZONES	100	Inferred	8,469,000	492	130	0.33	6.13	1,617	11.03

Notes:

Mineral Resources are stated as of June 21, 2026.

Stated at a cut-off grade of 100 g/t AgEq based on prices of C\$6.30/lb for copper, C\$1.28/lb for lead, C\$1.76/lb for zinc and C\$51.39/oz for silver.

Average processing recovery factors of 85% for copper, 85.7% for zinc, 86.5% for lead, and 94.6% for silver.

Average payables of 95% for copper, 85% for zinc, 87.9% for lead, and 85% for silver.

AgEq = (grade of Ag in g/t + [(grade of copper in ppm * price of copper in C\$/g * recovery of copper in % * payable copper in %)+(grade of lead in % * price of lead in C\$/lb * 22.046 * recovery of lead in % * payable lead in %) + (grade of zinc in % * price of zinc in C\$/lb * 22.046 * recovery of lead in % * payable lead in %)/ price of silver in C\$/g * recovery of silver in % * payable silver in %).

Numbers may not compute exactly due to rounding.

Table 14-10 through Table 14-12 show the mineral resource estimates for the MQV, SMS and STK zones respectively for a range of AgEq cut-offs and with the same rounding of tonnes and grades as for Table 14-9. It should be noted that the AgEq average grade is relatively insensitive to the cut-off grade, with the exception of the STK Indicated and Inferred. At all cut-offs the AgEq grade is significantly higher than 100% and therefore the use of a threshold grade has little impact on the total mineral resource. Note there are no Measured mineral resources reported for SMS. Base Case is highlighted.

Table 14-10: PC Silver Mine MQV Domain Mineral Resource Estimate at a Range of Cut-off Grades

Domain	Cut-off AgEq g/t	Classification	Tonnes	AgEq g/t	Ag pg/t	Cu %	Pb %	Sb ppm	Zn %
MQV	1000	Measured	61,000	1,109	345.64	0.75	18.06	2,345	19.13
MQV	1000	Indicated	86,000	1,099	339.25	0.67	19.91	2,593	17.88
MQV	1000	M&I	147,000	1,103	341.91	0.70	19.14	2,490	18.40
MQV	1000	Inferred	1,071,000	1,121	325.47	0.97	9.72	4,834	26.07
MQV	750	Measured	309,000	909	274.79	0.60	14.27	1,781	16.53
MQV	750	Indicated	967,000	862	246.79	0.53	15.93	2,211	14.65
MQV	750	M&I	1,276,000	873	253.58	0.55	15.53	2,107	15.11
MQV	750	Inferred	1,445,000	1,052	305.97	0.90	9.75	4,449	24.03
MQV	500	Measured	742,000	741	221.17	0.43	11.86	1,146	13.68
MQV	500	Indicated	3,401,000	693	201.07	0.42	13.19	1,874	11.40
MQV	500	M&I	4,144,000	701	204.67	0.42	12.95	1,743	11.81
MQV	500	Inferred	3,634,000	791	218.93	0.59	9.27	2,931	17.44
MQV	450	Measured	809,000	719	214.41	0.41	11.65	1,066	13.21
MQV	450	Indicated	3,861,000	666	194.02	0.40	12.77	1,791	10.90

Domain	Cut-off AgEq g/t	Classification	Tonnes	AgEq g/t	Ag pg/t	Cu %	Pb %	Sb ppm	Zn %
MQV	450	M&I	4,669,000	676	197.55	0.40	12.58	1,666	11.30
MQV	450	Inferred	3,857,000	772	213.58	0.57	9.17	2,839	16.98
MQV	400	Measured	845,000	707	210.43	0.40	11.51	1,030	12.96
MQV	400	Indicated	4,269,000	644	187.50	0.38	12.37	1,724	10.51
MQV	400	M&I	5,115,000	654	191.28	0.38	12.23	1,609	10.91
MQV	400	Inferred	4,388,000	730	200.17	0.54	8.66	2,648	16.13
MQV	350	Measured	854,000	703	209.21	0.40	11.45	1,021	12.90
MQV	350	Indicated	4,552,000	627	182.24	0.37	12.01	1,672	10.30
MQV	350	M&I	5,407,000	639	186.50	0.37	11.92	1,569	10.71
MQV	350	Inferred	4,555,000	716	196.38	0.52	8.56	2,589	15.82
MQV	300	Measured	856,000	702	208.96	0.40	11.44	1,020	12.89
MQV	300	Indicated	4,791,000	612	177.73	0.36	11.69	1,627	10.08
MQV	300	M&I	5,647,000	626	182.46	0.36	11.66	1,535	10.51
MQV	300	Inferred	4,664,000	707	193.62	0.52	8.48	2,548	15.62
MQV	250	Measured	856,000	702	208.96	0.40	11.44	1,020	12.89
MQV	250	Indicated	4,957,000	601	174.28	0.35	11.46	1,592	9.92
MQV	250	M&I	5,813,000	616	179.39	0.36	11.46	1,507	10.36
MQV	250	Inferred	4,769,000	698	190.82	0.51	8.38	2,506	15.41
MQV	200	Measured	856,000	702	208.96	0.40	11.44	1,020	12.89
MQV	200	Indicated	5,137,000	587	170.34	0.34	11.20	1,553	9.72
MQV	200	M&I	5,993,000	604	175.86	0.35	11.24	1,477	10.17
MQV	200	Inferred	4,860,000	689	188.26	0.50	8.30	2,470	15.21
MQV	150	Measured	856,000	702	208.96	0.40	11.44	1,020	12.89
MQV	150	Indicated	5,318,000	574	166.12	0.33	10.94	1,513	9.50
MQV	150	M&I	6,174,000	591	172.06	0.34	11.01	1,445	9.97
MQV	150	Inferred	4,932,000	681	186.11	0.49	8.21	2,439	15.04
MQV	100	Measured	856,000	702	208.96	0.40	11.44	1,020	12.89
MQV	100	Indicated	5,407,000	566	163.92	0.33	10.79	1,494	9.38
MQV	100	M&I	6,263,000	585	170.07	0.34	10.88	1,429	9.86
MQV	100	Inferred	4,958,000	679	185.33	0.49	8.18	2,429	14.98

Table 14-11: PC Silver Mine SMS Domain Mineral Resource Estimate at a Range of Cut-off Grades

Domain	Cut-off AgEq ppm	Classification	Tonnes	AgEq ppm	Ag ppm	Cu %	Pb %	Sb ppm	Zn %
SMS	500	Indicated	21,000	543	87.31	0.02	8.85	103	14.59
SMS	500	M&I	21,000	543	87.31	0.02	8.85	103	14.59
SMS	500	Inferred	9,000	537	86.40	0.01	7.13	85	15.69
SMS	450	Indicated	64,000	495	78.39	0.02	7.78	103	13.57
SMS	450	M&I	64,000	495	78.39	0.02	7.78	103	13.57
SMS	450	Inferred	20,000	502	81.37	0.01	6.77	88	14.55
SMS	400	Indicated	131,000	457	73.63	0.02	7.10	99	12.51
SMS	400	M&I	131,000	457	73.63	0.02	7.10	99	12.51
SMS	400	Inferred	42,000	460	71.63	0.01	6.22	83	13.45
SMS	350	Indicated	249,000	417	67.19	0.02	6.41	95	11.51
SMS	350	M&I	249,000	417	67.19	0.02	6.41	95	11.51
SMS	350	Inferred	91,000	412	62.02	0.01	5.43	74	12.26
SMS	300	Indicated	354,000	390	61.84	0.02	5.91	91	10.85
SMS	300	M&I	354,000	390	61.84	0.02	5.91	91	10.85
SMS	300	Inferred	150,000	377	56.35	0.01	5.04	76	11.17
SMS	250	Indicated	436,000	368	57.93	0.02	5.54	87	10.30
SMS	250	M&I	436,000	368	57.93	0.02	5.54	87	10.30
SMS	250	Inferred	214,000	346	51.42	0.01	4.64	76	10.28
SMS	200	Indicated	488,000	353	55.05	0.02	5.28	85	9.93
SMS	200	M&I	488,000	353	55.05	0.02	5.28	85	9.93
SMS	200	Inferred	243,000	332	48.99	0.01	4.45	75	9.87
SMS	150	Indicated	507,000	347	53.85	0.02	5.17	84	9.75
SMS	150	M&I	507,000	347	53.85	0.02	5.17	84	9.75
SMS	150	Inferred	253,000	326	48.04	0.01	4.37	74	9.69
SMS	100	Indicated	510,000	345	53.62	0.02	5.15	83	9.72
SMS	100	M&I	510,000	345	53.62	0.02	5.15	83	9.72
SMS	100	Inferred	255,000	325	47.88	0.01	4.35	73	9.67

Table 14-12: PC Silver Mine STK Domain Mineral Resource Estimate at a Range of Cut-off Grades

Domain	Cut-off AgEq ppm	Classification	Tonnes	AgEq ppm	Ag ppm	Cu %	Pb %	Sb ppm	Zn %
STK	750	Inferred	3,000	765	189.67	0.30	12.45	1,537	16.34
STK	500	Measured	16,000	535	122.88	0.26	5.83	1,273	13.88
STK	500	Indicated	11,000	521	124.85	0.22	5.51	1,169	13.55
STK	500	M&I	27,000	529	123.71	0.24	5.70	1,230	13.74
STK	500	Inferred	108,000	599	128.51	0.26	8.17	1,322	14.85
STK	450	Measured	39,000	500	114.13	0.23	5.20	1,180	13.22
STK	450	Indicated	76,000	477	101.71	0.19	4.61	972	13.36
STK	450	M&I	115,000	485	105.90	0.20	4.81	1,042	13.31
STK	450	Inferred	134,000	575	126.13	0.25	7.79	1,270	14.18
STK	400	Measured	76,000	463	106.34	0.22	4.66	1,103	12.32
STK	400	Indicated	235,000	439	94.24	0.18	4.19	905	12.29
STK	400	M&I	311,000	445	97.19	0.19	4.30	953	12.29
STK	400	Inferred	190,000	532	119.29	0.24	7.14	1,244	13.00
STK	350	Measured	92,000	447	103.86	0.22	4.50	1,076	11.84
STK	350	Indicated	553,000	401	86.55	0.17	3.90	857	11.12
STK	350	M&I	645,000	407	89.02	0.17	3.98	888	11.22
STK	350	Inferred	294,000	477	107.45	0.21	6.38	1,102	11.64
STK	300	Measured	104,000	435	102.31	0.21	4.45	1,063	11.40
STK	300	Indicated	1,038,000	365	79.54	0.16	3.67	815	9.98
STK	300	M&I	1,142,000	371	81.61	0.16	3.74	837	10.11
STK	300	Inferred	484,000	417	96.51	0.19	5.64	955	10.00
STK	250	Measured	120,000	414	98.77	0.20	4.27	1,042	10.74
STK	250	Indicated	1,660,000	331	73.24	0.15	3.44	774	8.90
STK	250	M&I	1,780,000	337	74.96	0.15	3.49	792	9.03
STK	250	Inferred	907,000	350	82.63	0.16	4.75	798	8.30
STK	200	Measured	128,000	401	97.28	0.20	4.15	1,028	10.32
STK	200	Indicated	2,285,000	301	66.94	0.14	3.25	712	7.97
STK	200	M&I	2,413,000	307	68.55	0.14	3.30	729	8.10
STK	200	Inferred	1,562,000	296	70.93	0.14	4.09	677	6.92
STK	150	Measured	128,000	401	97.28	0.20	4.15	1,028	10.32
STK	150	Indicated	3,301,000	262	57.83	0.12	2.94	618	6.85
STK	150	M&I	3,429,000	267	59.30	0.13	2.99	634	6.98
STK	150	Inferred	2,433,000	252	60.55	0.12	3.57	574	5.80
STK	100	Measured	128,000	401	97.28	0.20	4.15	1,028	10.32

Domain	Cut-off AgEq ppm	Classification	Tonnes	AgEq ppm	Ag ppm	Cu %	Pb %	Sb ppm	Zn %
STK	100	Indicated	4,727,000	221	48.94	0.10	2.60	522	5.69
STK	100	M&I	4,855,000	226	50.21	0.11	2.64	536	5.81
STK	100	Inferred	3,256,000	221	52.56	0.10	3.15	502	5.11

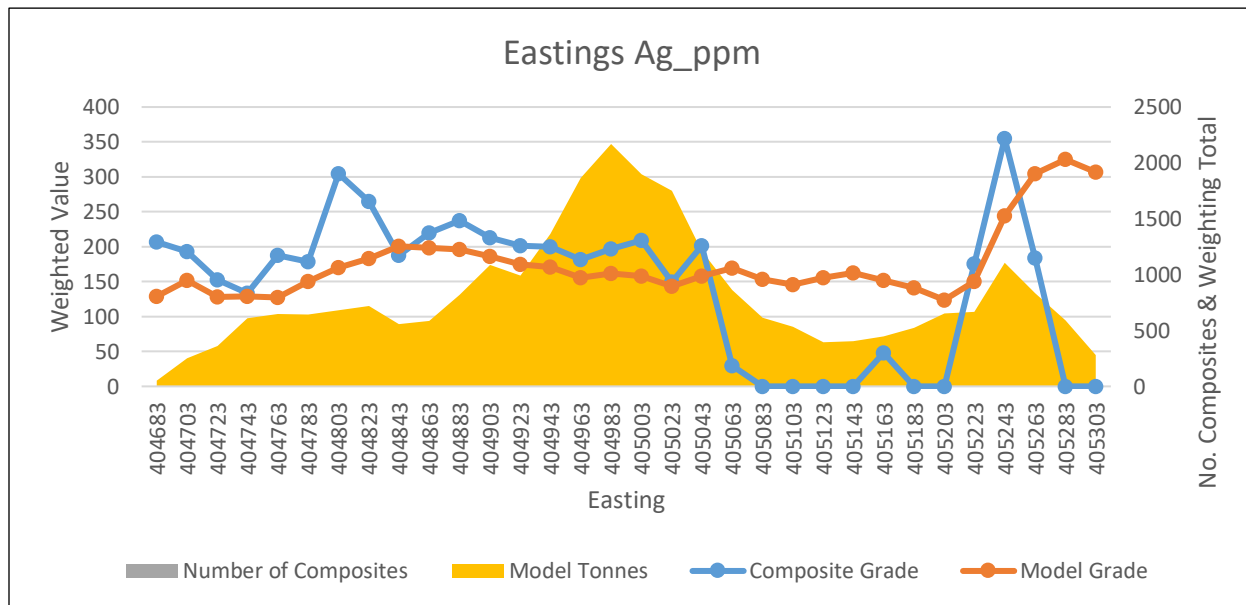
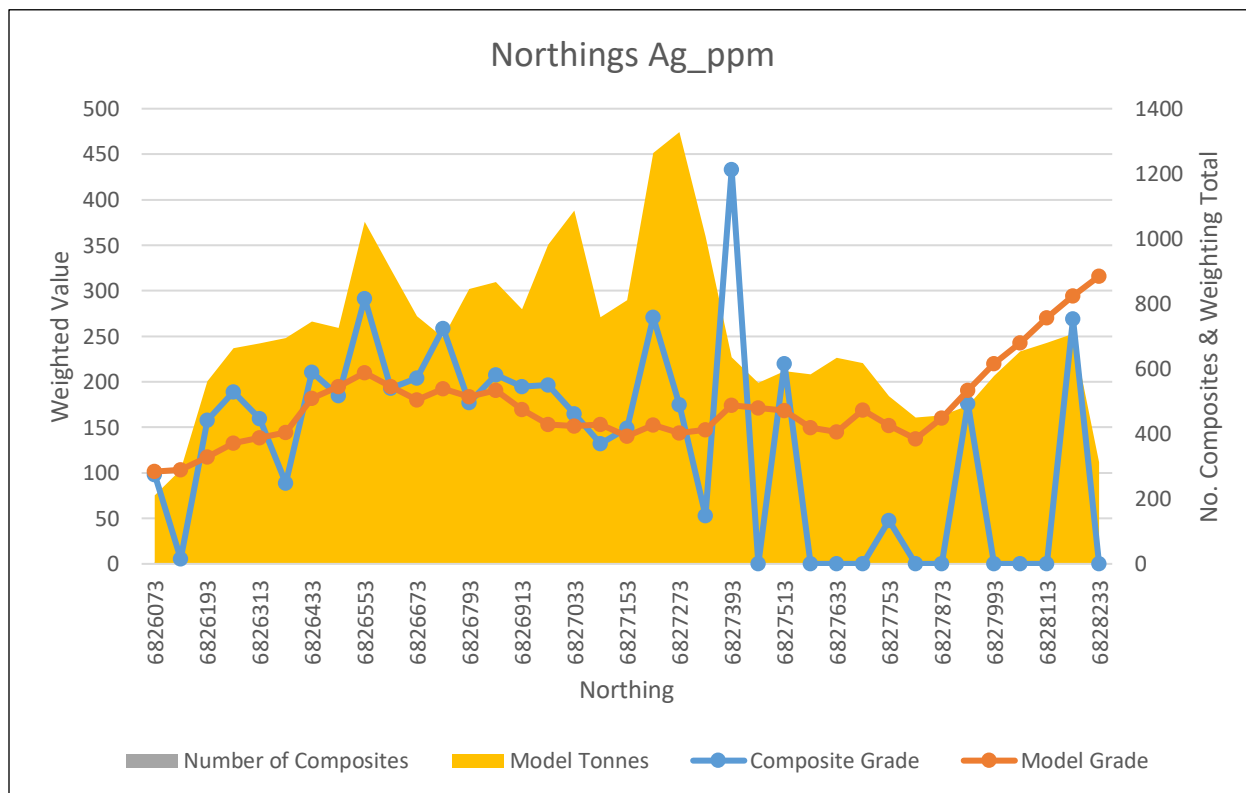
Other than the normal uncertainties that pertain to mineral properties, the author is not aware of any known environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other similar factors that could materially affect the stated mineral resource estimates.

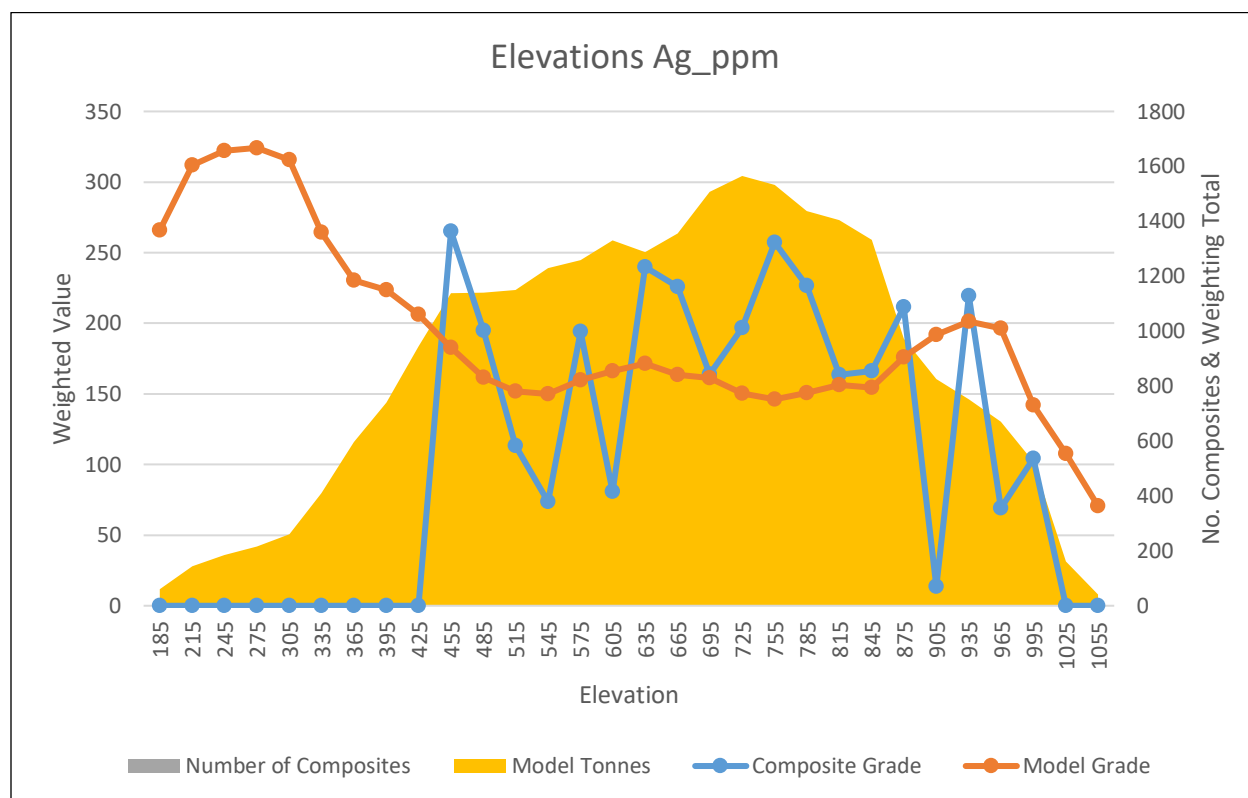
14.14 Block Model Validation

The block model was validated by visual comparison of drillhole assay grades relative to block grades, and numerically by comparison of composite average grades with corresponding block model average grades in the form of swath plots.

Visual inspection shows that the block model honors the boundaries of the wireframes and that the block grades correspond well with the relevant assay grades. Figure 14-7 shows the graphic comparison of composite and block grades on east-west, north-south and vertical bands through the MQV Zone and demonstrates the reasonable correspondence between the original and interpolated grades. These plots also demonstrate that interpolation smooths short-scale variability in composite grades.

Figure 14-7: PC Silver Mine MQV Zone Swath Plots





Source: Ausenco (2021)

14.15 Comparison with October 15, 2021 Mineral Resource Estimate

Table 14-13 shows a comparison between the current (June 2026) and most recent (September 2021) mineral resource estimate. The 2021 estimate is tabulated in ZnEq with an 8% cut-off and the 2026 estimate is tabulated in AgEq with a 100 g/t cut-off. The zinc price in 2021 was approximately CAD\$1.60, so the 8% cut-off had a dollar value of approximately CAD\$260; the current cut-off of 100 g AgEq has a dollar value of approximately CAD\$160. The current resource is approximately 20% larger and most of the additional tonnes are located in the MQV and STK Zones. The SMS resource has decreased by approximately 30%. The overall increase in tonnage is attributed to the increase in price of silver, lead and zinc relative to the 2021 prices as well as the lower cut-off grade.

Table 14-13: PC Silver Mine Comparison of Current and September 2021 Resource Estimates

Current (June 2026) Estimate (100 g/t AgEq Cut-off)					September 2021 Estimate (8% ZnEq Cut-off)			
	Tonnes	Ag g/t	Pb %	Zn %	Tonnes	Ag (g/t)	Pb (%)	Zn (%)
MQV								
Measured	856,000	209	11.44	12.89	903,000	206	11.2	12.9
Indicated	5,407,000	164	10.79	9.38	5,248,000	181	12	10.3
M & I	6,263,000	170	10.88	9.86	6,152,000	184	11.9	10.7
Inferred	4,958,000	185	8.18	14.98	3,849,000	207	8.4	16.7
SMS								
Measured					-	-	-	-
Indicated	510,000	54	5.15	9.72	722,000	53	5.1	9.7
Inferred	255,000	48	4.35	9.67	367,000	47	4.4	9.6
STK								
Measured	128,000	97	4.15	10.32	128,000	97	4.1	10.3
Indicated	4,727,000	49	2.60	5.69	2,754,000	63	3.21	7.63
M & I	4,855,000	50	2.64	5.81	2,882,000	653	3.21	7.74
Inferred	3,256,000	53	3.15	5.11	2,187,000	67	4	6.7
TOTAL								
Measured	984,000	194	10.5	12.6	1,031,000	193	10.3	12.6
Indicated	10,644,000	108	6.9	7.8	8,724,000	133	8.6	9.4
M & I	11,628,000	115	7.2	8.2	9,755,000	139	8.8	9.7
Inferred	8,469,000	130	6.1	11.0	6,403,000	150	6.7	12.9
TOTAL	20,097,000				16,158,000			

15 MINERAL RESERVE ESTIMATE

Mineral resources are not mineral reserves and have not demonstrated economic viability. There is no certainty that all or any part of the mineral resources would be converted into mineral reserves. Mineral reserves can only be estimated as a result of an economic evaluation as part of a preliminary feasibility study or a feasibility study of a mineral project. Accordingly, at the present level of development, there are no mineral reserves at the PC Silver Mine.

16 MINING METHODS

16.1 Introduction

The PC Silver Mine is a proposed underground mine comprised of three mining zones, the MQV, STK and SMS zones. Historically, three levels of adits (970 L, 930 L, 883 L) were developed, along with five shrinkage stopes that have been partially mined. Three portals were also established, each corresponding to one of the existing mining levels. These existing workings were incorporated as much as possible into the proposed mine plan to reduce new development requirements.

The MQV zone is the largest mining zone and is characterized by a narrow, steeply dipping vein, with most of the mineralization dipping between approximately 70° and 90°. The planned mining strike length is approximately 2.2 km across the domain. The STK zone is also steeply dipping but generally wider than the MQZ zone with a mining strike length of approximately 500 m. The SMS zone is flatter lying and wider, with a strike length of 800 m.

The MQV and STK zones will be mined using longitudinal retreat longhole stoping, with production drilling predominantly completed in the downward direction. Uphole drilling will be used from the uppermost drives where the drilling is completed blind. Stope dimensions in these zones are predominantly 20 m high and 20 m in length.

The SMS zone will be mined using a combination of longitudinal and transverse longhole stoping with upwards drilling. Stope heights will vary up to 20 m, with a greater proportion of shorter stopes required due to the flatter geometry of the mineralized zone varying in height up to 20 m and varying in width. Stoping will be completed in a retreating fashion, working back towards the level access.

Underground production will ramp up from 1,800 t/d in the first year of production before reaching a steady state rate of 2,400 t/d in production Year 2. At full run rate, annual production will be approximately 876,000 tonnes, with a mine life of approximately 22 years. For improved mining efficiencies, the underground mine was divided into four distinct zones (A,B,C,D), each accessed by a dedicated ramp. These areas and further broken up into production zones, with mining occurring in a bottom up fashion in these production zones with the higher level not commencing until production on the lower level has been finalized.

Approximately 5 km of historical underground workings have been completed across the mine, including historical development associated with the MQV zone, where five shrinkage stopes were partially mined on the 883 and 930 levels.

The 883 L portal will serve as the primary mine access and main ventilation intake. Existing workings will be utilized where practical, with rehab and slashing to larger dimensions completed as required.

Dense media separation will be used as a material pre-filtering step prior to sending the material to be processed, with approximately 37% of material being rejected and sent to the waste rock surface facility. Tailings will be primarily returned underground as paste backfill, with sufficient

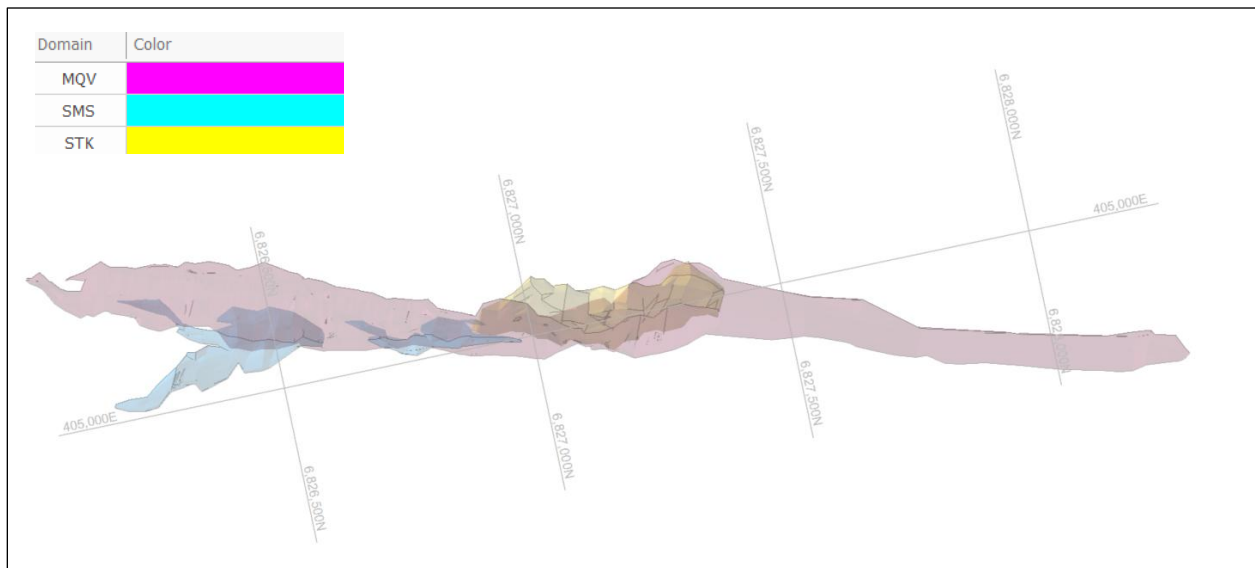
stope void capacity available for most of the mine life. During Years 1 to 3, approximately 90 kt of excess tailings are expected to report to the surface tailings storage facility.

16.2 Deposit Characteristics

The MQV and STK zones comprise hydrothermal quartz-vein mineralization and are both steep dipping. The MQV is relatively narrow and extends over a strike length of approximately 2.3 km and a vertical extent of 630 m, with dips ranging from approximately 60° to 90°. The STK zone has a shorter strike length of approximately 500 m and a vertical extent of 450 m but is far wider than the MQV, with widths up to 40 m. Both zones are generally continuous along their strike lengths.

The SMS zone is shallowly dipping to sub-horizontal, with limited continuity, and a strike length of approximately 800 m and a vertical extent of up to 30 m. The SMS zone ranges in width from approximately 3 m to 120 m. Figure 16-1 and Figure 16-2 show the three mineralized zones, with the MQV being represented by pink, the STK by yellow, and the SMS by blue.

Figure 16-1: Deposit Plan View



16.3.1 Ground Conditions & Rock Quality

The range in rock mass quality parameter Q' within each of the three mineralized systems and the surrounding 3 m of hanging wall and footwall are shown separately in Figure 16-3 and within stopes and the surrounding 3 m of hanging wall and footwall in Figure 16-4.

Figure 16-3: Rock Quality Q' by Mineralized Domain and Immediate Hanging Wall / Footwall

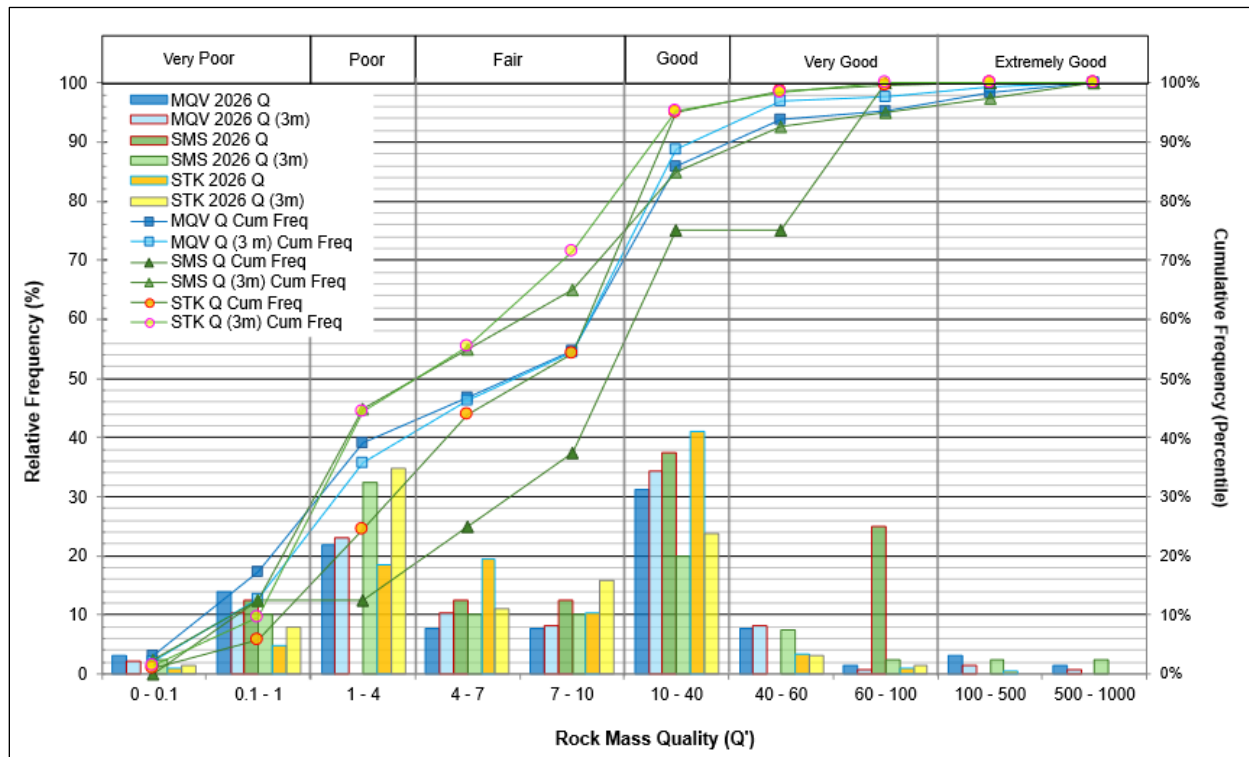
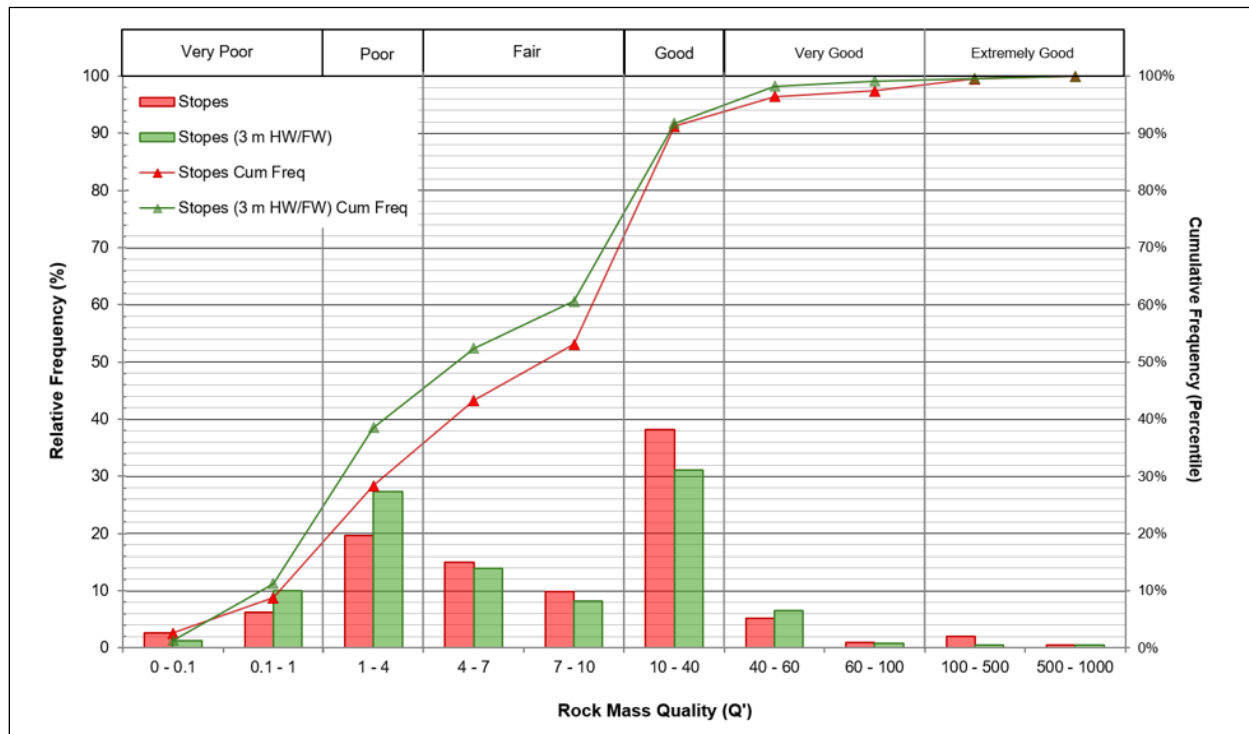


Figure 16-4: Rock Quality Q' within Stopes and Immediate Hanging Wall / Footwall



The range in intact rock strength as indicated from Point Load Testing of core samples is shown in Figure 16-5 where each of the three mineralized systems and the surrounding 3 m of hanging wall and footwall are shown separately. Test results within stopes and the surrounding 3 m of hanging wall and footwall are shown separately in Figure 16-6.

Figure 16-5: Point Load Index Strength by Mineralized Domain and Immediate Hanging Wall / Footwall

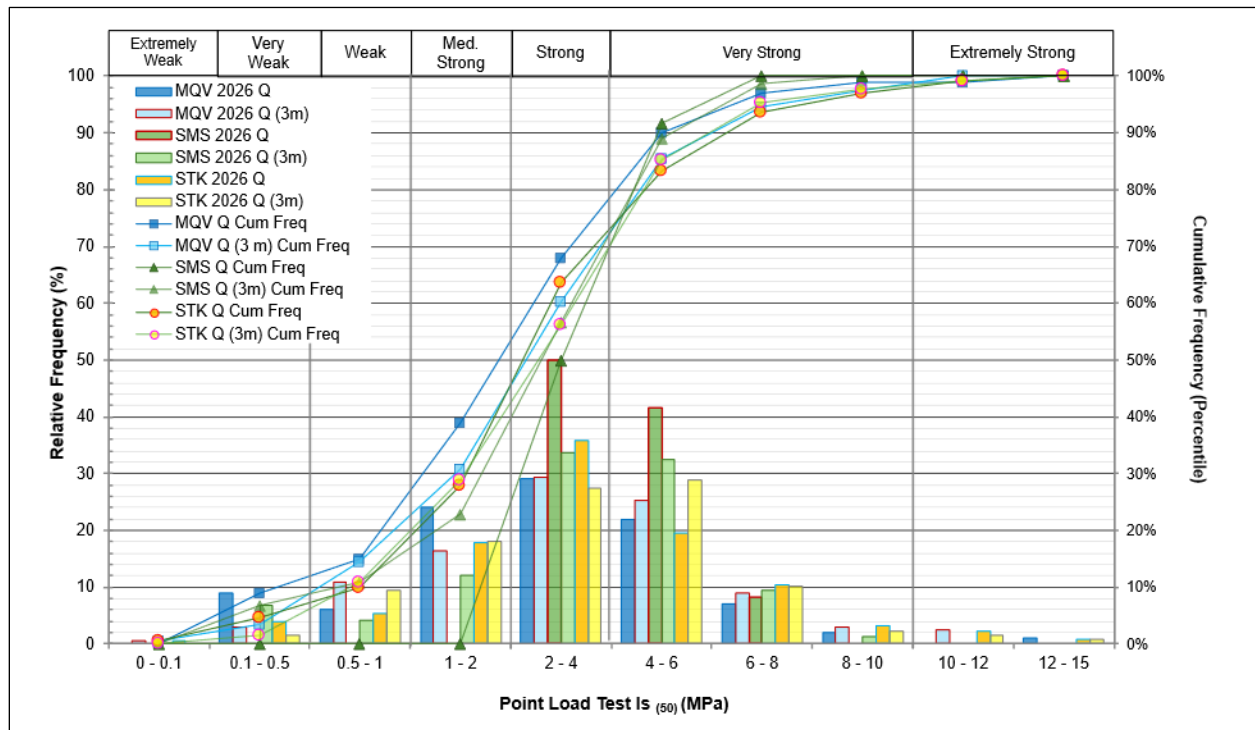
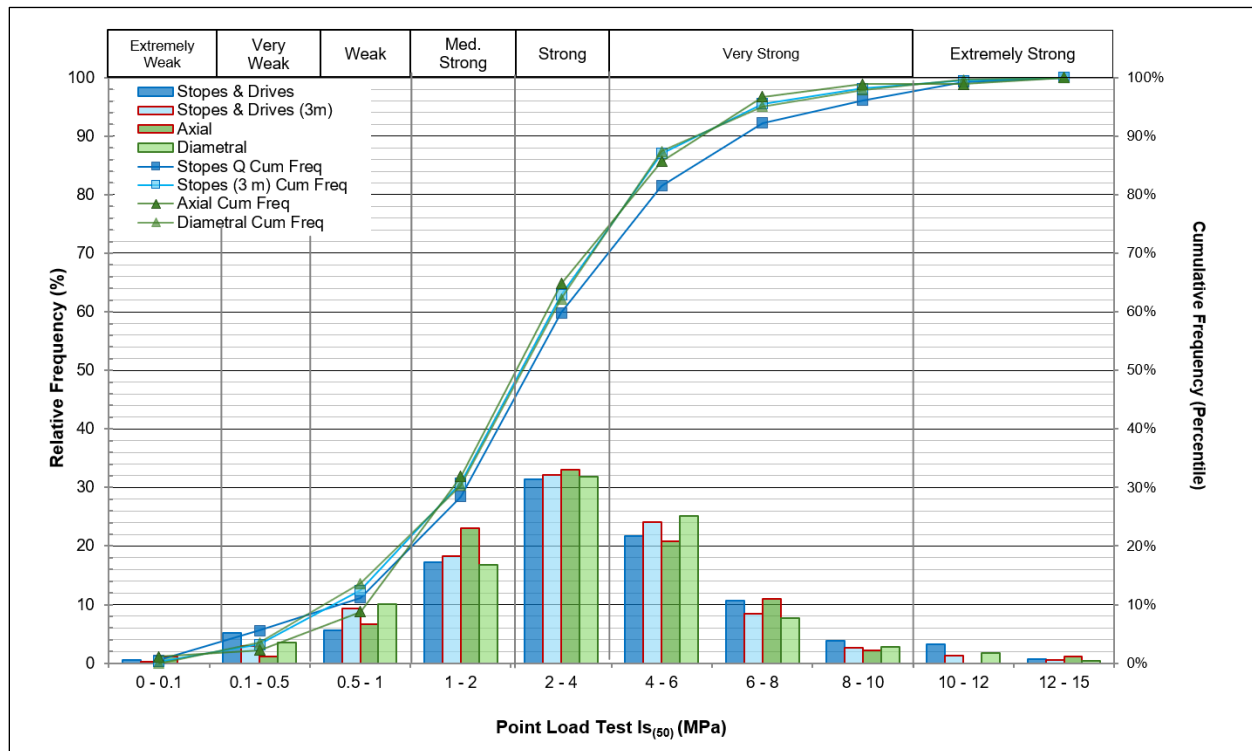


Figure 16-6: Point Load Index Strength within Stopes and Immediate Hanging Wall / Footwall



16.3.2 Stope Stability & Dilution

Maximum stable stope dimensions were estimated using the Potvin (2001) empirical method for the anticipated range of ground conditions. The Trueman & Mawdesley (2003) Stable' line was used as an alternative 'check' against the Potvin (2001) results.

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 (face area divided by perimeter) of the stope face on the x axis.

Stability Number (N') is calculated for each stope face according to the following:

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

Descriptions of each input parameter and the corresponding values selected for the analysis are summarized as follows:

- Rock quality (Q') ranging between 2 ('Poor') for fault zones and 26 ('Good') for typical conditions outside of fault zones. The third quotient in the Q equation (stress reduction factor,

Jw/SRF) has been excluded, stresses are accounted for with the 'A' (stress) parameter below;

- Induced stress parameter (A) is defined as the ratio of the intact rock strength (UCS) to the maximum stress induced by mining and varies between 1.0 for near surface/low stress and 0.1 for very deep and/or high horizontal stress environments. Values of 0.4 and 1.0 were selected to represent the range in depths and anticipated stress conditions;
- Joint orientation factor (B) considers the influence of the dominant structure orientation relative to the stope face with a maximum value of 1.0, where the dominant structures strike perpendicular to the stope wall. Values used in the analysis varied from 0.2 (most conservative) to 0.3 for the hanging wall and to account for potential sub-parallel structures as described above; and
- Gravity factor (C) considers the effect of gravity on potential wedges or slabs that may fall from the walls or back. Factor C is calculated based on the dip angle of each stope face using the equation: $C = 8 - 6 \times \cos(\text{face dip angle})$ and varies from 2, for the most adverse orientation (horizontal/flat backs), up to 8.0 for vertical walls or inward dipping footwalls (most favorable orientation).

A range in stope dimensions was used to assess the maximum stable hydraulic radius for each stope wall based on the following assumptions:

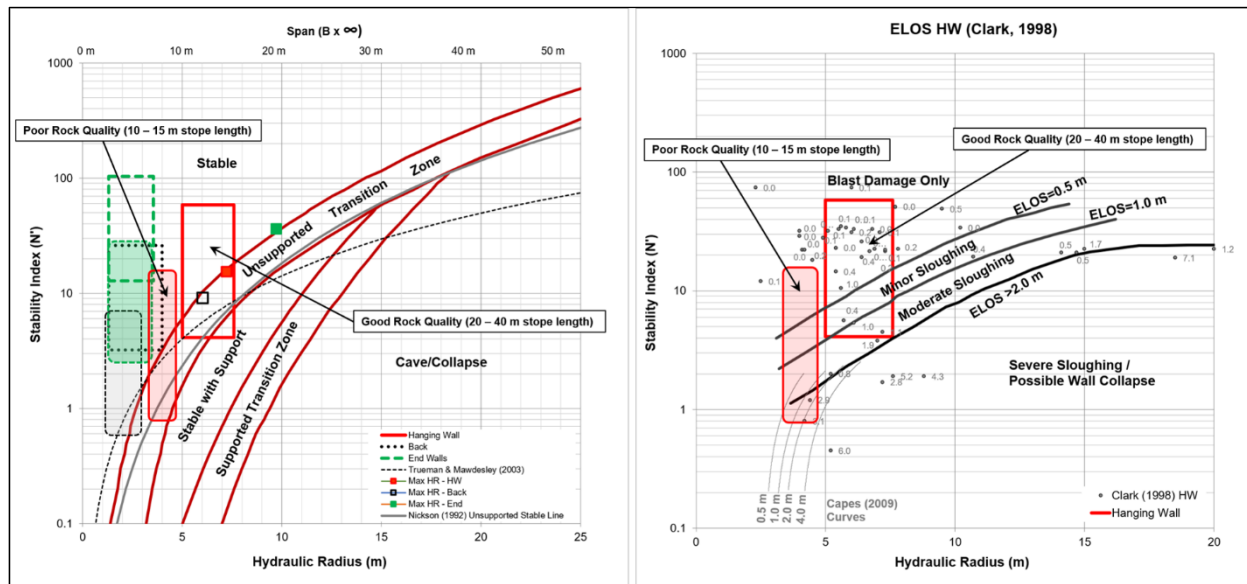
- Stope heights ranging between 20 m (level spacing) up to 24.5 m (20 m plus 4.5 m top access drive);
- Stope widths ranging between 3 and 10 m; and
- Maximum stable stope lengths are estimated from the stability graph considering the range of stope widths and heights listed above.

Based on the analysis discussed above, maximum stope lengths of 10 m for stopes within the poor-quality fault zones and 20 m in remaining areas were recommended for the PEA. Additional characterization and domaining will be required as the Project advances to better define the location and extents of poor-quality fault zones.

Potential for unplanned dilution or overbreak was estimated for the hanging wall using the equivalent linear overbreak/slough (ELOS) method developed by Clark (1998). The amount of unplanned dilution is estimated from an empirical chart, similar stope stability conditions, but contours of dilution/overbreak are provided rather than stability conditions. The contours are shown in terms of the average thickness of overbreak across the full hanging wall area.

Results indicate overbreak or dilution will be primarily blast controlled up to potentially 0.5 m where typical conditions are encountered. Greater dilution of up to 1 m is possible in the poor rock quality fault zones if lengths are not limited to 10 m. Results from the empirical stability and dilution analyses are shown in Figure 16-7.

Figure 16-7: Empirical Slope Stability Analysis and Dilution Estimates



16.3.3 Ground Support

Ground support has been designed based rock mass classification and empirical designs methods such as the Barton & Grimstad (2014) Q system and industry standard rules of thumb.

Ground support requirements were initially assessed according to the Barton criteria for the anticipated range of rock quality considering excavation sizes and primary use during operation. The Q-system accounts for usage of openings by applying a more conservative excavation support ratio (ESR) for man-entry or permanent infrastructure compared to short term development or excavations that will have equipment only access.

The ESR is used to adjust the design span providing an equivalent span (D_e) for use in the Q Support Diagram. This adjustment to the design span in effect, imposes a higher factor of safety on LOM, critical infrastructure (such as ramps, declines, underground workshops, crusher chambers and man-entry stopes) by applying a more conservative ESR than temporary excavations (such as drives, non-man-entry stopes and other mine areas requiring only short-term access).

An ESR of 1.6 for permanent infrastructure and 3 for temporary drives and non-entry stopes has been assumed for the PEA. The anticipated range of rock quality and dimensions are shown on the Barton Q Design Chart on Figure 16-8 for the expected development and intersection spans.

Figure 10: Design Chart for Rock Mass Quality (Q) and Bolt Spacing

The chart relates Rock Mass Quality (Q) to various parameters for design purposes. The top horizontal axis shows RMR₇₆ values (20, 40, 60, 80) and qualitative ratings (Very Poor, Poor, Fair, Good, Very Good). The left vertical axis shows ESR (1 to 100) and Span or Height (m) (1 to 20). The bottom horizontal axis shows Q values (0.001 to 1000) and qualitative ratings (Exceptionally Poor, Extremely Poor, Very Poor, Poor, Fair, Good, Very Good, Extrem. Good, Exc. Good). The right vertical axis shows Bolt length for ESR = 1 (m) (1.5 to 20).

Key Features:

- Bolt spacing in shotcreted area:** Indicated by red diagonal lines with labels (e.g., 1.0 m, 1.2 m, 1.3 m, 1.5 m, 1.7 m, 2.1 m, 2.3 m, 2.5 m).
- Bolt spacing in unsupported area:** Indicated by blue diagonal lines with labels (e.g., 1.0 m, 1.2 m, 1.5 m, 2.0 m, 2.5 m, 3.0 m, 4.0 m, 5.0 m).
- ESR Curves:** Curves for ESR = 1000 J, E = 400 J, and E = 700 J are shown.
- Span or Height (m):** Horizontal dashed lines indicate spans for unsupported conditions (e.g., 1.5 m, 2.4 m, 3 m, 4 m, 5 m, 7 m, 11 m, 20 m).
- Rock Mass Quality (Q):** The bottom axis shows Q values and qualitative ratings.
- Legend:**
 - Typical Rock Quality (Blue background)
 - Local Fault Zones (Red background)

Rock Mass Quality (Q) Formula:

$$Q = \frac{RQD}{J_n} \times \frac{J_r}{J_a} \times \frac{J_w}{SRF}$$

Minimum ground support recommendations for nominal 5 m x 5 m permanent and temporary development drifts are as follows:

- Permanent / LOM Development:
 - 1.5 m rebar with resin at 1.2 m spacing;
 - Welded wire mesh to 1.5 m above sill; and
 - Estimated 10-20% fiber shotcrete requirement.

- Temporary / Mineralized Development:
 - 1.5 m split sets at 1.2 m spacing; and
 - Welded wire mesh to 1.5 m above sill.
- Vertical Development:
 - 1.8 m forged head rebar and resin at 1 m spacing in walls; and
 - 1.2 m mechanical bolts and welded wire mesh in face.

16.4 Hydrogeology Analysis and Recommendations

The mine dewatering system is designed to maintain the underground workings in an operational condition while retaining sufficient pumping capacity to manage potential water inflow events. Peak groundwater inflow is estimated at approximately 64 L/s with an average of 50 L/s.

16.5 Mining Methods

Mining will be completed using long-hole stoping (LHS), with predominantly downward drilling with upward drilling in places and a backfill method of cement paste. Stopes will be mined in a retreating order to the level access. A long-hole production drill will be used to drill each stope, and after blasting a Load-Haul-Dump (LHD) and truck combination will be used to muck and haul the mineralized material.

- Long-hole stoping has a mining recovery of 95%;
- 5% external mining dilution (post stope optimization inventory creation) was applied in addition to the ELOS allowance on the footwall and hanging wall;
- Stopes in the MQV and STK zones were developed with a 20 m strike length and 20 m height, with further optimizations completed at 5 m intervals. Stope dimensions within the SMS zone vary according to the thickness of the mineralized domain; and
- Stopes in the MQV and STK zones are mined longitudinally whereas the SMS zone has a combination of longitudinal and transverse stoping.

Figure 16-9 and Figure 16-10 show plan and isometric views of the proposed development and stopes, colored by attribute type. The isometric view is oriented looking from west to east.

Figure 16-9: Mining Plan View

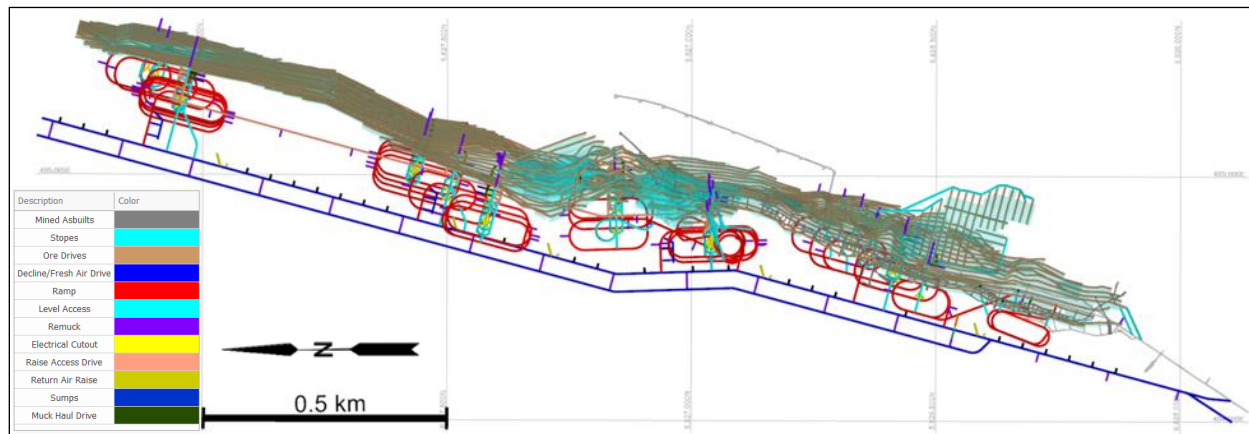
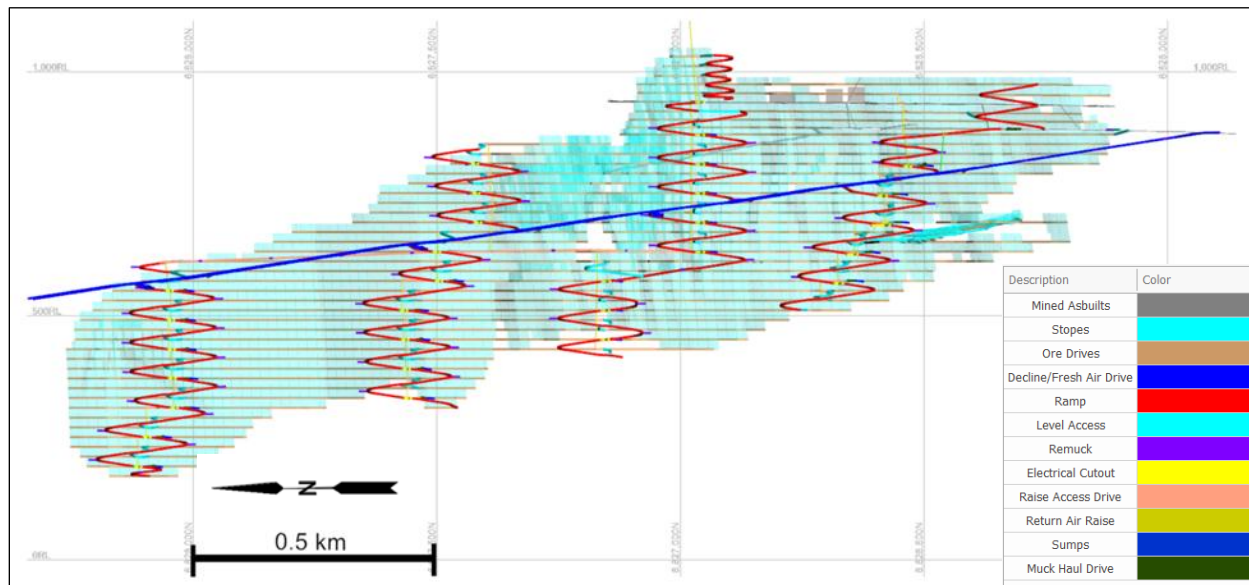


Figure 16-10: Mining Isometric View Looking East (North to South)



16.6 Mine Design

16.6.1 Optimization

The design criteria used for stope optimization and generating the mining inventory are summarized in Table 16-1.

Table 16-1: Stope Optimization Parameters

Parameter	Unit	Values
Revenue, Smelting, and Refining		
Zinc Price	US\$/lb	1.27
Lead Price	US\$/lb	0.92
Silver Price	US\$/oz	37.00
Copper Price	US\$/lb	5.57
Exchange Rate	USD:CAD	0.72
Zinc Payable	%	85.0
Lead Payable	%	87.9
Silver Payable	%	85.0
Copper Payable	%	96.5
Silver Eq (AgEq)	g/t	488
Indicative Value of Contained Metals	CAD\$/tonne	634
Net Smelter Royalty	%	2.2
Operating Cost Estimate		
Mining Cost - Underground Only	CAD\$/t mined	89.73
Processing Cost	CAD\$/t milled	44.29
G&A	CAD\$/t milled	20.15
Site Services	CAD\$/t milled	29.18
Transportation	CAD\$/t milled	34.50
Sustaining Capital Cost Estimate	CAD\$/t milled	33.25
Total Operating Cost Estimate	CAD\$/t milled	217.84
Recovery and Dilution		
Estimated Mining Dilution for Stope Optimization Input	%	12.0
Equivalent Linear Overbreak/Slough	m	0.5 + 0.5
External Mining Dilution for Mine Planning	%	5.0
Mining Recovery	%	95
Mining Dimensions		
Minimum Dip (MQV & STK)	°	55
Minimum Dip (SMS)	°	0
Minimum Mining Width	m	3
Stope Height (MQV & STK)	m	20
Stope Height (SMS)	m	Variable
Stope Length	m	5 to 20
Mill Recovery		
Zinc Recovery	%	85.7

Parameter	Unit	Values
Lead Recovery	%	86.5
Silver Recovery	%	94.6
Copper Recovery	%	85.0
Cut-off value		
In-Situ Diluted/Recovered Cut-Off Value (including royalty)	CAD\$/t	262.60
In-Situ Diluted/Recovered mill breakeven cut-off value	CAD\$/t	151.04
Other Parameters		
Resource Classification	-	M+I+I
Mill Throughput	t/yr	876,000

The LOM mineralized material, average grades and contained metal by primary commodities are summarized in Table 16-2. Note that 42% of the resources in the mine plan are classified as Inferred.

Table 16-2: LOM Inventory Summary

Parameter	Unit	LOM Total/Grade	Unit	Contained Metal
Mined Mineralized Material	kt	18,588		
Zinc	%	8.52	kt	1,583
Lead	%	6.14	kt	1,141
Silver	g/t	112.5	koz	67,212
Copper	%	0.25	kt	47

16.6.2 Access

Primary Access to the underground deposit will be through the existing 870 L portal. Back slashing will be required to expand the existing development to 5.0 m width by 5.0 m height, with remaining track removed where present. The existing portals at the 930 L and 970 L will support mining of the upper MQV and STK levels. A main decline will be developed to the extent of the mineralization, with a parallel fresh air drive will twin the main decline below approximately the 810 m elevation. Four internal ramps will be developed up and down from the decline to access the majority of the mining zones (A, B, C, and D). Each zone will be accessed by its respective ramp.

16.6.3 Development Types

The dimensions and total life of mine quantities for each development type are summarized in Table 16-3. The life of mine development totals approximately 104,445 m of lateral development and 2,406 m of vertical development. Sublevels spacing is generally 20 m except between existing mine workings where level spacing has been optimized to split the vertical height between the existing development.

Table 16-3: Life of Mine Development Dimensions and Quantities

Development Type	Unit	Values (W x H)	Total
Lateral Development			
As-built Slashing Rehab	m	5 x 5	3,366
Decline	m	5 x 5	2,509
Ramp	m	5 x 5	14,640
Diamond Drill Bay	m	4 x 4	311
Electrical Cut Out	m	5 x 5	1,669
Fresh Air Drive	m	4 x 4	2,059
Level Access	m	5 x 5	12,648
Mine Haul Drive (Loading Bay)	m	5 x 5	1,338
Main Sump	m	5 x 5	106
Return Air Drive	m	4 x 4	3,800
Remuck	m	5 x 5	3,807
Sump	m	4 x 4	797
Substation	m	4 x 4	385
Operating Mineralized Development	m	4 x 4	46,758
Operating Waste Drive	m	4 x 4	9,908
Operating Paste Fill Drive	m	4 x 4	338
Vertical Development			
Fresh Air Raise	m	2.3 Ø	82
Return Air Raise	m	2.3 Ø	2,024
Vertical Rehab	m	2.3 Ø	300

16.7 Mine Services

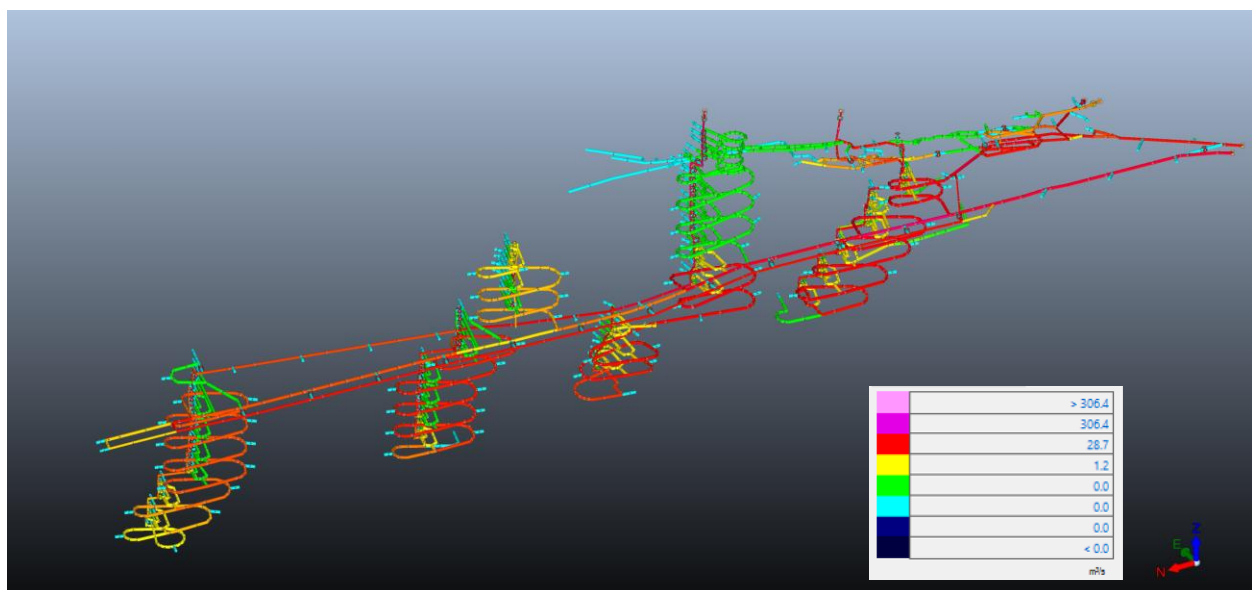
16.7.1 Mine Ventilation

The underground ventilation system was designed using two exhaust raises, with primary fans located on surface at the breakthrough of each raise. Fresh airflow will be the 883 L portal and main decline, while the 930 L and 970 L portals will provide additional flexibility for either intake or exhaust ventilation. One primary fan will provide a fixed airflow of 150 m³/s, while the second will provide 100 m³/s, resulting in a combined airflow of 250 m³/s. Based on ventilation requirements of 0.06 m³/s per kW engine power and 0.025 m³/s for personnel, the peak ventilation demand is estimated at 243.9 m³/s once contingency and leakage factors are applied.

Ventilation raises will be used as exhaust airways, while the main ramps will provide fresh air intake to the active mining areas. Auxiliary fans ranging from 10 hp to 100 hp will be positioned as required to distribute airflow to active development and production headings and will be relocated as mining progresses. Auxiliary fans are to be moved as mining progresses and active mining areas change. Ventilation ducting will range from 36 to 54 inches. in diameter, depending on the development type and airflow requirements, and will direct fresh air to the working faces.

The proposed ventilation design was incorporated into Ventsim, with simulations completed to verify that airflow progressed throughout the entirety of the mine to reach all potential working areas. During periods when certain zones are not actively being mined, ventilation to those areas will be reduced or shut off, allowing airflow to be directed to active mining areas. The ventilation layout is shown in Figure 16-11, with development colored according to airflow rate in m³/s.

Figure 16-11: Mining Ventilation



16.7.2 Mine Air Heating

Mine air heating requirements were estimated using the average monthly temperatures recorded at Fort Simpson Airport (185 km ENE from mine). To maintain a target underground air temperature of 1.5° C and prevent freezing and ice buildup, heating will be required from approximately October through April. The peak heating demand is estimated at 32,000 kBTU/h, with an average heating demand is 20,600 kBTU/h. Mine air heaters will be located at the fresh air intakes at the 883 L and 930 L portals, with additional heating required if the 970 L portal is also used as an intake portal.

16.7.3 Water Supply

Water will be distributed to the underground workings through piping ranging from approximately 2 to 8 inch diameter. Two main underground pump stations are planned, supported by approximately 900 m of sump development. During periods of lower underground water inflow, make up water will be supplied from a surface well. During periods of increased inflow, such as seasonal snowmelt, water will be recirculated where practical, with the surface well potentially replenished during these periods.

16.7.4 Dewatering

Underground dewatering will be managed through a series of level sumps located through the active mining areas with approximately 900 m of planned sump development. Water collected in these sumps will be transferred through satellite sumps system to the two main pump/sump stations, where it will be pumped to surface through piping and managed in accordance with the wastewater management plan. Additional pumps sized from 1 hp to 50 hp will be used at level sumps and active working faces as required. Where practical, collected mine water will be recirculated for underground use, reducing the volume of make-up water required from surface.

16.7.5 Electrical Distribution

Electrical power will be distributed underground through a system of main power stations and level substations. A total of four main power stations is planned, with one assigned to each mining zone (A,B,C and D), along with 18 level substations over the LOM. The level substations will be relocated as required to support active mining areas. Approximately 1,670 m of electrical cut-out development and 385 m of main and substation development have been incorporated into the mine design.

16.8 Back Fill

16.8.1 Basis of Design

The PC Silver Mine will have a paste backfill system to support underground mining and to facilitate the permanent underground storage of flotation tailings. The paste backfill system will

manage all flotation tailings generated by the mill and is not expected to produce backfill from other waste or material streams, such as dense media separation rejects. With this, the paste backfill system is composed of the following areas:

- Filtration system;
- Tailings stockpile;
- Paste production system; and
- Underground distribution system.

During the first year of mine life, there will be approximately 50,000 tonnes of surplus flotation tailings, as the mining sequence results in a lower volume of underground voids requiring backfill. The surplus tailings will be stored in a temporary Tailings Management Facility. In Years 2, 3, and 4, the tailings will be reclaimed to produce paste backfill for permanent underground storage.

At full production, the nominal paste demand (the annual paste demand distributed evenly over a year) is expected to be 30.5 m³/h. To support this demand, the nominal filtration demand (the annual filtration demand distributed evenly over a year) is 41.0 dry tonnes of tailings/h.

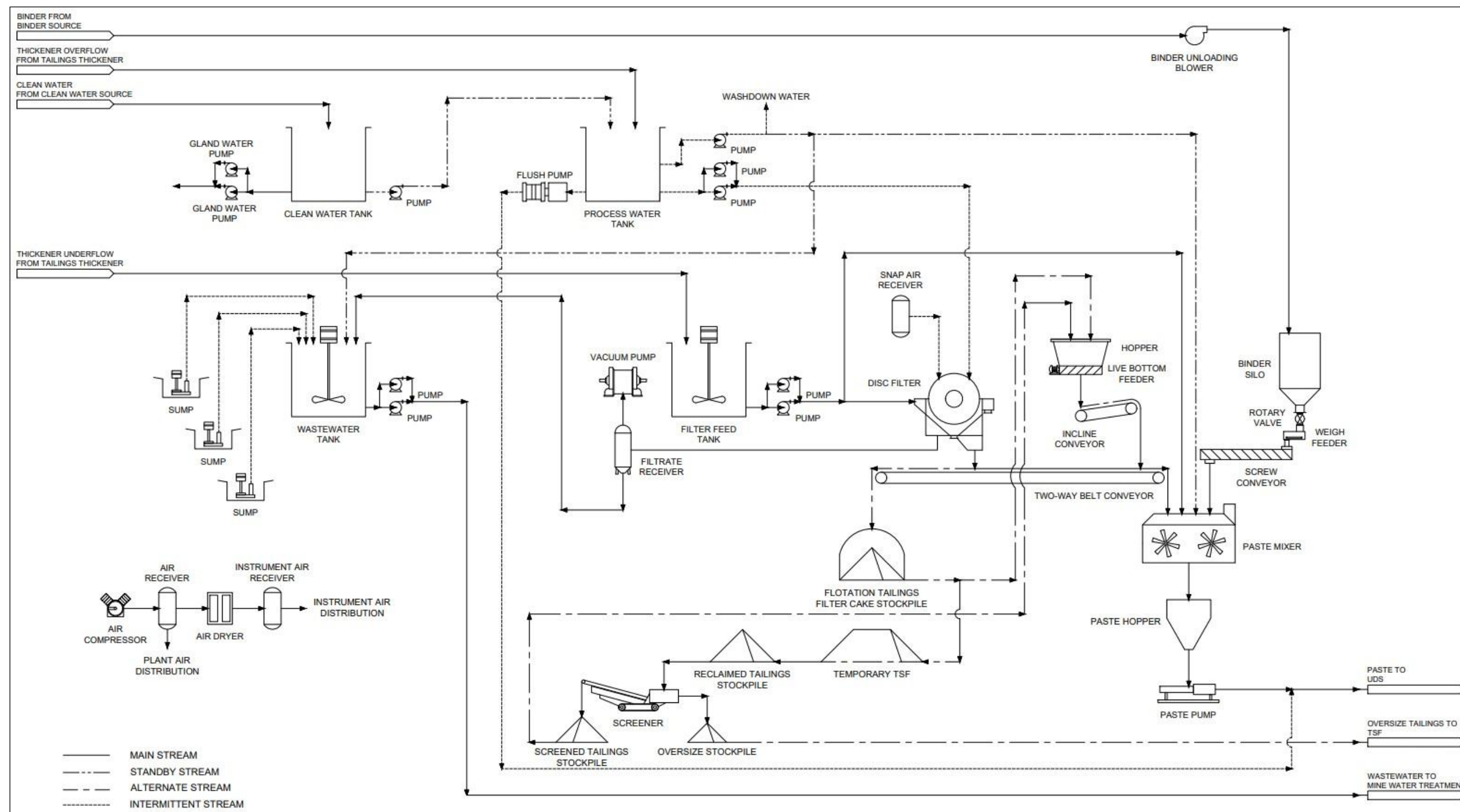
The paste backfill plant will receive flotation tailings as thickener underflow with a solids content of approximately 65% by weight. Filtration will increase the solids content to approximately 80%, producing a material suitable for handling by front-end loaders and storage at the temporary Tailings Management Facility and plant stockpile.

Because the filtration system will not be redundant (there will only be a single disc filter), but is expected to operate continuously, the filter feed tank will be required to have sufficient surge capacity to receive tailings from the Mill should there be an interruption with the filtration system. Surge capacity will also be provided in the form of a tailings stockpile, which will allow the filtration system and the paste production system to operate independently, i.e., paste can be produced when the filtration system is offline and vice versa. Paste production and backfilling will occur intermittently in campaigns. The paste production system will also have a production capacity greater than the nominal paste demand, providing flexibility for planned maintenance downtime. There is an opportunity to optimize project cost and improve system availability by installing a second disc filter. The additional filtration capacity may allow for a reduced filter feed tank capacity and provide operational flexibility during maintenance or downtime.

16.8.2 Process Description

A simplified Process Flow Diagram for the paste backfill plant is provided in Figure 16-12.

Figure 16-12: Simplified Backfill Process Flow Diagram



Source: T Engineering (2026)

Flotation tailings will be delivered from the mill to the paste backfill plant as thickener underflow. The underflow will be pumped to an agitated filter feed tank, sized to store approximately 24 hours of mill tailings production. This surge capacity will enable the mill to operate and manage flotation tailings during short-term interruptions to the filtration system.

Tailings will be continuously filtered using a single disc filter. A fraction of thickener underflow may bypass filtration and be pumped directly to the paste mixer to reduce the load on the filter. Filter cake from the disc filter will be discharged onto a two-way belt conveyor. During paste production, the conveyor will direct filter cake to the paste mixer. When the paste mixer is not operating, the conveyor will reverse direction and direct the filter cake to the tailings stockpile. The stockpile will be housed within an enclosed, heated building to help prevent freezing, control the moisture content of the filter cake, and minimize dust generation. A front-end loader will manage the stockpile tailings.

Filter cake reclaimed from the stockpile will be fed by front-end loader into a hopper equipped with a live-bottom feeder. The feeder will provide a controlled feed of filter cake to an incline conveyor, which will transfer the material to the two-way belt conveyor feeding the paste mixer. The incline conveyor will also be used when tailings are reclaimed from the temporary Tailings Management Facility. A belt scale on the conveyor feeding the mixer will measure the total tailings feed rate and provide input for process control.

Tailings reclaimed from the temporary Tailings Management Facility will be processed through a mobile screener to remove debris and oversize material. Undersize material will be placed in a dedicated stockpile for paste production, with oversize material returned to the Tailings Management Facility disposal.

Cement will be stored in a cement silo. During paste production, cement will be discharged from the silo via a rotary valve, weigh feeder, and screw conveyor for transfer to the paste mixer.

The paste mixer will combine tailings, cement, and trim water to produce paste backfill. The mixer will discharge to a paste hopper below, which will feed into a piston-type positive-displacement pump. The pumps will deliver paste backfill underground.

16.8.3 Underground Distribution System

The underground distribution system will begin at the discharge of the paste pump and comprise of a 4-inch pipeline. Pumped delivery is required because of the long lateral distance between the paste backfill plant and the underground workings. Hydraulic modelling was performed by assessing a variety of flow paths across the mine, with the modelling determining that a 150-bar pump would be required to successfully transfer paste to all mine stopes.

The underground distribution system will follow the mine decline before branching through interlevel boreholes to serve mine levels. The underground distribution system will be composed of Schedule 160 carbon-steel piping along the ramp before transitioning to Schedule 120 and Schedule 80 piping, where reduced pressure permits. Stope piping will be built with HDPE piping.

Stopes will be filled using a plug-body-cap sequence. An initial plug will be placed at the bottom of each stope to above the top of the barricade, reducing the hydrostatic pressure applied to the

barricade. The plug will cure for 48 hours before the body fill can be placed. The body fill will then be poured until the top of the stope.

LOM Backfill quantities are shown in Table 16-4.

Table 16-4: LOM Backfill Quantities

Parameter	Unit	Value
Total Underground Waste Mined	ktonnes	3,495
Underground Waste to Underground Backfill	ktonnes	651
Underground Waste to Waste Rock Surface Facility	ktonnes	2,844
Dense Media Separation Rejects to Waste Rock Surface Facility	ktonnes	6,878
Total to Waste Rock Surface Facility	ktonnes	9,722

16.9 Unit Operations

16.9.1 Drilling

Development drilling will be completed with a 2-boom jumbo to provide efficient advance rates and flexibility for larger developments when required. Development rounds are planned to be predominantly approximately 4 m in length.

Production drilling will be completed using top-hammer long-hole drill, with most stope holes planned at approximately 16 m in length. The use of top-hammer drilling allows for smaller-diameter holes, approximately 2 inches, providing greater flexibility in powder factor selection to help control dilution.

16.9.2 Blasting

Development blasting will use ANFO, pneumatically loaded into the main blast holes, with packaged explosives used in lifter and perimeter holes. Electronic detonators will be used to help improve blast timing and blast control.

Production blasting will be completed using pneumatically loaded ANFO with boosters as required. Electronic detonators will be used to provide accurate blast timing, with all blasting initiated from a centralized location away from return airways and at a safe distance from active underground blast areas.

16.9.3 Ground Support

Ground support will be completed using a mechanized bolter for drilling and installation of ground support. Ground support types are assumed for the purpose of this study to be a combination of rebar with resin and shotcrete where required. For certain ground support installations requiring longer holes, production drills or development jumbos will assist with drilling, including holes for cable bolts.

16.9.4 Mucking

Development mucking will be completed with a 3.0 m³ or 4.0 m³ LHDs. Production mucking will primarily use 4.0 m³ LHD. Material will generally be transported to remuck bays before being loaded onto haul trucks for transport from the active mining areas. Remote-controlled LHDs will be used for production mucking near open stopes where required to keep personnel outside hazardous areas beneath the stope brow. Ancillary mucking activities will be completed using 2.0 m³ loaders.

16.9.5 Hauling

Development and production haulage will be completed using 40 tonne haul trucks, including both conventional haul trucks and ejector trucks. All underground development has been designed to accommodate 40 tonne haul trucks.

16.10 Mine Equipment

Fleet requirements were estimated based on the mine ramp-up and production schedule, with equipment rebuilds and replacements planned as operating-hour thresholds are reached. The underground mine equipment quantities are summarized in Table 16-5.

Table 16-5: Mobile Equipment Fleet On Site

Underground Equipment	LOM Max	Year -1	Year 1	Year 5	Year 10	Year 15	Year 20
Jumbo 2 Boom	2	1	2	3	3	3	3
Long-hole Drill Large	3	0	2	3	3	3	3
Bolter	2	1	2	3	3	3	3
40 tonne Truck	6	2	4	5	6	6	6
2.0 cu. m LHD	1	1	1	1	1	1	1
3.0 cu. m LHD	1	1	1	1	1	1	1
4.0 cu. m LHD	3	1	2	3	3	3	3
Scissor Lifts	1	1	1	1	1	1	1

Underground Equipment	LOM Max	Year -1	Year 1	Year 5	Year 10	Year 15	Year 20
ANFO Loader	2	1	2	2	2	2	2
Boom Truck	2	2	2	2	2	2	2
Fuel/Lube Truck	1	1	1	1	1	1	1
Light Vehicles	4	6	8	8	8	8	8
Personnel Carriers	3	1	3	3	3	2	2
Shotcrete Machine	1	1	1	1	1	1	1
Grader	1	1	1	1	1	1	1
Agitator Truck	1	1	1	1	1	1	1
Telehandler	2	2	2	2	2	2	2
Tractor	2	2	2	2	2	2	2
Total	38	26	38	43	44	43	43

16.11 Mine Personnel

Underground mining personnel requirements and peak staffing levels are summarized in Table 16-6. Staffing is predominantly based on a fly-in fly-out rotation, with workforce requirements increasing as the mine ramps up to steady-state production.

Table 16-6: Underground Personnel

Position	LOM Max	Year -1	Year 1	Year 5	Year 10	Year 15	Year 20
Mine Rescue / Safety Officer / Trainer	2	2	2	2	2	2	2
Mining Foreman	2	2	2	2	2	2	2
Mine Supervisor / Shift Boss	4	4	4	4	4	4	4
Jumbo Operator	8	8	8	8	8	8	4
Production Drill Operator	12	0	8	12	12	12	12
Blaster	8	4	8	8	8	8	8
Drill & Blast Nipper	8	8	8	8	8	8	8
LHD Operator	16	4	12	16	16	16	16
Underground Truck Driver	24	8	12	20	16	16	16
Electrician	3	3	3	3	3	3	3
Development Services Miner	4	4	4	4	4	4	4

Position	LOM Max	Year -1	Year 1	Year 5	Year 10	Year 15	Year 20
Ground Support / Bolter / Shotcrete	8	8	8	8	8	8	8
Utility Vehicle Operator	4	4	4	4	4	4	4
Services Worker	4	4	4	4	4	4	4
Maintenance Planner	1	1	1	1	1	1	1
Lube / PM Mechanic	2	2	2	2	2	2	2
HD Mechanic	4	2	4	4	4	4	4
Drill Mechanic	2	2	2	2	2	2	2
Overhaul Mechanic	2	0	0	0	2	2	2
Welder	2	2	2	2	2	2	2
Apprentice	4	2	4	4	4	4	4
Dry / Lampman / Bitman	4	2	4	4	4	4	4
Total	128	76	106	122	120	120	116

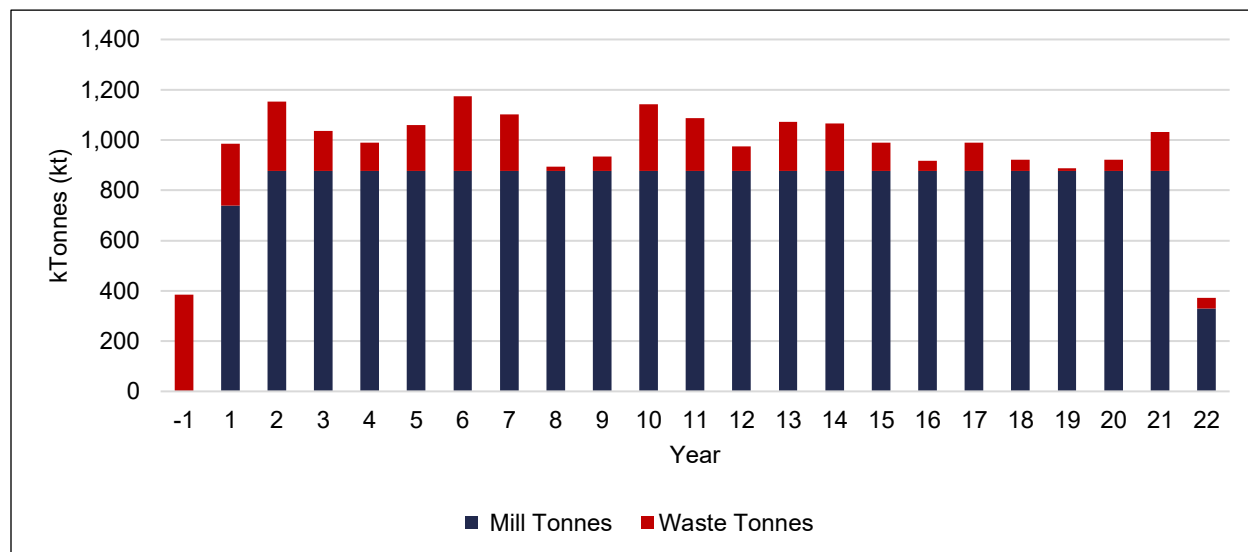
16.12 Mine Production Schedule

A steady-state mining rate of 2400 t/d was used in the mine schedule after a ramp up in Year 1, with an estimated mine life of approximately 22 years. A detailed mine and mill production schedule is provided in Table 16-7, with annual mill and waste tonnes represented graphically in Figure 16-3.

Table 16-7: Mine and Mill Production Schedule

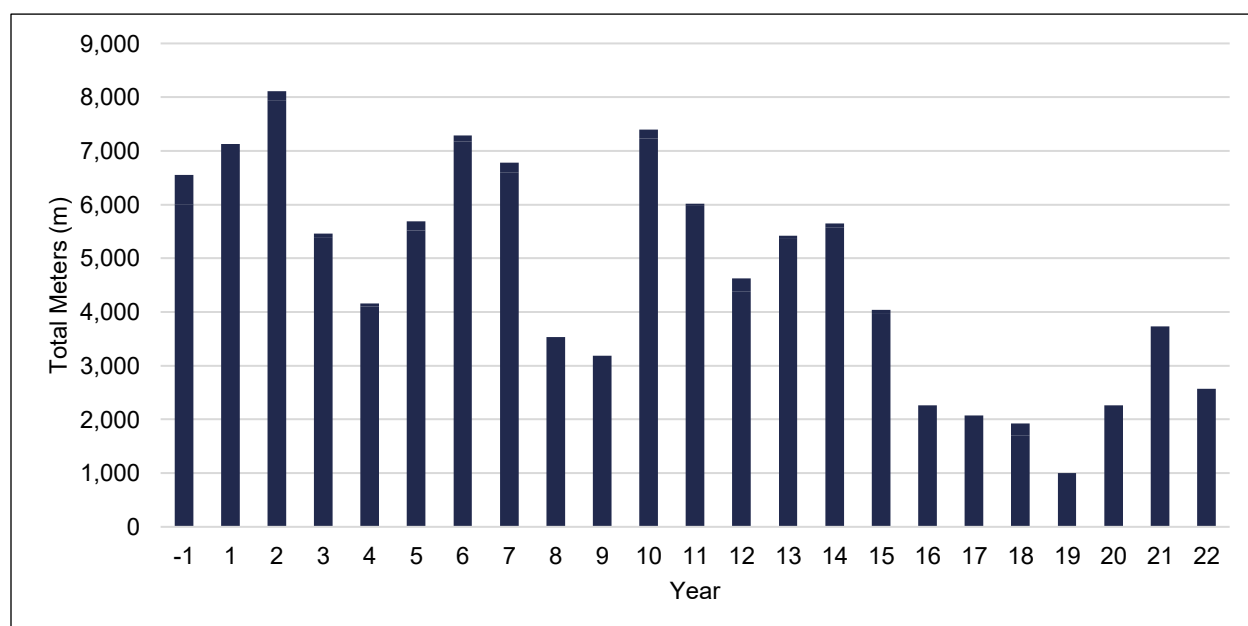
Year	Unit	LOM Total/Avg	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22
MINING, DEVELOPMENT & PROCESSING																									
Mined Mineralized Material	kt	18,588	0	739	876	876	876	876	876	876	876	876	876	876	876	876	876	876	876	876	876	876	876	876	329
Total Waste	kt	3,495	384	246	277	160	113	183	298	226	19	58	266	210	99	196	190	113	42	114	46	11	45	155	42
Total Mined	kt	22,083	384	985	1,153	1,036	989	1,059	1,174	1,102	895	934	1,142	1,086	975	1,072	1,066	989	918	990	922	887	921	1,031	371
Total Lateral Development	m	104,445	5,992	6,983	7,928	5,378	4,102	5,522	7,177	6,598	3,532	3,185	7,226	5,974	4,377	5,360	5,571	3,983	2,260	2,074	1,700	997	2,259	3,700	2,565
Vertical Development	m	2,406	555	149	180	86	57	168	112	185	0	0	170	39	248	61	73	56	0	0	229	0	0	36	0
Total Development	m	106,850	6,548	7,132	8,108	5,464	4,160	5,690	7,289	6,783	3,532	3,185	7,397	6,013	4,625	5,421	5,645	4,039	2,260	2,074	1,929	997	2,259	3,736	2,565
Backfill Volume	km³	6,749	0	189	282	281	278	282	285	296	330	333	338	336	347	337	359	354	324	318	299	297	312	355	216
Dense Media Separation Product	kt	11,710	0	466	552	552	552	552	552	552	552	552	552	552	552	552	552	552	552	552	552	552	552	552	207
Dense Media Separation Rejects	kt	6,878	0	273	324	324	324	324	324	324	324	324	324	324	324	324	324	324	324	324	324	324	324	324	122
MILL HEAD GRADES																									
Silver	g/t	112.5	0.0	145.9	144.9	145.0	158.6	177.0	177.7	176.2	98.3	102.6	102.6	97.2	94.0	116.3	102.8	83.7	79.1	79.0	80.5	80.6	73.6	65.2	76.1
Lead	%	6.1	0.0	8.4	9.1	8.0	7.2	7.4%	7.3	6.5	5.3	5.3	7.0	6.7	6.3	7.5	6.8	5.1	4.8	4.6	4.5	4.4	3.9	3.8	4.9
Zinc	%	8.5	0.0	8.4	8.0	9.8	12.0	13.2%	13.5	13.9	8.4	7.8	6.4	5.4	5.8	6.4	5.9	7.9	7.7	8.0	8.3	8.2	7.6	6.7	7.0
Copper	%	0.25	0.0	0.29	0.31	0.34	0.40	0.47%	0.48	0.48	0.24	0.25	0.18	0.17	0.18	0.22	0.22	0.19	0.18	0.17	0.16	0.15	0.14	0.13	0.15
CONTAINED METAL																									
Silver	koz	67,212	0	3,467	4,081	4,084	4,467	4,986	5,004	4,962	2,767	2,890	2,889	2,737	2,647	3,276	2,896	2,356	2,229	2,224	2,267	2,270	2,072	1,836	804
Lead	kt	1,141	0	62	79	70	63	65	64	57	47	46	61	59	55	65	59	45	42	41	39	38	34	33	16
Zinc	kt	1,583	0	62	70	86	105	115	119	122	74	68	56	48	51	56	52	69	68	70	73	72	66	59	23
Copper	kt	47	0	2.1	2.7	3.0	3.5	4.1	4.2	4.2	2.1	2.2	1.6	1.5	1.5	1.9	1.9	1.7	1.5	1.5	1.4	1.4	1.2	1.1	0.5

Figure 16-13: Annual Production



An average of 13 m per day of lateral development is scheduled for the LOM, with a peak rate of 35 m per day. Vertical development averages 1.0 m per day. The total annual development metres are shown in Figure 16-14.

Figure 16-14: Annual Development Schedule



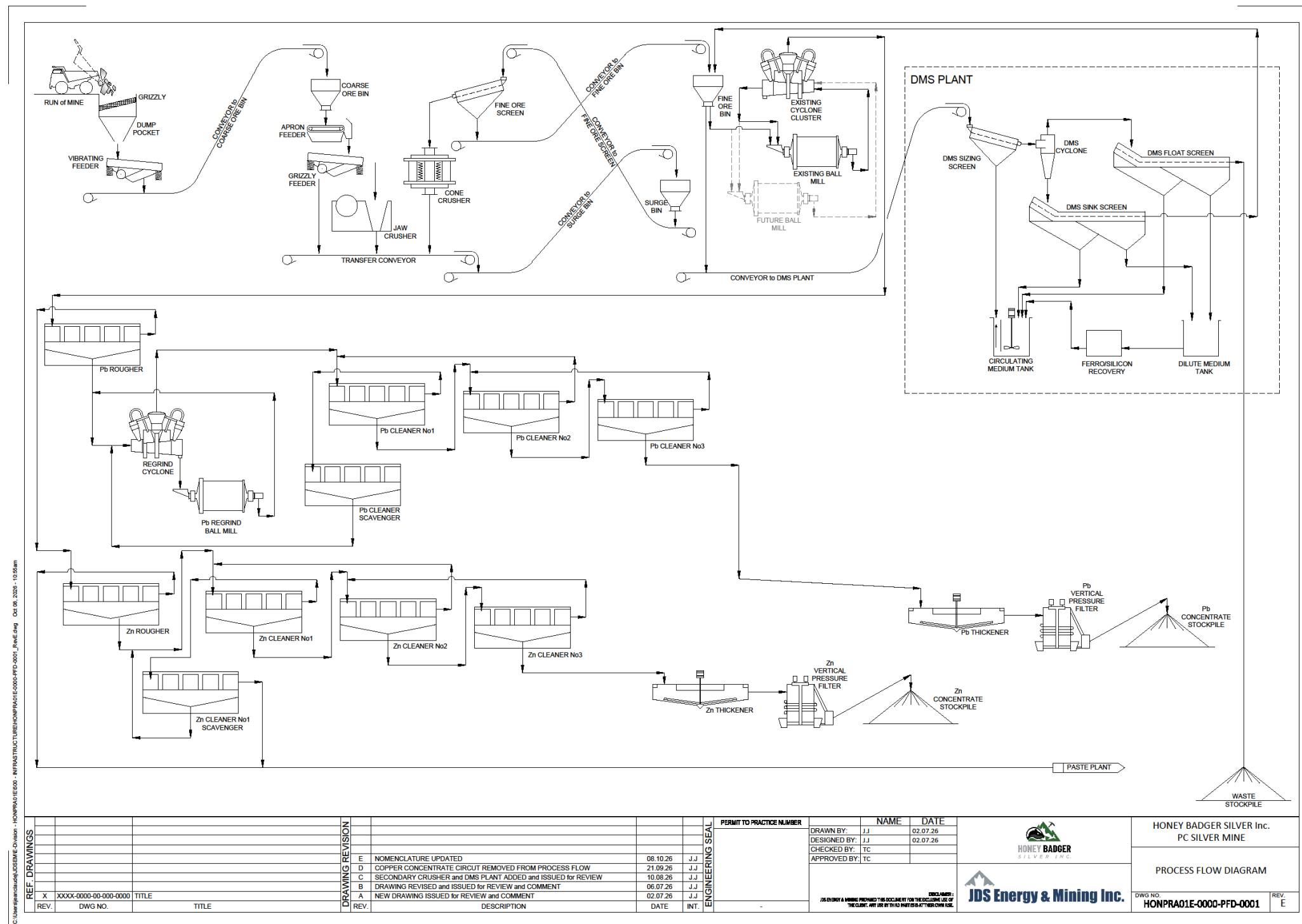
17 PROCESS DESCRIPTION / RECOVERY METHODS

17.1 Plant Design

The current process design incorporates the existing equipment, which was moved from another mine and installed at PC Silver Mine in 1981/1982. With the increase in throughput to 2,400 t/d, the crushing plant is the only area to be retained without modification. The dense media separation pre-concentration plant is new and will be fed using a new conveyor from the existing fine mill feed bin. A new ball mill will be added into the grinding circuit with the existing mill being refurbished. The flotation and dewatering circuits will be refurbished. All the pumps and pump boxes will be purchased new.

The proposed process flow diagram for the PC Silver Mine processing facility is shown in Figure 17-1.

Figure 17-1: Process Flow Diagram



17.2 Process Plant Description

The processing plant consists of crushing, dense media separation pre-concentration, grinding, lead and zinc sequential flotation, concentrate dewatering, and tailings dewatering/paste preparation units.

The process plant design criteria can be found in Table 17-1.

Table 17-1: Process Design Criteria

Criteria	Description	Unit	Design	Source
Plant Throughput		kt/d	2,400	D
		t/a	876,000	D
Crusher Availability		%	70%	D
Crusher Throughput		t/h	143	E
Crusher Selection		Size	24 x 36	D
		Number	1	D
Dense Media Separation Plant Rejection	Mass or Feed Rejected	%	36.5	T
	Silver	%	97.40	T
	Lead	%	97.50	T
	Zinc	%	95.60	T
	Copper	%	97.20	T
	Iron	%	100.00	T
Mill/Flotation Availability		%	92	D
Mill Throughput		t/h	69	E
Physical Characteristics	BWI	kWh/t	13	T
Primary Grind Size	P ₈₀	µm	150	T
Lead Concentrate Grind Size	P ₈₀	µm	40	T
Head Grade (Average)		%Pb	6.14	E
		%Zn	8.52	E
		% Cu	0.25	E
		g/t Ag	112.5	E
Flotation Recovery		%Pb	87.0	T
		%Zn	86.9	T
		% Cu	95	T
		% Ag	93.8	T
Concentrate Grade	Lead	%	60	D
	Zinc	%	58	D

17.2.1 Crushing

The existing refurbished crushing circuits, consisting of a primary crushing unit and a secondary crushing unit in closed circuit with a vibrating screen, will reduce the run-of-mine (ROM) mill feed to a particle size of 80% passing 12 mm.

The major equipment and facilities in this area include:

- ROM mill feed dump pocket, (40 tonnes live capacity) with a fixed grizzly and a vibrating feeder;
- Coarse mill feed surge bin (136 tonnes) with an apron feeder fitted with grizzly bars;
- Kue-Ken 36" x 24" (914 mm by 610 mm) jaw crusher;
- Secondary crushing feed surge bin (45 tonnes) with a belt feeder;
- Double deck screen with apertures of 25 mm and 15 mm;
- Symons Nordberg 5.5' (1.7 m) shorthead cone crusher;
- Conveyors including a metal detector and a magnetic separator;
- Fine mill feed bin (1,800 tonnes) with a reversible belt feeder; and
- Dust collection systems.

17.2.2 Dense Media Separation Plant

The dense media separation plant (new equipment) is designed to reject gangue material to reduce effective feed tonnage and increase feed grades to the downstream grinding and flotation circuits.

The mineralized material is fed to a desliming screen which separates and bypasses material that is too fine to be processed in the dense media separation plant. The screen oversize is then fed into the dense media separator where it will be separated based on density of the rocks.

The dense media separation rejects (the light, float fraction) will be conveyed to a temporary 200 tonne stockpile (uncovered) and will be loaded onto haul trucks by a front-end loader for transport to the waste rock storage facility. The coarse dense media separation sink fraction, will be conveyed to the ball mills in the grinding circuit.

The major equipment and facilities in this area will be located within a new heated building connected to the mill building and will include:

- Desliming screen;

- Condor two-stage DM separator;
- Sepro HB 1.8 x 4.8 m drain and rinse screens;
- Heavy media preparation system;
- Circulating heavy media handling system;
- Dilute heavy medium handling system including a wet magnetic separator; and
- Tanks, pumps, and conveyors.

17.2.3 Grinding

The grinding circuit will consist of two ball mills (one existing, one new) in closed circuit with classifying hydro-cyclones located within the existing mill building.

Major equipment and facilities in this area include:

- Existing refurbished 10' (3.05 m) diameter x 14' (4.27 m) longball mill with a 700 horsepower (522 kW) motor;
- New 10' (3.05 m) diameter x 14' (4.27 m), or similar ball mill with a 700 horsepower (522 kW) motor;
- New classifying hydro-cyclone pack;
- New ball mill discharge pump box;
- New hydro-cyclone feed pumps;
- Existing and new ball mill feed conveyors; and
- Ancillary equipment including a steel ball storage bin and a ball bucket.

17.2.4 Flotation / Leaching

Polymetallic, sequential flotation will be employed to separate lead and zinc sulphide minerals into concentrates. All flotation cells in both circuits will be new, and a new ball mill will be installed to regrind lead rougher concentrate to maximize grade of the final lead concentrate.

17.2.4.1 Lead Flotation

The lead flotation circuit consists of rougher, regrind and three stages of cleaner flotation; the major equipment for lead flotation circuit includes:

- One existing rougher conditioning tank, equipped with a mechanical agitator;
- Existing lead rougher flotation cells;
- One existing regrind ball mill, 6.5' (2 m) diameter x 5' (1.5 m) long mill and cyclones;
- Existing refurbished cleaner conditioning tanks, equipped with mechanical agitators;
- Existing primary cleaner flotation cells;
- Existing scavenger flotation cells;
- Existing secondary cleaner flotation cells;
- Existing tertiary cleaner flotation cells; and
- Ancillary equipment including pumps, pump boxes and sump pumps.

17.2.4.2 Zinc Flotation

The zinc flotation circuit consists of rougher flotation followed by three stages of cleaner flotation; major equipment includes:

- Existing rougher conditioning tanks, each equipped with mechanical agitators;
- Existing rougher flotation cells;
- Existing cleaner conditioning tank, equipped with mechanical agitator;
- Existing primary cleaner flotation cells;
- Existing scavenger flotation cells;
- Existing secondary cleaner flotation cells;
- Existing tertiary cleaner flotation cells; and
- Ancillary equipment including pumps, pump boxes and sump pumps.

17.2.5 Reagents and Compressed Air

Various chemical reagents will be added to the flotation circuits to facilitate lead, zinc, and silver recovery. Specific reagent requirements for the PC Silver Mine processes have been identified, along with packaging and estimated dosages.

A typical preparation unit of a solid reagent will include:

- Bulk handling system;
- Mixing tank, for mixing reagent with fresh water to required strength;
- Holding tank; and
- Reagent pumps.

Liquid reagents will be diluted prior to delivery to the flotation circuits or pumped directly to the flotation circuits without dilution. The existing reagent preparation area will be refurbished and utilized. Storage tanks will be equipped with level indicators and instrumentation to minimize spills. Appropriate ventilation, fire and safety protection, and Material Safety Data Sheet (MSDS) stations will be provided at the facility. Each reagent line and addition point will be labelled in accordance with Workplace Hazardous Materials Information Systems (WHMIS) standards. All operation personnel will receive WHMIS training, along with additional training for the safe handling and use of reagents.

Storage of bulk reagents will be located inside the mill building.

The two existing plant air compressors (duty/standby) will be refurbished to provide high-pressure air for general plant use, pressure filters and instrumentation. The instrumentation air stream will be dried, and the dry compressed air will be stored separately in a new dedicated air receiver.

The Paste Plant will have its own dedicated air system.

The two existing blowers will be refurbished and supply air for all flotation cells.

17.2.6 Concentrates

The final concentrates will each be dewatered by thickening and pressure filtration. The filtered lead and zinc concentrates will be stored on separate stockpiles prior to being shipped offsite. Both concentrates will be thickened in the existing thickeners and will be filtered using the existing (refurbished) Larox pressure filter.

17.2.6.1 Lead Concentrate Dewatering

Lead concentrate will be thickened in the existing 10.7 m diameter thickener. Underflow will be pumped using a new pump to the existing lead concentrate surge tank (equipped with a mechanical agitator) at approximately 65% solids.

The flocculant addition rate will be adjusted, based on the lead concentrate thickener overflow clarity and underflow density. The lead concentrate surge tank will be capable of holding the thickened concentrate for approximately four (3) hours to offset any minor maintenance required for the filter and load-out system. Any additional storage capacity required for filter maintenance will be achieved by holding concentrate in the thickener. The lead concentrate thickener overflow will be pumped back to required circuits as process water.

The thickened concentrate will be further dewatered to a moisture level of 8% using the existing refurbished Larox pressure filter. Filtrate will be returned to the lead concentrate thickener and concentrate filter cake conveyed to the lead concentrate stockpile.

The concentrate storage building will provide temporary storage of the lead and zinc concentrate products in their own dedicated stockpiles. The new building will include concrete foundations, concrete slab, HVAC, dust control system and a vacuum system to clean up any concentrate spills from the container before removal from the building.

The major equipment used in the lead concentrate dewatering circuit includes:

- Existing 10.7 m diameter lead concentrate thickener;
- Existing refurbished lead concentrate surge tank equipped with a mechanical agitator;
- Existing refurbished Larox pressure filter;
- Existing refurbished concentrate filter discharge conveyors; and
- Ancillary equipment including pump boxes and pumps.

17.2.6.2 Zinc Concentrate Dewatering

Zinc concentrate will be thickened in the existing 10.7 m diameter thickener. Underflow will be pumped using a new pump to the existing zinc concentrate surge tank (equipped with a mechanical agitator) at approximately 65% solids. The flocculant addition rate will be adjusted, based on the thickener overflow clarity and underflow density.

The zinc concentrate surge tank will be capable of holding the thickened concentrate for approximately three (3) hours to offset any minor maintenance required for the filter and load-out system. Similar to the lead concentrate system, additional zinc concentrate can be stored in the thickener to conduct major zinc filter and load-out system maintenance activity.

The thickened concentrate will be further dewatered to a moisture level of 8% using a new Larox (or similar) pressure filter. Filtrate will return to the zinc concentrate thickener, while concentrate filter cake is conveyed to a dedicated zinc concentrate stockpile (adjacent to the lead concentrate stockpile).

The major equipment used in the zinc concentrate dewatering circuit includes:

- Existing 10.7 m diameter zinc concentrate thickener;

- Existing refurbished zinc concentrate surge tank equipped with a mechanical agitator;
- Existing Larox pressure filter;
- Existing refurbished concentrate filter discharge conveyors; and
- Ancillary equipment including pump boxes and pumps.

17.2.7 Tailings

The final tailings from the zinc flotation circuit will be pumped to the paste plant thickener and then to the backfill plant to produce paste for backfilling underground slopes.

There will be a new paste plant and paste delivery system as discussed in Section 16.8.

17.2.8 Water Management

Water management is described in Section 18.6. Fresh water will be supplied the mine. Fresh water will be supplied from the water treatment plant and used primarily for:

- Fire water for emergency use, and
- Gland services for the slurry pumps (only in rare exceptions when contact water can't be used).

Process water will come from the concentrate thickener overflows and from the water storage pond cell A. It will be pumped back to the respective flotation circuits and re-used. The paste thickener overflow and excess water from the paste plant will be pumped to Cell A of the water storage pond to be re-used after the flotation reagents are allowed to degrade for approximately two months. No treatment of the process water is required. Cell A water will also be used for gland services and reagent make-up.

Process water is supplied throughout the plant via a ring main process water pipe arrangement.

17.2.9 Process Plant Personnel

Table 17-2: Process Plant Personnel

Position	Staff/Hourly	Quantity
Mill Operations		
Mill Operations Manager	Staff	1
Mill Operations Superintendent	Staff	2

Position	Staff/Hourly	Quantity
Admin Assistant	Staff	1
Shift Foremen	Staff	4
Control Room Operator	Hourly	4
Crusher Operator	Hourly	4
Grinding Operator	Hourly	4
Flotation Operator	Hourly	4
Dense Media Separation Operator	Hourly	4
Filter Operator	Hourly	4
Mill Labourer	Hourly	8
Mill Maintenance		
Maintenance Superintendent	Staff	2
Maintenance Foreman	Staff	2
Maintenance Planner	Staff	2
Electrician	Hourly	4
Millwrights	Hourly	4
Instrumentation	Hourly	2
Maintenance Apprentice	Hourly	4
Mill Technical		
Chief Metallurgist	Staff	1
Senior Metallurgist	Staff	2
Metallurgist	Staff	4
Chief Assayer	Staff	1
Senior Assayer	Staff	2
Assay Technician (Fire Assay)	Hourly	4
Assay Technician	Hourly	8
Total Process Plant Personnel		82

18 PROJECT INFRASTRUCTURE AND SERVICES

18.1 General Site Arrangement

The Project infrastructure is designed to support continuous mining and processing operations on a 24-hour-per-day, seven-day-per-week basis. The process plant and site-wide infrastructure will be located within the main project area, with additional mining and support facilities established at the 883 Portal to support operations underground.

The infrastructure has been developed to provide a practical and cost-effective solution for the planned production rate of 2,400 t/d, while accounting for the remote location, seasonal constraints, site topography, climatic conditions, and the need to transport personnel, equipment, consumables, and fuel throughout the property.

The overall site layout identifies the proposed locations of the mine portal, process plant, tailings management facilities, paste backfill plant, waste rock storage facilities, accommodations complex, and supporting infrastructure. The PC Silver Mine area will include the mining facilities and services required to support underground operations, equipment, personnel, water management, and material movement. See Figure 18-1 and Figure 18-2 for general site layout drawings.

The infrastructure considered for the Property includes:

- The utilization of the ASR;
- Site access roads, haul roads, and other on-site roads;
- Liard River crossing (winter ice bridge and summer barge) and associated seasonal transportation infrastructure;
- Airstrip and aviation support facilities;
- Power generation, transmission, and site electrical distribution;
- Accommodations complex and supporting camp facilities;
- Administration offices and mine dry facilities;
- Mine, process plant, and mobile equipment maintenance facilities;
- Mining support infrastructure at underground portal;
- Warehousing and laydown areas;
- Bulk fuel, lubricant, and reagent storage facilities;
- Explosives storage and handling facilities;

- Laboratory and sample preparation facilities;
- Freshwater supply, potable water treatment, wastewater treatment, and firewater systems;
- Site-wide water collection, diversion, treatment, and storage facilities;
- Waste rock storage facilities;
- Tailings management facilities;
- Emergency response, medical, and training facilities;
- Solid waste management, landfill / incinerator facilities; and
- Information technology and communications infrastructure.

PLAN

NOTES:
1. ALL DIMENSIONS IN MILLIMETRES, ELEVATIONS IN METRES

PRELIMINARY
NOT TO BE USED FOR CONSTRUCTION

1:5,000

NO.	DATE	BY	DESCRIPTION
D	2025-06-17	RR	ISSUED FOR INFORMATION
C	2025-06-19	RR	ISSUED FOR INFORMATION
B	2025-06-12	RR	ISSUED FOR APPROVAL
A	2025-07-29	RR	ISSUED FOR REVIEW

PROJECT NO.	26062
DISCIPLINE	CIVIL
SCALE	1:5000
DRAWN BY	R. KULAR
DESIGNED BY	R. KULAR
CHECKED BY	C. PATEL
APPROVED BY	C. PATEL
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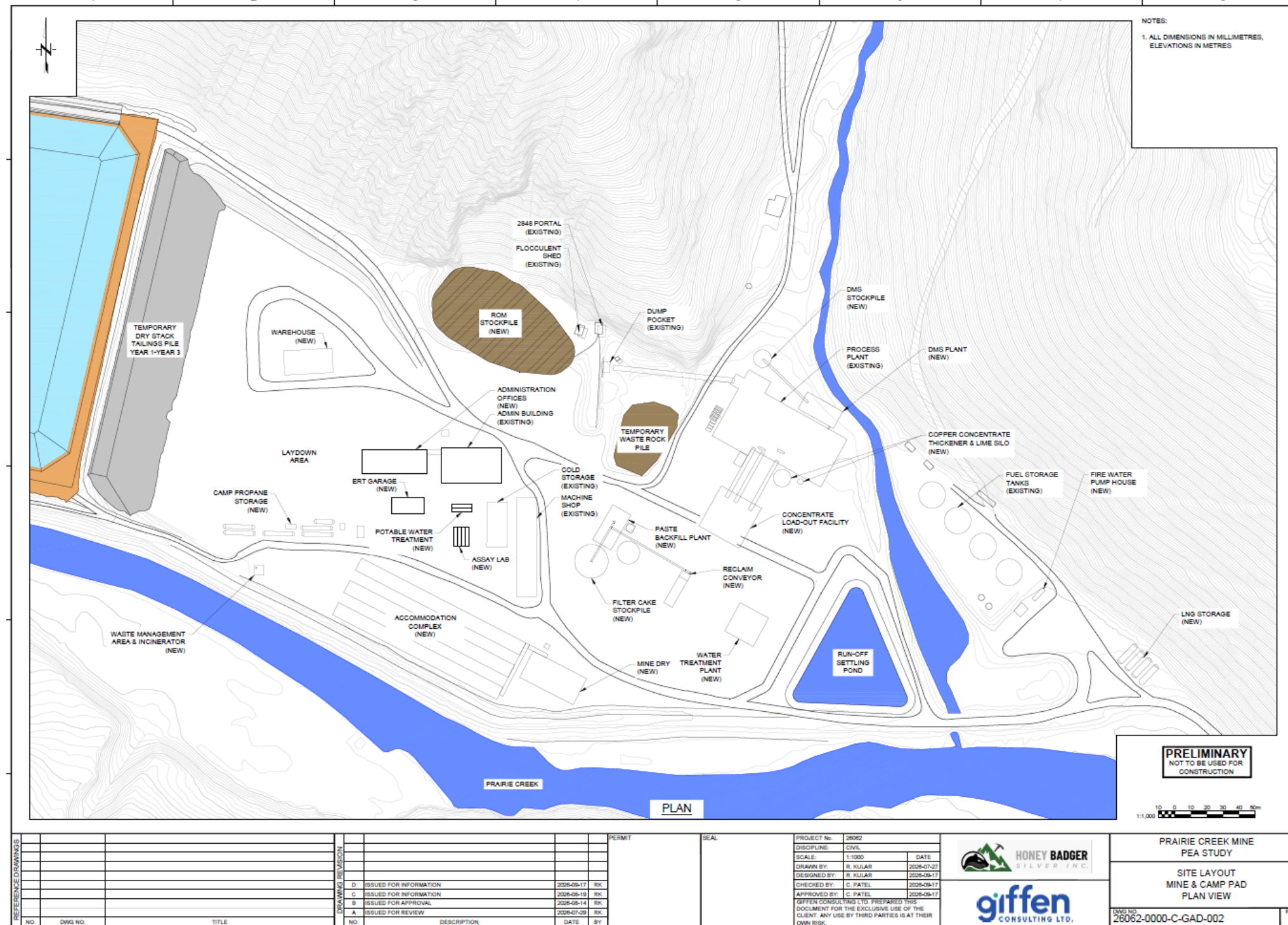
PRAIRIE CREEK MINE PEA STUDY

MINE PAD & ACCESS ROAD PLAN VIEW

DWG NO. 26062-0000-C-GAD-001

PAGE 18-3

Figure 18-2: Preliminary Site General Layout Arrangement – Extended



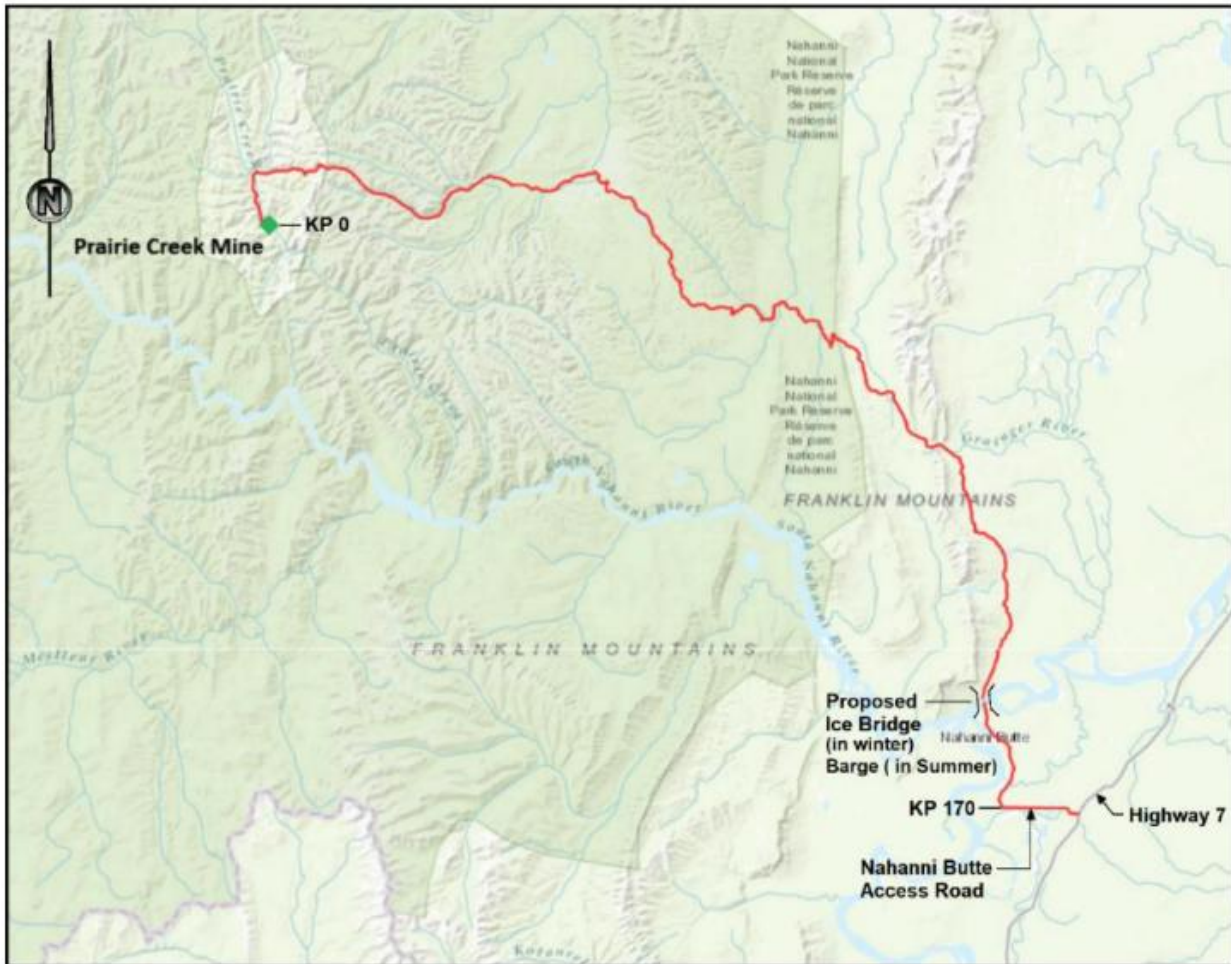
Source: Giffen (2026)

18.2 Site Access

18.2.1 Access Road to Site

The ASR is a critical component of infrastructure that will be constructed over three years to provide main access to the Project. The road will be comprised of approximately 170 km of roadway that connects the PC Silver Mine to the Liard Trail / Northwest Territories Highway 7 corridor. Construction efforts will be driven by remote location and camp-based operations. Construction operations will be planned to accommodate seasonal access constraints such as freeze-up and breakup. The road access route includes the Liard River crossing (barge river crossing in open-water and ice bridge in winter) and extensive watercourse crossings including bridges, arches, and major culverts. Construction materials and earthworks will consist of limited sources of fine-grained soils and majority utilization of borrow pit sources located along the access road (see Figure 18-3).

Figure 18-3: All-Season Access Road into PC Silver Mine



Seasonal interruption-based winter road access work will be conducted to open access to the mine-site prior to the ASR construction being completed. The winter road will include temporary measures such as rig mats, snow fills, and portable bridges that will be removed prior to spring run-off. Crews will conduct ice drilling, water pumping, area flooding, and subsequent testing to achieve sufficient cover on the road surfaces.

The road access includes a seasonal crossing of the Liard River, with barge access anticipated during the open-water season. Barge landings will be constructed on both sides of the river crossing to accommodate barge activities.

18.2.2 On Site Roads

The PC Silver Mine is a small, already-established site, and on-site road requirements are limited primarily to short-haul routes supporting underground operations. Existing roads will be utilized and upgraded where practical, with new road construction undertaken only where required.

Mineralized material will be trucked from the 883 Portal to the ROM Stockpile and Dump Pocket, with trucks returning underground to continue haulage. Waste rock from the underground mine and dense media separation rejects from the mill will be hauled to the Waste Rock Pile. Additional short on-site hauls will connect the process plant and stockpile areas to the Dry Stack Tailings Pile and other site facilities.

18.2.3 Airstrip

The existing 1,000 m gravel airstrip located approximately 1 km from the camp will be used and is registered with NAV CANADA as CBH4. The airstrip will serve as the primary means of transporting mine personnel to and from the site and will also support priority freight, emergency response, and medical evacuation requirements. It is located beside PC Silver Mine at the bottom of a narrow valley with obstructed approaches in either direction. Passenger aircraft up to a DHC-7 size can use the strip. The site does not allow for a sufficient runway extension to accommodate any larger and more economical freight aircraft, such as Hercules.

18.3 Power

A centralized on-site power plant will be developed to meet the Project's electrical generation requirements. The main power plant will be next to the process plant and will supply power to the crushing and material-handling systems, process facilities, mine infrastructure, underground mine activities, accommodations complex, administration buildings, maintenance facilities, water management systems, and other ancillary loads. The current onsite load capacity is estimated at approximately 1 MW.

The PEA assumes that the legacy power generation units in the process facility, comprising four Cooper Bessemer LS-8-GDT generators, will need to be fully disassembled and removed from site. Further studies are recommended to determine if these units can be economically restarted.

The power plant will initially be established using eight rented 1,500 kW diesel generators operating at 480V, with the same configuration and specifications carried through to permanent ownership. After two years of rental operation, a dual fuel (diesel/LNG) genset setup will be purchased outright as the site's permanent power solution, retaining rental units as backup support during the transition.

The generators are configured in pairs, with each pair connected to a transformer that steps voltage up to the 4,160V site distribution voltage. Transformer outputs feed a common medium-voltage switchgear lineup, which connects to a secondary switchgear containing the five feeders required for underground operations, the mill, the camp, and two ancillary loads. This configuration provides approximately 11,640 kW standby capacity, 10,400 kW prime capacity, and 9,048 kW continuous capacity.

The package also includes grounding transformers and neutral grounding resistors, auxiliary power distribution for generator support loads, a site office trailer, an oil storage container, a parts storage container, cable, and cable tray.

After the second year, generators are expected to operate on diesel, with LNG substitution or dual-fuel capability subject to fuel availability due to freeze-up and break-up conditions at the Liard River crossing. Site distribution at 4,160V will be performed through medium voltage pole lines, with local substations stepping down power for low voltage supply.

18.4 Fuel Storage and Distribution

18.4.1 Diesel Fuel Storage

Fuel storage will be sized to support approximately 12 weeks of diesel consumption at full production, including fuel requirements for mining equipment, ancillary vehicles, and on-site power generation. This storage capacity is intended to maintain operations during periods when fuel deliveries are interrupted by freeze-up and break-up conditions at the Liard River crossing.

Diesel fuel will be stored in the existing diesel storage facilities located east of the process plant. There are four 1.7-million-liter tanks with a total volume capacity of 6.8 million liters. A fueling station adjacent to the tank farm will be operational and easily accessible by roadway from the mine site.

18.4.2 Liquefied Natural Gas (LNG) and Propane

The LNG storage facility will be located on the eastern side of the site and will be connected and piped to the power plant. The LNG storage facility will include 4 tanks that are readily accessible by road. Camp propane storage will be located in the laydown area, near the ERT garage and warehouse.

18.5 Ancillary Facilities

18.5.1 Accommodations Complex

The existing workforce camp accommodation at the PC Silver Mine consists of fifteen single-story modules. These units will need to be replaced due to deterioration.

The new camp will consist of skid-mounted, hard-walled, prefabricated buildings designed for year-round operation in the Northwest Territories climate. The facilities will include appropriate insulation, heating, ventilation, fire protection, and utility systems suitable for cold-weather conditions. There will be several dormitories, each organized with bedrooms blocks for a total of 250 fully furnished rooms.

The construction/permanent camp and administration office complex will include the following:

- Dormitory units;
- Kitchen and food storage;
- Dining room;
- Arrivals/departure building including reception and first aid;
- Recreation facilities and gymnasium;
- Utility rooms (mechanical, electrical, domestic potable/hot water, fire protection);
- Laundry;
- IT/server room; and
- Wastewater treatment plant.

18.5.2 Administration Buildings

The new administration and mining offices will be combined into one single module, connected to the existing administration building. The administration building will be a modular prefabricated structure shipped to site for final assembly and will be connected to the water and sewage systems.

18.5.3 Mine Dry

The mine dry will be a modular prefabricated structure shipped to site for final assembly and will be connected to water and sewage systems. This building will be connected to the new accommodation complex.

18.5.4 Mine Maintenance Facility

The existing warehouse and mechanics shop are located north of the camp facilities. These existing facilities require minimal rehabilitation. There will be a new warehouse facility and laydown area built north-west of the camp accommodation complex to support the Projects' storage and laydown requirements.

18.5.5 Laboratory

The onsite Laboratory will consist of prefabricated modules and ancillary equipment, such as drying ovens, dust and fume control, and heating equipment. It will be utilized for various monitoring and testing activities.

18.5.6 Additional Facilities

Additional planned ancillary facilities not discussed previously include, but are not limited to:

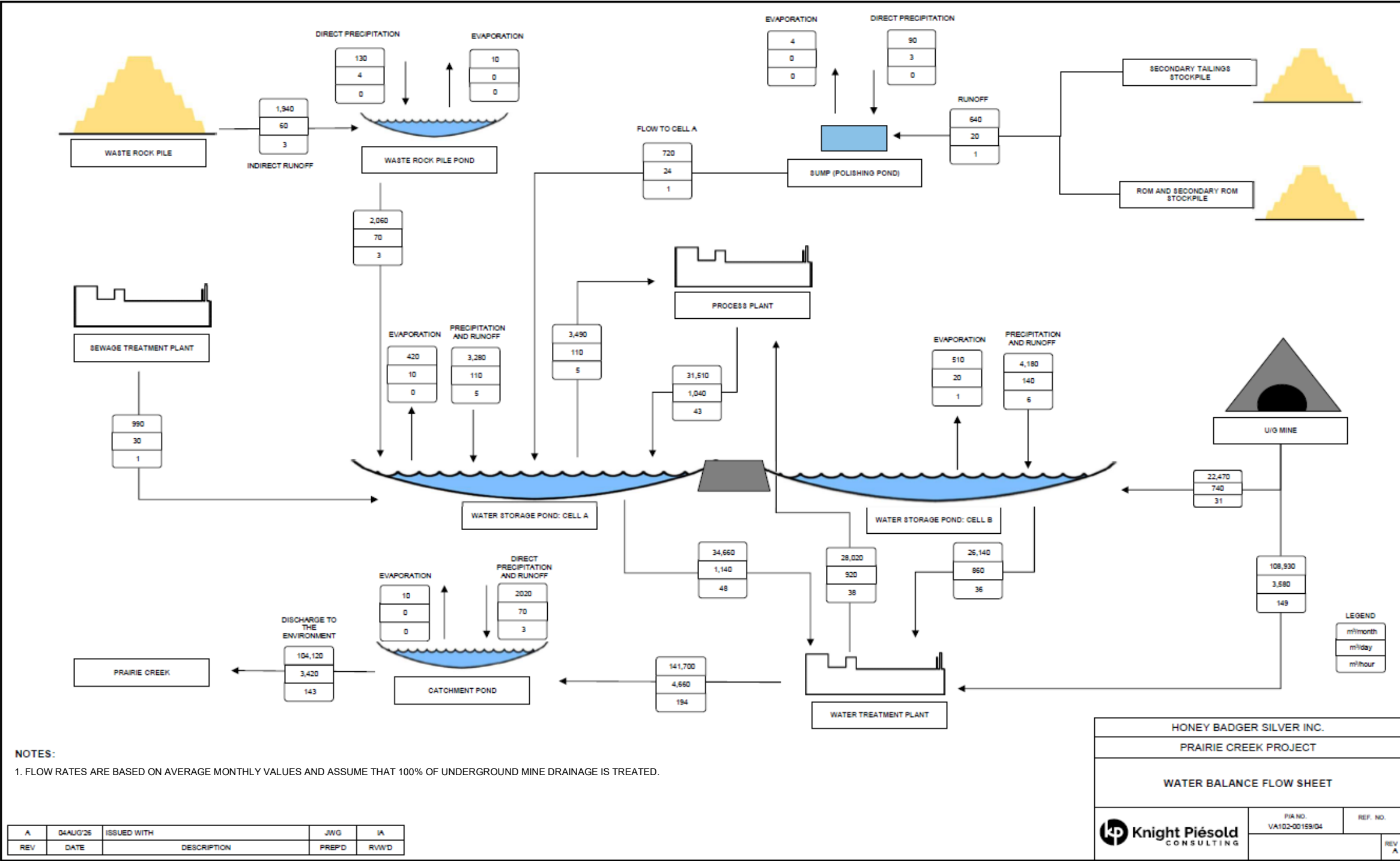
- Emergency Response Team garage;
- Fire Water Pump House;
- Reagent & Site Storage;
- Waste Management Area / Incinerator;
- Underground explosives and detonator cap magazines storage;
- Cold Storage (existing);
- Flocculent Shed (existing); and
- Dump pocket (existing).

18.6 Water

The site-wide water balance flow sheet is shown on Figure 18-4. The primary inflow to the system is groundwater pumped from underground dewatering. Additional inflows include precipitation and runoff, waste rock seepage, and treated sewage effluent, though these represent a small fraction of the total inflows. The flow rates displayed are based on the monthly average.

Water is primarily stored in the water storage pond, with the catchment pond and the waste rock pile pond providing smaller storage capacities. The catchment pond will be equipped with an emergency back-pump in the event that water quality does not meet discharge requirements, and the waste rock pile pond will include an emergency overflow spillway.

Figure 18-4: Site Wide Water Balance Flow Sheet

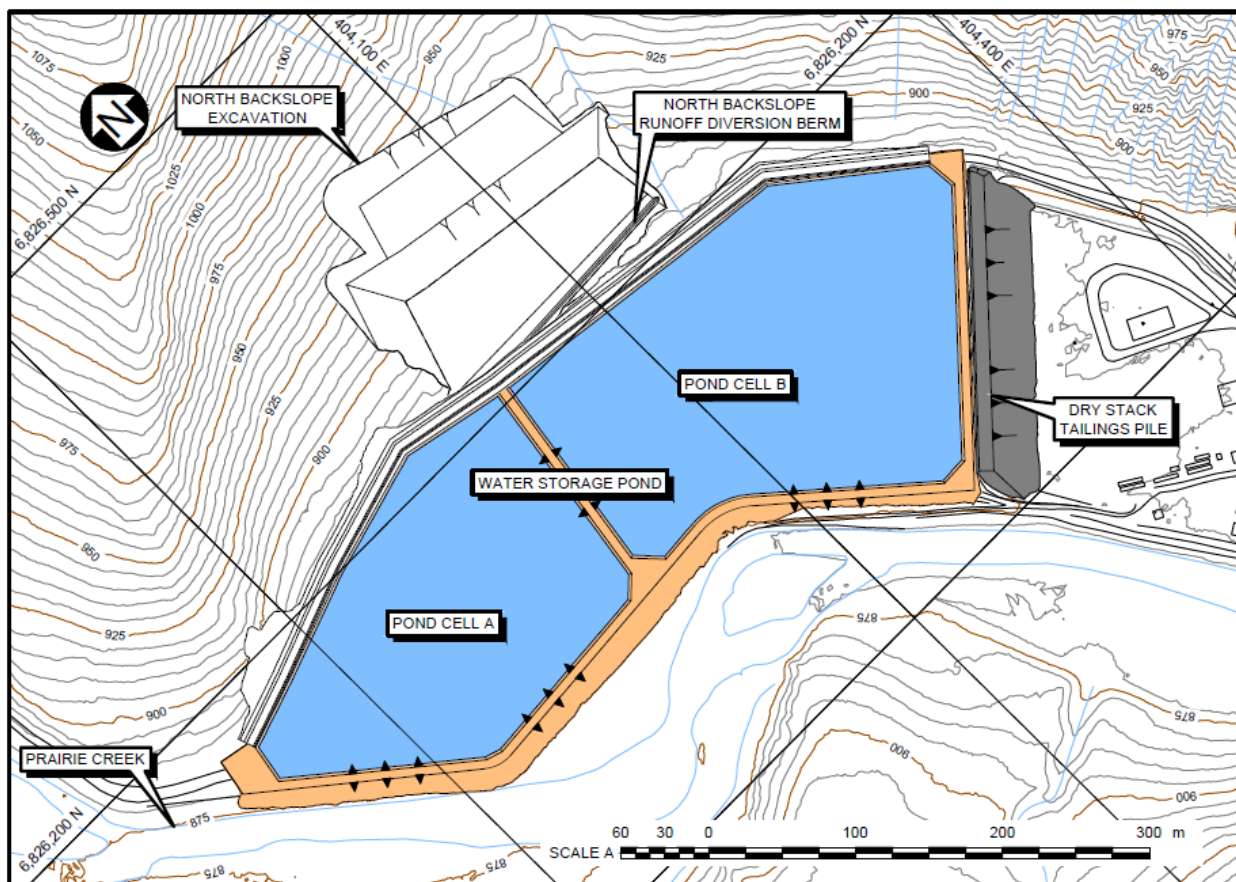


Source: KP (2026)

The water storage pond will consist of two lined cells as shown on Figure 18-5. Cell A, with an approximate total volume of 300,000 m³, collects process water from the mill, contact mine drainage, and seepage or runoff collected from the waste rock pile and stockpiles. Process water is held in Cell A for several months to allow flotation reagents to degrade naturally without chemical treatment. Cell A serves as the primary water source for the mill, operating as a closed loop with no discharge to the environment.

Cell B, with an approximate total volume of 400,000 m³, collects clean, non-contact mine groundwater intercepted up-gradient of the orebody through the mine's pre-drainage program, as well as treated sewage effluent and clean site drainage. Water from Cell B supplies fresh water for mine utilities, emergency fire suppression, and gland services. Excess non-contact water in Cell B is routed to the on-site Water Treatment Plant for polishing and controlled discharge to PC Silver Mine via an underwater exfiltration. The ponds are required to have a minimum freeboard of 1 m.

Figure 18-5: Water Storage Pond and Temporary Tailings Storage Facility



Source: KP (2026)

18.6.1 Water Supply

Fresh water for domestic and potable use will be sourced from a groundwater well in the Prairie Creek Valley Aquifer. Process water will be sourced primarily from recycled mill effluent stored in Cell A of the water storage pond, with make-up transferred from Cell B (non-contact groundwater).

18.6.2 Water Treatment

All non-contact water (primarily from the underground mine) will be treated as a contingency prior to discharge. Excess Cell A and Cell B water will be treated at the water treatment plant prior to discharge to Prairie Creek via the exfiltration trench. Discharge rates will vary depending on real-time flows in Prairie Creek to meet the minimum 19:1 creek-to-effluent dilution ratio. The plant has an initial capacity of 75 L/s and is modular to allow expansion. Assuming all water from the underground mine pumped at a rate of 50 L/s is required to be treated, along with assuming the most conservative Prairie Creek flow rates, there is an annual surplus of 345,000 m³ of water that is required to be treated. Increasing the water treatment capacity by a minimum of 22 L/s would allow for the excess water to be treated and discharged over the summer months.

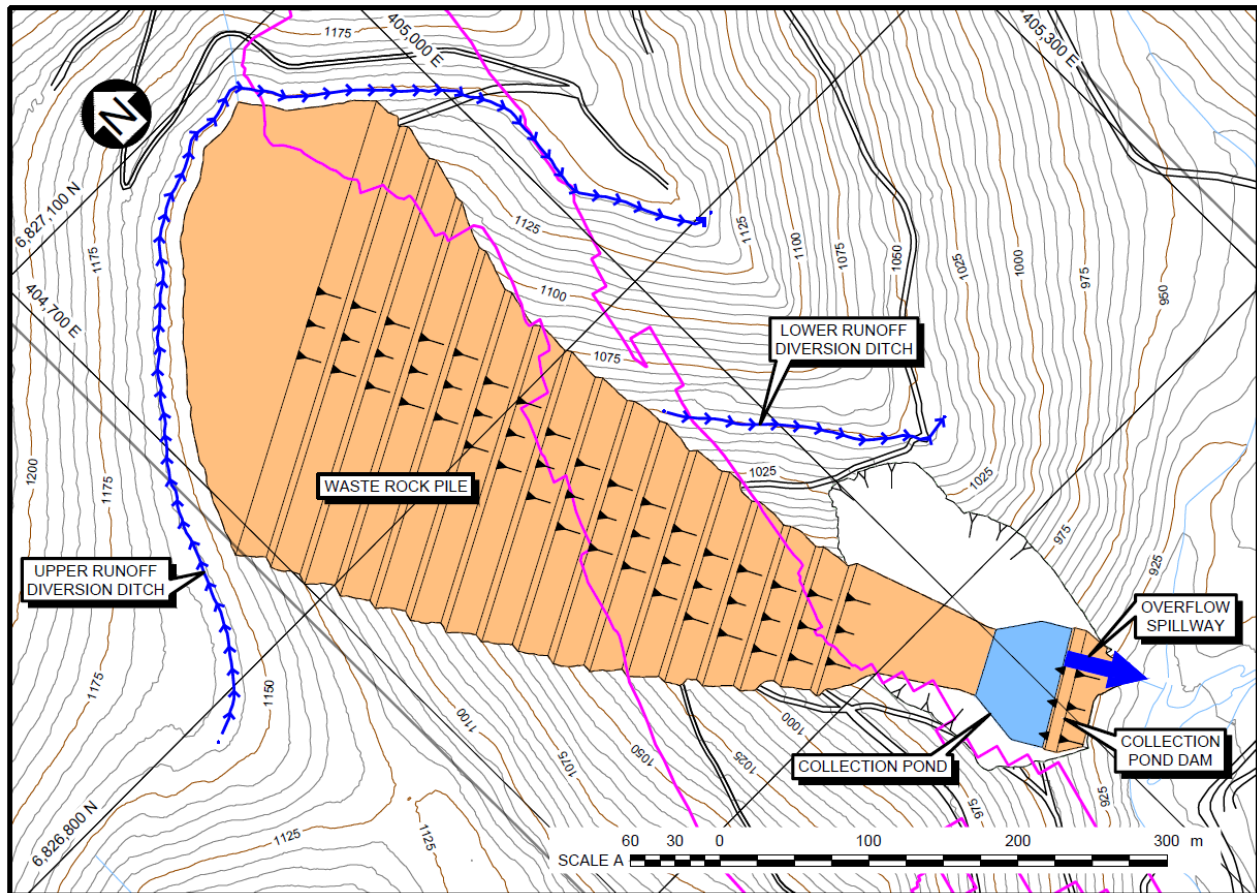
18.7 Waste Rock Management

The waste rock pile will store approximately 5 Mt of combined development waste rock and dense media separation rejects, along with 35,000 m³ of non-hazardous solid waste (wood, metal, paper, etc.). The waste rock pile will be constructed in a ravine approximately 1 km north of the plant site, as shown on Figure 18-1. The capacity is variable, as the footprint can migrate upslope within the ravine as additional waste is placed (see Figure 18-6).

The waste rock pile begins at a toe elevation of approximately 937 m and is planned to proceed upslope with an overall external slope gradient of 2:1, reaching an elevation of approximately 1,145 m. Rock will be placed from the bottom up in stable 6 to 7 m high benches, with benches along the exterior slopes acting as catchments to intercept surface runoff, which drains to Harrison Creek.

The waste rock pile is integrated into the site's water management plan. A geomembrane-lined seepage collection pond with a capacity of approximately 16,000 m³ is situated below the toe of the pile to capture seepage and runoff, with collected contact water piped to the water storage pond for recycling or treatment. Lined diversion channels run along the east and west flanks of the pile to divert non-contact upslope runoff safely into Harrison Creek.

Figure 18-6: Waste Rock Pile



Source: KP (2026)

18.8 Temporary Tailings Management Facility

Filtered tailings will be temporarily stored in a stockpile to the south of water storage pond Cell B, as shown on Figure 18-5, prior to being backfilled to the underground. The maximum tailings volume that needs to be stored at any given point during operations is approximately 35,000 m³. The tailings stockpile was designed for the following specifications:

- Height of 10 m, up to a maximum elevation of 881 m;
- Top width of 10 m; and
- Outer slopes of 3 horizontal to 1 vertical (3H:1V).



The stockpile will be founded on an existing sand and gravel pad that will be graded and lined with a geomembrane liner system. The liner will be covered by a crushed granular material to promote drainage towards the sump at the southeast corner.

19 MARKET STUDIES AND CONTRACTS

19.1 Market Studies

The Company engaged Howard Okamura, an experienced concentrate marketing consultant, to undertake a market assessment and develop a marketing strategy for the zinc and lead concentrates anticipated to be produced at the PC Silver Mine.

Mr. Okamura has over 30 years of experience in the global base metals industry, including experience with Teck Resources and extensive involvement in concentrate marketing, sales and offtake arrangements. His experience encompasses both the producer and purchaser sides of concentrate transactions, providing an understanding of international concentrate markets, commercial negotiations and smelter procurement requirements.

As part of the PEA process, Mr. Okamura prepared a detailed marketing report, which was provided to JDS for consideration in its preparation of the Technical Report.

The marketing assessment addresses the potential commercialization of the zinc and lead concentrates produced at the PC Silver Mine, with particular emphasis on opportunities in the major global concentrate markets of Asia, Europe and North America.

19.1.1 Marketing Strategy

The marketing strategy developed for the PC Silver Mine contemplates the sale of zinc and lead concentrates to international smelters and other potential concentrate purchasers.

The assessment considers opportunities across multiple geographic markets, providing the Company with a framework for evaluating potential customers and developing sales arrangements as the Project advances toward production.

The Company's marketing approach is intended to provide flexibility in the allocation of concentrate production among potential purchasers, taking into consideration concentrate specifications, commercial terms, transportation requirements and regional market conditions.

The marketing strategy will be refined as discussions with prospective customers advance.

The end-products of the Project are a silver-rich lead concentrate containing copper components and a zinc concentrate. The terms are shown in Table 19-1 and Table 19-2.

Note that smelter terms may vary significantly over time based on the supply and demand of particular concentrates.

The supply of both lead and zinc concentrates in the market have been underperforming the demand for the concentrates over recent years. Therefore, treatment and refining charges, which a smelter charges the mine for the processing of the concentrates have been at historically low levels, which indicates a deficit market. This deficit market is forecast for the future when PC

Silver Mine will be coming into production which will provide strong support for sales of the concentrate.

Table 19-1: Lead Concentrate Smelter Terms

Lead Concentrate	Unit	Value
Lead Recovery	%	87.0
Silver Recovery	%	87.9
Copper Recovery	%	95.0
Lead Concentrate Grade		Est. LOM Average
Lead	%	60
Silver	g/t	1,110
Copper	%	2.7
Antimony	g/t	10,142
Arsenic	g/t	5,658
Cadmium	g/t	426
Mercury	g/t	458
Moisture Content	%	8.0
Smelter Payables		
Lead payable	%	95
Min. lead deduction	%	3.0
Silver Payable	%	95.0
Min. Silver deduction	Ag g/dmt	50.0
Copper payable	%	30
Treatment & Refining Costs		
Pb concentrate TC	US\$/dmt conc	100.00
Ag RC	US\$/payable oz	0.85
Mercury Penalty Threshold	g/dmt	200
Mercury Penalty	\$/dmt for each 1g/dmt above threshold	0.30
Calculated Penalties	US\$/dmt conc	7.74
Transport Costs	US\$/wmt conc	271.36

Table 19-2: Zinc Concentrate Smelter Terms

Zinc Concentrate	Unit	Value
Zinc Recovery	%	86.9
Zinc Concentrate Grade		Est. LOM Average
Zinc	%	58
Cadmium	g/t	3,229
Mercury	g/t	1,604
Moisture Content	%	8.0
Smelter Payables		
Zinc payable	%	85
Min. zinc deduction	%	8.0
Treatment & Refining Costs		
Zn concentrate TC	US\$/dmt conc	185.00
Cadmium Penalty Threshold	g/dmt	3,000
Cadmium Penalty	\$/dmt for each 1g/dmt above threshold	0.0015
Mercury Penalty Threshold 1	g/dmt	300
Mercury Penalty 1	\$/dmt for each 1g/dmt above threshold 1 up to threshold 2	0.0225
Mercury Penalty Threshold 2	g/dmt	800
Mercury Penalty 2	\$/dmt for each 1g/dmt above threshold 2 up to threshold 3	0.0425
Mercury Penalty Threshold 3	g/dmt	1,200
Mercury Penalty 3	\$/dmt for each 1g/dmt above threshold 3	0.0725
Calculated Penalties	US\$/dmt conc	59.71
Transport Costs	US\$/wmt conc	244.96

19.2 Concentrate Quality

19.2.1 Lead Concentrate

The PC Silver Mine lead concentrate is an attractive concentrate for processing by smelters, especially in China, which is the largest market. The significant levels of by-product metals such as silver, copper and zinc make the concentrate beneficial for processing by a lead smelter.

The lead concentrate is expected to contain elevated levels of Antimony, and Mercury. At this time, the Antimony levels are assumed to be considered neutral (i.e., not high enough to be payable, but not low enough to be considered deleterious) but there is a possibility that specific concentrates produced may not be considered neutral. Levels of Antimony will need to be monitored. Average Mercury levels are expected to exceed smelter thresholds and will incur penalties.

19.2.2 Zinc Concentrate

The forecast PC Silver Mine zinc concentrate contains a significant amount of mercury but otherwise is an attractive concentrate for processing by smelters because of its high zinc grade, low iron and low silica content. PC Silver Mine is not unique for producing a zinc concentrate containing significant levels of mercury. For this reason, most zinc smelters have capability to remove mercury but will charge tiered penalties when the concentrate exceeds certain thresholds.

19.3 Contracts

Although there are no smelting, transport, mining or other significant contracts in place as of the date of this report, Honey Badger Silver has been in conversations with smelters and metals traders. Letters of Intent and Memorandum of Understanding have been signed with multiple parties showing support for the Project. Contracts for the sales and purchase of the concentrates will be entered as the Project advances. The primary market for zinc concentrates will be Canada and Europe while the primary market for lead concentrate will be China.

19.4 Royalties

The Property is subject to the following royalties:

- A 1.2% NSR royalty payable to Sandstorm Gold on the Project. Sandstorm Gold and Equinox Gold created a company called Sandbox Royalites in 2022 (Sandbox Royalties, 2022). In 2024, Sandbox Royalties entered into an agreement with B2Gold and changed their name to Versamet Royalties (Versamet Royalties, 2024); and
- A 1% NSR royalty payable to RCF VI CAD LLC.

19.5 Metal Prices

The base case metal prices selected for this study were based on a combination of long-term consensus projections, current prices, three-year trailing averages and timing of production.

The assumed metal prices and exchange rate are shown in Table 19-3.

Table 19-3: Assumed Metal Pricing and Exchange Rate

Parameter	Unit	Value
Silver (Ag) Price	US\$/oz	50.00
Lead (Pb) Price	US\$/lb	0.90
Zinc (Zn) Price	US\$/lb	1.50
Copper (Cu) Price	US\$/lb	5.00
FX Rate	CAD\$:US\$	0.72

For reference, historical silver, lead, zinc and copper prices are shown in Figure 19-1 through Figure 19-4.

Figure 19-1: Historical Silver Price

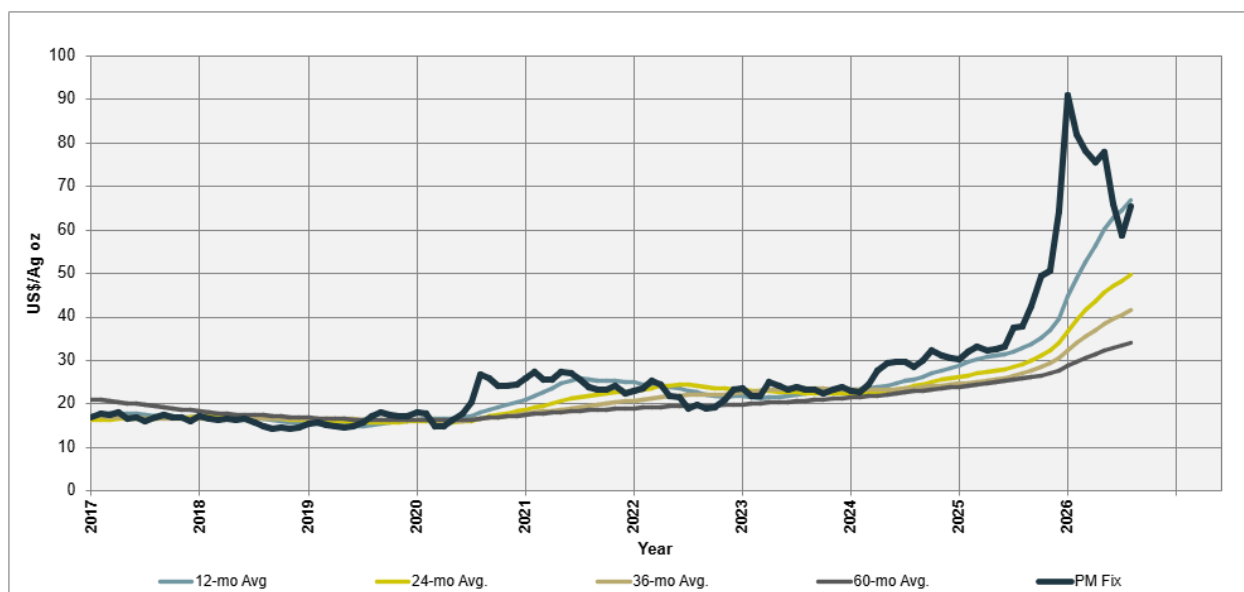


Figure 19-2: Historical Lead Price

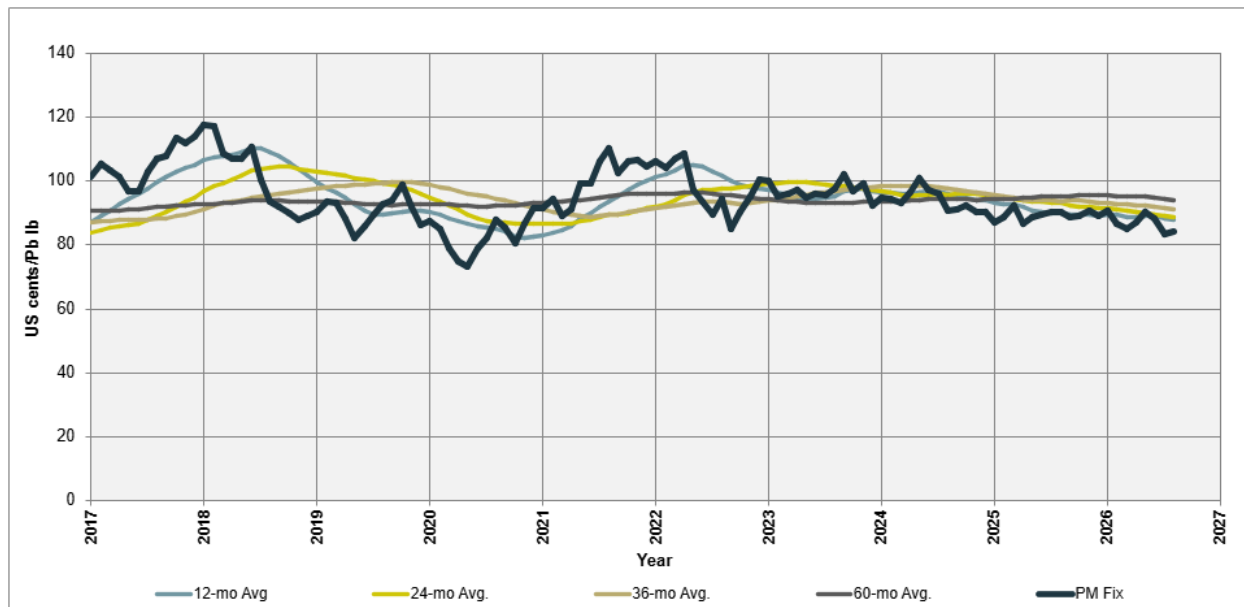


Figure 19-3: Historical Zinc Price

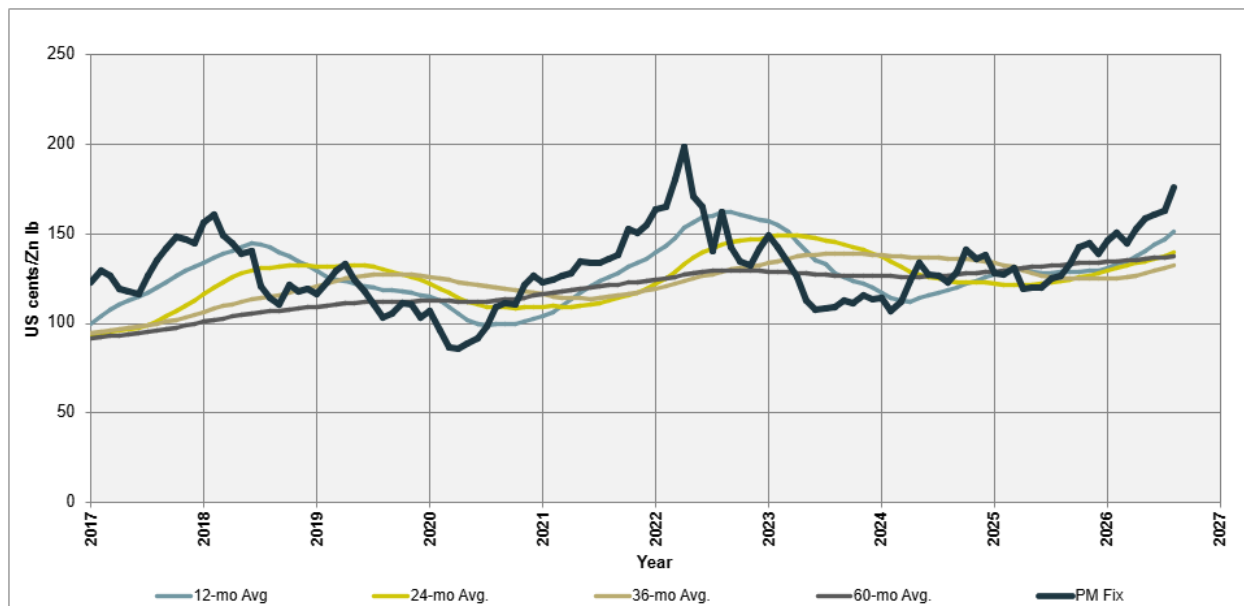
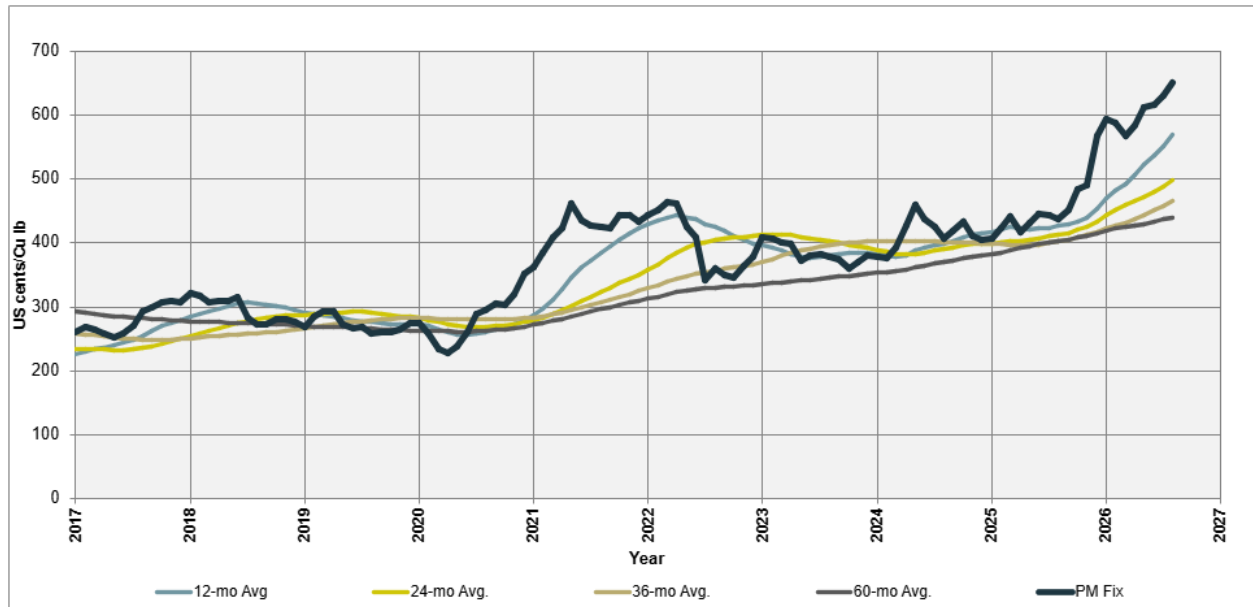


Figure 19-4: Historical Copper Price



A sensitivity analysis of metal price and exchange rate was completed as part of the economic analysis and is discussed further in Section 22.

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACTS

20.1 Environmental Studies

20.1.1 Biophysical Setting

The Project is in the southern Mackenzie Mountains in south-west Northwest Territories. The Mine site facilities are situated on the eastern side of and adjacent to PC Silver Mine, about 43 km upstream from its confluence with the South Nahanni River, and approximately 7 km upstream of the point where PC Silver Mine crosses the boundary of the expanded Nahanni National Park Reserve. The South Nahanni River flows into the Liard River near Nahanni Butte, 100 km downstream from PC Silver Mine. The Liard River merges with the Mackenzie River at Fort Simpson, a further 175 km downstream.

The Mine site is at an elevation of 870 m above mean sea level (AMSL), and is situated in topography characterized by low mountains and narrow valleys with an average relief of 300 m. The Mine site is located within the Alpine Forest-Tundra section of the Boreal Forest, characterized by stunted fir with limited undergrowth and open areas dominated by lichen.

Overland access to the site is currently by winter road. A proposed 170 km ASR will connect the Mine to the Liard Highway. The ASR, departing the Mine site at Km 0, will head north along the Prairie Creek valley for about 7 km before turning east to cross the Mackenzie Mountains. The road will climb out of the Prairie Creek valley; it enters Sub-Alpine Shrub and Alpine Tundra from an elevation of approximately 1,000 m AMSL at Km 10. The road continues to climb through the Alpine to a summit of 1,530 m at Km 17, then dropping down and leaving the Sub-Alpine again at the 1000 m elevation around kilometre 25. As the road drops from the 1,000 m elevation to the 900 m elevation, it passes through a spruce-lichen Alpine forest zone similar to that found at the Mine site and then into Riparian Alluvial habitat in the Sundog tributary valley bottom, a key landscape feature along the ASR.

From Km 40 to km 55, the road crosses forest developed on glacial depositional deposits, and crosses Polje Creek which drains the Poljes (karst lakes) before ascending the Ram Plateau. As the road crosses the Ram Plateau, it passes through an open forest Black Spruce/Pine Parkland setting between the 830 to 930 m elevations, before dropping down into the Tetcela River valley. The valley consists of a mixed coniferous/deciduous closed forest. The road then passes through a short distance of muskeg open shrub/sedge wetland at the headwaters of Fishtrap Creek, and climbs up and over the Silent Hills, again a closed mixed coniferous/deciduous forest. The road alignment then runs along the eastern slopes of the Silent Hills, an area of black spruce, before passing through mixed coniferous-deciduous-pine parkland prior to entering the Grainger River headwaters at Grainger Gap (Second Gap), staying north of, and never crossing into, the Bluefish Creek basin.

Once through the Grainger Gap, the road alignment turns south along the foothills of the Front Range through mixed deciduous coniferous forest towards Nahanni Butte, avoiding the Grainger

Tillplain. The road crosses the Liard River near the community and continues through forest to the Nahanni Butte access road which after 10 km connects with the Liard Highway.

Both Mine and ASR have been the subject of extensive baseline studies to support separate environmental assessments and to provide data for project plans. The Developer's Assessment Report (DAR) for the Mine was submitted to the Mackenzie Valley Review Board (MVRB) in 2010 (Canadian Zinc Corp., 2010), and the DAR for the ASR was submitted to the MVRB in 2015 (Canadian Zinc Corp., 2015). Studies relating to the Mine environment include operation of a hydrometric station on Prairie Creek, collection of data on water quality, sediment, fish and other aquatic biota, as well as climate and wildlife studies. An aquatic effects monitoring plan (AEMP) was developed in 2023. Baseline studies for the ASR included hydrometric stations, water sampling, extensive wildlife studies, terrain assessment and permafrost investigation.

There is extensive background information available on the MVLWB's public registries' (www.mvlwb.com), which includes environmental baseline studies; regulatory reports/documents; and socio-economic data. Much of this information has been compiled by Canadian Zinc or has been collected as part of baseline and environmental assessment activities by consultants. Some of the baseline information collected at the site dates to the 1970's.

20.1.1.1 Terrestrial Flora and Fauna

Prevalent wildlife within the Project area includes Dall's sheep; mountain woodland caribou; wood bison; wolverine; and grizzly bear. While potential impacts to mammalian megafauna from mine operations are expected to be limited and largely avoidable, there is potential for effects associated with road use, such as mortality (primarily caribou and bison), and noise disturbance due to air traffic (primarily Dall's sheep). The possibility exists for potential bear-human encounters at the site; however, programs to limit any attraction of bears will be implemented.

To help avoid potential interactions of wildlife with humans and project-related activities, a wildlife sighting and notification system will be adopted as part of a broader mitigation plan. Other mitigation measures include posted and enforced speed limits, implementing access control measures to monitor and manage non-Project related travel a hunting pressure along the ASR, and the management of flight paths for air traffic.

No significant impacts to vegetation communities are expected due to the relatively small area of disturbance that will result from Project construction and operations.

20.1.1.2 Terrain and Stability

No large-scale landslide features are evident near the mine and access road. The Mine site is protected by an armoured flood protection berm designed to withstand the 1:100-year return period flood. The main engineered structure associated with the Project will be the Water Storage Pond.

The ASR follows a steep and remote mountainous route. Terrain issues such as landslides, avalanches and permafrost thaw have potential impacts on human safety and the environment along the proposed road. The MVRB, in its Report of Environmental Assessment and Reasons for Decision (MVRB, 2017), required Canadian Zinc to establish an Independent Technical

Review Panel (ITRP) for road design. The mandate of the ITRP is to provide independent expert advice and recommendations on the design and construction of the road to minimize traffic related accidents, road failure or malfunctions, and any resulting significant adverse impacts on human safety or the environment. Several records on the MVRB public registry include updates from Honey Badger Silver Inc. note that the ITRP has been established, and terms of reference for the ITRP was approved in January 2020 (Honey Badger Silver Inc., 2026).

20.1.1.3 Aquatic Environment

Bull trout and mountain whitefish have been documented in Prairie Creek near the PC Silver Mine in multiple surveys Ker Priestman (July 1980), Beak Consultants (March, April, May, September 1981), Rescan (May-June, September 1994), and Mochnacz, DFO (August 2001). No evidence of spawning has been found adjacent to or downstream of the Mine site; however, bull trout were found to spawn in Funeral Creek (DFO, 2005), an upstream tributary of PC Silver Mine, the valley of which is used for part of the road route.

Based on the site water management plan and water quality predictions (Canadian Zinc, 2021), effluent discharge via an exfiltration trench will not impact on the aquatic environment and will meet downstream water quality objectives. An exfiltration trench for effluent discharge will be installed below the bed of Prairie Creek and only part-way across the channel (Northwest Hydraulics, 2011). These plans and predictions formed the basis of the effluent regulation contained in the Water Licence issued by the MVLWB September 23, 2022, and amended August 7, 2025.

20.1.2 Socioeconomic Setting

The Project is within the claimed traditional territories of the Nahʔą Dehé Dene Band and the Łíídlı́ Kúę First Nation. The nearest community is Nahanni Butte, home of the Nahʔą Dehé Dene Band, located approximately 90 km to the southeast of the Project site. Other communities within 200 km of the site include Fort Simpson, Fort Liard, Trout Lake and Jean-Marie River.

Resource development and other industrial projects are next to non-existent in the region. The major employer is government. Limited education, high unemployment, low income levels, sub-standard housing and social issues are typical of local communities. With Mine development, there would be a period of adjustment as people and communities integrate into the wage economy.

20.1.3 Environmental Risks and Opportunities

Permitting the PC Silver Mine and ASR has been challenging given that the Mine site is on land surrounded by a national park and world heritage site, and the access road must cross the park to connect to transportation links. Expectations for environmental protection are high and translate into extensive monitoring and management plans and oversight. PC Silver Mine is an essentially pristine mountain stream, and as a result downstream water quality objectives must be met for the protection of aquatic life.

Approximately 5 km of underground workings were previously developed at the Mine site and include an adit that discharges mine water by gravity, requiring treatment for discharge. Mine development will include backfill and sealing of the mine openings such that mine water discharge will not exist after mine closure.

20.2 Water Management and Water Management

20.2.1 Waste Management

Canadian Zinc evaluated the potential for acid rock drainage and metal leaching (ARD/ML) from mine wastes. Mesh Environmental Inc. (Mesh) undertook a broad geochemical study in 2005 and 2006, which analyzed mineralized rock samples, tailings and waste rock. Laboratory work conducted as part of this study to assess acid rock drainage included: acid-base accounting (ABA); total inorganic carbon and multi-element Inductively Coupled Plasma (ICP) analyses on all samples; mineralogy; expanded ABA (pyritic sulphur, siderite correction, acid-buffering characterization curves); and grain size analyses on a sub-set of samples. Mesh made the following conclusions regarding the study:

- All tested host rock units are non-acid generating due to low quantities of sulphur and the substantial effective buffering capacity provided by reactive Mg containing carbonates (dolomite);
- Dolomite presence in some waste materials represents a low risk of elevated salinity associated with neutral mine drainage; however, environmental factors (higher rainfall, lower year-round temperatures) are likely to reduce incidence;
- Vein and stratabound mineralization classify as potentially acid generating due to an abundance of sulphide mineralization (although Mesh's kinetic test data collected up to December 2006 suggests that it may take a substantial amount of time for acidity to be generated, due to the significant amount of buffering capacity available from the carbonate host rocks);
- Dense media separation rock is non-PAG and contains relatively low sulphur values; and
- Flotation tailings are classified as non-PAG and contain sufficient buffering capacity to maintain neutral conditions under laboratory conditions.

Mesh also evaluated potential metal leaching as part of their study program. Samples were collected from underground seeps and portal discharge. Short-term leach extraction tests were completed on rock, tailings and concentrate samples. In addition, kinetic testwork was carried out on two mine wall-wash stations (one host rock and one mineralized sample) and on seven humidity cells. The following conclusions were made:

- Waste/host rock have the potential to release soluble metals such as cadmium, copper, mercury, lead, strontium, and zinc at neutral pH conditions, mainly as a result of metal carbonate dissolution and, to a lesser extent, sulphide oxidation (note predicted rates of soluble metal release were considered to reflect a worst-case scenario);

- Under neutral pH conditions, dense media separation rock could potentially release elevated concentrations of several metals of environmental concern such as arsenic, strontium, cadmium, copper, lead, mercury, selenium, gold, and zinc;
- Under neutral pH conditions, tailings have the potential to release metals such as arsenic, cadmium, copper, lead, mercury, selenium and zinc at levels of potential environmental concern (release rates like those for dense media separation rock material); and
- Dissolved metals are typical for flotation supernatant.

The current Project plan includes the placement of the flotation tailings from the mill underground into the mined-out voids as a paste backfill mix. Dense media separation reject rock, together with waste rock from mine development, will be placed in an engineered waste rock pile located in a draw of Harrison Creek. This approach has two clear advantages:

- 1) Following mine closure, there will be no mine waste on the PC Silver Mine floodplain; and
- 2) The underground workings will be backfilled and sealed, reducing pathways for mine drainage egress.

During operations, seepage from the waste rock pile will be collected at the toe of the pile in a lined seepage collection pond. The pond will be connected to the site water management system by pipeline.

Testing has confirmed that mine and mill wastes have the potential to leach metals at neutral pH. For this reason, a closure and reclamation strategy has been selected specifically to minimize metal leaching, primarily by placing tailings underground and placing an engineered cover over the waste rock pile.

20.2.2 Water Management

In 2010, Canadian Zinc commissioned the Saskatchewan Research Council to complete a study of background metal concentrations in PC Silver Mine to assist with the development of site-specific water quality guidelines for the Project. Hatfield Consultants continued this work in 2012. Based on the findings of these studies and site-specific water balances, it was predicted that the planned discharges from the Mine during operations would result in metal concentrations in PC Silver Mine that would not exceed the proposed objectives, which were adopted by the MVLWB when the Water Licence was issued. Canadian Zinc has developed a discharge strategy based on monitoring flows in the creek and determining the effluent volume that can be safely discharged without causing exceedance of objectives, with water being stored temporarily in the on-site Water Storage Pond.

There are three main sources of water that will need to be managed during mine operations. These are:

- Waste rock dump and stockpiles runoff and seepage water;
- Groundwater intercepted underground (non-contact water from a dewatering well and contact water from sumps in the mine workings) pumped to the water storage pond; and

- Process water from the mill.

The groundwater may contain elevated metals. The process water is expected to contain high concentrations of several metals plus residues from flotation chemicals.

A large, ponded facility was originally built on site with dykes and a clay lining and intended for tailings disposal. This pond will be re-engineered (including the installation of a new synthetic liner) as a water storage pond for the Project (Section 18.5). The water storage pond will consist of two cells, one for mainly the groundwater from underground, and the other for mainly mill process effluent.

During operations, the water storage pond will supply feed water to the mill. Mine drainage coming into contact with the mine workings will be recycled to stope drills, and the remainder sent to the process water cell of the water storage pond (Cell A). Cell A will also receive seepage and runoff from the waste rock pile and stockpiles. A water balance will be maintained due to water losses to concentrates and tailings backfill. Cell A will provide the main source of water to the Mill, with no discharge to the environment.

The groundwater drawn from the vein structure up-gradient of the mine workings will be sent to Cell B for temporary storage. Cell B water will be treated in a new treatment plant to reduce metal concentrations, as necessary, and then released to PC Silver Mine. A schematic of the water management system and narrative can be found in Section 18.5.

The water level in Cell B of the water storage pond will fluctuate seasonally, increasing in the winter as water accumulates in storage, and decreases in the summer when water is discharged at a higher rate.

The treatment and release rate of water will vary depending on flows in PC Silver Mine at the time of discharge in order to meet in-stream water quality objectives. Effluent discharge in the Northwest Territories is regulated by a Water Licence that specifies end-of-pipe concentrations, and in some cases, volume restrictions. This will be the case for the PC Silver Mine, with volume restrictions in the form of an established creek to effluent flow ratio. The current Water Licence for the mine requires all effluent to be discharged from the exfiltration trench at a dilution rate of 19 or more (the discharge rate should be 1/19th of the flow in the PC Silver Mine) up to a maximum effluent discharge rate of 0.56 m³/s. The Water Licence also specifies the effluent quality criteria (EQC) that must be met. Based on the latest modelling (Canadian Zinc, 2021), the effluent is expected to meet the EQCs.

Canadian Zinc will establish a permanent, automated flow monitoring station on PC Silver Mine. flows would be monitored continuously, with data relayed to the water treatment plant in real time. A flow monitoring protocol has been developed to convert creek water levels to reliable flow rates, no matter the circumstance. This accounts for seasonal effects, high water events, and ice cover. This ensures the effluent flow calculation will always be based on creek flows that are known to exist with a high degree of certainty. Monitoring of discharge flows will be automated, with data relayed continuously.

The discharge of the final combined effluent from the site will be achieved using perforated pipes installed in an exfiltration trench located below the bed of PC Silver Mine. This exfiltration system will promote mixing of the effluent with the receiving waters.

The majority of mine activities, and those associated with chemicals, fuel and hazardous material, will take place within a dyke-protected area, isolated from PC Silver Mine. Any spills or contamination can be contained on site, and discharge of site water to the environment can be stopped temporarily. Spilled liquid could infiltrate, but the shallow water table would carry the liquid to the final site collection pond where it could be collected and managed. Specific chemicals and fuels, such as diesel, will have their own dedicated containments. Most other chemicals will be non-liquid in nature and will be stored in warehouses.

The potential for spills or leaks along the access road will be minimized by controlling road use in terms of vehicle numbers and speeds and using industry-standard containers for transport and storage. Response equipment will be carried by every vehicle and will also be stored on the road at specific locations to facilitate a rapid response. Response efforts and spill collection will be focused on control points in the event of a spill. Six control points have been defined.

Mine concentrates will be transported in sealed containers and thus concentrate dust should be minimal. Any spills would be completely recovered. To confirm the absence of impacts, roadbed soil and vegetation samples will be collected along the route annually and compared to a baseline to confirm material is not being dispersed.

Given the remote location of the Project, there is currently no development nearby, and it is expected that there will be very little additional activity in the future which could contribute to cumulative effects.

20.2.3 Management Plans

Permits for project development include requirements to submit a number of detailed plans and conduct programs that are expected to mitigate such effects. For mine operations, documents that will need to be submitted and activities to be undertaken are:

- Engagement Plan;
- Final design, Construction drawings Waste Rock Pile;
- Final design, Construction drawings Feed Stockpile;
- Final design, Construction drawings Water Storage Pond;
- Final design, Construction drawings Exfiltration Trench;
- Exfiltration Trench construction as-built report;
- Final design, Construction drawings Engineered Structures;
- Engineered Structures construction as-built reports;
- Waste Management Plan;
- Waste Rock and Feed Storage Monitoring Plan;

- Contaminant Loading Management Plan;
- Tailings and Backfill Management Plan;
- Explosives Management Plan;
- Water Management Plan;
- Report on water treatment effluent quality optimization;
- Update the Protocol for Real-Time Estimation of PC Silver Mine Flows;
- Terms of Reference for Plume Delineation Study;
- Results of Plume Delineation Study;
- AEMP Design Plan;
- Spill Contingency Plan;
- Failure Modes and Effects Analysis (FMEA);
- Mine Site Contingency Plan;
- Closure and Reclamation Plan; and
- QA/QC Plan for SNP Water Sampling.

For the winter road and subsequent ASR, the following documents will need to be submitted:

- Spill Contingency Plan;
- Engagement Plan;
- Sediment and Erosion Control Plan;
- Permafrost Management Plan;
- Rare Plant Management Plan;
- Invasive Species Management Plan;
- Geochemical Management Plan;
- Traffic Control Mitigation, Operation and Maintenance Plan;
- Explosives Management Plan;
- Water Management Plan;

- Interim Closure and Reclamation Plan;
- Waste Management Plan;
- Wildlife Management and Monitoring Plan;
- Avalanche Hazard Management Plan;
- Design and Construction Plan; and
- Cultural Heritage Protection Plan.

20.3 Closure and Reclamation Planning

20.3.1 Closure Schedule and Cost Estimate

Upon cessation of operations, closure activities at the PC Silver Mine site are envisaged to occur in three phases, as follows:

Phase I – On-site and off-site reclamation of all facilities not required for long-term monitoring and water treatment:

- Backfill underground workings;
- Mine and mill equipment removal to waste rock pile;
- Building and on-site infrastructure demolition;
- Water storage pond – Tailings and Sediment removal and disposal;
- Water storage pond – Liner removal and dyke breaching; and
- Substantial reclamation of the waste rock pile, leaving a portion available for the disposal of final site items.

Phase II – Post-closure monitoring and water treatment:

- Monitoring over a period of 13 years, as described above; and
- Treatment of mine water for six years, if required.

Phase III – Reclamation of post-closure facilities:

- Demolition of the post-closure facilities;
- Disposal of final demolition waste in the waste rock pile; and

- Final reclamation of the waste rock pile.

Following Phase III at the Mine site, reclamation of the access road will occur.

The total amount of security deposits is described in the issued permits associated with future operations and summarized in Table 20-1.

20.3.2 Closure and Reclamation Plan

Following mine closure, it is expected that there will be no surface drainage from mine portals as the underground workings and access tunnels will be backfilled with a paste tailings mix and sealed with bulkheads. Some groundwater seepage from the bedrock surrounding the underground workings may occur, with the water containing some metals, and to a lesser extent from the backfilled waste mixture. A small quantity of seepage from the covered Waste Rock Pile is also possible.

Predictions for PC Silver Mine water quality after mine closure were made by Robertson Geoconsultants, with geochemical source terms provided by phase Geochemistry. The predictions indicate that all metal concentrations will remain within the water quality objectives for average creek flows year-round, although if creek flows are lower than normal in winter, zinc concentrations may exceed objectives but will be similar to those predicted to have occurred before any mine development. Post-mine predictions also indicate higher cadmium concentrations in PC Silver Mine during the winter if creek flows are unusually low.

An interim Closure and Reclamation Plan for the PC Silver Mine was prepared during the 2012-2013 permitting process and was updated in 2021 for the mine permit renewal applications. The principal difference is the assumption of a much larger waste rock pile. The plan adheres to the 2017 MVLWB “Guidelines for the Closure and Reclamation of Advanced Mineral Exploration and Mine Sites in the Northwest Territories”.

A Closure and Reclamation Plan is prepared at the permitting stage to demonstrate how the mine site can be reclaimed to protect the environment, and as a basis for estimating reclamation costs which allow a decision regarding a reclamation bond to be made.

The following sections briefly describe temporary and permanent activities to close and reclaim the site.

20.3.3 Temporary Closure Activities

According to MVLWB, temporary closure is defined as a mine ceasing operations with the intent to resume mining activities in the future. Temporary closures can last for periods of weeks, or for several years based on economic, environmental, political or social factors.

20.3.3.1 Waste Rock Pile

Activities planned for the Waste Rock Pile during temporary mine closures include continued collection and management of seepage, maintenance of diversion ditches, and monitoring of physical stability and water quality.

Underground

The focus of activities underground would be on maintaining stability, safety and water management systems. Specific activities would include the following:

- Inspect open faces and access ways, and fence-off or install temporary support for any unstable areas;
- Ensure that mine drainage flows to sumps and that water pumping stations are active and maintained;
- Continue to monitor the operation of underground pumps and their flow rates;
- Ensure all explosives and detonators are removed from temporary storage areas and placed in secure magazines on surface;
- Remove all mobile electrical and other equipment not required during the shutdown to surface; and
- Review ventilation requirements to assess if, and what, reductions can be implemented.

20.3.3.2 Process Plant

All process equipment, tanks and piping will be emptied to prevent problems on restart. Process water will be sent to the Water Storage Pond. All concentrates within the processing circuit will be filtered and containerized. For extended temporary closures, the ball mill will be jacked up off its bearings to prevent wear.

20.3.3.3 Water Storage Pond

The water storage pond will continue to receive water from underground and the Waste Rock Pile. The latter would be directed to Cell B for later treatment and discharge, as necessary, to avoid accumulation in Cell A.

The Process Plant will not be producing process water.

Cell B of the water storage pond will be kept in balance by sending water to the water treatment plant.

20.3.3.4 Water Treatment Plant

The water treatment plant will remain at full operational status and will be operated as required to maintain an annual balance in Cell B of the Water Storage Pond.

20.3.3.5 On-site Infrastructure

Any infrastructure facilities on-site that are not required during temporary closure will be taken off-line, such as some of the generator sets. Most facilities will need to stay in operation but at a lower utilization rate, such as the Sewage Treatment Plant, incinerator and power plant. All infrastructure will be maintained.

20.3.3.6 Off-site Infrastructure

Off-site infrastructure is limited to the access road where non temporary closure activities are planned over and above the normal seasonal closures each year.

20.3.4 Permanent Closure Activities

The MVLWB defines permanent closure as when a mine exhausts material that can be economically extracted and ceases operations without the intent to resume mining activities in the future.

20.3.4.1 Salvage

Due to the remote location of the mine, only a portion of the mine assets are expected to have sufficient salvage value at mine closure to warrant transport off-site to a suitable market. The majority of the assets are expected to be buried either underground or in the waste rock pile:

- All mobile equipment;
- All stationery equipment;
- All building structural materials;
- All construction materials; and
- All other solid materials.

At the completion of the mine reclamation landfilling, the landfill within the waste rock pile will be covered with a minimum 1-metre thick layer of waste rock. A cover will be placed over the waste rock pile during mine reclamation to promote runoff and minimize infiltration and the generation of leachate. An initial cover design study recommended a 1 to 2 m thick 'till' (clayey soil) layer be

applied. The cover is part of the current closure plan, but the design will be reviewed during operations.

The selected cover design will be based on data from seepage monitoring during the mine life, and predictions of cover behaviour, long-term waste rock seepage, and the resulting groundwater and surface water quality.

At mine closure, the seepage from the waste rock pile may need to be temporarily directed to water treatment (see 'Underground' below re receiving water quality predictions).

20.3.4.2 Water Storage Pond

At the point that the water storage pond is no longer required for mine operations or reclamation activities, it will be reclaimed as follows:

- Water in both cells will be processed through the water treatment plant and discharged;
- Sediment and any remaining tailings in the water storage pond will be removed and sent to the paste backfill plant to be subsequently deposited underground;
- When the water storage pond is free of contaminated solids and water, the liner will be removed and placed in the waste rock pile; and
- The water storage pond embankment will be breached in two places to prevent the structure from impounding water, and the locations of discharge to PC Silver Mine will be stabilized as necessary to limit erosion and sedimentation.

20.3.4.3 Underground

The intent is to backfill and seal the underground workings and all access tunnels. The workings will be sealed all the way out to the portals. Hydraulic bulkheads have been included in closure plans to ensure the tunnels do not provide seepage pathways.

After mine closure and underground backfill, groundwater levels will slowly rebound in the mine area, flooding any remnants of the workings. Some groundwater movement may occur along the edges of the backfilled area where the wall rocks are fractured, and within the workings where gaps remain between the backfill mix and the roof that could not be filled, or where the mix has settled. Predictions have been made regarding the quality and movement of this groundwater to surface, and the resulting impact to the quality of surface water (see above). These predictions indicate that surface water quality objectives will be met without a need for further actions. While the predictions will be refined during operations and at mine closure, as a contingency, a short-term groundwater pumping and water treatment scheme has been devised and provided for in security estimates.

The pumping system will consist of a well installed from surface into the core of the backfilled workings. Pumping would occur during open water months to depress groundwater levels, and pumping would stop in winter allowing water levels to rebound. In this way, the Mine void would be used as a sump. Pumped water would be sent to a scaled-down mine water treatment circuit

in the water treatment plant, followed by discharge to the environment. The water treatment plant would also receive seepage from the waste rock pile if deemed necessary. Over time, the quality of groundwater underground and waste rock pile seepage is expected to improve as leachable metals diminish. Therefore, the contingency pumping scheme, if required, would be expected to operate for four to eight years. Six years was assumed for security estimates.

20.3.4.4 Mine Equipment

Mine equipment containing hydrocarbons will be removed from underground before mine closure. Inert equipment will be left.

Equipment and material that is salvageable will also be removed. Equipment and material that has no salvage value will be decontaminated and either moved back underground or placed in the waste rock pile.

20.3.4.5 Process Plant and On-site Infrastructure

All surface facilities including the Process Plant, Paste Plant, Administration, Camp, Sewage Treatment Plant and Tank Farm will be reclaimed as follows:

- Evaluate and store for removal all waste that does not qualify for disposal in the waste rock pile; and
- Dismantle all equipment and building structures, reduce the material to manageable pieces, and place them in the waste rock pile.

For the post-closure monitoring phase, a scaled-down mine water treatment circuit may remain, along with reduced accommodations, fuel storage and warehouse facilities.

20.3.4.6 Off-site Infrastructure

The road will be reclaimed by removing stream-crossing structures and culverts, grading/pulling back fill slopes, and scarifying surfaces to promote revegetation by the natural invasion of native species. Cross ditches/water bars will be installed as necessary to limit erosion and sedimentation.

20.3.5 Post-Closure Monitoring, Maintenance, and Reporting Program

Post-closure monitoring at the Mine site will include inspection of mine access barricades, the waste rock pile cover and runoff controls, and reclaimed surfaces for erosion, and the collection of water samples. Water samples will be collected from Harrison Creek and Prairie Creek, and a limited number of groundwater wells. Three locations on Harrison Creek (one upstream and two downstream), three locations on Prairie Creek (one upstream and two downstream) are envisaged. The number and location of groundwater wells to be included will be determined during operations.

For the first three years after closure and reclamation, monitoring and inspections will occur monthly over the period March to November. In the following five years, monitoring and inspections will occur bi-monthly from May to September. In the final five years, monitoring and inspections will occur once a year in July (post-freshet). The intent of monitoring is to track the revegetation and stabilization of surfaces and confirm that water quality is as expected. An annual monitoring report will be provided to regulators.

Provision has also been made to operate the previously described mine water pumping and treatment system for six years after the groundwater level has rebounded to elevation 865 m.

Post-closure monitoring of the access road over 2-3 years will focus on surface stabilization and sediment controls, with water quality sampling primarily for turbidity/TSS to confirm the absence of issues.

20.4 Permitting Considerations

20.4.1 Overview of the Regulatory Process

As the Mine Site is located within the Mackenzie Valley, all activity relating to land and water use at the site is subject to the *Mackenzie Valley Resource Management Act* (MVRMA). The MVLWB is responsible for regulating the use of land and waters and the deposit of waste on Crown Land used by the mine and its infrastructure. The MVLWB issues land use permits (LUP) and Water Licences for projects outside settled land claim areas in the Mackenzie Valley.

Applications for a LUP or a Water Licence are made to the MVLWB. Each application requires the inclusion of certain baseline and technical information, in the form of a Project Description Report (PDR). The information in a PDR is used to undertake preliminary screenings of applications to determine whether an application should be referred to the MVRB for Environmental Assessment (EA) or can proceed directly to regulatory review for the issuance of a LUP and / or Water Licence.

If an application is referred to an EA, the MVRB develops a work plan and terms of reference for the EA, for the proponent to prepare a Developers Assessment Report (DAR). On completion of an EA, the MVRB, in their Report of Environmental Assessment (REA), can either reject the Project, approve it with or without measures to enforce environmental mitigation actions, or refer the Project to Environmental Impact Review (EIR) by an appointed panel. The REA is forwarded to the Minister of Crown-Indigenous Relations and Northern Affairs Canada (CIRNAC) for consideration. The Minister may do nothing, in which case the MVRB's decision stands, or the Minister may seek to modify the decision in a consult-to-modify process. If the Project is approved, the file reverts to the MVLWB for the processing of permits.

20.4.2 Environmental Assessments

20.4.2.1 Mine

An EA of the Mine was completed in the 2010-2011 time period, with a Developer's Assessment Report submitted to the MVRB in 2010, and the MVRB recommended approval of the Project proposal for operations on December 8, 2011. The MVRB concluded, pursuant to paragraph 128(1)(a) of the MVRMA, that the proposed development as described in the EA (including the list of commitments made by Canadian Zinc) is "not likely to have any significant impacts on the environment or to be a cause for significant public concern". As part of their decision, the MVRB provided a series of suggestions that, in their opinion, would improve the monitoring and management of potential impacts from the Project:

- Either option proposed by Canadian Zinc to increase water storage on site will improve water quality in PC Silver Mine; however, construction of a second pond may address a broader range of risks and result in better water management on site;
- A Tailings Management Plan should be prepared for both the permanent storage of tailings underground and the temporary storage of tailings on surface at the Mine Site; and
- There are better ways to contain concentrate during transport along the winter road than the bag method proposed. Secondary containment of concentrate during transport was recommended.

20.4.2.2 All-Season Road

On April 16, 2014, Canadian Zinc made applications to the MVLWB and Parks Canada for permits to construct, maintain and operate an ASR from the mine to the Nahanni Butte access road, which connects to the Liard Highway. An ASR will enable the haul of concentrates from the mine site to rail year-round, with temporary closures relating to freeze-up and ice break-up on the Liard River. The ASR would use a similar alignment to the permitted winter road with some re-alignments to cross ground more suitable for ASR construction. The MVLWB referred the applications to the MVRB on May 22, 2014 for EA1415-001.

Canadian Zinc submitted its DAR to the MVRB in April 2015, and an Addendum to the DAR in September 2015. The MVRB issued its Report of Environmental Assessment and Reasons for Decision (the EA Report) on September 12, 2017 and submitted the Report to the federal Minister of CIRNAC. The MVRB recommended approval of the ASR subject to implementation of measures described in the Report, which it considers are necessary to prevent significant adverse impacts on the environment and local people. Some of these measures include requirements for:

- Create an ITRP, to ensure that the road is designed to a standard that is highly protective of people and the environment;
- Create a Traffic Control Mitigation and Management Plan;

- Conduct systematic wildlife monitoring and adaptive management incorporating Traditional Knowledge;
- Prepare a Wildlife Management and Monitoring Plan;
- Install stations to collect baseline water flow data, to use when designing water crossings;
- Collect detailed baseline information, monitor effects and make an adaptive management framework for the Sundog Creek diversion;
- Further engage Traditional Knowledge holders about cultural and heritage resources in the Project area, and conduct an Archaeological Impact Assessment incorporating this Traditional Knowledge;
- Conduct a rare plant survey to form the basis of a Rare Plant Management Plan;
- Conduct permafrost investigations to inform road design and appropriate permafrost mitigations;
- Create a Permafrost Management Plan with systematic permafrost monitoring and adaptive management; and
- Support independent community monitoring of the Project.

The federal Minister approved the MVRB's decision regarding the ASR EA on October 9, 2018.

20.4.3 Current Permits and Licences

Canadian Zinc currently holds the land use permits and water licences necessary to advance the Project, and these are identified in Table 20-1. The ASR crosses through the jurisdictions of both the MVLWB and Parks Canada, and thus both agencies have issued land use permits and water licences to Canadian Zinc. For Parks Canada, the permits/licences are for the portion of the ASR that crosses the Nahanni National Park Reserve.

As per the *Mackenzie Valley Resource Management Act*, land use permits are issued for a five year period, with the option to extend an additional two years (a maximum of seven years). Two land use permits issued by the MVLWB (for the ASR and construction staging area) are pending replacement with a single new land use permit MV2026F0014, expected to be issued in November 2026 in advance of expiration. The staging area land use permit will be incorporated into the ASR land use permit with this renewal and a Storage Authorization was issued on September 22, 2026 between the time when the LUP expires, and the new LUP is issued. The Parks Canada land use permit also expires in November 2026 and the renewal process is currently underway with the issuance of the new LUP in November 2026 in advance of expiration.

Table 20-1: Summary of Current Permits and Licences

Agency	Type	Permit	Project Component	Description	Date of Issuance (duration)	Date of Expiry
MVLWB	Land Use Permits	MV2014F0013	ASR	Construction and operation of THE ASR on territorial land	Nov 13, 2019	Nov 12, 2026; to be replaced by MV2026F0014
		MV2019T0021	Construction staging area	Staging Area – North side of the Liard River Barge Landing	Sept 3, 2019	Replaced by Storage Authorization issued Sep 22, 2026 and then included as a part of the scope to of the new LUP
		MV2026F0014	ASR and construction staging area	This will replace MV2014F0013 and MV2019T0021	5-year term expected	(Application in progress)
	Water Licence	MV2021L2-0004 (Type A)	Mine	Mineral exploration and mining and milling	Sept 23, 2022	Sep 22, 2032
		MV2023L8-0002 (Type A)	ASR	KP 0 to 17.14 and KP 101.3 to 169.3; non-federal land	July 12, 2024	Jul 11, 2044
		MV2023L8-0003 (Type A)	ASR	KP 156.1 to 161.7 and KP 163 to 164.1; federal land	July 12, 2024	Jul 11, 2044
Parks Canada	Land Use Permit	PC2014F0013	ASR	KP 17.14-101.3 of the ASR within the Nahanni National Park Reserve	Nov 22, 2019	Nov 21, 2026 to be replaced by new LUP
	Water Licence	PC2023L8-0002	ASR	KP 17.14-101.3 of the ASR within the Nahanni National Park Reserve	July 12, 2024	Jul 11, 2034

Source: KP (2026)

20.5 Social Considerations

The PC Silver Mine and access road is in an area that includes the claimed traditional territory of the Nahąą Dehé Dene Band and the Łíldıı Kúę First Nation, historically both part of the Dehcho First Nations.

The Dehcho Region hosts a distinct group of Aboriginal peoples, whose ancestors were among the South Slavey people of the Dene Nation of what is now the Northwest Territories, as well as Metis people. Many Dehcho people continue to rely heavily on the land, water and resources within Dehcho First Nations territory for sustenance, social and ceremonial purposes.

Dehcho First Nations is an organization representing Dene and Metis peoples in the Dehcho territory of the Northwest Territories. Dehcho First Nations have incorporated a society under the laws of the Northwest Territories to provide leadership, governance, administration and program delivery to their member communities. Dehcho First Nations is a governing body of the Dehcho peoples' lands and administers and oversees programs and services for its member communities, including those relating to health, employment, education, and land and resource management.

The Dehcho First Nations have been in negotiations since late 1990s. Several Agreements between Canada and Dehcho First Nations have been completed and guided the Dehcho Process Negotiations. Included in these are 21 Common Ground Principles, the Framework Agreement, the Interim Measures Agreement, the Interim Resource Development Agreement and Nahanni National Park Reserve.

The outcome of land claims negotiations between the Federal Government, GNWT and the Dehcho First Nations, referred to as the Dehcho Process, is expected to be a Final Agreement that will provide, amongst other things, for the implementation of a Dehcho government within the Dehcho territory. The negotiations have taken many years; however, the timing of completion is uncertain.

20.5.1 Nahᓵᓴ Dehé Dene Band

The community of Nahanni Butte is located at the confluence of the South Nahanni and Liard Rivers, 146 km downstream of the mine site and the home of the Nahᓵᓴ Dehé Dene Band. The population of Nahanni Butte is approximately 90 people.

There is no other existing land occupation, nor commercial land or water-based activities, in the vicinity of the Mine. No traditional use or trapping activity has occurred in the mine site area in recent history.

In October 2008, Canadian Zinc and the Nahᓵᓴ Dehé Dene Band entered into a MOU, to establish a mutually beneficial, co-operative and productive relationship. In the MOU, the Band agreed to maintain close communication links with Canadian Zinc, participate in good faith in current and pending environmental assessment and regulatory processes, and not to oppose, "in principle," mining operations at PC Silver Mine. Canadian Zinc has agreed to apply best efforts to employ Band members and to assist the Band and its community to benefit from business opportunities associated with the exploration and development of the PC Silver Mine. The MOU also provided for the subsequent negotiation of an Impact Benefits Agreement regarding mining operations.

The Company continued discussions and engagement with the Band throughout 2009 and 2010, specifically regarding their Traditional Knowledge and alternate routes for the access road to PC Silver Mine, taking into consideration the expressed preferences of the community of Nahanni Butte. The Band outlined their concerns with the Project and the Company's responses to date

include investigation of road realignment options and surveys of specific locations along the access road for heritage resources.

In January 2011, the Company signed the Nahʔą Dehé Dene Prairie Creek Agreement (the Nahanni Agreement), which provides for an ongoing working relationship between Canadian Zinc and the Nahʔą Dehé Dene Band that respects the goals and aspirations of each party and will enable the Nahanni community members to participate in the opportunities and benefits offered by the PC Silver Mine and confirms their support for the mine.

The Nahanni Agreement provides a framework such that training, employment and business contracts are made available to Nahʔą Dehé Dene Band members to ensure maximization of benefits from opportunities arising from the PC Silver Mine in a manner that will be to the mutual benefit of both parties.

The Nahʔą Dehé Dene Prairie Creek Agreement provides for a positive and cooperative working relationship between the Company and Nahanni Butte in respect of developing and operating an environmentally sound mining undertaking at PC Silver Mine, which will not have significant adverse environmental effects on the ecological integrity of the South Nahanni River or the Nahanni National Park Reserve.

Canadian Zinc engaged with the Nahʔą Dehé Dene Band during the course of the EA for the ASR. The parties completed a Traditional Land Use Agreement (TLUA) related to the ASR which covers road activities and will provide additional benefits to the Band. The agreement was signed on January 15, 2019.

On May 3, 2017 the Nahʔą Dehé Dene Band announced the Band's withdrawal from the Dehcho First Nation. This development has not negatively affected relations between the Band and Canadian Zinc.

20.5.2 Łíídlı́ Kúę First Nation

In June 2011, the Company signed an Impact Benefits Agreement (Łíídlı́ Kúę First Nation Agreement) with the Łíídlı́ Kúę First Nation of Fort Simpson. The Łíídlı́ Kúę First Nation Agreement is similar in many respects to the Nahanni Agreement entered into with the Nahanni Butte Dene Band. The Łíídlı́ Kúę First Nation agreed to support Canadian Zinc in obtaining all necessary permits and other regulatory approvals required for the PC Silver Mine Project. The Agreement is intended to ensure that Canadian Zinc undertakes operations in an environmentally sound manner.

The Agreement provides a framework such that training, employment and business contracts, and some financial provisions are made available to the Łíídlı́ Kúę First Nation to ensure maximization of benefits from opportunities arising from the PC Silver Mine in a manner that will be to the mutual benefit of all parties. The Łíídlı́ Kúę First Nation is the largest member of the Dehcho First Nation.

Similar to the TLUA signed with the Nahʔą Dehé Dene Band, on August 11, 2021 Canadian Zinc signed a Road Benefit Agreement with the Łíídlı́ Kúę First Nation ensuring economic and social benefits from the construction and use of the ASR.

20.5.3 Acho Dene Koe First Nation

The Company signed a Transportation Corridor Benefit Agreement on August 11, 2022 with Acho Dene Koe First Nation (ADKFN). Benefits from this Agreement include employment and training, and contract opportunities.

20.6 Agreements and Programs with Government Agencies

20.6.1 Nahanni National Park Reserve / Parks Canada Memorandum of Understanding

In June 2009, new legislation was enacted by the Canadian Parliament entitled “An *Act* to amend the *Canada National Parks Act* to enlarge Nahanni National Park Reserve of Canada” to provide for the expansion of Nahanni National Park Reserve. Nahanni National Park Reserve was expanded by 30,000 km², making it the third largest National Park in Canada. The enlarged Park covers most of the South Nahanni River watershed and completely encircles the PC Silver Mine. However, the Mine itself and a large surrounding area of approximately 300 km² are specifically excluded from the Park and are not part of the expanded Park.

The exclusion of the PC Silver Mine from the Nahanni National Park Reserve expansion area brought clarity to the land use policy objectives for the region and will facilitate various aspects of the environmental assessment process. The Government's decision on the expansion of Nahanni National Park Reserve reflects a balanced approach to development and to conservation which allows for Mineral Resource and energy development in the Northwest Territories and, at the same time, protects the environment.

Section 7(1) of the new *Act* amended the *Canada National Parks Act* to enable the Minister of the Environment to enter into leases or licences of occupation of, and easements over, public lands situated in the expansion area for the purposes of a mining access road leading to PC Silver Mine, including the sites of storage and other facilities connected with that road. Heretofore, an access road to a mine through a National Park was not permitted under the *Canada National Parks Act*, and the *Act* was amended solely for Nahanni National Park Reserve and specifically for the purpose of providing access to PC Silver Mine.

On July 29, 2008, Parks Canada Agency (Parks Canada) and Canadian Zinc entered into a MOU with regard to the expansion of the Nahanni National Park Reserve and the development of the PC Silver Mine, whereby:

- Parks Canada and Canadian Zinc agreed to work collaboratively, within their respective areas of responsibility, authority and jurisdiction, to achieve their respective goals of an expanded Nahanni National Park Reserve and an operating PC Silver Mine;
- Parks Canada recognized and respects the right of Canadian Zinc to develop the PC Silver Mine and was to manage the expansion of Nahanni National Park Reserve so that the expansion did not in its own right negatively affect development of, or reasonable access to and from, the PC Silver Mine; and

- Canadian Zinc accepted and supported the proposed expansion of the Nahanni National Park Reserve and will manage the development of the PC Silver Mine so the mine does not, in its own right, negatively affect the expansion of the Nahanni National Park Reserve.

The 2008 MOU was intended to cover the period up to the development of the PC Silver Mine (Phase I). In February 2012, and subsequently in November 2015, Canadian Zinc and Parks Canada signed a renewed Memorandum of Understanding regarding the operation and development of the PC Silver Mine and the management of Nahanni National Park Reserve.

The Phase III MOU, signed November 2015 was valid for five years, replaced the previous MOU signed between the Parties in 2008. The Phase IV MOU was signed in 2021 and was valid for five years. The MOU states:

- Parks Canada and Canadian Zinc agree to work collaboratively, within their respective areas of responsibility, authority and jurisdiction, to achieve their respective goals of managing Nahanni National Park Reserve and an operating PC Silver Mine;
- Parks Canada recognizes and respects the right of Canadian Zinc to develop the PC Silver Mine and has granted a Land Use Permit Parks and a Water Licence to provide road access through the Park to the Mine area; and
- Canadian Zinc acknowledges the cooperative management relationship Parks Canada shares with the Dehcho First Nations in the management of Nahanni National Park Reserve. This includes recognition of the 2003 Parks Canada - Dehcho First Nation Interim Park Management Arrangement and the role of the cooperative management mechanism – Naha Dehé Consensus Team.

Since the signing of the Phase IV MOU, Parks Canada, Nahᓂᓂ Dehé Dene Band, and the Dehcho First Nation have signed an Impact Benefit Agreement for the cooperative management through the Naha Dehe Consensus Team of the Nahanni National Park Reserve.

In the MOU Parks Canada and Canadian Zinc agreed to make every reasonable effort to address issues of common interest and build a strong working relationship.

The parties have agreed to make reasonable efforts to be timely in regard to permit requests being submitted, with ample time for review and consultation. Such review, consultation and approval will occur without unreasonable delay.

To the extent that the PC Silver Mine is subject to regulatory or government processes, including hearings, Parks Canada reserves the right, while recognizing the intent of the MOU, to participate in any such process and take such positions as it sees fit and the MOU does not, and is not intended to constrain Parks Canada from doing so, subject only to the understanding that in doing so Parks Canada will not object to or oppose, in principle, the development of the PC Silver Mine.

20.6.2 Government of the Northwest Territories Department of Infrastructure

In August 2012, Canadian Zinc and the GNWT Department of Transportation (now Infrastructure) signed a Collaboration Agreement to ensure effective co-operation related to the public transportation infrastructure that will support the PC Silver Mine project and will help ensure that both public needs and mine activities are supported.

Canadian Zinc plans to use the existing Northwest Territories public transportation system to bring goods, fuel and equipment by road to the mine and to transport its mineral products from the mine to world markets. As part of this Collaborative Agreement, to assist in priority setting, Canadian Zinc will provide reports to the Department of Transportation on its anticipated road transportation requirements for the construction and operation of the PC Silver Mine, which will help the Department of Transport to plan future work on these roads and to maintain and enhance these roads effectively; also the Department agreed to work closely with Canadian Zinc to ensure public safety by identifying areas of Highway 7 and the Nahanni Butte access road that require enhancement or upgrading.

20.6.3 The Northwest Territories Power Corporation

On February 14, 2017 an MOU was signed with Northwest Territories Power Corporation (NTPC) to examine the supply of electrical power for the development and operation of the PC Silver Mine in the Northwest Territories, Canada. In the MOU, NTPC has agreed to work with Canadian Zinc to assess (a) how NTPC could supply the primary electrical energy source to the mine and (b) how NTPC could install generating and connection facilities or other infrastructure assets to provide such electricity supply. Canadian Zinc and NTPC have also agreed to evaluate the integration of other energy alternatives and specifically LNG, as part of the energy supply for the mine.

21 CAPITAL AND OPERATING COST ESTIMATES

21.1 Capital Cost Summary

The capital cost estimate has been developed to reflect the current level of Project definition and incorporates a combination of benchmark data, quantity-based estimating, vendor and contractor input, and first-principles calculations, as appropriate to each cost area. The estimate also draws on experience from comparable mining projects and operating sites.

At this stage of the study, detailed engineering has not been completed for all Project components. Accordingly, the estimate includes assumptions, allowances, and factors to account for items that remain to be further defined during subsequent phases of engineering. Final procurement and construction costs may therefore vary from the values presented as the design is advanced and market conditions are confirmed.

For the purposes of this study, the estimate has been classified as an AACE Class 5 estimate and has been developed to a target accuracy of approximately $\pm 30\%$. This level of accuracy is considered appropriate for the current degree of engineering definition and the intended use of the estimate.

Capital costs are summarized in Table 21-1.

Table 21-1: Capital Cost Summary

Capital Costs	Pre-Production (\$M)	Sustaining / Closure (\$M)	Total (\$M)
Site Development	5.7	-	5.7
Mining	64.6	330.3	394.9
Mineral Processing	66.0	-	66.0
Tailings & Waste Rock Management	15.4	2.9	18.4
On-site Infrastructure	44.7	46.9	91.6
Off-site Infrastructure	205.2	-	205.2
Project Indirects	114.9	5.0	119.9
Engineering & EPCM	41.2	6.5	47.7
Owner's Costs	17.5	-	17.5
Closure	-	49.0	49.0
Subtotal	575.3	440.6	1,015.9
Contingency	91.8	42.5	134.3
Total Capital Costs	667.1	483.1	1,150.2

21.1.1 Basis of Estimate

The capital cost estimate includes all costs to develop the site and sustain the operation. The cost estimate consists of Direct, Indirect, Owner's, Contingency costs and Growth Allowances:

- Direct costs: Costs of all permanent equipment and bulk materials and the installation costs for all permanent facilities including contractor's supervision and management costs, and contractor's administration and profits;
- Indirect costs: Costs of construction support labour, support equipment, field procurement, field indirects, camp & catering, temporary construction facilities and services, freight and logistics, commissioning and start-up, first fills, spares, and EPCM support;
- Owner's costs: Costs associated with pre-production operating costs, owner's facilities and services during construction, and owner's team project management; and
- Contingency costs: A construction contingency to cover necessary work within the defined scope of the Project which cannot be identified or itemized at this stage of the Project development but is expected to be incurred.

The following key assumptions were made during development of the capital cost estimate:

- The capital cost estimate is based on the development schedule, execution strategy, and key dates described within the Mine Plan included in Section 16 of this Technical Report;
- All construction (civil, structural, architectural, mechanical, piping, electrical, and instrumentation) will be performed by Contractors; and
- Contractors would mobilize only once to carry out their work and are not already mobilized on site performing other work.

The following key parameters apply to the capital cost estimate:

- Estimate Class: The capital cost estimate is considered an AACE Class 5 cost estimate with a target accuracy of $\pm 30\%$;
- Estimate Base Date: The base date of the capital cost estimate is Q3 2026. 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 cost estimate; and
- Currency: Costs are expressed in C\$ and do not include allowances for exchange rate fluctuations unless stated otherwise. A foreign exchange rate of C\$1.00:US\$0.72 is used where applicable.

21.1.2 Capital Cost 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 Property schedule;
- Warehouse inventories, other than those supplied in initial fills, capital spares, or commissioning spares;
- Any Property sunk costs (studies, exploration programs, existing access roads, etc.);
- Closure bonding; and
- Escalation cost.

21.1.3 Mine Capital Cost Estimate

The capital cost estimate for the Project was based on the mine design and production schedule, including one year of pre-production mine development, a ramp-up to a nominal throughput of 2,400 t/d and a 22-year operating mine life.

The mining estimate was broken into three primary components: mine development and associated costs, underground mine infrastructure and underground mining equipment.

The mine design utilized existing underground workings where practical to reduce the amount of new development required. Existing underground workings will be widened to design to allow for mining equipment to fit.

For lateral and vertical development capital cost estimate, underground mine solids were assigned profiles and attributes based on their development purpose. Unit development costs were then estimated on a per metre basis for each development type, with rates varying primarily according to excavation dimensions and profile. Lateral development dimensions ranged from approximately 4.0 m × 4.0 m arched headings to 5.0 m × 5.0 m arched headings, while vertical development was predominantly based on circular openings with a diameter of 2.3 m.

Development was scheduled on a quarterly basis from pre-production to the end of the LOM. The associated development quantities and costs were then aggregated by year to generate a

detailed annual capital cost estimate, with costs separated between the pre-production and production periods.

The primary underground mining cost categories included mobile equipment, ventilation, dewatering and water supply, and compressed air.

Mine air heating requirements were estimated based on underground ventilation airflow and seasonal temperature conditions. Fuel costs were estimated from annual equipment operating requirements and associated diesel consumption.

Mine labour costs were developed from annual staffing requirements and position specific, fully burdened labour rates. These rates were benchmarked against similar projects and employment market data, with consideration given to the remote nature of the Project. Workforce requirements were linked to the mine schedule and equipment fleet, with the labour model predominantly assuming a fly-in fly out rotation that required multiple employees per each continuously staffed position. Mining labour includes mining supervisors and foremen, technical services personnel, drill and blast, loading and hauling, mine support services and underground backfill operations. Capital haulage costs were estimated from scheduled capital development waste tonnes with location specific haulage distances, with rates based on equipment cycle times, productivity assumptions and operating costs.

These costs were allocated between capital and operating based on the proportion of development activity classified as capital versus operating development throughout the mine schedule. The resulting mine development and associated capital costs are summarized in Table 21-2.

Table 21-2: Mine Development and Associated Costs

Mining Capital Costs	Pre-Production (\$M)	Sustaining (\$M)	Total (\$M)
Lateral Capital Development	9.6	77.0	86.6
Vertical Capital Development	3.0	13.6	16.6
Power	4.7	16.8	21.5
Heating	3.3	9.7	13.0
Fuel	1.2	8.3	9.5
Labour	15.5	63.0	78.5
Haulage	1.0	9.6	10.6
Subtotal	38.3	197.9	236.3

Underground mine infrastructure Capital Cost Estimate quantities and installation timelines were aligned with the planned development sequence and operational requirements. The infrastructure schedule considered existing inventory and equipment relocation where possible. Unit costs were primarily based on similar underground mining projects, contractor quotes and CostMine data, with the resulting costs summarized in Table 21-3.

Table 21-3: Underground Mine Infrastructure

Underground Mine Infrastructure Capital Costs	Pre-Production (\$M)	Sustaining (\$M)	Total (\$M)
Main Pump Stations	0.4	0.4	0.9
Level Sumps	0.1	3.0	3.1
Main Power Stations	0.6	1.8	2.3
Substations	0.6	2.9	3.5
Main Fans	0.7	0.7	1.3
Level Fans	0.2	0.5	0.6
Mine Air Heaters	0.8	0.8	1.6
Refuge Stations	0.3	1.6	1.9
Portal (rehab)	0.5	-	0.5
Subtotal	4.2	11.6	15.8

Equipment purchase quantities were based on the required fleet by equipment type, including additional units and replacements required over the mine life. Unit purchase costs were primarily based on CostMine data, supplemented by benchmark costs from comparable underground mining projects.

Equipment rebuilds were scheduled based on cumulative operating hours and equipment-specific rebuild and replacement intervals, with rebuild costs estimated at 40% of the applicable equipment purchase cost. Freight was estimated at 5% of equipment purchase costs. The resulting underground mining equipment capital costs are summarized in Table 21-4.

Table 21-4: Underground Mining Equipment

Underground Mine Equipment	Pre-Production (\$M)	Sustaining (\$M)	Total (\$M)
Purchase Cost	21.1	83.2	104.3
Rebuild	-	33.3	33.3
Freight	1.1	4.2	5.2
Subtotal	22.1	120.7	142.8

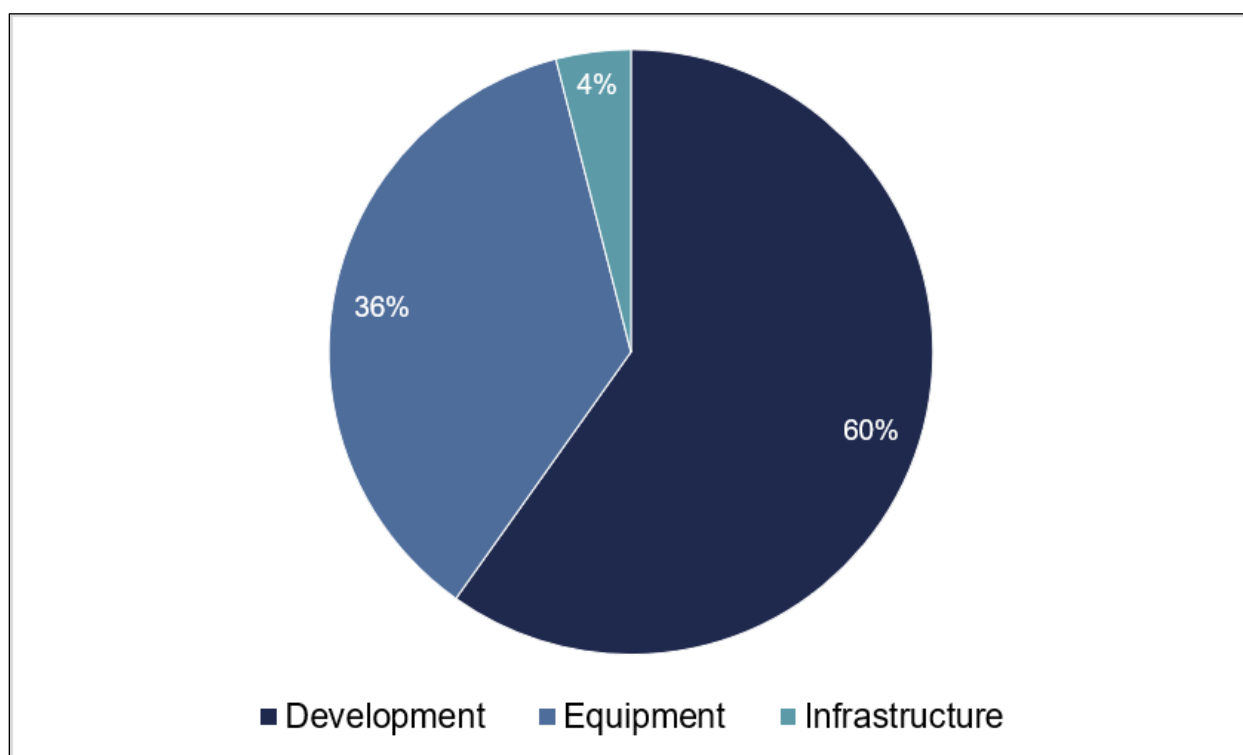
A full breakdown of mining capital costs across the pre-production and sustaining periods is provided in Table 21-5. A 5% contingency has been applied to all mining Capital Cost Estimate items and has been added in Table 21-1.

Table 21-5: Mining Capital Cost Summary

Mining Capital Costs	Pre-Production (\$M)	Sustaining (\$M)	Total (\$M)
Mine Development and Associated Costs	38.3	197.9	236.3
Underground Mine Infrastructure	4.2	11.6	15.8
Underground Mining Equipment	22.1	120.7	142.8
Subtotal	64.6	330.3	394.9

Figure 21-1 illustrates the relative contribution of each cost category to total mining Capital Cost Estimate.

Figure 21-1: Mine Capital Cost Breakdown



21.1.4 Processing Capital Cost Estimate

The process facility capital cost estimate and associated engineering basis for the Project includes definition of the major mechanical, electrical, and civil requirements for the proposed plant refurbishment and new installations. The process plant is based on the refurbishment and reuse of the existing plant where practical, supplemented by new equipment and systems required to support the proposed production rate and updated process flowsheet. The facility includes crushing, grinding, dense media separation, lead flotation, zinc flotation, concentrate dewatering and storage, and associated tailings and process-water handling systems.

The process facility capital cost estimate was developed using the process flow diagrams, design criteria, mechanical equipment list, and layout information prepared for the Project. Budgetary quotations were obtained, where available, for major mechanical and electrical equipment, packaged systems, and pre-engineered buildings. Installation labour requirements were estimated using JDS internal estimating data, historical project experience, and benchmarks from comparable mineral processing facilities.

The process facility capital costs are summarized in Table 21-6. Process facility sustaining capital requirements are included within the operating cost estimate and are therefore not presented as a separate sustaining capital allowance in this section.

Table 21-6: PC Silver Process Facility Capital Costs

Process Capital	Pre-Production (\$M)	Total (\$M)
Crushing, Screening & Handling	4.4	4.4
Dense Media Separation	11.1	11.1
Grinding	8.1	8.1
Flotation	3.5	3.5
Dewatering	2.5	2.5
Tailings	4.0	4.0
Paste Plant	17.6	17.6
Reagents & Plant Auxiliary	3.2	3.2
Process Plant Building & Utilities	11.5	11.5
Total Processing	66.0	66.0

21.1.5 Tailings and Waste Management Cost Estimate

The tailings and waste management capital cost estimate is based on PEA-level designs, facility layouts, and material quantities. The estimate was developed using unit rates primarily sourced from JDS' internal cost database and relevant project experience.

The estimate includes the pre-production and sustaining capital requirements associated with the development and ongoing management of the tailings and waste rock facilities. The tailings and waste management capital costs are summarized in Table 21-7.

Table 21-7: Tailings and Waste Management Capital Costs

Tailings & Waste Rock Storage Capital	Pre-Production (\$M)	Sustaining (\$M)	Total (\$M)
Total Tailings and Waste Management	15.4	2.9	18.4

21.1.6 Infrastructure Capital Cost Estimate

Infrastructure capital costs for the PC Silver Mine include on-site development, on-site infrastructure, and off-site infrastructure required to support construction and ongoing mine operations. The estimate reflects the brownfield nature of the Project, with certain existing facilities retained and reused, others refurbished or upgraded, and selected facilities replaced or constructed new where the existing infrastructure is not suitable for the proposed operating requirements.

On-site development includes the earthworks, site preparation, roads and laydown areas, surface water management works, airstrip-related development, and site services distribution required to support the proposed mine and process facilities. The scope incorporates both modifications to existing developed areas and new works required to accommodate the updated site layout.

On-site infrastructure includes the facilities and utilities required to support mine and process operations. Existing infrastructure has been assessed for continued use, refurbishment, replacement, or expansion depending on its condition and suitability for the proposed operating basis. The scope includes power generation and distribution, accommodation and administration facilities, mine dry and maintenance facilities, warehouses, water supply and treatment systems, wastewater treatment, fuel storage and distribution, waste management facilities, emergency response facilities, and other ancillary site services. Where practical, existing facilities are retained or refurbished; where existing systems are inadequate or no longer suitable, new facilities have been included in the capital cost estimate.

Off-site infrastructure includes the transportation and access infrastructure required to connect the Project to external supply routes. This includes the ASR, river crossing and barge-related infrastructure, and other supporting logistics infrastructure outside the main mine site. The

estimate considers rehabilitation and reuse of existing access infrastructure together with new or upgraded components required to support construction and operations.

Table 21-8 summarizes the basis of estimate for the infrastructure components included in the capital cost estimate.

Table 21-8: Infrastructure Basis of Estimate

Description	Basis of Estimate
Bulk Earthworks, Including On-site Roads	Quantities from site layouts and available design information; JDS unit rates and project benchmarks applied.
Surface Water Management	Quantities and allowances for rehabilitation and new ponds, drainage, ditches, culverts, and associated earthworks.
Power Supply & Distribution	Refurbishment and replacement scope based on vendor pricing and comparable project costs.
Water Supply & Treatment	Allowances and vendor pricing for refurbished, upgraded, and new water systems.
Waste Management	Allowances and benchmark costs for wastewater, solid waste, landfill, and incineration facilities.
Ancillary Buildings	Costs based on refurbishment, replacement, or new construction, using vendor pricing and recent project benchmarks.
Support Mobile Fleet	Fleet quantities based on site and access road requirements; costs from vendor pricing and recent projects.
Bulk Fuel Storage & Distribution	Allowances and benchmark costs for fuel storage, dispensing, and distribution systems.
IT & Communications	Database costs from similar projects.
Airstrip	Rehabilitation and upgrade costs based on available design information and comparable projects.
Off-site Access & Logistics	Costs based on construction quotations for the seasonal access route, river crossing, barge infrastructure, and associated logistics works.

The site development and infrastructure cost estimate are summarized as follows and shown on Table 21-9.

Table 21-9: Site Development and Infrastructure Cost Estimate

Infrastructure Capital	Pre-Production (\$M)	Sustaining (\$M)	Total
On-site Development			
Bulk Earthworks (Pads)	1.0	-	1.0
On-site Roads	0.5	-	0.5
Surface Water Management	1.2	-	1.2
Airstrip	1.0	-	1.0
Infrastructure	2.1	-	2.1
Total On-site Development	5.7	-	5.7
On-site Infrastructure			
Power Supply & Distribution	5.4	30.6	36.0
Water Supply, Distribution & Treatment	7.5	-	7.5
Waste Management	1.4	-	1.4
Accommodation & Offices	12.9	-	12.9
Ancillary Buildings	6.3	16.3	22.6
Support Mobile Fleet	8.4	-	8.4
Bulk Fuel Storage & Distribution	1.5	-	1.5
IT & Communications	1.3	-	1.3
Total On-site Infrastructure	44.7	46.9	91.6
Off-site Infrastructure			
Road Construction	116.2	-	116.2
River Crossings	65.6	-	65.6
Road Infrastructure	3.4	-	3.4
Winter Road Construction	20.0	-	20.0
Off-site Infrastructure	205.2	0.0	205.2
Total	255.6	46.9	302.5

21.1.7 Indirect Capital Cost Estimate

Indirect capital costs include project costs that are not directly attributable to a specific facility or work package. At the current level of study, these costs have been estimated primarily as factors applied to direct capital costs associated with all areas except for mining, using allowances and benchmarks from comparable projects.

Table 21-10 summarizes the indirect cost factors applied in the capital cost estimate.

Table 21-10: Indirect Costs Basis of Estimate

Description	Factor (%)	Description
Indirects		
Construction Support Equipment	0.5	Includes crane hire, etc. Site service support equipment is accounted for as Owner's costs, and discipline specific support equipment is included in direct costs
Construction Support Labour	3.0	Includes QA/QC, survey, scaffolding, etc.
Construction Materials Procurement	1.5	Field procurement
Contractor Field Indirects	5.0	Contractor mobilization/demobilization
Camp & Catering	-	\$95/day, project estimated at 150,000 man-days
Temporary Utilities	3.5	Temporary power, construction fuel and propane
Freight & Logistics	11	Factor applied for all non mining equipment related freight costs to site costs
Temporary Construction Facilities	2.3	Offices and 200 temporary beds to support construction
Start-up & Commissioning	6.0	Commissioning, vendor reps, capital spares and first fills. Mechanical equipment list and mining equipment capital spares and first fills have been included in indirects
EPCM		
Engineering & Major Procurement	4.0	Database costs from similar projects
Construction & Project Management	9.0	Database costs from similar projects

The indirect capital cost estimate is summarized as follows and shown on Table 21-11. Indirect costs related to pre-production capital results in \$114.9 million (or 29% of direct pre-production capital) and EPCM related costs result in \$41.2 million (or 10% of direct pre-production capital). Indirect sustaining and closure capital costs are entirely associated with tailings and waste management facilities.

Table 21-11: Indirect Capital Cost Estimate

Indirect & EPCM Capital	Pre-Production (\$M)	Sustaining (\$M)	Total (\$M)
Indirects			
Construction Support Contracts	15.8	5.0	20.8
Contractor Field Indirects	15.8	-	15.8
Camp Catering	15.0	-	15.0
Temporary Utilities/Supplies	11.1	-	11.1
Freight & Logistics	31.0	-	31.0

Indirect & EPCM Capital	Pre-Production (\$M)	Sustaining (\$M)	Total (\$M)
Temporary Construction Facilities	7.1	-	7.1
Startup & Commissioning	19.0	-	19.0
Total Indirects	114.9	5.0	119.9
EPCM			
Engineering & Procurement	12.7	2.0	14.7
Construction & Project Management	28.5	4.5	33.0
Total EPCM	41.2	6.5	47.7
Total Indirects & EPCM	156.1	11.5	167.6

21.1.8 Owners Cost Estimate

Owner's costs are included within the operating costs during production, but during the construction period these items are included in the initial capital costs and are capitalized:

- Pre-Production Mining: Costs for the Owner's mining labour and equipment operation are estimated from the mining cost model and are held within the capital cost estimate for pre-production development;
- Pre-Production Processing: Costs associated with the ramp-up period prior to full production, including process plant labour, power, and consumables.
- Pre-production General & Administration: Costs of the Owner's labour (admin, accounting, HR, community relations, safety, environmental, IT, procurement, warehouse, security, etc.) and expenses (admin, legal, insurance, recruitment, community relations, safety, environmental, IT, transport etc.) incurred prior to commercial production; and
- Owner's Project team: Costs of the Owner's Project team labour specific to the Project development period.

Table 21-12 presents the owner's costs within the capital cost estimate.

Table 21-12: Owner's Cost Estimate

Owner's Capital	Pre-Production (\$M)	Sustaining (\$M)	Total (\$M)
Mining (carried under mining capital cost estimate)	-	-	-
Processing	7.0	-	7.0
G&A and Owner's Project Team	10.6	-	10.6
Total	17.6	-	17.6

21.1.9 Closure Cost Estimate

The closure cost estimate includes direct costs for mine and site closure and reclamation, together with separate allowances for tailings management facility and waste rock storage facility, closure indirect, and contingency. The general closure allowance of \$42.1 million is based on closure cost requirements developed through the Mackenzie Valley Land and Water Board regulatory process, rather than the lower internal estimates previously prepared for the Project.

An additional \$1.6 million has been included for tailings management facility and waste rock pile closure and reclamation based on the current Project design. Closure indirect costs include allowances for EPCM, Owner's G&A, and construction management. A 30% contingency has been applied to reflect the current level of closure planning and design definition. No escalation has been included. The resulting total closure cost is approximately \$63.7 million.

21.1.10 Contingency

Contingency is a provision of funds for unforeseen or inestimable costs within the defined Project scope relating to the level of engineering effort undertaken and estimate/engineering accuracy. Contingency is meant to cover events or incidents that occur during the course of the Project, which cannot be quantified during the estimate preparation and do not include any allowance for Project risk. No provision is made, or contingency allowed, for design changes or changes to the scope of work.

Individual contingency rates were applied to each of the capital cost categories, to reflect the level of detail in engineering design and associated pricing and quotes. This resulted in a blended contingency rate of 16.0%, or \$91.8 million, in contingency for the initial Capital Cost Estimate. Contingency was applied to sustaining costs at a blended overall rate of 9.6%, or a total of \$42.6 million which includes a 30% contingency applied to the closure capital costs.

Contingency for pre-production capital costs on the Project were applied as outlined in Table 21-13.

Table 21-13: Initial Capital Cost Estimate Contingency Cost Estimate

Capital Cost Area	Contingency (%)	Pre-Production (\$M)
Underground Mining	5	3.2
On-site Development	15	0.9
Mineral Processing	25	16.5
Tailings and Waste Management	25	3.9
On-site Infrastructure	20	8.9
Off-site Infrastructure	20	41.0
Indirects & Owner's Costs	10	17.4
Total	16	91.8

21.2 Operating Cost Summary

The operating cost estimate was prepared using first principles, applying project experience, and avoiding the use of general industry factors. Inputs are derived from engineers, contractors, and suppliers who have provided similar services to other projects. The operating cost is based on owner mining and minimal use of permanent contractors except where value is provided through expertise and/or packaged efficiencies/skills.

Operating costs in this section of the report include mining, processing, tailings, and administration up to the production of concentrate from the site. Mine operating costs incurred during the construction phase (pre-production Years -3,-2 and -1) are capitalized and form part of the capital cost estimate. Concentrate transportation, treatment and refining charges, and royalties are discussed in Section 19. Total operating costs over LOM are \$3,101 million or \$166.83 per tonne mined as shown in Table 21-14.

Table 21-14: Breakdown of Estimated Operating Costs

Operating Costs	\$/t Milled	LOM \$M
Mining	88.99	1,654.1
Processing	39.33	731.0
G&A	38.52	715.9
Total	166.83	3,101.1

Table 21-15: Summary of Personnel during Production

Position	Staff/Hourly	Quantity
Operations & Maintenance		
Mining Operations	Staff/Hourly	85
Mine Maintenance	Staff/Hourly	22
Technical Services	Staff	13
Total Mining	Staff/Hourly	120
Total Process Plant	Staff/Hourly	82
Subtotal - Operations & Maintenance	Staff/Hourly	202
G&A		
Subtotal - Administration	Staff	4
Subtotal – Accounting	Staff	4
Subtotal - Human Resources	Staff	3
Subtotal – Community Relations	Staff	2
Subtotal - Health & Safety	Staff	6
Subtotal – Environment	Staff	4
Subtotal - IT & Communications	Staff	2
Subtotal – Procurement and Logistics	Hourly	6
Subtotal – Site Services: Surface Operations	Hourly	25
Subtotal – Site Services: Road Operations	Staff	8
Subtotal – Site Services: Winter Road Construction	Hourly	2
Subtotal – Camp Catering and Cleaning	Hourly	26
Subtotal – Water Treatment Plant	Staff/Hourly	10
Subtotal – Power Plant	Staff/Hourly	4
Total General & Administration	Staff/Hourly	106
Total- EPCM & Construction Indirects		24
Total Personnel - All Areas		332

21.2.1 Basis of Estimate

The operating cost estimate was developed from several sources including quoted costs, database information, and costs from similar operations in Northern Canada. Mining costs were built up from first principles based on the mine plan. Process cost determinations were based on fixed and variable components relating to mill throughput and the plant flow sheet. G&A costs were built up from first principles utilizing information from similar projects in Northern Canada.

Operating costs are presented in 2026 Canadian dollars and do not include allowances for escalation or exchange rate fluctuations unless stated otherwise. A foreign exchange rate of C\$1.00:US\$0.72 is used where applicable. The estimate conforms to $\pm 30\%$ accuracy standards required for a PEA-level estimate.

Key operating cost component assumptions are listed in Table 21-16.

Table 21-16: Summary of Key Operating Cost Assumptions

Item	Unit	Value
Power cost (generated, Y-1 -Y2)	\$/kWh	0.459
Power cost (generated, Y3+)	\$/kWh	0.350
Diesel cost (delivered)	\$/litre	1.61

21.2.2 Mine Operating Cost Estimate

Mining costs were estimated over the life of the mine, starting in Year 1 and continuing through Year 22, based on the scheduled underground mining activities.

The estimate assumes an owner-operator mining model, where Honey Badger Silver employs the workforce and manages the underground mining operations.

Mining costs were benchmarked against comparable Canadian underground mining operations, with contractor quotations and current market data applied where available for key cost inputs such as diesel, propane and power.

The operating cost estimate includes labour, power, heating, fuel, consumables, equipment usage, supplies, and the costs associated with underground development, production, haulage and backfill activities.

Table 21-17 summarizes the key components of the mining operating cost estimate, which total LOM expenditures of \$1,654 million and an average unit cost of \$89 per milled tonne.

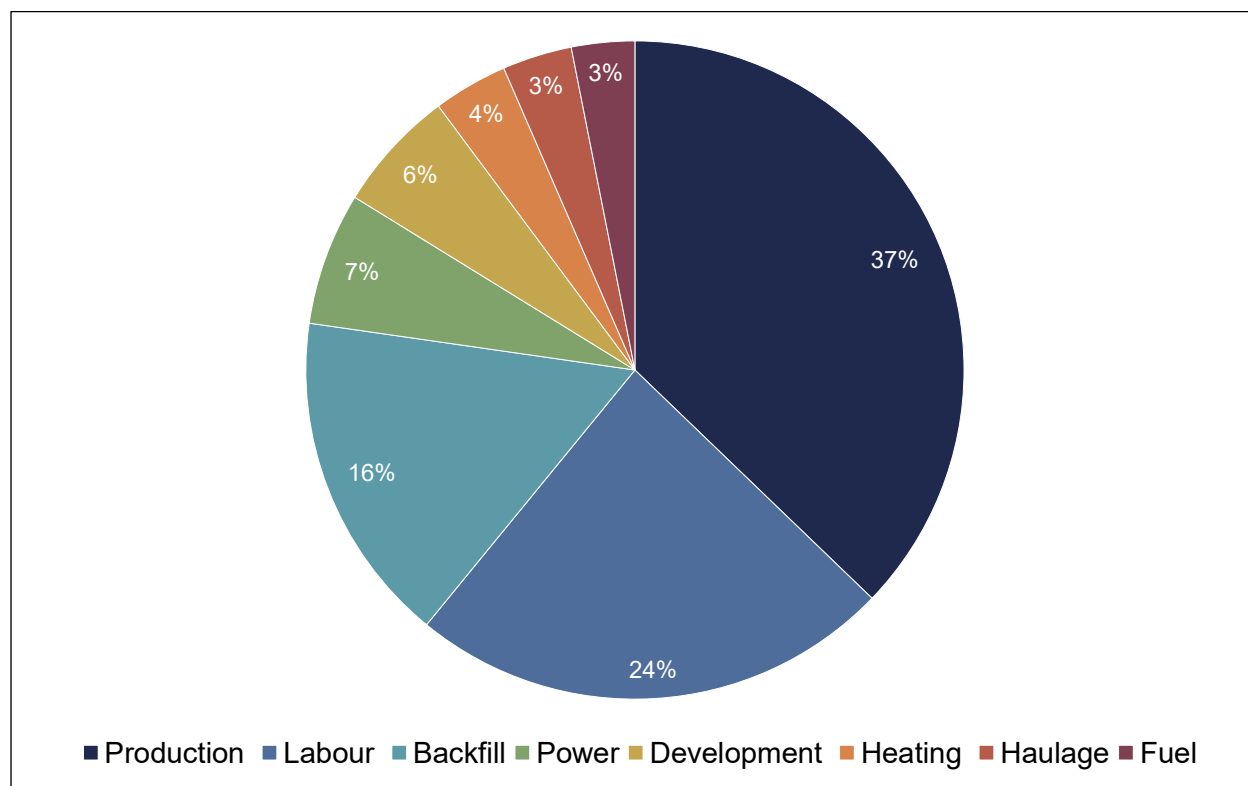
Table 21-17: Total Mining Operating Expenditures

Operating Costs	\$/t Milled	LOM \$M
Development	5.36	99.6
Production	33.10	615.2
Haulage	3.04	56.6
Backfill	14.56	270.6

Operating Costs	\$/t Milled	LOM \$M
Power	5.80	107.8
Heating	3.26	60.6
Fuel	2.76	51.3
Labour	21.11	392.3
Total Operating Cost Estimate	89.00	1,654

Figure 21-2 illustrates the relative contribution of each cost category to total mining Operating Cost Estimate.

Figure 21-2: Mine Operating Cost Breakdown



21.2.3 Processing Operating Cost Estimate

Process operating costs were estimated for the 2,400 t/d process facility which utilizes crushing, dense media separation, grinding and differential flotation to produce 2 concentrates: a lead concentrate and a zinc concentrate. Processing costs were estimated on an annual basis in the following functional areas:

- Labour - Process plant labour costs include mill management, mill operations, mill maintenance, and mill technical services (including assay lab). The labour rates are drawn from the JDS database on current and historical labour costs;
- Power and Fuel - Power costs calculated from the estimated mill power draw, diesel power generation, and an allowance for fuel for mobile equipment;
- Maintenance – Maintenance costs include wear items such as mill and crusher liners, and a maintenance factor of liner pricing and vendor recommended spare parts for one year of operation, and plant maintenance costs calculated by applying a factor of 5% per year to the expected mill capital cost;
- Operations - Grinding media and reagent costs developed from metallurgical test results for consumptions and a combination of recent vendor quotes and prices in the JDS database; and
- Assay Lab - Allowance for annual operation.

The mill operating costs are estimated to be \$39.33/t processed based on 2,400 t/d feeding the crusher and dense media separation plant. The breakdown of the costs can be found in Table 21-18.

Table 21-18: Annual Processing Operating Cost

Area	Total (\$M)	Unit Cost (\$/tonne)	% of Total
Labour	13.0	14.95	38
Power & Fuel	7.8	8.97	23
Maintenance	5.2	6.02	15
Operations	7.6	8.76	22
Assay Lab	0.6	0.63	2
Total	34.2	39.33	100

21.2.4 General and Administration Operating Cost Estimate

The costs of general and administrative expenses include two main components:

- Labour staff for the following areas: administration, accounting, human resources, community relations, health & safety, environmental, IT, security, maintenance consumables & parts, equipment operation, camp catering & cleaning, transportation to site, power consumption; and
- Expenses for the following areas: administration, accounting, human resources, community relations, health & safety, environmental, IT, security, maintenance consumables & parts, equipment operation, camp catering & cleaning, transportation to site, power consumption.

Average camp requirements are expected to be approximately 200 people. The breakdown of G&A operating costs is presented in Table 21-19.

Table 21-19: Summary of General & Administrative Operating Cost Estimate

G&A Operating Cost	Average Annual (\$M)	LOM Cost (\$M)	Operating Cost Estimate (\$/tonne)	Total (%)
Management & Administration	2.2	48.1	2.5	7
Accounting	0.6	13.1	0.68	2
Human Resources	0.9	20.3	1.08	3
Community Relations	0.9	19.4	1.03	3
Health & Safety	1.2	26.1	1.36	4
Environment	1.1	23.0	1.20	3
IT & Communications	0.4	8.9	0.47	1
Procurement & Logistics	1.2	25.7	1.36	4
Security	0.1	1.7	0.10	0
Camp Catering & Cleaning	6.8	150.7	7.80	20
Personnel Transportation	5.0	108.9	5.70	15
Water Treatment Plant Operations	2.3	49.5	2.62	7
Site Electrical Power Consumption	3.1	68.9	3.50	9
Site Services & Road Maintenance	7.5	163.6	8.56	22
Total	33.2	727.8	37.93	100

22 ECONOMIC ANALYSIS

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 calculated during operations and, as such, the after-tax results are only approximations.

Univariate sensitivity analyses were performed for variations in metal prices, head grades, 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 of this Technical Report (presented in 2026 Canadian dollars). The economic analysis has been run on a constant dollar basis, with no inflation.

22.1 Summary of Results

The summary of the mine plan and payable metals produced is outlined in Table 22-1.

Table 22-1: LOM Summary

Parameter	Unit	Value
Mine Life	Years	22
Resource Mined	Mt	18.6
Silver (Ag) Grade	g/t	112
Lead (Pb) Grade	%	6.1
Zinc (Zn) Grade	%	8.5
Copper (Cu) Grade	%	0.253
Processing Rate (pre-sorter)	t/d	2,400
Processing Rate (post-sorter)	t/d	1,500
Silver (Ag) Payable	koz	56,063
	koz/a	2,548

Parameter	Unit	Value
Lead (Pb) Payable	MIbs	2,080
	MIbs/a	94.5
Zinc (Zn) Payable	MIbs	2,577
	MIbs/a	117.2
Copper (Cu) Payable	MIbs	29.5
	MIbs/a	1.3

It must be noted that 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 the preliminary economic assessment will be realized.

22.2 Basis of Analysis

The economic analysis was based on the following factors:

- Discount rate of 8%;
- Nominal 2026 dollars;
- Revenues, costs, and taxes are calculated for each period in which they occur rather than actual outgoing/incoming payment;
- Results are based on 100% ownership;
- No management fees or financing costs (equity fund-raising was assumed); and
- 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 22-2 outlines the metal prices and exchange rate assumptions used in the economic analysis. The silver prices selected were loosely based on consensus metals pricing as of August 2026.

Table 22-2: Metal Prices and Exchange Rates

Parameter	Unit	Value
Silver (Ag) Price	US\$/oz	50.00
Lead (Pb) Price	US\$/lb	0.90
Zinc (Zn) Price	US\$/lb	1.50
Copper (Cu) Price	US\$/lb	5.00
FX Rate	CAD\$:US\$	0.72

22.3 Assumptions

Mine revenue is derived from the sale of concentrate into the international marketplace. No contractual agreements for refining currently exist. Table 22-3 indicates the NSR parameters that were used in the economic analysis.

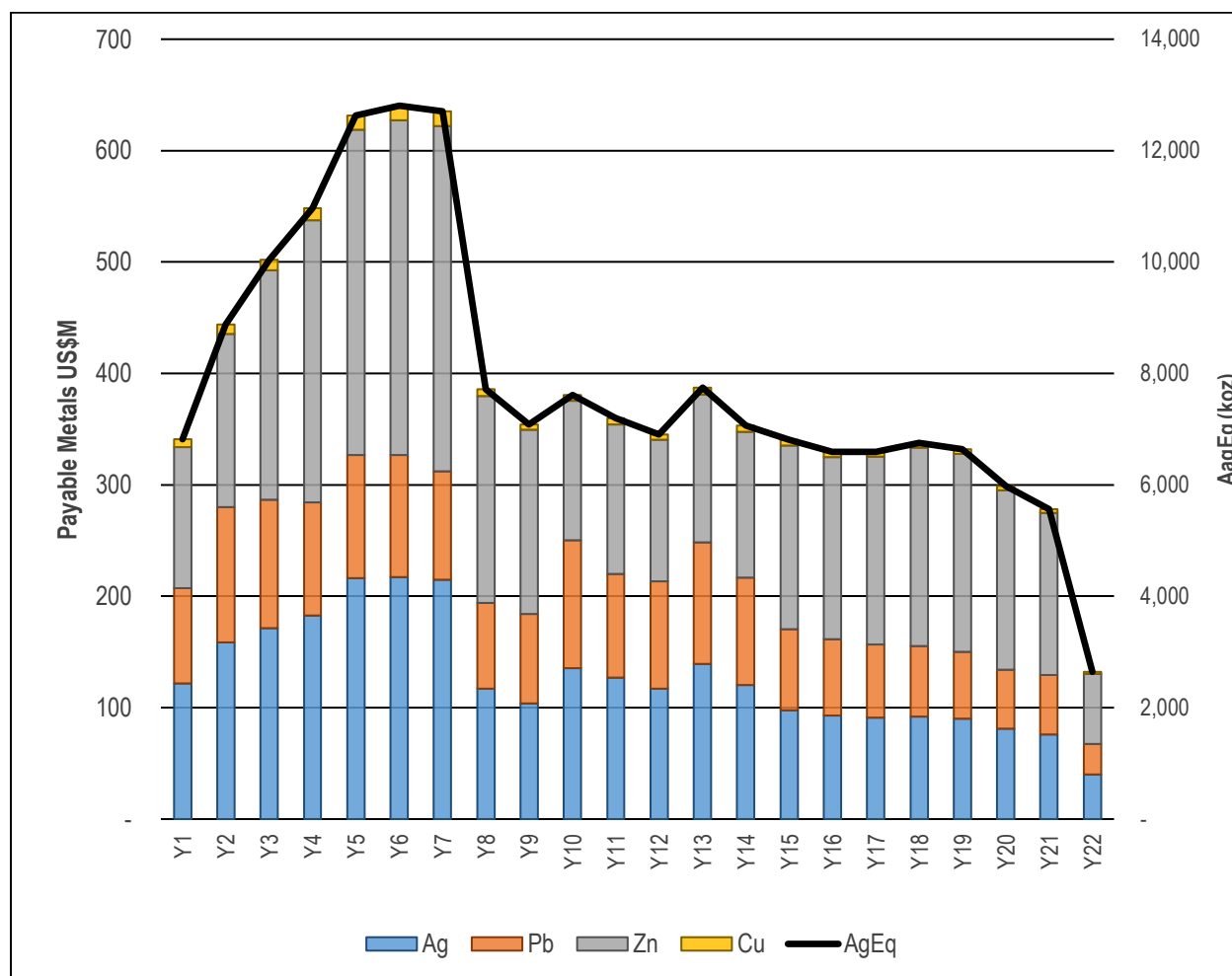
Table 22-3: NSR Parameters

Parameter	Unit	Value
Lead (Pb) Concentrate		
Silver (Ag) Recovery	%	87.9
Lead (Pb) Recovery	%	87.0
Copper (Cu) Recovery	%	95.0
Silver (Ag) Payable	koz	56,063
Lead (Pb) Payable	Mlbs	2,080
Copper (Cu) Payable	Mlbs	29.5
Penalties (Hg)	US\$M	12.8
TCRCs and Transportation	US\$M	701.9
Royalties	US\$M	90.4
Lead (Pb) Concentrate NSR	US\$M	4,017.3
Zinc (Zn) Concentrate		
Silver (Ag) Recovery	%	5.9
Zinc (Zn) Recovery	%	86.7
Silver (Ag) Payable	koz	0.0
Zinc (Zn) Payable	Mlbs	2,577.3
Penalties (Cd & Hg)	US\$M	141.6

Parameter	Unit	Value
TCRCs and Transportation	US\$M	437.8
Royalties	US\$M	58.4
Zinc (Zn) Concentrate NSR	US\$M	2,595.9
Total NSR	US\$M	6,613.2

Figure 22-1 shows the value of the payable metals on an annual basis. A total of 56,063 koz of Silver, 2,080 Mlbs of Lead, 2,577 Mlbs of Zinc, and 29.5 Mlbs of Copper are projected to be produced over the LOM.

Figure 22-1: Annual Payable Metals



22.4 Taxes

The property 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 from first principles for this evaluation. Current tax pools were used in the analysis. The tax model contains the following assumptions:

- Federal Income Tax: 15.0%;
- Northwest Territories Income Tax Rate: 11.5%;
- Northwest Territories Mining Royalties were assessed on a marginal rate structure based on designated taxable income thresholds; and
- Capital cost allowance applied on units of production basis and at specific rates in the *Tax Act*.

The total amount of taxes applied to the Project amount to \$1,762 million.

22.5 Results

The results for the PC Silver Mine PEA show a post-tax IRR of 29.3% and a Net Present Value using an 8% discount rate (NPV_{8%}) of \$1,172 million using the metal prices described in Section 22.2. Figure 22-2 shows the projected cash flows, and Table 22-4 summarizes the economic results of the Project.

The LOM all-in sustaining cost (AISC) is US\$ 26.79/AgEq oz. The AISC cost is calculated by adding the refining, royalty, operating, sustaining, and closure costs together and then dividing that number by the total payable AgEq ounces.

At base case pricing, Zinc provides 44% of the Project revenues, followed by Silver (32%), then Lead (22%).

This economic assessment is preliminary in nature and includes the use of inferred mineral resources that are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the preliminary economic assessment will be realized.

Figure 22-2: After-Tax Cashflow

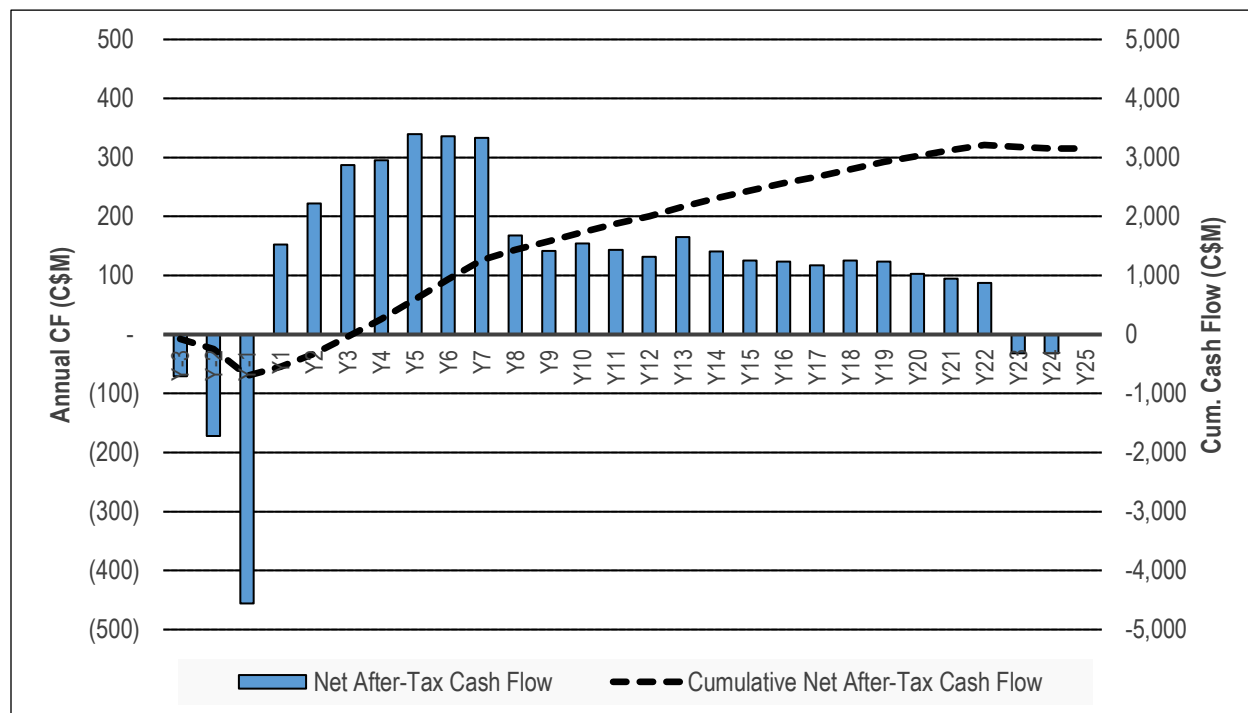


Table 22-4: Summary of Economic Results

Summary of Results	Unit	Value
Operating Costs	US\$	2,233
Offsite Charges	US\$	2,075
Sustaining / Closure	US\$	348
By-Product Credits (Pb, Zn, Cu)	US\$	5,885
Payable Ag	koz	56,063
Payable AgEq***	koz	173,766
AISC* (AgEq)***	US\$/AgEq oz	26.79
AISC (net by-product credits)**	US\$/Ag oz	(21.93)
Capital Costs		
Pre-Production Capital	\$M	575.3
Pre-Production Contingency	\$M	91.8
Total Pre-Production Capital	\$M	667.1
Sustaining and Closure Capital	\$M	440.6
Sustaining and Closure Contingency	\$M	42.5

Summary of Results	Unit	Value
Total Sustaining and Closure Capital	\$M	483.1
Total Capital Costs Incl. Contingency	\$M	1,150.2
Working Capital	\$M	31.7
Pre-Tax Cash Flow	LOM \$M	4,933.8
	\$M /a	224.3
Taxes	LOM \$M	1,761.5
After-Tax Cash Flow	LOM \$M	3,172.3
	\$M /a	144.2
Economic Results		
Pre-Tax NPV _{8%}	\$M	1,915
Pre-Tax IRR	%	37.0
Pre-Tax Payback	Years	2.7
After-Tax NPV _{8%}	\$M	1,172.4
After-Tax IRR	%	29.3
After-Tax Payback	Years	3.1

Note:

*All-in Sustaining Cost is calculated as: (TC/RC + transportation + royalties + operating costs + sustaining and closure capital) / payable oz.

**All-in Sustaining Cost net by-product credits is calculated as: (TC/RC + transportation + royalties + operating costs + sustaining and closure capital less the revenues from Pb, Zn, Cu) / payable oz.

***AgEq oz = Total payable revenues / Silver price

22.6 Sensitivities

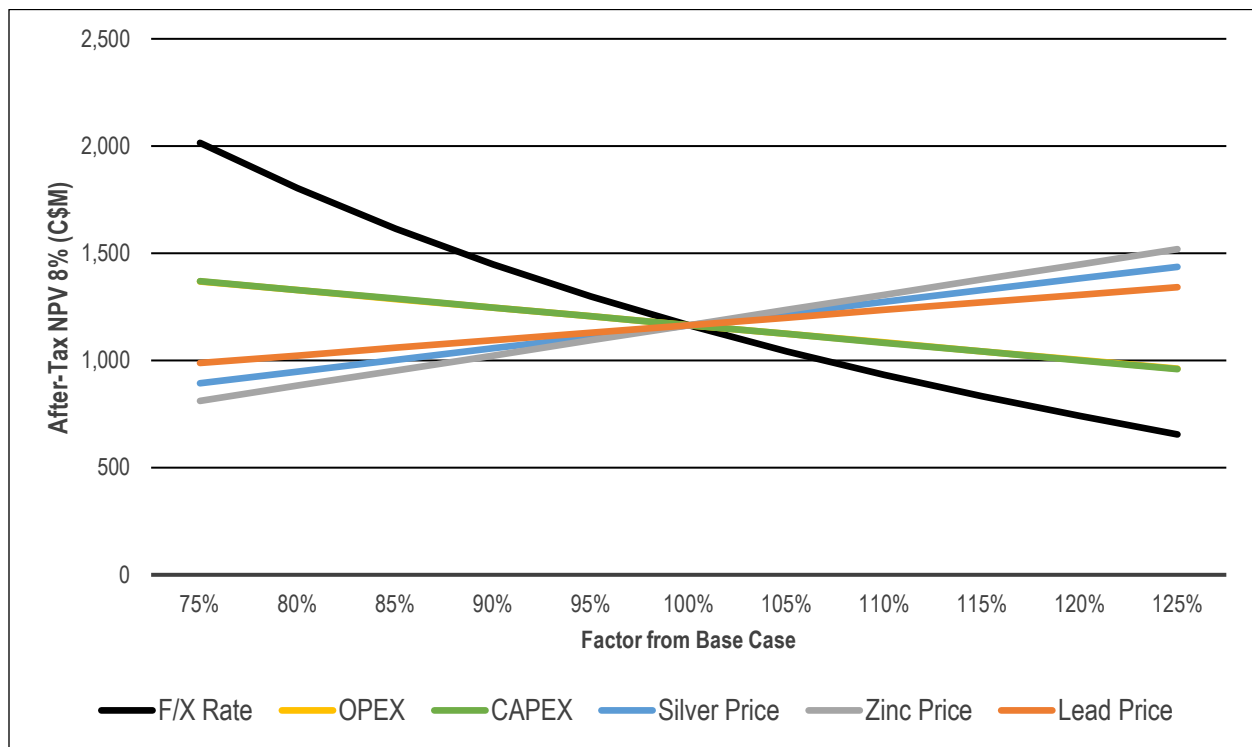
A univariate sensitivity analysis was performed to examine which factors most affect the Project economics when acting independently of all other cost and revenue factors. Each variable evaluated was tested using the same percentage range of variation, from -25% to +25%, although some variables may actually experience significantly larger or smaller percentage fluctuations over the LOM. For instance, the metal prices were evaluated at a +/- 25% range to the base case, while the Capital Cost Estimate and all other variables remained constant. This may not be truly representative of the market scenarios, as metals 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 aforementioned limitations to the analysis, which are common to studies of this sort, the analysis revealed that the Project is most sensitive to exchange rates and zinc price, and the least sensitive to Capital and Operating Cost Estimates. Table 22-5 and Figure 22-3 show the results of the sensitivity tests, while Table 22-6 shows the NPV at various discount rates.

Table 22-5: Univariate Sensitivity Analysis Results

Variable	After-Tax NPV _{8%} (\$M)		
	-25% Variance	0% Variance	+25% Variance
Silver Price	900	1,172	1,446
Lead Price	995	1,172	1,350
Zinc Price	816	1,172	1,529
Foreign Exchange Rate	2,029	1,172	659
Operating Cost Estimate	1,377	1,172	968
Capital Cost Estimate	1,378	1,172	967

Figure 22-3: After-Tax Sensitivity Analysis



The base case results above were further sensitized by comparison to a spot price case, a 12-month high price case and a 36-month low price case. Table 22-6 shows the results of the alternative pricing scenarios.

Table 22-6: Alternative Pricing Scenarios

		Base Case	Spot Prices*	12-month High**	36-month Low***
Pre-Tax Results	NPV _{0%}	4,934	7,300	12,214	778
	NPV _{5%}	2,683	4,054	6,942	272
	NPV _{8%}	1,915	2,951	5,147	93
	IRR	37.0%	48.2%	69.1%	10.1%
	Payback	2.7	2.1	1.3	6.0
Post-Tax Results	NPV _{0%}	3,172	4,678	7,813	528
	NPV _{5%}	1,683	2,559	4,403	136
	NPV _{8%}	1,172	1,836	3,240	-5
	IRR	29.3%	38.2%	54.8%	7.9%
	Payback	3.1	2.4	1.6	6.2

Note:

*Spot prices refers to closing prices on August 31, 2026 (Ag = US\$ 66.41/oz, Pb = US\$ 0.90/lb, Zn = US\$ 1.50/lb, Cu = US\$ 5.00/lb, FX = 0.72).

**12-month high prices refer to the highest closing prices reported between September 2025-September 2026 (Ag = US\$ 121.67/oz, Pb = US\$ 0.95/lb, Zn = US\$ 1.84/lb, Cu = US\$ 6.92/lb, FX = 0.7019)

***36-month low prices refer to the lowest closing prices reported between September 2023-September 2026 (Ag = US\$ 21.59/oz, Pb = US\$ 0.85/lb, Zn = US\$ 1.04/lb, Cu = US\$ 3.36/lb, FX = 0.7450)

Table 22-7: Discount Rate Sensitivity using Base Case Pricing

Discount Rate (%)	Pre-Tax NPV (\$M)	Post-Tax NPV (\$M)
0	4,934	3,172
5	2,683	1,683
6	2,393	1,490
7	2,139	1,321
8	1,915	1,172
10	1,541	924
12	1,246	726

The economic cash flow model for the Project is illustrated in Table 22-8 below:



Table 22-8: Economic Cash Flow Model

	Unit	LOM Total	Y-3	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14	Y15	Y16	Y17	Y18	Y19	Y20	Y21	Y22	Y23	Y24	
		Tax Year-->	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039	2040	2041	2042	2043	2044	2045	2046	2047	2048	2049	2050	2051	2052	2053	
MINE PRODUCTION																														
Ore Mined	ktonnes	18,588	-	-	-	739	876	876	876	876	876	876	876	876	876	876	876	876	876	876	876	876	876	876	876	876	329	-	-	
Ag	g/t	112	-	-	-	146	145	145	159	177	178	176	98	87	110	103	96	116	101	82	79	79	81	81	73	65	87	-	-	
Pb	%	6.1	-	-	-	8.4	9.1	8.0	7.2	7.4	7.3	6.5	5.3	5.5	7.6	6.2	6.4	7.4	6.6	5.0	4.7	4.6	4.5	4.4	3.9	3.8	4.8	-	-	
Zn	%	8.5	-	-	-	8.4	8.0	9.8	12.0	13.2	13.5	13.9	8.4	7.5	5.7	6.1	5.9	6.3	6.3	7.8	7.7	8.0	8.3	8.2	7.5	6.7	7.6	-	-	
Cu	%	0.253	-	-	-	0.290	0.313	0.342	0.403	0.471	0.478	0.485	0.237	0.175	0.194	0.218	0.177	0.225	0.217	0.189	0.175	0.166	0.157	0.154	0.138	0.131	0.194	-	-	
Contained Metal																														
Ag	kg	2,090,517	-	-	-	107,829	126,929	127,025	138,925	155,095	155,629	154,340	86,072	76,551	96,685	90,454	84,418	101,666	88,114	71,598	68,988	69,202	70,627	70,580	64,036	57,274	28,480	-	-	
Pb	ktonnes	1,141.0	-	-	-	62.0	79.3	69.7	63.2	64.6	64.0	57.1	46.6	48.3	66.2	54.3	56.5	65.0	57.6	43.8	41.4	40.5	39.4	38.3	34.1	33.0	15.9	-	-	
Zn	ktonnes	1,583.2	-	-	-	61.8	70.2	86.1	105.3	115.3	118.6	122.0	73.5	65.6	49.8	53.2	51.5	55.4	55.1	68.0	67.6	69.9	72.8	71.9	65.7	58.8	24.8	-	-	
Cu	ktonnes	47.0	-	-	-	2.1	2.7	3.0	3.5	4.1	4.2	4.2	2.1	1.5	1.7	1.9	1.6	2.0	1.9	1.7	1.5	1.5	1.4	1.4	1.2	1.1	0.6	-	-	
MILL PRODUCTION																														
Ore Milled	ktonnes	18,588	-	-	-	739	876	876	876	876	876	876	876	876	876	876	876	876	876	876	876	876	876	876	876	876	876	329	-	-
Dense Media Separation Mass Rejection	%		-	-	-	25.0	37.0	37.0	37.0	37.0	37.0	37.0	37.0	37.0	37.0	37.0	37.0	37.0	37.0	37.0	37.0	37.0	37.0	37.0	37.0	37.0	37.0	-	-	
Contained Metal																														
Ag	kg	2,036,164	-	-	-	105,026	123,629	123,722	135,313	151,062	151,583	150,327	83,835	74,561	94,171	88,102	82,223	99,023	85,823	69,737	67,194	67,403	68,791	68,745	62,371	55,785	27,739	-	-	
Pb	ktonnes	1,112.5	-	-	-	60.45	77.37	67.92	61.62	63.03	62.40	55.67	45.48	47.13	64.54	52.97	55.06	63.40	56.20	42.71	40.40	39.48	38.42	37.36	33.22	32.21	15.46	-	-	
Zn	ktonnes	1,513.6	-	-	-	59.09	67.12	82.34	100.69	110.24	113.41	116.66	70.31	62.67	47.64	50.83	49.25	53.01	52.64	65.05	64.66	66.85	69.58	68.77	62.81	56.23	23.75	-	-	
Cu	ktonnes	45.7	-	-	-	2.08	2.66	2.91	3.43	4.01	4.07	4.13	2.02	1.49	1.65	1.86	1.51	1.92	1.85	1.61	1.49	1.41	1.34	1.31	1.18	1.12	0.62	-	-	
NET SMELTER RETURN																														
Pb CONCENTRATE																														
Metal in Pb Concentrate	Ag kg	1,837,565	-	-	-	79,633	104,002	112,250	119,665	141,599	142,267	140,761	76,486	68,172	89,456	83,108	77,002	91,323	79,041	63,910	60,876	59,729	60,336	59,092	53,094	49,587	26,176	-	-	
	Ag koz	59,079	-	-	-	2,560	3,344	3,609	3,847	4,553	4,574	4,526	2,459	2,192	2,876	2,672	2,476	2,936	2,541	2,055	1,957	1,920	1,940	1,900	1,707	1,594	842	-	-	
	Pb ktonnes	992.9	-	-	-	45.4	64.5	61.0	54.0	58.5	58.0	51.6	41.1	42.7	60.7	49.5	51.1	57.9	51.3	38.8	36.3	34.7	33.4	31.8	28.0	28.4	14.5	-	-	
	Pb Mlbs	2,189.1	-	-	-	100.1	142.1	134.5	119.0	129.0	127.9	113.8	90.6	94.1	133.9	109.1	112.6	127.7	113.0	85.5	79.9	76.4	73.6	70.1	61.7	62.5	31.9	-	-	
	Cu ktonnes	44.6	-	-	-	2.0	2.6	2.8	3.4	3.9	4.0	4.0	2.0	1.5	1.6	1.8	1.5	1.9	1.8	1.6	1.5	1.4	1.3	1.3	1.1	1.1	0.6	-	-	
	Cu Mlbs	98.4	-	-	-	4.5	5.7	6.3	7.4	8.6	8.8	8.9	4.4	3.2	3.6	4.0	3.3	4.1	4.0	3.5	3.2	3.0	2.9	2.8	2.5	2.4	1.3	-	-	
Pb Concentrate Produced	dmt	1,654,896	-	-	-	75,657	107,437	101,717	89,956	97,535	96,679	86,046	68,488	71,141	101,201	82,489	85,121	96,524	85,448	64,606	60,422	57,755	55,625	53,018	46,676	47,267	24,088	-	-	
	wmt	1,798,800	-	-	-	82,236	116,779	110,562	97,779	106,016	105,086	93,528	74,444	77,327	110,001	89,662	92,523	104,917	92,878	70,224	65,676	62,778	60,462	57,628	50,735	51,377	26,183	-	-	
Pb Revenues	US\$M	1,871.6	-	-	-	85.6	121.5	115.0	101.7	110.3	109.3	97.3	77.5	80.5	114.5	93.3	96.3	109.2	96.6	73.1	68.3	65.3	62.9	60.0	52.8	53.5	27.2	-	-	



	Unit	LOM Total	Y-3	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14	Y15	Y16	Y17	Y18	Y19	Y20	Y21	Y22	Y23	Y24	
		Tax Year-->	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039	2040	2041	2042	2043	2044	2045	2046	2047	2048	2049	2050	2051	2052	2053	
Cu Revenues	US\$M	147.6	-	-	-	6.7	8.6	9.4	11.1	13.0	13.1	13.3	6.5	4.8	5.3	6.0	4.9	6.2	6.0	5.2	4.8	4.6	4.3	4.3	3.8	3.6	2.0	-	-	
Ag Revenues	US\$M	2,803.2	-	-	-	121.6	158.6	171.4	182.7	216.2	217.3	215.0	116.8	103.9	135.7	126.9	116.9	139.0	120.2	97.5	93.0	91.2	92.1	90.2	81.1	75.7	40.0	-	-	
Concentrate Penalties	US\$M	12.8	-	-	-	0.3	0.4	0.9	1.3	1.7	1.8	2.0	0.7	0.4	0.2	0.4	0.2	0.2	0.2	0.3	0.3	0.3	0.3	0.3	0.3	0.2	0.2	-	-	
Pb Treatment Charge	US\$M	165.5	-	-	-	7.6	10.7	10.2	9.0	9.8	9.7	8.6	6.8	7.1	10.1	8.2	8.5	9.7	8.5	6.5	6.0	5.8	5.6	5.3	4.7	4.7	2.4	-	-	
Pb Concentrate Transport Cost	US\$M	488.1	-	-	-	22.3	31.7	30.0	26.5	28.8	28.5	25.4	20.2	21.0	29.8	24.3	25.1	28.5	25.2	19.1	17.8	17.0	16.4	15.6	13.8	13.9	7.1	-	-	
Cu Concentrate Transport Cost	US\$M	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	-	-	
Cu Refining Charge	US\$M	0.6	-	-	-	0.0	0.0	0.0	0.0	0.1	0.1	0.1	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	-	-
Ag Refining Charge	US\$M	47.7	-	-	-	2.1	2.7	2.9	3.1	3.7	3.7	3.7	2.0	1.8	2.3	2.2	2.0	2.4	2.0	1.7	1.6	1.6	1.6	1.5	1.4	1.3	0.7	-	-	
Versamet NSR Royalty	US\$M	49.3	-	-	-	2.2	2.9	3.0	3.1	3.5	3.6	3.4	2.1	1.9	2.6	2.3	2.2	2.6	2.2	1.8	1.7	1.6	1.6	1.6	1.4	1.4	0.7	-	-	
RCF NSR Royalty	US\$M	41.1	-	-	-	1.8	2.4	2.5	2.6	3.0	3.0	2.9	1.7	1.6	2.1	1.9	1.8	2.1	1.9	1.5	1.4	1.4	1.4	1.3	1.2	1.1	0.6	-	-	
Pb Net Smelter Return	US\$M	4,017.3	0.0	0.0	0.0	177.6	237.7	246.3	249.9	289.0	289.5	279.7	167.3	155.3	208.3	186.9	178.3	209.0	182.7	145.0	137.3	133.5	132.6	128.8	115.0	110.1	57.6	0.0	0.0	
Zn CONCENTRATE																														
Metal in Zn Concentrate	Ag kg	123,869	-	-	-	5,408	7,063	7,624	8,127	9,617	9,662	9,560	5,195	4,630	5,295	5,537	5,230	6,202	5,368	4,340	4,134	4,057	4,098	4,013	3,606	3,368	1,734	-	-	
	Ag koz	3,982	-	-	-	174	227	245	261	309	311	307	167	149	170	178	168	199	173	140	133	130	132	129	116	108	56	-	-	
	Zn ktonnes	1,375.3	-	-	-	45.1	55.2	73.4	90.0	103.9	106.9	110.1	65.9	58.7	44.6	47.6	45.3	47.3	46.4	58.5	58.2	60.0	63.4	63.2	57.4	51.7	22.4	-	-	
	Zn Mlbs	3,032.1	-	-	-	99.5	121.8	161.7	198.3	229.1	235.7	242.8	145.3	129.5	98.3	105.0	99.9	104.3	102.4	129.0	128.2	132.3	139.8	139.3	126.5	114.1	49.4	-	-	
Zn Concentrate Produced	dmt	2,371,291	-	-	-	77,800	95,228	126,488	155,107	179,154	184,353	189,895	113,594	101,250	76,849	82,117	78,094	81,589	80,085	100,890	100,266	103,479	109,334	108,955	98,939	89,213	38,611	-	-	
	wmt	2,577,490	-	-	-	84,566	103,509	137,487	168,595	194,733	200,384	206,407	123,472	110,054	83,532	89,258	84,885	88,684	87,049	109,663	108,985	112,477	118,841	118,429	107,542	96,970	41,969	-	-	
Zn Revenues	US\$M	3,866.0	-	-	-	126.8	155.3	206.2	252.9	292.1	300.6	309.6	185.2	165.1	125.3	133.9	127.3	133.0	130.6	164.5	163.5	168.7	178.2	177.6	161.3	145.4	62.9	-	-	
Concentrate Penalties	US\$M	141.6	-	-	-	4.5	7.6	10.8	13.6	17.7	18.4	18.9	6.9	4.3	5.5	5.7	3.7	4.5	3.8	2.9	2.4	2.0	1.8	1.8	1.8	1.8	1.4	-	-	
Zn Treatment Charge	US\$M	438.7	-	-	-	14.4	17.6	23.4	28.7	33.1	34.1	35.1	21.0	18.7	14.2	15.2	14.4	15.1	14.8	18.7	18.5	19.1	20.2	20.2	18.3	16.5	7.1	-	-	
Zn Concentrate	US\$M	631.4	-	-	-	20.7	25.4	33.7	41.3	47.7	49.1	50.6	30.2	27.0	20.5	21.9	20.8	21.7	21.3	26.9	26.7	27.6	29.1	29.0	26.3	23.8	10.3	-	-	

	Unit	LOM Total	Y-3	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14	Y15	Y16	Y17	Y18	Y19	Y20	Y21	Y22	Y23	Y24
		Tax Year-->	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039	2040	2041	2042	2043	2044	2045	2046	2047	2048	2049	2050	2051	2052	2053
Transport Cost																													
Versamet NSR Royalty	US\$M	31.9	-	-	-	1.0	1.3	1.7	2.0	2.3	2.4	2.5	1.5	1.4	1.0	1.1	1.1	1.1	1.1	1.4	1.4	1.4	1.5	1.5	1.4	1.2	0.5	-	-
RCF NSR Royalty	US\$M	26.5	-	-	-	0.9	1.0	1.4	1.7	1.9	2.0	2.1	1.3	1.2	0.9	0.9	0.9	0.9	0.9	1.2	1.2	1.2	1.3	1.3	1.1	1.0	0.4	-	-
Zn Net Smelter Return	US\$M	2,595.9	-	-	-	85.3	102.4	135.3	165.5	189.2	194.6	200.5	124.2	112.5	83.3	89.1	86.4	89.7	88.6	113.5	113.3	117.4	124.4	123.8	112.3	101.1	43.2	-	-
Ag Revenues	US\$M	2,803.2	-	-	-	121.6	158.6	171.4	182.7	216.2	217.3	215.0	116.8	103.9	135.7	126.9	116.9	139.0	120.2	97.5	93.0	91.2	92.1	90.2	81.1	75.7	40.0	-	-
Pb Revenues	US\$M	1,871.6	-	-	-	85.6	121.5	115.0	101.7	110.3	109.3	97.3	77.5	80.5	114.5	93.3	96.3	109.2	96.6	73.1	68.3	65.3	62.9	60.0	52.8	53.5	27.2	-	-
Zn Revenues	US\$M	3,866.0	-	-	-	126.8	155.3	206.2	252.9	292.1	300.6	309.6	185.2	165.1	125.3	133.9	127.3	133.0	130.6	164.5	163.5	168.7	178.2	177.6	161.3	145.4	62.9	-	-
Cu Revenues	US\$M	147.6	-	-	-	6.7	8.6	9.4	11.1	13.0	13.1	13.3	6.5	4.8	5.3	6.0	4.9	6.2	6.0	5.2	4.8	4.6	4.3	4.3	3.8	3.6	2.0	-	-
Total Revenues	US\$M	8,688.3	-	-	-	340.7	443.9	502.1	548.5	631.6	640.3	635.2	386.0	354.2	380.7	360.1	345.4	387.4	353.4	340.3	329.6	329.8	337.6	332.1	299.0	278.2	132.2	-	-
Ag equivalent	koz	173,766	-	-	-	6,815	8,878	10,042	10,969	12,632	12,806	12,704	7,720	7,084	7,615	7,202	6,908	7,749	7,067	6,806	6,592	6,596	6,753	6,642	5,980	5,565	2,643	-	-
Total Offsite Charges	US\$M	1,926.3	-	-	-	71.9	96.1	111.9	123.6	142.6	145.3	144.2	87.9	80.3	82.7	77.9	74.7	82.0	76.0	75.9	73.4	73.3	74.9	73.8	66.5	62.3	29.2	-	-
Total Royalties	US\$M	148.8	-	-	-	5.9	7.7	8.6	9.3	10.8	10.9	10.8	6.6	6.0	6.6	6.2	6.0	6.7	6.1	5.8	5.6	5.6	5.8	5.7	5.1	4.8	2.3	-	-
Total NSR	US\$M	6,613.2	-	-	-	263.0	340.1	381.6	415.5	478.3	484.1	480.2	291.5	267.9	291.5	276.0	264.7	298.7	271.3	258.6	250.6	250.9	256.9	252.6	227.3	211.2	100.7	-	-
	C\$M	9,185.1	-	-	-	365.2	472.4	530.0	577.1	664.3	672.4	667.0	404.9	372.1	404.9	383.3	367.7	414.9	376.8	359.1	348.0	348.4	356.9	350.9	315.7	293.3	139.9	-	-
	C\$/t processed	494.14	-	-	-	494.13	539.28	605.03	658.74	758.30	767.55	761.38	462.19	424.73	462.21	437.55	419.71	473.65	430.16	409.95	397.29	397.75	407.38	400.52	360.41	334.86	425.47	-	-
Operating Cost Estimate																													
Mining	C\$M	1,654.1	-	-	-	62.04	76.05	76.60	77.00	77.10	75.49	77.87	81.11	80.07	75.97	76.87	78.58	77.20	79.63	78.72	79.97	77.69	79.32	80.13	79.83	76.94	29.91	-	-
	C\$/t mined	88.99	-	-	-	83.9	86.8	87.4	87.9	88.0	86.2	88.9	92.6	91.4	86.7	87.7	89.7	88.1	90.9	89.9	91.3	88.7	90.5	91.5	91.1	87.8	91.0	-	-
Processing	C\$M	731.0	-	-	-	30.90	36.62	34.25	34.25	34.25	34.25	34.25	34.25	34.25	34.25	34.25	34.25	34.25	34.25	34.25	34.25	34.25	34.25	34.25	34.25	34.25	12.85	-	-
	C\$/t mined	39.33	-	-	-	41.8	41.8	39.1	39.1	39.1	39.1	39.1	39.1	39.1	39.1	39.1	39.1	39.1	39.1	39.1	39.1	39.1	39.1	39.1	39.1	39.1	39.1	-	-
G&A	C\$M	715.9	-	-	-	34.00	34.00	33.23	33.23	33.23	33.23	33.23	33.23	33.23	33.23	33.23	33.23	33.23	33.23	33.23	33.23	33.23	33.23	33.23	33.23	33.23	16.61	-	-
	C\$/t mined	38.52	-	-	-	46.0	38.8	37.9	37.9	37.9	37.9	37.9	37.9	37.9	37.9	37.9	37.9	37.9	37.9	37.9	37.9	37.9	37.9	37.9	37.9	37.9	50.5	-	-
Total Operating Cost Estimate	C\$M	3,101.1	-	-	-	126.9	146.7	144.1	144.5	144.6	143.0	145.3	148.6	147.5	143.4	144.3	146.1	144.7	147.1	146.2	147.4	145.2	146.8	147.6	147.3	144.4	59.4	-	-
	C\$/t mined	166.83	-	-	-	171.74	167.44	164.47	164.93	165.04	163.20	165.91	169.61	168.43	163.75	164.77	166.73	165.15	167.92	166.89	168.32	165.72	167.57	168.50	168.16	164.85	180.61	-	-
Net Operating Income	C\$M	6,084.0	-	-	-	238.3	325.7	385.9	432.6	519.7	529.4	521.6	256.3	224.5	261.5	239.0	221.6	270.2	229.7	212.9	200.6	203.3	210.1	203.3	168.4	148.9	80.5	-	-
	C\$/tonne	327.31	-	-	-	322.39	371.85	440.56	493.81	593.26	604.35	595.46	292.58	256.30	298.46	272.78	252.98	308.50	262.24	243.06	228.97	232.03	239.81	232.02	192.26	170.01	244.86	-	-

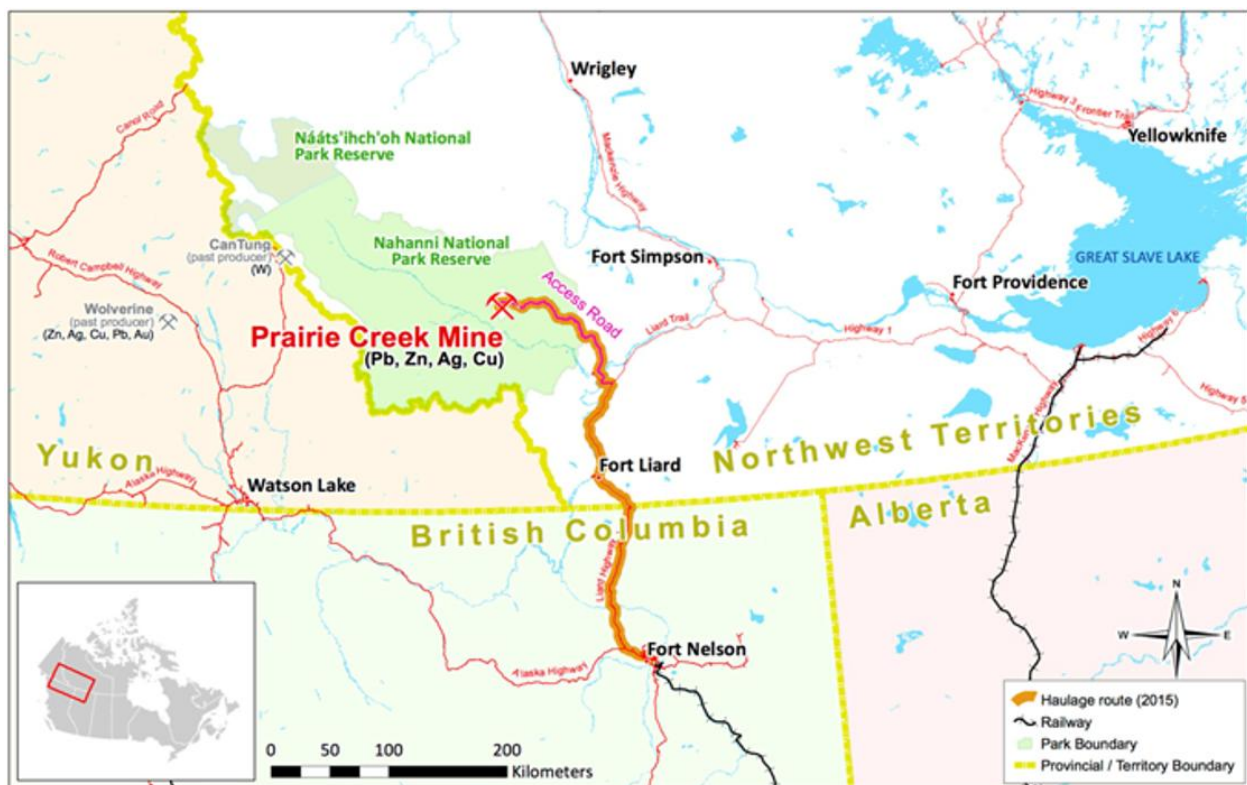


	Unit	LOM Total	Y-3	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14	Y15	Y16	Y17	Y18	Y19	Y20	Y21	Y22	Y23	Y24
		Tax Year-->	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039	2040	2041	2042	2043	2044	2045	2046	2047	2048	2049	2050	2051	2052	2053
Capital Cost Estimate																													
Mining	C\$M	394.9	0.0	0.0	64.6	36.3	28.8	13.4	10.5	17.6	26.2	20.8	8.5	11.1	22.0	17.5	17.5	14.7	12.3	17.8	10.8	11.8	6.4	3.2	12.7	8.6	2.0	0.0	0.0
Onsite Development	C\$M	5.7	0.0	1.4	4.3	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	0.0
Mineral Processing	C\$M	66.0	0.0	0.0	66.0	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	0.0	0.0
Tailings & Waste Management	C\$M	18.4	0.0	0.0	15.4	2.9	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
Onsite Infrastructure	C\$M	91.6	0.0	21.3	23.5	16.3	30.6	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
Offsite infrastructure	C\$M	205.2	44.0	84.0	77.3	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	0.0
Owners Costs	C\$M	17.5	0.0	0.0	17.5	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	0.0
Indirects & EPCM	C\$M	167.6	17.1	41.4	97.6	4.4	7.0	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
Closure	C\$M	49.0	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	0.0	0.0	24.5	24.5	
Subtotal	C\$M	1,015.9	61.1	148.1	366.2	60.0	66.4	13.4	10.5	17.6	26.2	20.8	8.5	11.1	22.0	17.5	17.5	14.7	12.3	17.8	10.8	11.8	6.4	3.2	12.7	8.6	2.0	24.5	24.5
Contingency	C\$M	134.3	10.0	24.4	57.4	6.3	8.3	0.7	0.5	0.9	1.3	1.0	0.4	0.6	1.1	0.9	0.9	0.7	0.6	0.9	0.5	0.6	0.3	0.2	0.6	0.4	0.1	7.3	7.3
Total Capital Cost Estimate	C\$M	1,150.2	71.1	172.4	423.6	66.3	74.7	14.1	11.0	18.5	27.5	21.8	8.9	11.6	23.1	18.3	18.3	15.4	12.9	18.7	11.4	12.4	6.7	3.4	13.3	9.1	2.1	31.8	31.8
Pre-Production	C\$M	667.1	71.1	172.4	423.6																								
Sustaining	C\$M	483.0				66.3	74.7	14.1	11.0	18.5	27.5	21.8	8.9	11.6	23.1	18.3	18.3	15.4	12.9	18.7	11.4	12.4	6.7	3.4	13.3	9.1	2.1	31.8	31.8
Working Capital	C\$M	0.0	0.0	0.0	31.7	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	-31.7	0.0	0.0
Net Pre-Tax Cash Flow	C\$M	4,933.8	-71.1	-172.4	-455.3	172.0	251.0	371.9	421.6	501.2	501.9	499.8	247.4	212.9	238.4	220.6	203.3	254.8	216.8	194.2	189.2	190.8	203.4	199.9	155.1	139.9	110.1	-31.8	-31.8
Cumulative Net Pre-Tax Cash Flow	C\$M	4,933.8	-71.1	-243.5	-698.9	-526.9	-275.8	96.0	517.6	1,018.8	1,520.8	2,020.6	2,267.9	2,480.8	2,719.2	2,939.8	3,143.1	3,397.9	3,614.8	3,809.0	3,998.2	4,189.0	4,392.4	4,592.3	4,747.5	4,887.3	4,997.5	4,965.6	4,933.8
Taxes	C\$M	1,761.5	0.0	0.0	0.0	20.0	28.8	63.9	132.2	165.7	169.3	168.5	81.5	72.7	85.1	77.8	71.8	90.2	76.3	69.5	66.3	67.4	71.2	69.8	47.3	40.7	25.5	0.0	0.0
Net After-Tax Cash Flow	C\$M	3,172.3	-71.1	-172.4	-455.3	152.0	222.2	307.9	289.4	335.5	332.6	331.3	165.9	140.2	153.3	142.8	131.5	164.6	140.5	124.7	123.0	123.4	132.2	130.1	107.8	99.1	84.7	-31.8	-31.8
Cumulative Net After-Tax Cash Flow	C\$M	3,172.3	-71.1	-243.5	-698.9	-546.9	-324.7	-16.7	272.7	608.2	940.8	1,272.1	1,438.0	1,578.3	1,731.5	1,874.3	2,005.8	2,170.5	2,310.9	2,435.6	2,558.6	2,682.0	2,814.2	2,944.3	3,052.1	3,151.3	3,235.9	3,204.1	3,172.3

23 ADJACENT PROPERTIES

There are no mineral properties immediately adjacent to the PC Silver Mine since the site is somewhat uniquely located in relation to the Nahanni National Park Reserve. The Nahanni National Park Reserve was expanded in 2009 and, as part of the expansion agreement, the PC Silver Mine itself, and a large surrounding area of approximately 300 km², was specifically excluded from the Park. This excluded area remains under the jurisdiction of the GNWT and is surrounded by the Nahanni National Park Reserve as shown in Figure 23-1. In addition, road access into the PC Silver Mine area through the expanded Park area was also provided through an amendment to the *Canada National Parks Act*, solely for the Nahanni National Park Reserve.

Figure 23-1: PC Silver Mine Area



Source: NorZinc (2021)

The closest mineral property of significance is the Cantung Mining Property, located 185 km to the west of PC Silver Mine. The Cantung Mine was formerly owned by North American Tungsten Corporation Ltd. and operated intermittently since 1968 until operations ceased in 2015. Mining consisted of both open pit and underground operations extracting tungsten ore from a scheelite-chalcopyrite bearing skarn. The Cantung Mine, located in the Northwest Territories, is distant from and had little influence on the PC Silver Mine, since the Cantung Mine was accessed by a road from Watson Lake in the Yukon Territory. On November 19, 2015 the GNWT announced that it had acquired the leasehold interests to the mineral rights of North American Tungsten Corporation's undeveloped Mactung Property, and the Government of Canada had assumed responsibilities for the Cantung Mine. The Wolverine Mine in the Yukon Territory, 100 km west of Cantung Mine, was the closest base metal underground mine to PC Silver Mine but also closed in 2015.

24 OTHER RELEVANT DATA AND INFORMATION

There is no other relevant data or information available that is necessary to make the Technical Report understandable and not misleading.

25 INTERPRETATIONS AND CONCLUSIONS

It is the conclusion of the QPs that the PEA summarized in this Technical Report contains adequate preliminary detail and information to support a PEA-level report. Standard industry practices, equipment and design methods were used in the PEA.

Based on the assumptions used and the information gathered for this preliminary evaluation, the Project shows positive economics.

To date, the QPs are not aware of any fatal flaws for the PC Silver Mine.

A total of 323 holes for 86,383 m has been drilled on the PC Silver Mine by previous project owners. Honey Badger Silver has completed no drilling on the property. The PEA is based on the Company's mineral resource estimate with an effective date of June 21, 2026.

The PEA contemplates underground mining, with MQV and STK zones mined using longitudinal retreat stoping, primarily with downhole drilling, while the SMS zone will use a combination of longitudinal transverse longhole stoping and mostly upwards drilling. Underground production will ramp up from an initial level of 1,600 t/d to a steady state rate of 2,400 t/d, or approximately 876,000 t/y. The mine life is estimated at approximately 22 years, and the mine will provide approximately 18.6 Mt at average grades of 112.5 g/t silver, 8.5% zinc, 6.1% lead and 0.23% copper.

The current process design incorporates some existing equipment, which was moved from another mine and installed at PC Silver Mine in 1981/1982. With the increase in throughput to 2,400 t/d, the crushing plant is the only area to be retained without modification. The dense media separation pre-concentration plant is new and will be fed using a new conveyor from the existing fine mill feed bin. A new ball mill will be added into the grinding circuit with the existing mill being refurbished. The flotation cells and dewatering circuits will be refurbished. Reagent preparation system will be completed to modern standards for lead and zinc concentrate production. Estimated processing recoveries are 93.8% for silver, 87% for lead, 86.9% for zinc, and 95% for copper.

PC Silver Mine is an established remote site that will be upgraded to support continuous underground mining and processing at 2,400 t/d, with mining support facilities at the 883 Portal.

A 170 km ASR will be built over three years to connect the site to the Northwest Territories Highway 7 corridor. It will include a seasonal barge and ice bridge crossing of the Liard River, and a winter road will provide access until construction is finished. The existing 1,000 m gravel airstrip will be used for personnel transport, priority freight, and medical evacuation.

Power will be generated onsite by eight 1,500 kW diesel generators. They will be rented for the first two years and then replaced with dual fuel units purchased. Diesel will be stored in the existing onsite tanks with a combined capacity of 6.8 ML.

The site will include freshwater supply, potable and wastewater treatment, and firewater systems. Surface runoff and contact water will be collected, diverted, and treated before discharge.

Tailings will be stored in a dry stack facility, with the vast majority used as paste backfill underground. Waste rock will be hauled from the underground mine to a surface waste rock pile.

A new 250-room camp will replace the deteriorated existing camp. It will be supported by new modular administration, mine dry, and laboratory buildings, a new warehouse and laydown area, and the existing maintenance shop will be rehabilitated.

The Project is within the claimed traditional territories of the Nahᓃᓴ Dehé Dene Band and the Łı́ı́dlı́ Kúę First Nation, both historically part of the Dehcho First Nations. The Dehcho First Nations have been in land claim negotiations since late 1990s. The nearest community is Nahanni Butte, home of the Nahᓃᓴ Dehé Dene Band, located approximately 90 km to the southeast of the Project site.

Permitting of the PC Silver Mine and All-Season Access Road has been completed by the previous owners.

Honey Badger Silver, through its 100% owned subsidiary Canadian Zinc Corp. currently holds the land use permits and water licences necessary to advance the Project. The ASR crosses through the jurisdictions of both the Mckenzie Valley Land and Water Board and Parks Canada, and thus both agencies have issued land use permits and water licences to the Company.

25.1 Risks

The most significant Project risks are summarized in Table 25-1.

Table 25-1: Main Project Risks

Risk	Explanation/Potential Impact	Possible Risk Mitigation
Mining Dilution	Higher than expected dilution has a severe impact on project economics. The mine must ensure accurate drilling and blasting practices are maintained to minimize dilution from wall rock backfill and other mineralized zones, minimize secondary breaking and optimize extraction. The ability to segregate higher grade material, early in the mine life, is critical to project economics.	Well planned and executed ground control and grade control plans are necessary immediately upon commencement of mining.
Stope Stability	Ground conditions worse than anticipated based on existing geotechnical data could adversely impact stability of production stopes increasing dilution and potentially decreasing recovery. The development sequence could be delayed and require additional support.	Geotechnical drilling and testing are required to better define rock mass quality and delineation of geotechnical domains. A geologic structural model is needed to define locations and continuity of the main fault structures. Findings will be confirmed through geotechnical mapping of development headings.
Resource Modelling	All mineral resource estimates carry some risk and are one of the most common issues with project success. 42.4% of the resources in the mine plan are Inferred.	Infill drilling may be recommended in order to provide a greater level of confidence in the resource.
Condition of existing process plant equipment	The major process plant components (mill, crushers, filters) may need more extensive and possibly offsite refurbishment.	Preliminary investigations have been completed, but more thorough equipment inspections and testing are required to confirm condition and extent of refurbishment required.
Metallurgical Recoveries	Negative changes to metallurgical assumptions could lead to reduced metal recovery, increased processing costs, and/or changes to the processing circuit design. If metal recovery is lower than assumed, the Project economics would be negatively impacted.	Additional sampling and testwork is needed at the next level of study.
Capital and Operating Cost Estimates	<p>The ability to achieve the estimated Capital and Operating Cost Estimates are important elements of project success.</p> <p>If operating costs increase the size of the mineable resource will be reduced.</p>	Further cost estimation accuracy with the next level of study, as well as the active investigation of potential cost-reduction measures would assist in the support of reasonable cost estimates.

Risk	Explanation/Potential Impact	Possible Risk Mitigation
Waste Rock Storage Facility Design	Risks associated with the waste rock storage facility design relate to the uncertainties in the geotechnical and hydrogeological foundation conditions of the proposed infrastructure. Modifications to the current design of the structures could result in changes to the capital cost estimate.	Site investigation programs will be required as part of future studies to characterize the geotechnical and hydrogeological foundation conditions of the proposed infrastructure.
Water Management and Water Balance	The water management strategy was developed based on the current understanding of the Project climate, make-up water requirements, and estimated surplus water. Refinements to these inputs at future study phases may present changes to the water management strategy and project water balance, resulting in changes to the capital and operating cost estimates.	The collection of site-specific meteorological and hydrology data will be required as part of future studies to refine seasonal run-off values, design storms, and water balance calculations.
Climate Change	Climate change in the area may impact the regional precipitation, making low precipitation and high precipitation more frequent. As approved discharge to Prairie Creek of treated water is dependent on flow, storage capacity through the winter months during low flows may need to be higher than originally calculated.	Reviewing and updating water balance, discharge volumes, storage capacity and amendments to the Water Licence should be reviewed to ensure storage and discharge can support continuous milling.
Schedule Delays	Seasonal restrictions for access are variable and could affect Project schedule and alter project economics.	Experienced project management, logistics planning and detailed operating plans.
Generic Risks	Almost all mining projects have largely uncontrollable risks from changing commodity prices, government approval delays, regulatory changes, exchange rates, inflation, supply shortages, etc.	These risks are difficult to mitigate, and the company must be diligent in monitoring changes in operating conditions.
Ability to Attract Experienced Professionals	<p>The ability to attract and retain competent, experienced professionals is a key success factor for the Project.</p> <p>High turnover or the lack of appropriate technical and management staff on the Project could result in difficulties meeting project goals.</p>	The early search for professionals as well as competitive salaries and benefits identified will attract and retain critical people.

Risk	Explanation/Potential Impact	Possible Risk Mitigation
Concentrate Smelting	The produced concentrates contain deleterious elements that will likely at times exceed current allowable smelter thresholds. Blending will likely be required and finding smelters to take all the available material may be challenging. Specific treatment terms and transportation costs to customers may be highly variable.	Ongoing discussions with smelters and brokers to understand options and likely customer locations.

25.2 Opportunities

The main opportunities identified for the Project are listed in Table 25-2.

Table 25-2: Identified Project Opportunities

Opportunity	Explanation	Potential Benefit
Taxation	The Productivity Mega Deduction announced on September 15, 2026 has not been incorporated in the PEA economics.	This will provide immediate expensing for a broad-based range of depreciable property on a permanent basis. Immediate expensing allows the Company to fully write off the cost of an investment in the year that it becomes available for use.
Antimony, Germanium and other Critical Minerals Recovery	There is evidence of antimony, gallium and germanium at the PC Silver Mine, though no formal resource has been defined. Future drilling and metallurgical testwork may lead to including these revenues in future studies.	Increased revenues.
Expansion of the Mine	The mineral resource has not been fully delineated along the 16 km of mineralization on the property and there is an opportunity to expand the mineable resource.	Increased mine life.
Increased Production	Increased production may be possible at high tonne per vertical meter levels. There is an opportunity for the mine to produce more tonnes for short durations on the high tonnage levels of the mine.	Reduced unit operating costs and increased revenue.
Backfill Cement Content	Paste backfill testing may reduce the cement content assumption.	Reduce mining costs.
Concentrate Smelting	Concentrates are currently assumed to be shipped overseas. There may be potential to source North American smelter capacity to reduce concentrate transport costs. It may be possible to obtain better treatment and/or refining terms from smelters through formal negotiations in the future	Reduced transportation and concentrate shipping costs. Reduced concentrate treatment and refining costs

26 RECOMMENDATIONS

It is recommended that the PC Silver Mine proceed to the Feasibility Study stage in line with Honey Badger Silver Inc.'s desire to advance the Project. It is also recommended that environmental and permitting continue as needed to support project development plans. The Feasibility Study work plan will consist of:

26.1 Drilling & Exploration

Approximately 2,000 – 3,000 m of diamond drilling will be done in Q4 2026 from the 930-level drift. This drilling has three main goals:

- 1) Find new STK and MQV mineralization in an area that has yet to be tested to the north of and above the current STK geological model that will add new ounces to the deposit;
- 2) Simultaneously convert Inferred resource blocks to M&I with drilling that is proximal to the existing resource model; and
- 3) Test this area for additional critical mineral potential including tungsten, germanium, gallium, and indium, etc.

26.2 Mine Geotechnical

A geotechnical drilling and testing program is suggested to better define rock mass quality, strength and delineation of domains. The program should consist of 8 to 12 drillholes crossing through the mineralized areas and extend approximately 20 to 30 m into the footwall. It is recommended that at a minimum, 50% of the geotechnical holes are also surveyed with a downhole acoustic televiewer. Laboratory strength testing including uniaxial compressive, triaxial compressive and Brazilian indirect tensile strengths should be completed on 10 to 20 samples of each of the main lithology and vein types.

A geologic structural model is also recommended to define locations and continuity of the main fault structures within and around the stoping areas and development. The existing drillhole database, core photographs and supplemental data acquired from the geotechnical specific drilling program will provide sufficient data to develop the model.

26.3 Metallurgical Testing

A metallurgical program focused on material variability will be required to confirm the process design. The program will consist of 40 – 70 interval composites which will be divided up to form a 1-3 year master composite, approximately 15 variability composites and each interval will form a geometallurgical composite. The testwork conducted will confirm the flowsheet design and provide recovery and operating cost data that can be incorporated into the block model.

26.4 Waste & Water Management

A geotechnical site investigation program will be required to confirm the assumed foundation conditions and material properties at the key infrastructure locations. This will be particularly important at locations such as the water storage pond, the north backslope excavation, and the waste rock pile, where there may be the possibility of slope instability.

26.5 Process Plant Assessment

Tusk Automation has been contracted to evaluate opportunities to accelerate and minimize the cost of re-starting the extensive processing facilities already in place at the PC Silver Mine. They are spending two weeks at the site to fully assess the existing installations. A written report to AACESCE Class 3 (Feasibility Study) level will be issued and this work is expected to be incorporated into the upcoming Feasibility Study.

26.6 Environment & Permitting

Environment and permitting work includes the Preparation of Design and Construction Plans and the accompanying Management Plans, Water licence amendments; and Impact Benefit Agreement engagement.

26.7 Feasibility Study Project Budget

It is estimated that a Feasibility Study and supporting field work would cost approximately \$5.0 million. A breakdown of the key components of the next study phase is as follows in Table 26-1.

Table 26-1: Estimated Cost of Proposed Programs to Support a Feasibility Study

Program Description	Estimated Total Cost (\$M)
Drilling	0.3
Mine Geotechnical (excludes drilling)	0.4
Metallurgical Testing	0.5
Waste & Water Management	0.5
Process Plant Assessment	0.5
Environment & Permitting	0.4
Feasibility Study Engineering and Technical Report	2.4
Total	5.0

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28 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)
AANDC	Aboriginal Affairs and Northern Development Canada
ABA	acid-base accounting
ac	acre
Acfm	actual cubic feet per minute
Ag	silver
Ai	Abrasion
AISC	all-in sustaining cost
ADK	Acho Dene Koe First Nation
AN	ammonium nitrate
ARD	acid rock drainage
As	arsenic
ASR	all-season road
Au	gold
AWR	all-weather road
B	billion
BD	bulk density
Bt	billion tonnes
BBMWi	bond ball mill work index
BTU	British thermal unit
bya	billion years ago
C\$	dollar (Canadian)
Cd	cadmium
CDE	Canadian development expense
cfm	cubic feet per minute
CIM	Canadian institute of Mining and Metallurgy
CIRNAC	Minister of Crown-Indigenous Relations and Northern Affairs Canada

Symbol / Abbreviation	Description
cm	centimetre
cm ²	square centimetre
cm ³	cubic centimetre
CRM	certified reference materials
CSMRI	Colorado School of Mines Research Institute
Cu	copper
CZC	Canadian Zinc Corporation
d	day
d/a	days per year (annum)
d/wk	days per week
DAR	developer's assessment report
dB	decibel
dBa	decibel adjusted
DFO	Déline First Nation
dmt	dry metric tonne
EA	environmental assessment
ELOS	equivalent linear overbreak/slough
EIR	Environmental Impact Review
EIS	environmental impact statement
EQC	effluent quality criteria
ERT	emergency response team
ESR	excavation support ratio
Fe	iron
FMEA	failure modes and effects analysis
FPB	flood protection berm
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

Symbol / Abbreviation	Description
GNWT	Government of the Northwest Territories
GPa	gigapascal
gpm	gallons per minute (US)
GSC	geological survey of Canada
GW	gigawatt
h	hour
h/a	hours per year
h/d	hours per day
h/wk	hours per week
ha	hectare
Hg	mercury
hp	horsepower
HQ	drill core diameter of 63.5 mm
Hz	hertz
IBA	Impact Benefit Agreement
ICP	inductively coupled plasma
ICP-MS	inductively coupled plasma mass spectrometry
in	inch
in ²	square inch
in ³	cubic inch
IRR	internal rate of return
ITRP	independent technical review panel
JDS	JDS Energy & Mining Inc.
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
KM	Kamloops Metallurgy
km	kilometre
km/h	kilometres per hour
km ²	square kilometre
kPa	kilopascal
kt	kilotonne
kV	kilovolt
kVA	kilovolt-ampere

Symbol / Abbreviation	Description
kW	kilowatt
kWh	kilowatt hour
kWh/a	kilowatt hours per year
kWh/t	kilowatt hours per tonne
L	litre
L/min	litres per minute
L/s	litres per second
LDD	large-diameter drill
LHS	long-hole stoping
LNG	liquified natural gas
LOM	life of mine
LUP	land use permits
m	metre
M	million
m/min	metres per minute
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	metal leaching

Symbol / Abbreviation	Description
mL	millilitre
mm	millimetre
Mm ³	million cubic metres
MMER	metal mining effluent regulations
mo	month
MOU	memorandum of understanding
MPa	megapascal
MSDS	material safety data sheet
Mt	million metric tonnes
MVA	megavolt-ampere
MVLWB	Mackenzie Valley Land and Water Board
MVRB	Mackenzie Valley Review Board
MVRMA	<i>Mackenzie Valley Resource Management Act</i>
MVT	Mississippi Valley-type
MQV	main quartz vein / hydrothermal quartz veins
MQV-Cu	higher copper content
MQV-Fe	higher iron content
MQV-H	upper MQV zone
MQV-L	lower MQV zone
MQV-Pbox	variable lead oxide content
MW	megawatt
NAD	North American datum
Na ₂ S	sodium sulphide
Ni	nickel
NI 43-101	national instrument 43-101
NQ	drill core diameter of 47.6 mm
NSR	net smelter return
NT	Northwest Territories
NTPC	Northwest Territories Power Corporation
OK	ordinary kriging
OP	open pit
OSA	overall slope angles
oz	troy ounce
P.Geo.	professional geoscientist
Pa	pascal
PAG	potentially acid generating
PDR	Project Description Report

Symbol / Abbreviation	Description
PEA	preliminary economic assessment
Pb	lead
PC	Prairie Creek
PCA	Prairie Creek Anticline
ppb	parts per billion
ppm	parts per million
PSA	particle size analyses
psi	pounds per square inch
QA/QC	quality assurance/quality control
QEMSCAN	quantitative evaluation of minerals by scanning electron microscopy
QP	qualified person
RC	reverse circulation
RCF	resource capital funds
REA	Report of Environmental Assessment
RMR	rock mass rating
ROM	run-of-mine
RPEEE	reasonable prospects of eventual economic extraction
rpm	revolutions per minute
RQD	rock quality designation
s	second (time)
Sb	antimony
S.G.	specific gravity
SDr	Road River
Scfm	standard cubic feet per minute
SG	specific gravity
SMS	stratabound massive sulphide
STK	stockwork-style
t	tonne (1,000 kg) (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
TLUA	Traditional Land Use Agreement
TOC	total organic carbon
tph	tonnes per hour
US	united states

Symbol / Abbreviation	Description
US\$	dollar (American)
UTM	universal transverse mercator
V	volt
VMS	volcanic massive sulphide
WHMIS	Workplace Hazardous Materials Information Systems
wk	week
wmt	wet metric ton
µm	microns
Zn	zinc
ZnSO ₄	zinc sulphate

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

29 CERTIFICATES

CERTIFICATE OF QUALIFIED PERSON

Michael Makarenko, P.Eng.

I, Michael Makarenko, P.Eng., do hereby certify that:

1. This certificate applies to the Technical Report entitled "Preliminary Economic Assessment for the PC Silver Mine" (the Technical Report) with an effective date of June 21, 2026 prepared for Honey Badger Silver Inc.;
2. I am currently employed as Vice President, Technical Services with JDS Energy & Mining Inc. with an office at Suite 900, 999 West Hastings St., Vancouver, BC V6C 2W2;
3. I am a graduate of the University of Alberta with a B.Sc. in Mining Engineering, 1988. I have practiced my profession continuously since 1988;

I have spent the last 38 years working on mining projects. I have worked in technical, operations and management positions at mines in Canada, the United States, Brazil and Australia. I have been an independent consultant for over twenty-one years and have performed mine design, mine planning, cost estimation, operations & construction management, technical due diligence reviews and technical report writing for mining projects worldwide;

I am a Registered Professional Engineer in Alberta (#48091), British Columbia (#49223) and the Northwest Territories/Nunavut (#1359);

I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101. I am independent of the issuer, vendor, property and related companies applying all of the tests in Section 1.5 of NI 43-101;

4. I have visited the PC Silver Mine on June 16, 2026;
5. I am responsible for Sections 1.1-1.3, 1.8, 1.9, 1.11, 1.15, 1.16, 2, 3, 12.2, 12.3, 15, 16 (except 16.3 and 16.8), 18 (except 18.6, 18.7), 23-28 of this Technical Report;
6. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;
7. I have had no prior involvement with the property that is the subject of this Technical Report;
8. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading; and
9. I have read NI 43-101, and the Technical Report has been prepared in accordance with NI 43-101 and Form 43-101F1.

Effective Date: June 21, 2026

Signed Date: October 15, 2026

(Original signed and sealed) "Michael Makarenko, P.Eng."

Michael Makarenko, P.Eng.

CERTIFICATE OF QUALIFIED PERSON

Shane Tad Crowie, P.Eng.

I, Shane Tad Crowie, P.Eng., do hereby certify that:

1. This certificate applies to the Technical Report entitled "Preliminary Economic Assessment for the PC Silver Mine" (the Technical Report) with an effective date of June 21, 2026 prepared for Honey Badger Silver Inc.;
2. I am currently employed as a Senior Metallurgist with JDS Energy & Mining Inc. with an office at Suite 900, 999 West Hastings St., Vancouver, BC V6C 2W2;
3. I am a graduate of the University of British Columbia in 2001, with a B.A.Sc. in Mining and Mineral Process Engineering. I have practiced my profession continuously since 2001. I have been involved with various mining projects and studies; where I have performed, technical, operations and management positions at mines in Canada. I have been responsible for recovery optimization projects, capital improvement projects, budgeting, planning and pilot plant operations. I also have been responsible for writing technical reports, managing metallurgical testwork, and performing due diligence audits on mines and development properties.

I am a Professional Engineer (P. Eng) registered with the Association of Professional Engineers and Geoscientists of B.C. (No. 34052), and I am a 'Qualified Person' in relation to the subject matter of this Technical Report;
4. I have visited the PC Silver Mine on June 16, 2026;
5. I am responsible for Sections 1.6, 1.10, 12.4, 13, 17, and parts of the Executive Summary and Conclusions and Recommendations pertaining to Mineral Processing and Recovery Methods of this Technical Report;
6. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;
7. I have had prior involvement with the property that is the subject of this Technical Report. I have been responsible for managing metallurgical testwork;
8. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading; and
9. I have read NI 43-101, and the Technical Report has been prepared in accordance with NI 43-101 and Form 43-101F1.

Effective Date: June 21, 2026

Signed Date: October 15, 2026

(Original signed and sealed) "Shane Tad Crowie, P.Eng."

Shane Tad Crowie, P.Eng.



CERTIFICATE OF QUALIFIED PERSON

MICHAEL LEVY, P.Eng.

I, Michael Levy, P.Eng., do hereby certify that:

1. This certificate applies to the Technical Report entitled "Preliminary Economic Assessment for the PC Silver Mine" (the Technical Report) with an effective date of June 21, 2026, prepared for Honey Badger Silver Inc.;
2. I am currently employed as Geotechnical Manager with JDS Energy & Mining Inc. with an office at Suite 900 – 999 West Hastings Street, Vancouver, British Columbia, V6C 2W2;
3. I am a graduate of the University of Iowa with a B.Sc. degree in Geology in 1998 and the University of Colorado in 2004 with a M.Sc. degree in Geotechnical Engineering. I have practiced my profession continuously for 27 years (since 1999) and have been involved in numerous mining geotechnical projects across the Americas. I am a registered Professional Engineer and member in good standing in the state of Colorado (P.E. #40268) and in British Columbia (P.Eng. #216542).

I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101;

4. I have not visited the PC Silver Mine;
5. I am responsible for Sections 12.5 and 16.3 of this Technical Report;
6. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;
7. I have had no prior involvement with the property that is the subject of this Technical Report;
8. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading; and
9. I have read NI 43-101, and the Technical Report has been prepared in accordance with NI 43-101 and Form 43-101F1.

Effective Date: June 21, 2026

Signed Date: October 15, 2026

(Original signed and sealed) "Michael Levy, P.Eng."

Michael Levy, P.Eng.

CERTIFICATE OF QUALIFIED PERSON

Tysen Hantelmann, P.Eng.

I, Tysen Hantelmann, P.Eng., do hereby certify that:

1. This certificate applies to the Technical Report entitled "Preliminary Economic Assessment for the PC Silver Mine" (the Technical Report) with an effective date of June 21, 2026 prepared for Honey Badger Silver Inc.;
2. I am currently employed as an Engineering Manager with JDS Energy & Mining Inc. with an office at Suite 900, 999 West Hastings St., Vancouver, BC V6C 2W2;
3. I am a graduate of the University of Alberta with both a B.Sc. in Mining Engineering, 2001 and a M.Eng. in Mining Engineering, 2003. I have practiced my profession continuously since 2001. I have worked in technical and operational positions at several mines in Canada. I have been an independent consultant for over sixteen years and have performed all aspects of mine planning design and costing on over a hundred projects and studies worldwide. I am a Registered Professional Engineer and member in good standing in Alberta (#71697), Yukon (#2631), and Northwest Territories (L2810);

I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101. I am independent of the issuer, vendor, property and related companies applying all of the tests in Section 1.5 of NI 43-101;

4. I have not visited the PC Silver Mine;
5. I am responsible for Sections 1.13, 1.14, 19, 21, 22 of this Technical Report and parts of the Executive Summary and Conclusions and Recommendations pertaining to Market Studies, Costing and Economic Analysis of this Technical Report;
6. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;
7. I have had no prior involvement with the property that is the subject of this Technical Report;
8. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading; and
9. I have read NI 43-101, and the Technical Report has been prepared in accordance with NI 43-101 and Form 43-101F1.

Effective Date: June 21, 2026

Signed Date: October 15, 2026

(Original signed and sealed) "Tysen Hantelmann, P.Eng."

Tysen Hantelmann, P.Eng.

CERTIFICATE OF QUALIFIED PERSON

Gregory Zale Mosher, P.Geo.

I, Gregory Zale Mosher, P.Geo., do hereby certify that:

1. This certificate applies to the Technical Report entitled "Preliminary Economic Assessment for the PC Silver Mine" (the Technical Report) with an effective date of June 21, 2026 prepared for Honey Badger Silver Inc.;
2. I am currently employed as Principal Geologist with Global Mineral Resource Services, with an office at 304-3373 Capilano Crescent, North Vancouver, BC;
3. I am a graduate of Dalhousie University (B.Sc. Hons., 1970) and McGill University (M.Sc. Applied, 1973). I am a member in good stand of the Association of Professional Engineers and Geoscientists of British Columbia, License #19267;

My relevant experience with respect to base metal deposits includes over 30 years of exploration for and evaluation of such deposits and over 20 years of mineral resource estimation. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101. I am independent of the issuer, vendor, property and related companies applying all of the tests in Section 1.5 of NI 43-101;

4. I have most recently visited the PC Silver Mine on June 16, 2026;
5. I am responsible for Sections 1.4-1.5, 1.7, 4-11, 12.1, and 14 of this Technical Report;
6. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;
7. I have previously prepared several mineral resource estimates and related technical reports on the property that is the subject of this Technical Report;
8. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading; and
9. I have read NI 43-101, and the Technical Report has been prepared in accordance with NI 43-101 and Form 43-101F1.

Effective Date: June 21, 2026

Signed Date: October 15, 2026

(Original signed and sealed) "Gregory Zale Mosher, P.Geo."

Gregory Z. Mosher, P.Geo.

CERTIFICATE OF QUALIFIED PERSON

Les J. Galbraith, P.Eng.

I, Les J. Galbraith, P.Eng., do hereby certify that:

1. This certificate applies to the Technical Report entitled "Preliminary Economic Assessment for the PC Silver Mine" (the Technical Report) with an effective date of June 21, 2026 prepared for Honey Badger Silver Inc.;
2. I am currently employed as a Specialist Engineer | Associate with Knight Piésold Ltd. with an office at Suite 1400 - 750 West Pender Street, Vancouver, British Columbia, Canada V6C 2T8.
3. I am a graduate of the University of British Columbia in Vancouver, Canada (B.A.Sc (Civil Engineering) in 1995). I am a member in good standing of Engineers Yukon (Membership # 2368), and Engineers and Geoscientists British Columbia (Membership #25493). I have been involved with or associated with the mining industry since 1996. My experience includes tailings and water dam design, geotechnical investigations, construction supervision, stability modelling, and foundation assessment.

I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101. I am independent of the issuer, vendor, property and related companies applying all of the tests in Section 1.5 of NI 43-101;

4. I visited the PC Silver Mine on June 16, 2026;
5. I am responsible for Sections 12.6, 18.6, and 18.7 of this Technical Report;
6. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;
7. I have had no prior involvement with the property that is the subject of this Technical Report;
8. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading; and
9. I have read NI 43-101, and the Technical Report has been prepared in accordance with NI 43-101 and Form 43-101F1.

Effective Date: June 21, 2026

Signed Date: October 15, 2026

(Original signed and sealed) "Les Galbraith, P.Eng."

Les J. Galbraith, P.Eng.

CERTIFICATE OF QUALIFIED PERSON

Richard A. Cook, P.Ge.

I, Richard A. Cook, P.Ge., do hereby certify that:

1. This certificate applies to the Technical Report entitled "Preliminary Economic Assessment for the PC Silver Mine" (the Technical Report) with an effective date of June 21, 2026 prepared for Honey Badger Silver Inc.;
2. I am currently employed as a Specialist Environmental Scientist | Associate with Knight Piésold Ltd. with an office at 200 - 1164 Devonshire Avenue, North Bay, ON Canada P1B 6X7;
3. I graduated from Queen's University with a Bachelor of Science (Honours) in 1996 and completed additional geoscience coursework at the University of Waterloo and Athabasca University. I have practiced my profession continuously since 1996;

I have spent the last 26 years working on environmental management in the mining industry. This includes environmental assessments, water management planning, and mine closure planning. As a qualified person, I have certified Item 20 of numerous technical reports ranging from preliminary economic assessments to pre-feasibility studies and feasibility studies;

I am a Professional Geoscientist registered with Professional Geoscientists Ontario (No. 2199; limited designation in hydrogeology and contaminated sites);

I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101. I am independent of the issuer, vendor, property and related companies applying all of the tests in Section 1.5 of NI 43-101;

4. I have not visited the PC Silver Mine.
5. I am responsible for Sections 1.12 and 20 of this Technical Report;
6. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;
7. I have had no prior involvement with the property that is the subject of this Technical Report;
8. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading; and
9. I have read NI 43-101, and the Technical Report has been prepared in accordance with NI 43-101 and Form 43-101F1.

Effective Date: June 21, 2026

Signed Date: October 15, 2026

(Original signed and sealed) "Richard Cook, P.Ge."

Richard A Cook, P.Ge.

CERTIFICATE OF QUALIFIED PERSON

Yen Jui (Bernie) Ting, P.Eng.

I, Yen Jui Bernie Ting, P.Eng., do hereby certify that:

1. This certificate applies to the Technical Report entitled "Preliminary Economic Assessment for the PC Silver Mine" (the Technical Report) with an effective date of June 21, 2026 prepared for Honey Badger Silver Inc.;
2. I am currently employed as a Lead Engineer with T Engineering. with an office at 201-75 Front Street Toronto;
3. I am a graduate of the University of Toronto in 2012, with a Master of Applied Science in Electrical and Computer Engineering. I have practiced my profession continuously since 2012;

I have spent the last 15 years working on mining projects; where I have worked with engineers from different disciplines to design backfill systems. I have also worked in labs where backfill mix designs are studied. I also supported and operated a backfill plant.;

I am a Registered Professional Engineer in Ontario, Northwest Territories, Newfoundland and Labrador;

I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101. I am independent of the issuer, vendor, property and related companies applying all of the tests in Section 1.5 of NI 43-101;

4. I have not visited the PC Silver Mine;
5. I am responsible for Sections 16.8 of this Technical Report;
6. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;
7. I have had no prior involvement with the property that is the subject of this Technical Report;
8. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading; and
9. I have read NI 43-101, and the Technical Report has been prepared in accordance with NI 43-101 and Form 43-101F1.

Effective Date: June 21, 2026

Signed Date: October 15, 2026

(Original signed and sealed) "Yen Jui (Bernie) Ting, P.Eng."

Yen Jui (Bernie) Ting, P.Eng.